



OreWin
Independent Mining Consultants

Turquoise Hill **Resources Ltd.**

Oyu Tolgoi

2016 Oyu Tolgoi Technical Report

October 2016

Job No. 15016



IMPORTANT NOTICE

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Title Page

Project Name:	Oyu Tolgoi
Title:	2016 Oyu Tolgoi Technical Report
Location:	Ömnögovi Aimag, Mongolia
Effective Dates:	
Effective Date of Technical Report:	14 October 2016
Effective Date of Mineral Reserve Estimates:	
• Oyut	31 December 2015
• Hugo North and Hugo North Extension	20 September 2014
Effective Dates of Mineral Resource Estimates:	
• Oyut	19 March 2012
• Hugo North and Hugo North Extension	28 March 2014
• Hugo South	1 November 2003
• Heruga	30 March 2010

Qualified Persons:

Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation of the 2016 Technical Report and, the Mineral Reserve estimates.

Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for the preparation of the Mineral Resources.

Signature Page

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Effective Date of 2014 Technical Report: 14 October 2016

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- Heruga 30 March 2010

Overall Preparation of the Technical Report and Mineral Reserve Estimates:

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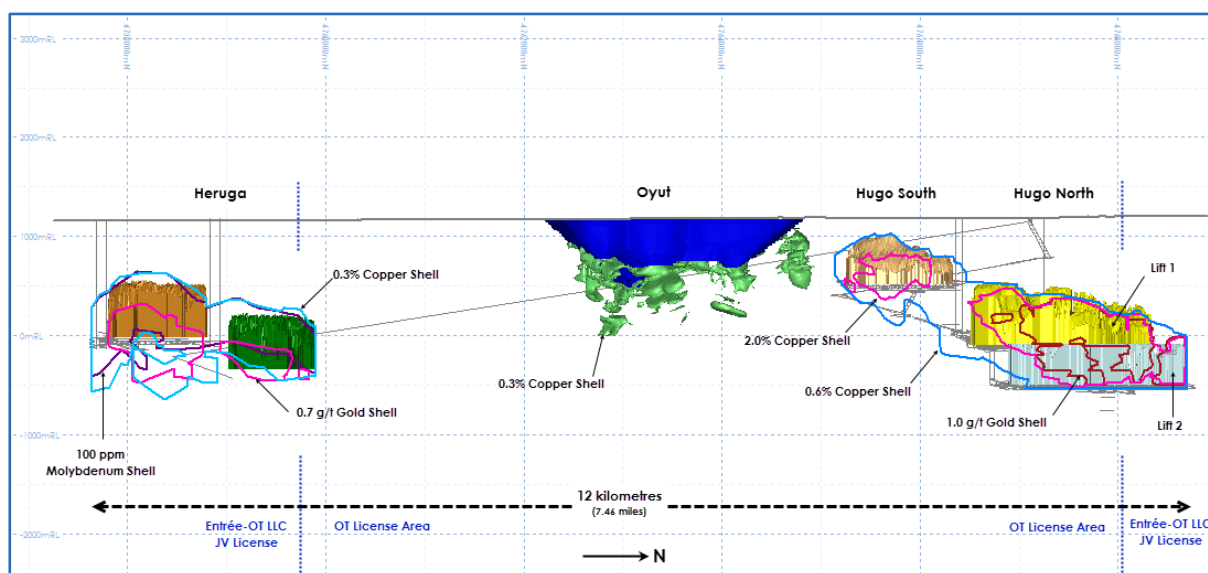
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1 SUMMARY

1.1 Project Overview and Development

The Oyu Tolgoi copper and gold project (Oyu Tolgoi) is located in the Southern Gobi region of Mongolia and is being developed by Oyu Tolgoi LLC (OT LLC). Oyu Tolgoi consists of a series of deposits containing copper, gold, silver, and molybdenum. The deposits lie in a structural corridor where mineralization has been discovered over a 26 km strike length from Ulaan Khud in the north and Javkhlant in the south. The Oyu Tolgoi deposits stretch over 12 km, from the Hugo North deposit in the north through the adjacent Hugo South, down to the Oyut deposit (formerly known as Southern Oyu Tolgoi (SOT)), and extending to the Heruga deposit in the south as shown in Figure 1.1.

Figure 1.1 Oyu Tolgoi Projected Long Section



After accounting for depletion due to mining up until 31 December 2015, the series of deposits contain an estimated Measured and Indicated Mineral Resource of 45.9 billion pounds of contained copper and 23.5 million ounces of contained gold and an estimated Inferred Mineral Resource of 51.5 billion pounds of contained copper and 36.0 million ounces of contained gold. The Oyu Tolgoi trend is still open to the north and south and the deposits have not been closed off at depth.

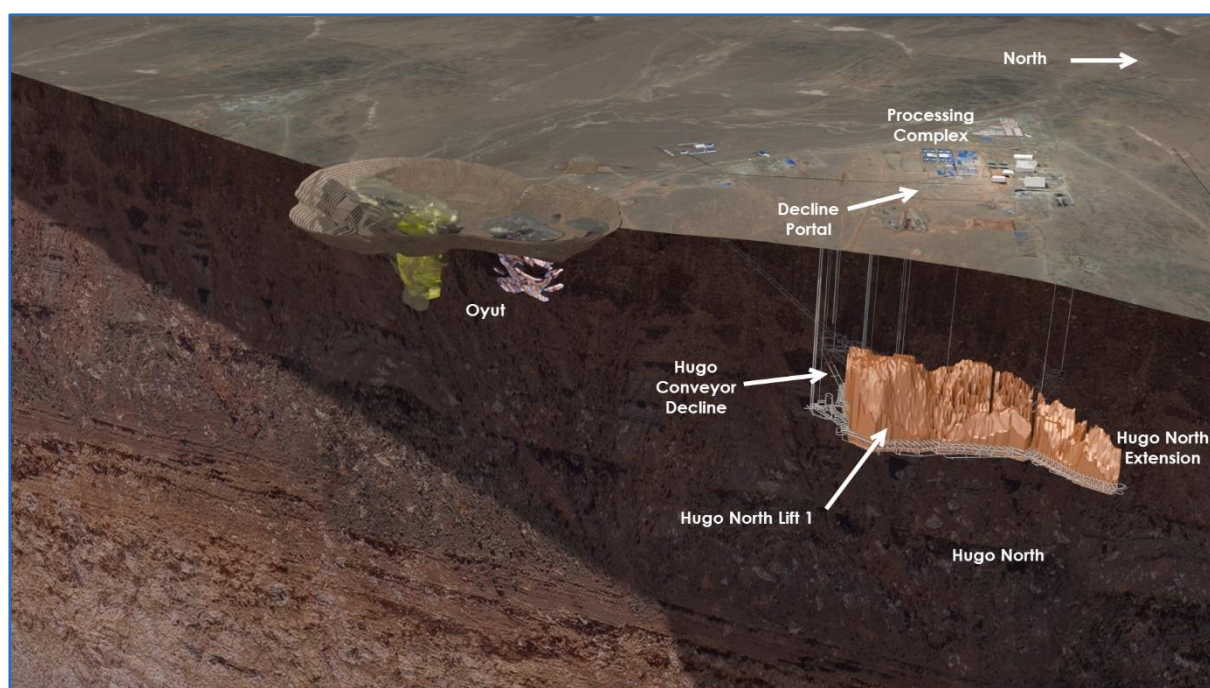
OT LLC is 66% owned by Turquoise Hill Resources Ltd (TRQ) and 34% owned by Erdenes OT LLC. Rio Tinto plc (Rio Tinto) owns 50.8% of TRQ and Erdenes OT LLC is owned by the Government of Mongolia (GOM). Rio Tinto is appointed by OT LLC to provide strategic and operational management to Oyu Tolgoi.

This report, the 2016 Oyu Tolgoi Technical Report (2016 OTTR), was prepared by Independent Qualified Persons (QPs), acting on behalf of TRQ and is based on the April 2016 Oyu Tolgoi Feasibility Study 2016 (OTFS16), prepared by OT LLC. OTFS16 has been approved by the OT LLC board of directors and shareholders. The 2016 OTTR is based on a review of the latest

technical, production, and cost information prepared by OT LLC and is based on feasibility-level study work complying with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). The 2016 OTTR meets the standards of US SEC Industry Guide 7 requirements for reporting Mineral Reserves.

The 2016 Reserves Case assumes processing of 1.4 billion tonnes of ore, mined from the Oyut open pit and the first lift in the Hugo North underground block cave. The mining areas included in the 2016 Reserves Case are shown schematically in Figure 1.2.

Figure 1.2 2016 Reserves Case Mining Areas



Over time, there are expected to be multiple investment decisions made for Oyu Tolgoi and an evaluation of each development option, as and when it is required, ensuring that the commitments made for the project represent the optimum use of capital to develop Oyu Tolgoi for Mongolia.

The initial investment decision was made in 2010 to construct Phase 1 of Oyu Tolgoi. Phase 1 consisted of the Oyut open pit mine, a concentrator and supporting infrastructure. These facilities are complete and the operation has commenced. Processing operations have been in production since December 2012, commercial production was achieved in September 2013, and first concentrate exported in October 2013.

Part of the initial investment decision included continued investment into the development of the Hugo North underground mine in parallel with mining the open pit. Lift 1 of Hugo North is the most significant value driver for the project. The Phase 2 scope, which includes the Hugo North underground block cave, supporting conveyor decline and shafts, concentrator conversion, and supporting infrastructure expansion has now commenced development.

The 2016 OTTR project scope from OTFS16 that has been used for the Mineral Reserves evaluation is the 2016 Reserves Case. A summary of the production and financial results for the 2016 Reserves Case are shown in Table 1.1.

Table 1.1 Summary Production and Financial Results – 2016 Reserves Case

Description	Units	2016 Reserves Case
Total Processed	bt	1.4
Cu Grade	%	0.86
Au Grade	g/t	0.30
Ag Grade	g/t	1.95
Copper Recoverable	blb	23.9
Gold Recoverable	Moz	10.4
Silver Recoverable	Moz	74.3
Life	Years	38
Expansion Capital	US\$b	4.63
NPV8% After Tax	US\$b	6.94
IRR After Tax	%	21
Payback Period	Years	8

Notes:

1. NPV8% is Net Present Value (NPV) at a discount rate of 8% for all years.
2. IRR is Internal Rate of Return.
3. For mine planning the metal prices used to calculate block model Net Smelter Returns (NSR) were copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t and the underground costs are based on US\$15.34/t.
4. 2016 OTTR financial analysis long-term metal prices used are: copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
5. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes low grade Indicated Mineral Resources. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
6. The Oyut open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
7. For Oyut, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
8. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves.
9. The Mineral Reserves reported above are not additive to the Mineral Resources.
10. Economic analysis has been calculated from the start of 2017 and excludes \$0.46b expansion capital from 2016. Costs shown are real costs not nominal costs. Expansion capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

The Mineral Resource models and mine designs and cut-off grades for the Mineral Reserves in 2016 OTTR are the same as those in the 2014 OTTR except for depletion from the Oyut open pit. The production schedule was updated in OTFS16 to allow for the delayed mid-2016 project restart. OTFS16 also includes updated capital and operating costs.

The 2016 OTTR includes Mineral Resources from the Oyu Tolgoi project (wholly owned by OT LLC) and Entrée–OT LLC Joint Venture (EJV) license areas. The Shivee Tolgoi license and the Javkhlant license are held by Entrée Gold Inc. (Entrée). The Shivee Tolgoi license and the Javkhlant license areas 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.

Four deposits have been identified in the Mineral Resources at Oyu Tolgoi; they are Oyut, Hugo Dummett, comprising Hugo North and Hugo South, and Heruga. Heruga is a separate deposit that lies south of the Oyut 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
- Oyut
 - Southwest Zone
 - Central Zone
- Hugo South
- Heruga

Currently and in the initial years the predominant source of ore is the Oyut open pit. In parallel to this surface works, underground infrastructure, and mine development is ongoing for the Hugo North underground block cave. Stockpiling allows the higher grade ore from Hugo North to gradually displace the open pit ore as the underground production ramps-up to reach 95 kt/d.

Ore is processed through the existing concentrator using conventional crushing, grinding, and flotation circuits. The concentrate produced is trucked to smelters and traders in China.

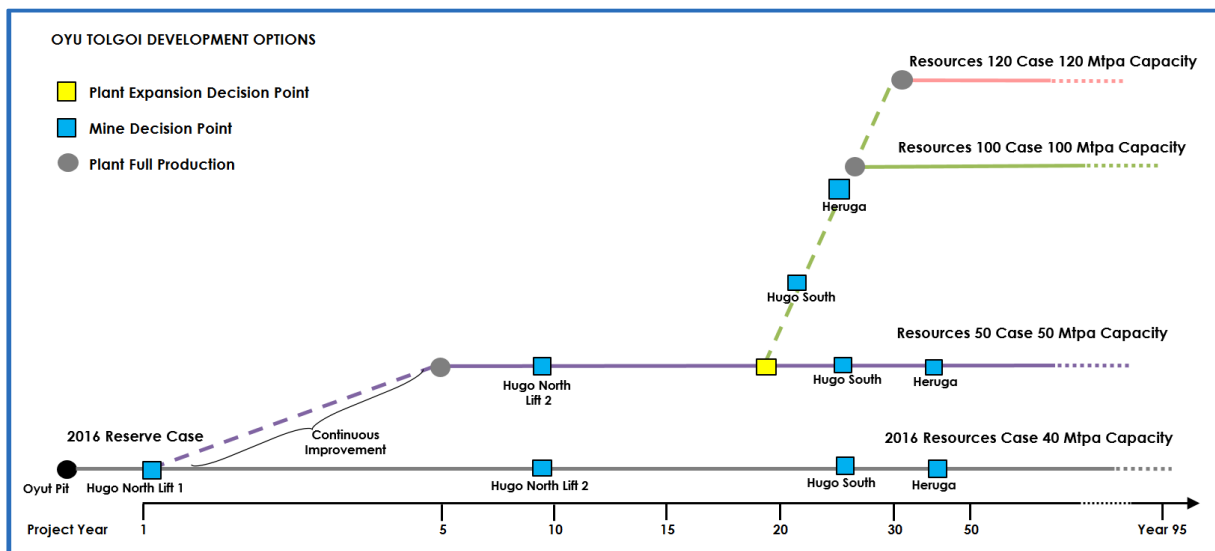
Oyu Tolgoi is a remote brownfields project and extensive infrastructure has been constructed in addition to the concentrating facilities. The major initial infrastructure elements include:

- Water Borefield,
- Water Treatment,
- Housing,
- Airstrip,
- Supporting Facilities, and
- Power.

Development of the entire resource is the objective of all stakeholders and over the life of Oyu Tolgoi OT LLC will continue to progress its understanding of these resources and ultimately make decisions on development of the entire resource.

The overall strategy for the development of Oyu Tolgoi remains the same as it has been in previous studies. Oyu Tolgoi's large resource base represents outstanding opportunities for production expansion. Figure 1.3 shows an example of the decision tree for the possible development options at Oyu Tolgoi. This has been updated to include alternative production options that take advantage of productivity improvements in plant throughput that have begun to be recognized in the process plant. The decision tree shows options assuming that continuous improvements in plant productivity are achieved over the next five years. After this time period there would be key decision points for plant expansion and the development of new mines at Hugo North Lift 2, Hugo South, and eventually Heruga. This provides an opportunity as the Oyu Tolgoi project will have the benefit of incorporating actual performance of the operating mine into the study before the next investment decisions are required. OT LLC plans to continue to evaluate alternative production cases in order to define the relative ranking and timing requirements for overall development options.

Figure 1.3 Oyu Tolgoi Development Options



1.2 Qualified Persons

The following Qualified Persons (QPs) were responsible for the preparation of the 2016 OTTR:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation of the report and the Mineral Reserve estimates.
- Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for the preparation of the Mineral Resources.

1.3 Project Location and Ownership

The majority of the identified mineralization at Oyu Tolgoi occurs within the mining license MV-006709 (OT License) at the Hugo Dummett and Oyut deposits. OT LLC holds its rights to the Oyu Tolgoi through this mining license, which comprises approximately 8,496 ha. The GOM granted the OT License to Ivanhoe Mines Mongolia Incorporated (IMMI, precursor to OT LLC) in 2003, along with mining licenses for three other properties, identified as MV-006708, MV-006710, and MV-006711. Subsequently, MV-006711 has been relinquished.

The OT License includes the right to explore, develop mining infrastructure and facilities, and conduct mining operations on Oyu Tolgoi. In 2006, the Mongolian Parliament passed new mining legislation and changed the term of mining licenses to a 30-year term with two 20-year extensions. Figure 1.4 shows the location of Oyu Tolgoi regionally relative to the Mongolian-Chinese border. Figure 1.5 shows the deposits and license boundaries.

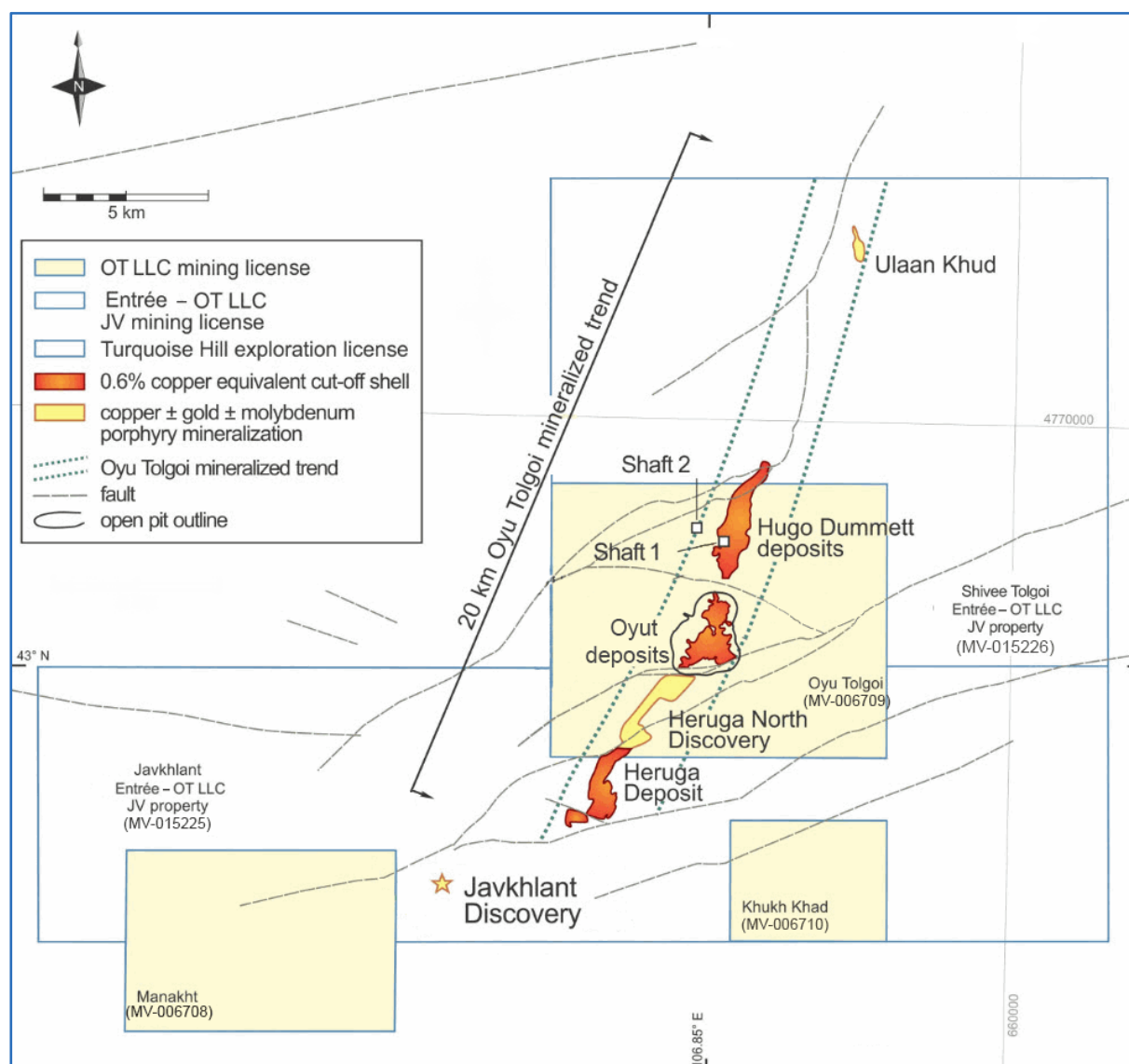
Figure 1.4 Project Location



OT LLC has an economic interest in MV-015225 (Javkhlant) and MV-015226 (Shivee Tolgoi) pursuant to an Equity Participation and Earn-in Agreement with Entrée (as amended in 2005). This agreement contemplates the establishment of a joint venture (EJV) between the parties that provides for OT LLC to hold legal title in MV-015225 and MV-015226, subject to the terms of the agreement, and to OT LLC meeting prescribed earn-in expenditures. While a formal joint venture has not been entered into yet, the earn-in requirements have been met, and OT LLC's participating interest in the joint venture (including the licenses) will be:

- In respect of the proceeds from mining from the surface to 560 m below the surface, 70%, and
- In respect of the proceeds from mining from depths beneath 560 m, 80%.

Figure 1.5 Oyu Tolgoi Licenses



The vast majority of the identified mineralization for the project occurs at the Hugo Dummett and Oyut deposits within the OT License area. The northernmost extension of the Hugo Dummett deposits (Hugo North) crosses onto the Shivee Tolgoi property (Hugo North Extension). The Heruga deposit lies almost entirely within the Javkhlant property, with only the northern extent passing into MV-006709. There are numerous exploration targets across MV-006708, MV-006709, MV-006710, MV-015225, and MV-015226.

The OT License property was surveyed by an independent consultant in 2002, by a qualified Mongolian Land Surveyor in 2004 and again in 2011 after the GOM ordered a re-survey to establish the legal boundaries of the OT License concession.

On 8 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 Oyu Tolgoi. These State special needs areas are to be used for infrastructure facilities necessary in order to implement the development of Oyu Tolgoi.

Most of the 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 government authorities in respect of such licenses. OT LLC has entered into certain consensual arrangements with some of the 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 terms of Resolution 175 and the Minerals Law (2006).

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.

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 but may be significant. The description of Resolution 175 has been provided by OT LLC and has been relied on under Item 3 of NI 43-101 Reliance on Other Experts.

In April 2015, the Standing Committee of the Parliament of Mongolia requested the GOM to modify Resolution 175 due to an alleged inconsistency between Resolution 175 and the Minerals Law and Land Law. OT LLC understands that the GOM supports the validity and justification for Resolution 175 and that Resolution 175 will not be modified or revoked.

1.4 Mineral Resource

Mongolia has its own system for reporting Mineral Reserves and Mineral Resources. OT LLC registered a Mineral Reserve with the GOM in 2009. A key difference between the two standards is the classification of material contained in Hugo North Lift 2, Hugo South, and Heruga under Mongolian standards as reserves. This contrasts to the Canadian National Instrument (NI) 43-101 definitions, which include only Oyut and Hugo North Lift 1 in the Mineral Reserve category.

The base case copper-equivalent (CuEq) cut-off grade assumptions for each deposit were determined using cut-off grades applicable to mining operations exploiting similar deposits. The 0.22% open pit CuEq cut-off is equivalent to the open pit Mineral Reserve cut-off determined by OT LLC and the 0.37% underground CuEq cut-off is equivalent to the underground Mineral Reserve cut-off determined by OT LLC.

CuEq Formula

In order to assess the value of the total suite of minerals of economic interest in the mineral inventory, formulae have been developed to calculate copper equivalency (CuEq) based on given prices and metallurgical recovery factors.

The initial formula used to calculate the CuEq grade was developed in 2003 for Hugo North and Oyut. There have been numerous variants on the formulae used since that time.

1.4.1 2014 CuEq Formula Derivation

The 2014 CuEq formulae are developed for each deposit and incorporate copper, gold, and silver, and also molybdenum for Heruga. The assumed metal prices are US\$3.01/lb for copper, US\$1,250/oz for gold, US\$20.37/oz for silver, and US\$11.90/lb for molybdenum.

Copper estimates are expressed in the form of percentages (%), gold and silver are expressed in grams per tonne (g/t), and molybdenum is expressed in parts per million (ppm).

Metallurgical recovery for gold, silver, and molybdenum are expressed as a percentage relative to copper recovery.

The unit conversions used in the calculation are as follows:

1 tonne = 1 million grams

grams per tonne (g/t) to ounces per tonne (oz/t) = 31.103477

pounds per kilogram (lb/kg) = 2.20462

tonne to pounds (lb) = 2,204.62

This leads to a base formula of:

$$\text{CuEq}_{14} = \text{Cu} + ((\text{Au} \times \text{AuRev}) + (\text{Ag} \times \text{AgRev}) + (\text{Mo} \times \text{MoRev})^{\dagger}) / \text{CuRev}$$

[†] Mo and MoRev are only incorporated into CuEq calculations for Heruga

Where:

$$\text{CuRev} = (3.01 \times 2.20462)$$

$$\text{AuRev} = (1,250 / 31.103477 \times \text{RecAu})$$

$$\text{AgRev} = (20.37 / 31.103477 \times \text{RecAg})$$

$$\text{MoRev} = (11.90 \times 0.00220462 \times \text{RecMo})$$

$$\text{RecAu} = \text{Au Recovery} / \text{Cu Recovery}$$

$$\text{RecAg} = \text{Ag Recovery} / \text{Cu Recovery}$$

$$\text{RecMo} = \text{Mo Recovery} / \text{Cu Recovery}$$

Different metallurgical recovery assumptions lead to slightly different copper-equivalent formulas for each of the deposits; these are outlined in the following tables for Oyut, Hugo North, Hugo North Extension, Hugo South, and Heruga. In all cases, the metallurgical recovery assumptions are based on metallurgical testwork. For Oyut, actual mill performance has been used to further refine the recovery assumptions. Recoveries are relative to copper because copper contributes the most to the equivalence calculation.

All elements included in the copper-equivalent calculation have a reasonable potential to be recovered and sold, except for molybdenum. Molybdenum grades are only considered high enough to support construction of a molybdenum recovery circuit for Heruga mineralization; hence the recoveries of molybdenum are assumed to be zero for the other deposits.

Copper equivalence assumptions and calculations for the various deposits are shown in Table 1.2 through Table 1.6.

The total Mineral Resources for Oyu Tolgoi are shown in Table 1.7. A profile of Oyu Tolgoi deposits is shown in Figure 1.6.

Table 1.2 Oyut – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.794	0.704	0.754	0		
	Recovery Relative to Cu	1	0.887	0.949	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.537	35.63
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	0.45				0.45	29.86
	Au Grade		0.31			0.166	11.05
	Ag Grade			1.23		0.012	0.76
	Mo Grade				0	0	–
	CuEq Grade & Revenue	0.45	0.31	1.23	0.	0.628	41.67

From Table 1.2 above, the base formula is adjusted for Oyut as follows:

$CuEq_{14}(Oyut) =$

$$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.887) + (Ag \times 20.37 \times 0.0321507 \times 0.949)) / (3.01 \times 22.0462)$$

Table 1.3 Hugo North – Copper Equivalence Assumptions and Calculation based on Average Grades

	Cu	Au	Ag	Mo
Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb
Recovery (%)	0.92	0.83	0.86	0
Recovery Relative to Cu	1	0.906	0.941	0
Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.549	36.43
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.66				1.66	110.16
	Au Grade		0.34			0.187	12.38
	Ag Grade			3.37		0.031	2.08
	Mo Grade				27.43	0	–
	CuEq Grade & Revenue	1.66	0.34	3.37	27.43	1.878	124.62

From Table 1.3 above, the base formula is adjusted for Hugo North as follows:

$CuEq_{14(HN)} =$

$$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.906) + (Ag \times 20.37 \times 0.0321507 \times 0.941)) / (3.01 \times 22.0462)$$

Table 1.4 Hugo North Extension – Copper Equivalence Assumptions and Calculation based on Average Grades

	Cu	Au	Ag	Mo
Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb
Recovery (%)	0.92	0.84	0.86	0.00
Recovery Relative to Cu	1.00	0.913	0.942	0
Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.553	36.69
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.59				1.59	105.51
	Au Grade		0.55			0.304	20.18
	Ag Grade			3.72		0.035	2.29
	Mo Grade				25.65	0	–
	CuEq Grade & Revenue	1.59	0.55	3.72	25.65	1.929	127.98

From Table 1.4 above, the base formula is adjusted for Hugo North Extension as follows:

$CuEq_{14(HNE)} =$

$$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.913) + (Ag \times 20.37 \times 0.0321507 \times 0.942)) / (3.01 \times 22.0462)$$

Table 1.5 Hugo South – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.89	0.81	0.85	0		
	Recovery Relative to Cu	1	0.909	0.945	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.551	36.54
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.07				1.07	71.00
	Au Grade		0.06			0.033	2.19
	Ag Grade			2.07		0.019	1.28
	Mo Grade					0	–
	CuEq Grade & Revenue	1.07	0.06	2.07		1.122	74.48

From Table 1.5 above, the base formula is adjusted for Hugo South as follows:

$CuEq_{14(HS)} =$

$$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.909) + (Ag \times 20.37 \times 0.0321507 \times 0.945)) / (3.01 \times 22.0462)$$

Table 1.6 Heruga – Copper Equivalence Assumptions and Calculation based on Average Grades

	Cu	Au	Ag	Mo
Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb
Recovery (%)	0.86	0.79	0.82	0.635
Recovery Relative to Cu	1	0.911	0.949	0.736
Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.552	36.61
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	0.42				0.42	27.87
	Au Grade		0.41			0.226	15.01
	Ag Grade			1.47		0.014	0.91
	Mo Grade				138.47	0.055	2.67
	CuEq Grade & Revenue	0.42	0.41	1.47	138.47	0.70	46.47

From Table 1.6 above, the base formula is adjusted for Heruga as follows:

$CuEq_{14}(HERUGA) =$

$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.911) + (Ag \times 20.37 \times 0.0321507 \times 0.949) + (Mo \times 11.9 \times 0.0022046 \times 0.736)) / (3.01 \times 22.0462)$

Figure 1.6 Idealized Profile of Oyut, Hugo Dummett, and Heruga Deposits (long section looking west)

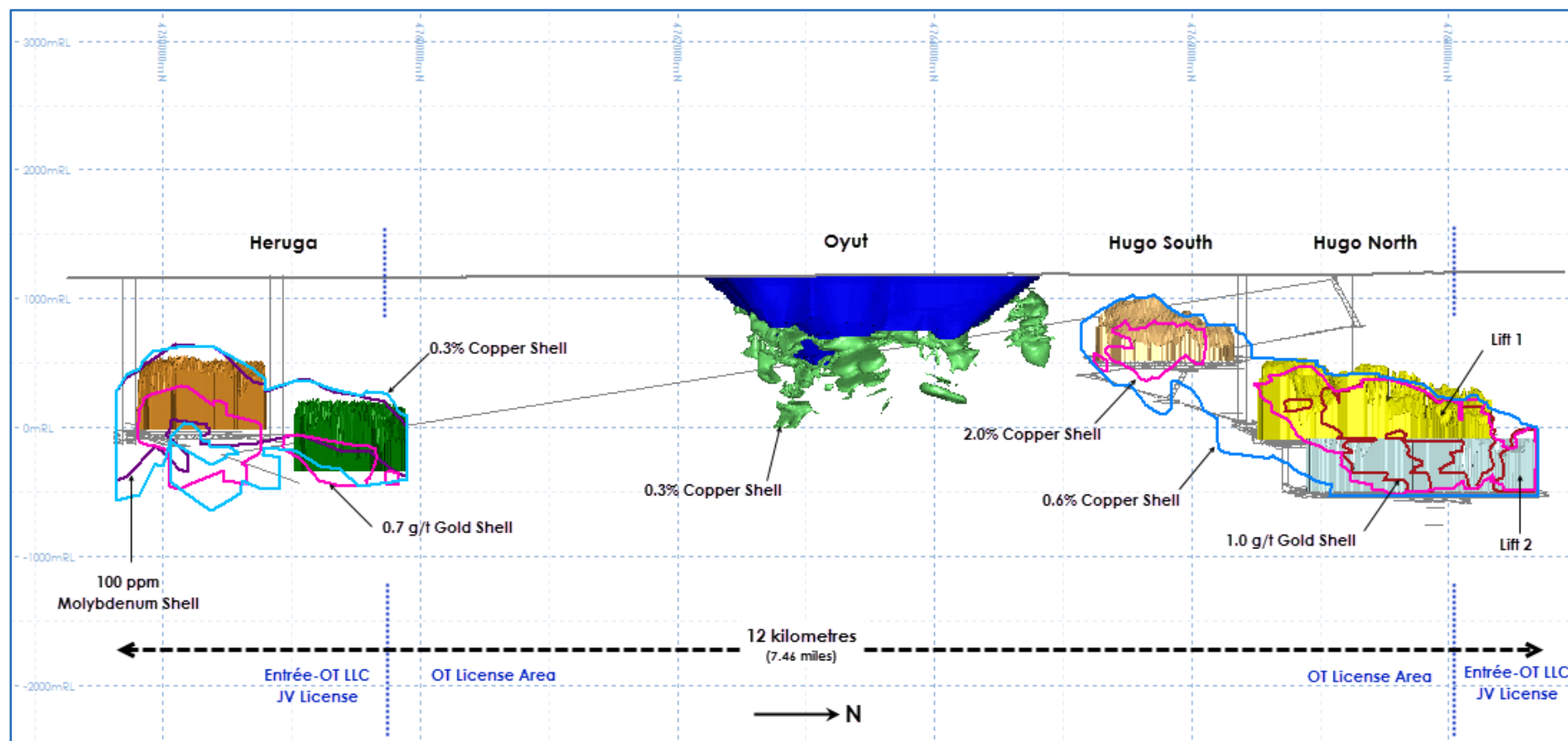


Table 1.7 Oyu Tolgoi Mineral Resource Summary – 31 December 2015

Classification	Deposit	Tonnage (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Contained Metal				
								Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)	CuEq (Mlb)
Oyut Deposit – Open Pit (0.22% CuEq Cut-off) (Excludes material mined up to 31 December 2015)												
Measured		377	0.52	0.35	1.35	53.9	0.72	4,335	4,038	15,804	45	5,947
Indicated		715	0.38	0.23	1.11	56.4	0.51	6,039	5,082	24,705	89	8,110
Measured + Indicated		1,092	0.43	0.27	1.19	55.5	0.58	10,374	9,120	40,509	134	14,057
Inferred		389	0.29	0.16	0.86	44.2	0.38	2,461	1,888	10,381	37	3,247
Oyut Deposit – Underground (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Measured		14	0.40	0.78	1.15	38.8	0.83	121	342	509	1.2	250
Indicated		93	0.35	0.59	1.19	34.3	0.67	713	1,766	3,562	7.1	1,386
Measured + Indicated		107	0.35	0.61	1.18	34.8	0.69	833	2,108	4,072	8.2	1,636
Inferred		159	0.39	0.32	0.85	25.4	0.56	1,354	1,638	4,382	8.9	1,985
Hugo Dummett Deposits (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Measured	OT LLC	98	1.97	0.46	4.48	30.3	2.26	4,231	1,446	14,046	6.5	4,865
	EJV	1	1.43	0.12	2.86	39.4	1.52	35	4	103	0.1	38
	All Hugo North	99	1.96	0.46	4.46	30.4	2.25	4,267	1,450	14,149	6.6	4,902
Indicated	OT LLC	749	1.56	0.34	3.35	34.3	1.78	25,737	8,268	80,718	57	29,362
	EJV	128	1.65	0.55	4.12	33.6	1.99	4,663	2,271	16,988	10	5,633
	All Hugo North	877	1.57	0.37	3.46	34.2	1.81	30,400	10,539	97,707	66	34,994
Measured + Indicated	OT LLC	847	1.61	0.36	3.48	33.85	1.83	29,968	9,714	94,764	63	34,226
	EJV	129	1.65	0.55	4.11	33.70	1.99	4,698	2,276	17,091	10	5,670
	All Hugo North	976	1.61	0.38	3.56	33.83	1.85	34,667	11,989	111,856	73	39,897
Inferred	OT LLC	811	0.77	0.27	2.34	34.8	0.94	13,807	7,058	60,964	62	16,851
	EJV	179	0.99	0.34	2.68	25.4	1.20	3,887	1,963	15,418	10	4,730
	All Hugo North	990	0.81	0.28	2.40	33.1	0.99	17,695	9,021	76,382	72	21,581
Inferred	Hugo South	845	0.77	0.07	1.78	66.4	0.83	14,372	1,861	48,406	124	15,384
Heruga Deposit (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Inferred Heruga EJV		1,700	0.39	0.37	1.39	113.2	0.64	14,610	20,428	75,955	424	24,061
Inferred Heruga TRQ		116	0.41	0.29	1.56	109.8	0.61	1,037	1,080	5,819	28	1,565
Inferred (All Heruga)		1,816	0.39	0.37	1.40	113.0	0.64	15,647	21,508	81,774	453	25,626
Oyu Tolgoi All Deposits Grand Total (Excludes material mined up to 31 December 2015)												
Measured		489	0.81	0.38	1.97	48.7	1.03	8,722	5,971	30,996	53	11,098
Indicated		1,686	1.00	0.32	2.34	43.6	1.20	37,152	17,572	126,797	162	44,486
Measured + Indicated		2,175	0.96	0.34	2.26	44.8	1.16	45,875	23,543	157,792	215	55,584
Inferred		4,200	0.56	0.27	1.64	75.1	0.73	51,531	35,980	221,670	695	67,821

Notes:

- The Mineral Resources include Mineral Reserves.
- The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries.
- The 0.22% CuEq cut-off is equivalent to the open pit Mineral Reserve cut-off determined by OT LLC.
- The 0.37% CuEq cut-off is equivalent to the underground Mineral Reserve cut-off determined by OT LLC.
- Oyut open pit Mineral Resources exclude material mined in the open pit as at 31 December 2015.
- CuEq has been calculated using assumed metal prices (US\$3.01/lb for copper, US\$1,250/oz for gold, US\$20.37/oz for silver, and US\$11.90/lb for molybdenum). Mo grades outside of Heruga are assumed to be zero for CuEq calculations.
 - Oyut CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.887) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.949)) / (3.01 \times 22.0462)$
 - HN (OT LLC) CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.906) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.941)) / (3.01 \times 22.0462)$
 - HN (EJV) CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.913) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.942)) / (3.01 \times 22.0462)$
 - HS CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.909) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.945)) / (3.01 \times 22.0462)$
 - Heruga CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.911) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.949) + (\text{Mo (ppm)} \times 11.9 \times 0.0022046 \times 0.736)) / (3.01 \times 22.0462)$
- Totals may not match due to rounding.
- EJV is the Entrée-OT LLC Joint Venture. The Shivee Tolgoi and Javkhant licenses are held by Entrée. The Shivee Tolgoi and Javkhant licenses 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. See Section 4.2.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

1.5 Mineral Reserves

The Mineral Reserves for the project have been estimated using the Oyut and Hugo North Mineral Resources. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT). 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 1.8. The Mineral Reserves for the 2016 OTTR are based on mine planning work prepared by OT LLC in OTFS16.

The Mineral Reserves for the Oyut open pit are based on the same modifying parameters and Mineral Resources, the change since the 2014 OTTR has been the mining depletion. The Hugo North Mineral Reserves in the 2016 OTTR are the same as the Mineral Reserves in the 2014 OTTR.

The Hugo North Mineral Reserve contains ore that is on the OT License and on the EJV Shivee Tolgoi license.

The 2016 OTTR Mineral Reserves are reported as at 31 December 2015. This date was selected for reporting of the Mineral Reserve to remain consistent with OTFS16.

The metal prices and assumptions used for the cut-off grades were denominated in NSR US\$/t and are the same as those used for cut-off grade determination in the 2014 OTTR. The economic analysis has been updated with current long-term metal prices and assumptions.

OT LLC undertook pit surveys and reported the depletion from the Oyut Mineral Reserve. The Oyut Mineral Reserve shown in shown in Table 1.8 is the Proven and Probable remaining in the pit. Stockpiles have not been included in the 2016 OTTR Oyut Mineral Reserve reporting and they will include some inferred and unclassified materials as well as low grade Measured and Indicated Mineral Resources. A comparison of the 2016 OTTR and 2014 OTTR Mineral Reserves is shown in Table 1.9.

The 2016 OTTR only considers Mineral Resources in the Measured and Indicated categories, and engineering that has been carried out to a feasibility level or better to estimate the open pit and underground Mineral Reserve. Mine designs were prepared using industry-standard mining software, assumed metal prices as described in the notes to the Mineral Reserves, and smelter terms as set forth in Section 22.

Table 1.8 Oyu Tolgoi Mineral Reserves 2016 – 31 December 2015

Deposit by Classification	Ore (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
					Cu (Mlb)	Au (koz)	Ag (koz)
Oyut Mineral Reserve							
Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
Probable	598	0.39	0.23	1.11	4,058	3,103	15,977
Oyut Total (Proven and Probable)	951	0.45	0.28	1.22	7,325	5,878	27,814
Hugo North Mineral Reserve							
Probable (OT LLC)	464	1.66	0.34	3.37	15,592	4,199	43,479
Probable (EJV)	35	1.59	0.55	3.72	1,121	519	3,591
Hugo North Total (Probable)	499	1.66	0.35	3.40	16,713	4,717	47,070
Total Mineral Reserve							
Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
Probable	1,097	0.97	0.29	2.15	20,771	7,820	63,047
Total (Proven and Probable)	1,450	0.86	0.30	1.97	24,037	10,595	74,884

Notes:

1. Metal prices used for calculating the financial analysis are as follows: long-term copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
2. For mine planning the metal prices used to calculate block model NSR were copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz.
3. The Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs at Oyu Tolgoi, therefore cut-off is denominated in US\$/t. By definition the cut-off is the point at which the costs are equal to the NSR. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t and the underground (including some mining costs) costs were based on US\$15.34/t.
4. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes Indicated Mineral Resources below the resource cut-off grade. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
5. The Oyut open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
6. For Oyut, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
7. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves.
8. EJV is the Entrée-OT LLC Joint Venture. The Shivee Tolgoi and Javkhant licenses are held by Entrée. The Shivee Tolgoi and Javkhant licenses 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. See Section 4.2.
9. The Mineral Reserves reported above were not additive to the Mineral Resources.
10. Totals may not match due to rounding.
11. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

Table 1.9 Mineral Reserves Reconciliation 2016 OTTR versus 2014 OTTR

Case	Mineral Reserves	Ore (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
						Cu (Mlb)	Au (koz)	Ag (koz)
2016 OTTR	Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
	Probable	1,097	0.97	0.29	2.15	20,771	7,820	63,047
	Total	1,450	0.86	0.30	1.97	24,037	10,595	74,884
2014 OTTR	Proven	410	0.54	0.42	1.38	3,829	3,952	13,768
	Probable	1,120	0.96	0.29	2.14	21,075	7,951	64,192
	Total	1,530	0.85	0.32	1.94	24,905	11,903	77,960
Absolute Difference	Proven	-57	0	-0.06	0.02	-563	-1,177	-1,931
	Probable	-23	0.01	0	0.01	-304	-130	-1,145
	Total	-81	0.02	-0.02	0.03	-867	-1,308	-3,076
Relative Difference	Proven	-16%	0%	-18%	2%	-17%	-42%	-16%
	Probable	-2%	1%	0%	1%	-1%	-2%	-2%
	Total	-6%	2%	-6%	2%	-4%	-12%	-4%

Notes:

1. 2014 OTTR Mineral Reserves have the effective date 20 September 2014.
2. 2016 Oyu Tolgoi Technical Report Mineral Reserves have the effective date of 31 December 2015.
3. Metal prices used in 2014 for calculating the Oyu open pit Net Smelter Return (NSR) and the Hugo North underground NSR are as follows: copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz, all based on long-term metal price forecasts at the beginning of the Mineral Reserves work. The analysis indicates that the Mineral Reserves are still valid at these metal prices.
4. Metal prices used in 2016 for calculating the financial analysis are as follows: long-term copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties. Prices are assumed to increase from current prices to the long-term prices which apply from 2021.
5. The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
6. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t.
7. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes Indicated Mineral Resources below the resource cut-off grade. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
8. For Oyu, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
9. The Oyu open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
10. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves (no Measured Mineral Resource for Hugo North in 2014 OTTR).
11. The Mineral Reserves reported above are not additive to the Mineral Resources.
12. Totals may not match due to rounding.

1.6 Economic Analysis

1.6.1 Economic Assumptions

The 2016 OTTR is an update of the Reserves Case previously presented in the 2014 Oyu Tolgoi Technical Report (2014 OTTR). The results of the 2016 Reserves Case show an after tax Net Present Value (NPV) at a discount rate of 8% for all years (NPV8%) of US\$6.94b. The case exhibits an after tax Internal Rate of Return (IRR) of around 21% and a payback period of around eight years. The estimates of cash flows have been prepared on a real basis from at 1 January 2017. The calendar year 2017 is Year 1 of the analysis and costs and revenues for 2016 have not been included in the 2016 Reserves Case analysis. Key economic assumptions in the analyses are shown in Table 1.10.

Table 1.10 Economic Assumptions

Parameter	Unit	Long-Term Financial Analysis Assumptions	Average Financial Analysis Assumptions
Copper Price	US\$/lb	3.00	2.97
Gold Price	US\$/oz	1,300	1,300
Silver Price	US\$/oz	19.00	19.00
Treatment Charges	US\$/dmt conc.	85.00	85.45
Copper Refining Charge	US\$/lb	0.085	0.085
Gold Refining Charge	US\$/oz	4.50	4.50

1.6.2 Investment Agreement (IA) and Taxation Assumptions

Both the process of negotiation and the final agreement in October 2009 of the Investment Agreement (IA) with the GOM, presented an opportunity to confirm how the Laws of Mongolia should be interpreted in their application to Oyu Tolgoi and provided for some specific terms to apply to Oyu Tolgoi. For OT LLC, the agreement has provided the confidence in the stability of the terms Oyu Tolgoi will operate under and reliably assess its intended investment. The IA itself is effective for an initial term of 30 years and an extension of a further 20 years. The term of a mining license under the Minerals Law is for 30 years with two 20-year extensions.

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 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 has the option to increase its shareholding to 50%, under terms to be agreed with TRQ 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 issuing of the GOM's equity share in Oyu Tolgoi and bringing the IA into full effect.

In 2011, an Amended and Restated Shareholders Agreement (ARSHA) was concluded, which reduced the applicable shareholder loan rate from 9.9% to LIBOR plus 6.5%. In addition, an agreement was reached to convert existing preference shares into ordinary shares.

Under the ARSHA, the TRQ Shareholders have appointed Rio Tinto OT Management Limited (RTOTM) to provide strategic and operational management. This entity or group of entities providing services and support to OT LLC is defined as the Management Team. RTOTM has been appointed as the Management Team with effect from 15 December 2010 for the purposes of clause 7.2 of the ARSHA and is the manager providing strategic and operational management pursuant to a Management Agreement (MA) executed between RTOTM and OT LLC dated 4 June 2015.

OT LLC met the IA requirement to achieve commencement of production within seven years of the effective date of the IA, which it did 1 September 2013.

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 as they applied as at the date of the IA. The taxes and fees payable to the GOM under Mongolian Law, and their stabilized rates, include:

- Corporate income tax 25%
- Mineral royalties 5% (gross sales value)
- Value added tax 10%
- Customs duties 5%
- Withholding tax 20%

OT LLC is also only subject to those taxes listed in the General Taxation Law as at the date of the IA and not taxes introduced at any future date. Non-stabilized taxes to which OT LLC is subject are payable in accordance with the applicable rates under Mongolian law from time to time; however, these taxes must apply to OT LLC on a non-discriminatory basis and as such cannot be imposed on OT LLC in any manner other than that applied to all taxpayers. Taxes to be withheld are calculated at the rates specified as in force at the signing of the IA, which includes in accordance with any applicable double tax treaties and which rates will be stabilized.

OT LLC may also apply to take advantage of any future law or treaty that comes into force and that would apply any rates lower than those specified in the IA.

In 2009, the GOM enacted amendments to the legislation governing the carry-forward of income tax losses. The loss carry-forward period has been extended to eight years and if sufficient, can be applied to offset 100% of taxable income. This was incorporated into the IA tax stability terms. This contrasts with the previous law, in which losses carried forward for two years 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.

On 18 May 2015, TRQ conceded that it has no entitlement to receive any payment from OT LLC under Mongolian law for the 2% NSR interest that TRQ had acquired from BHP (now BHP Billiton) in November 2003. This NSR entitlement arose out of TRQ's purchase of BHP's original exploration license (MEL) and all future entitlements covering what is now MV-006709.

1.6.3 Operating Assumptions

Although it has a requirement to make its self-discovered water resources available to be used for household purposes, the IA confirms that OT LLC holds the sole rights to use these water resources for the project. On 17 October 2014, a water use permit for 25 years was issued to OT LLC. In June 2016, OT LLC entered into a utilization agreement with a GOM water agency for 25 years (until June 2040). Together with water use conclusions issued annually and the approved water reserve rate, these arrangements enable OT LLC to utilize the water required to develop the project. As the Law on Water and IA provide that the term of water use permits for exploiting mineral deposits of strategic importance shall be the same as the term of mining licenses, OT LLC considers that it is entitled to extensions of its water permit and water utilization agreements for subsequent 20-year periods as its mining licenses are renewed.

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. The IA includes an overarching commitment from the GOM and OT LLC to work together to determine the optimal and most reliable solutions for power supply.

Under the IA, OT LLC is required to secure its power requirements from within Mongolia within four years of commencement of production. However, this timeframe is currently suspended pursuant to the Southern Region Power Sector Cooperation Agreement entered into by OT LLC and the GOM on 14 August 2014 that proposes an independently funded and operated coal fired power plant at Tavan Tolgoi (TTPP Project).

So long as OT LLC continues to participate in the TTPP Project the four-year timeframe for sourcing Mongolian power will be suspended. Upon a withdrawal from the TTPP Project by either OT LLC or the GOM, the four-year timeframe will be reinstated and recommence from the date of withdrawal. A request for proposals from potential investors in the TTPP coal fired power plant led to a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

OT LLC sources its present power under a four-year contract with a Chinese provider, the Inner Mongolia Power International Cooperation Company Ltd. (IMPIC) via the Mongolian National Power Transmission Grid (NPTG) authority. In May 2016 the parties agreed via a non-binding Memorandum of Understanding (MoU), which captured key agreed principles of the new Power Purchase Agreement (PPA), to extend the power supply agreement to at least 2021. OT LLC and the GOM have agreed under the Power Sector Cooperation Agreement that the GOM will assume responsibility for securing the extension of the power import arrangements through its national grid company NPTG. The agreed MoU includes comparable power pricing to the current agreement. OT LLC is endeavoring to execute binding agreements with the GOM and IMPIC within 2016.

OT LLC has the right to construct, manage, and use an aerodrome in connection with the project, based on permits issued in accordance with Mongolian law. A permanent domestic airport, capable of servicing Boeing 737-800 series aircraft, has been constructed at Oyu Tolgoi to support the transportation of people and goods to the site from Ulaanbaatar. It further serves as the regional airport for Khanbogd soum.

The GOM may construct or facilitate the construction and management of a railway in the vicinity of the project to the China-Mongolia border. The GOM will consult with OT LLC on the location and route of the railway, and, if the railway is constructed, then it will be made available to OT LLC on commercial and non-discriminatory terms. Energy Resources is currently constructing a single-track heavy-haul rail from its Ukhua Khudag coal mine (approximately 120 km to the north-west of Oyu Tolgoi) to Gashuun Sukhait, ultimately to be interconnected with the Chinese rail network at Ganqimaodao on the Chinese side of the border. Once constructed, the South Gobi Rail alignment would pass within 10 km of the Oyu Tolgoi project area and therefore represents an opportunity for eventual connection of the mine to the rail network.

OT LLC also has the right to construct roads for the transport of its product. A gravel road has been constructed to the town of Khanbogd and is being maintained. OT LLC intends to construct a paved road from the mine site to the town of Khanbogd. A 105 km sealed road is being constructed to the Chinese border crossing at Gashuun Sukhait, with sealing of the entire road expected to be completed in 2017. On the Chinese side of the border a provincial road connects the border town of Ganqimaodao with the Jingzang Expressway via the towns of Hailutu and Wuyuan.

Pursuant to the ARSHA, a Management Services Payment (MSP) equal to 3% of total operating and capital costs prior to commencement of production and 6% of operating and capital costs during production is payable by OT LLC to the Management Team (or other TRQ or Rio Tinto group entities to whom the Management Team may direct payment). Under the terms of the Underground Mining Development and Financing Plan (UDP) and Management Agreement (MA), OT LLC shareholders have agreed that in calculating the MSP, the rate to be applied to the capital costs of developing the Underground Stage shall be 3% instead of 6% as provided in the ARSHA. Underground Stage is defined by the UDP to mean the construction of the Hugo North Lift 1 underground mine and modification of the concentrator and infrastructure to handle underground ore.

The MA imposes a cap on the amount of capital costs that can be used to calculate the MSP and clarifies the categories of capital costs and operating costs that are to be included and excluded in calculation of the MSP. The MA clarifies the calculation of the MSP during the periods between 31 March 2010 and 1 September 2013, from 1 September 2013 and 4 June 2015 (the date of execution of the MA), and from 4 June 2015 onwards, and sets out a mechanism for OT LLC and RTOTM to carry out a reconciliation of the amounts of MSP payable against MSP payments already made. The MA further sets out the amount of chargeable costs to be borne by OT LLC, being costs incurred by the Management Team in providing Operational Management to OT LLC.

1.6.4 2016 Reserves Case Project Results

A summary of the 2016 Reserves Case project financial results is shown in Table 1.11. The estimates of cash flows have been prepared on a real basis from 1 January 2017 and discounted at a rate of 8%. Long-term metal prices used for the analysis are copper US\$3.00/lb, gold US\$1,300/oz, and silver US\$19.00/oz. The NPV results have been calculated starting from January 2017.

Table 1.11 Financial Results – 2016 Reserves Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	25.31	23.00
	5%	11.83	10.95
	6%	10.15	9.43
	7%	8.71	8.10
	8%	7.45	6.94
	9%	6.35	5.92
	10%	5.39	5.03
IRR (%)	–	21	21
Project Payback Period (Years)	–	8	8

C1 cash costs are shown in Table 1.12. C1 cash costs are those costs relating to the direct operating costs of the mine site, namely:

- Mining
- Concentration
- Tailings
- General and administrative (G&A) costs
- Operational Support Costs
- Infrastructure
- Realization Costs
- By-product Credits

The revenues and operating costs, have been presented in Table 1.13, along with the net sales revenue value attributable to each key period of operation. Interest charges from shareholder loans and project financing are not included in the operating and capital costs but are allowed for in the calculation of tax.

The 2016 Reserves Case Processing, and Concentrate and Metal Production are summarized in Figure 1.7 and Figure 1.8 respectively.

Table 1.12 C1 Cash Costs – 2016 Reserves Case

Description	US\$/lb Payable Copper			
	2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Mine Site Cash Cost	1.86	3.01	1.73	1.55
By-product Credit	0.62	1.15	0.71	0.53
C1 Cash Costs (Net of By-product Credit)	1.24	1.86	1.01	1.02

Table 1.13 Operating Costs and Revenues – 2016 Reserves Case

	US\$b	US\$/t Ore Milled		
	Total 2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Revenue				
Gross Sales Revenue	82.81	32.48	65.59	57.18
Less: Realization Costs				
Realization Costs	9.46	4.49	7.15	6.53
Government Royalty	4.31	1.69	3.41	2.97
Total Realization Costs	13.77	6.18	10.56	9.51
Net Sales Revenue	69.04	26.30	55.03	47.67
Less: Site Operating Costs				
Mining (all sources)	8.43	4.85	5.55	5.82
Processing and Tailings	11.91	7.45	7.77	8.22
G&A and Operations Support	2.77	2.52	2.44	1.91
Infrastructure and Other	1.76	1.83	1.69	1.22
Government Fees & Charges	2.83	1.98	2.06	1.95
Management and JV Payments	2.75	2.74	2.54	1.90
Total Site Operating Costs	30.44	21.36	22.05	21.02
Operating Margin	38.59	4.93	32.97	26.65

The total 2016 Reserves Case capital costs are shown in Table 1.14 these are the costs from 1 January 2017. The changes in financial results for a range of copper and gold prices are shown in Table 1.15.

Table 1.14 Total Project Capital Cost – 2016 Reserves Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.564	1.564
Underground	2.240	3.065	5.304
Concentrator	0.145	0.149	0.293
Infrastructure	0.354	0.230	0.585
Tailings Storage Facility (TSF)	–	0.912	0.912
Subtotal	2.739	5.920	8.659
Construction Indirect	0.858	–	0.858
Contractor Execution – EPCM	0.310	–	0.310
Owner Execution	0.429	0.168	0.597
GOM Fees & Charges – Mongolian VAT	0.298	0.669	0.967
Total	4.635	6.756	11.391

Notes:

1. Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. The 2016 Reserves Case total capital cost excludes capital costs for the year 2016. Expansion capital for 2016 excluded is US\$0.46b.

Figure 1.7 Processing – 2016 Reserves Case

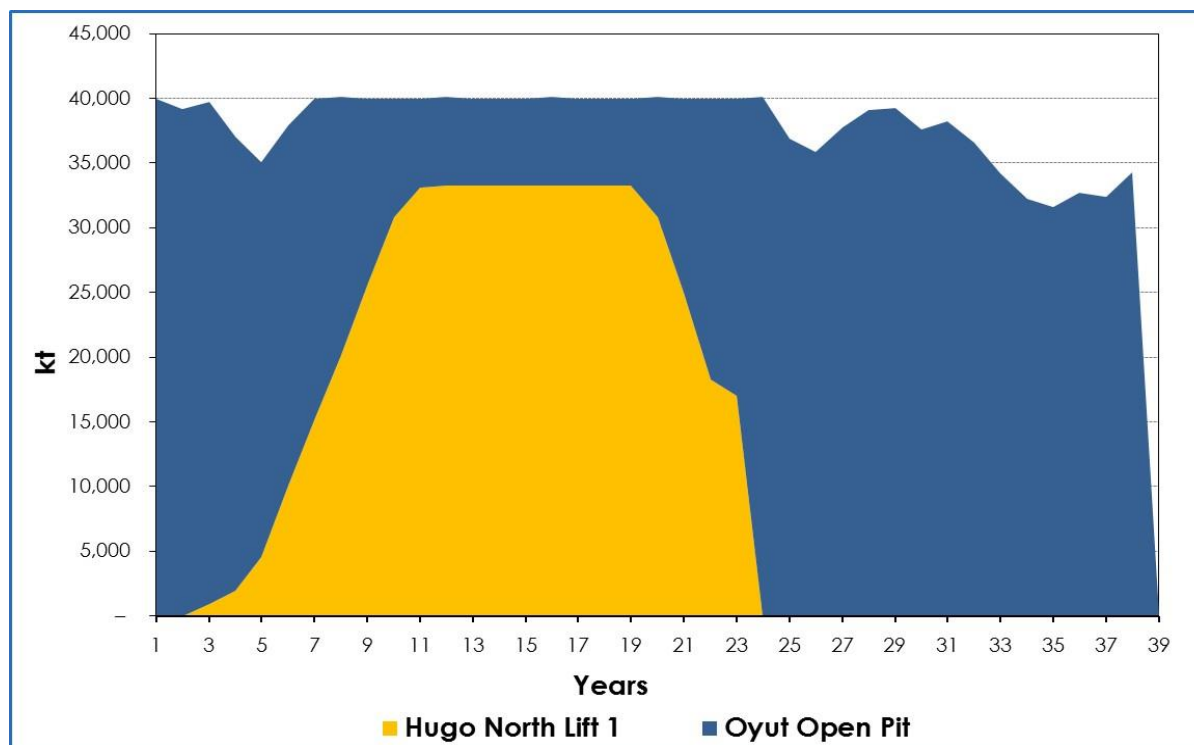


Figure 1.8 Concentrate and Metal Production – 2016 Reserves Case

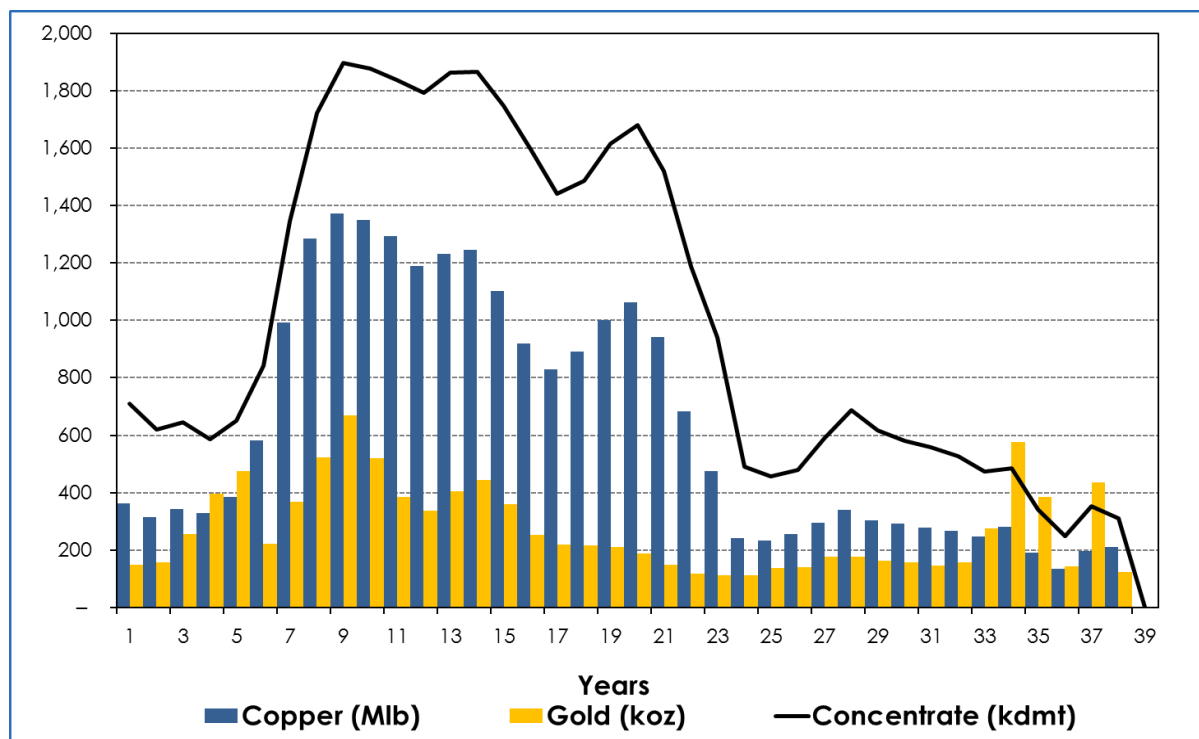


Table 1.15 After Tax Metal Price Sensitivity – 2016 Reserves Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.89	-2.54	-2.20	-1.86	-1.51	-1.17	-0.31
2.00	-1.48	-1.14	-0.79	-0.45	-0.11	0.24	1.10
2.50	2.04	2.39	2.73	3.07	3.42	3.76	4.62
2.80	4.16	4.50	4.84	5.19	5.53	5.87	6.73
3.00	5.56	5.91	6.25	6.59	6.94	7.28	8.14
3.50	9.09	9.43	9.77	10.12	10.46	10.80	11.66
4.00	12.61	12.95	13.29	13.64	13.98	14.32	15.18
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	7
2.00	–	–	5	6	8	9	11
2.50	13	14	14	15	16	17	18
2.80	16	17	18	18	19	20	21
3.00	19	19	20	20	21	22	23
3.50	23	23	24	25	25	26	27
4.00	27	27	28	28	29	30	31
Project Payback After Tax (Years)							
1.80	14.4	13.7	13.1	12.7	12.2	11.7	10.6
2.00	12.2	11.8	11.4	11.0	10.7	10.3	9.7
2.50	9.7	9.5	9.3	9.2	9.0	8.9	8.6
2.80	9.0	8.8	8.7	8.6	8.5	8.4	8.2
3.00	8.6	8.5	8.4	8.3	8.3	8.2	8.0
3.50	8.1	8.0	7.9	7.8	7.7	7.7	7.5
4.00	7.6	7.5	7.5	7.4	7.4	7.3	7.2
Cash Costs (Net of By-product Credit) (US\$/lb Payable Copper)							
1.80	1.35	1.31	1.27	1.23	1.18	1.14	1.04
2.00	1.36	1.32	1.28	1.24	1.19	1.15	1.05
2.50	1.38	1.34	1.30	1.26	1.22	1.18	1.08
2.80	1.40	1.36	1.32	1.27	1.23	1.19	1.09
3.00	1.41	1.37	1.33	1.28	1.24	1.20	1.10
3.50	1.43	1.39	1.35	1.31	1.27	1.23	1.12
4.00	1.46	1.42	1.37	1.33	1.29	1.25	1.15

[†] Calculated at a silver price of US\$19.00/oz

1.7 Infrastructure

Most of the infrastructure facilities required for the Oyu Tolgoi project were completed during Phase 1. The main facilities for Phase 2 will be associated with the underground mine, other key infrastructure buildings and services that will be expanded or added are: a power distribution system, some internal access roads, concentrate logistics facilities, camp accommodation, water distribution, information and communication (ICT), surface warehouse for underground, central heating and the waste and recycling facilities.

1.8 Environmental and Social Impact Assessment

OT LLC has completed a comprehensive Environmental and Social Impact Assessment (ESIA) for Oyu Tolgoi. The ESIA undertaken as part of the project finance process was publicly disclosed in August 2012. The culmination of nearly ten years of independent work and research carried out by both international and Mongolian experts, the ESIA identifies and assesses the potential environmental and social impacts of the project, including cumulative impacts, focusing on key areas such as biodiversity, water resources, cultural heritage, and resettlement.

The ESIA also sets out measures through all project phases to avoid, minimize, mitigate, and manage potential adverse impacts to acceptable levels established by Mongolian regulatory requirements and good international industry practice, as defined by the requirements of the Equator Principles, and the standards and policies of the International Finance Corporation (IFC), European Bank for Reconstruction and Development (EBRD), and other financing institutions.

Corporate commitment to sound environmental and social planning for the project is based on two important policies: TRQ's Statement of Values and Responsibilities, which declares its support for human rights, social justice, and sound environmental management, including the United Nations Universal Declaration of Human Rights (1948); and The Way We Work 2009, Rio Tinto's Global Code of Business Conduct that defines the way Rio Tinto manages the economic, social, and environmental challenges of its global operations.

OT LLC has implemented and audited an environmental management system (EMS) that conforms to the requirements of ISO 14001:2004. Implementation of the EMS during the construction phases to focus on the environmental policy; significant environmental aspects and impacts and their risk prioritization; legal and other requirements; environmental performance objectives and targets; environmental management programmes; and environmental incident reporting.

The EMS for operations consists of detailed plans to control the environmental and social management aspects of all project activities following the commencement of commercial production in 2013. The Oyu Tolgoi ESIA builds upon an extensive body of studies and reports, and Detailed Environmental Impact Assessments (DEIA's) that have been prepared for project design and development purposes, and for Mongolian approvals under the following laws:

- The Environmental Protection Law (1995);
- The Law on Environmental Impact Assessment (1998, amended in 2001); and
- The Minerals Law (2006).

These initial studies, reports, and DEIA's were prepared over a six-year period between 2002 and 2008, primarily by the Mongolian company Eco-Trade LLC, with input from RPS Aquaterra on water issues.

The original DEIA's provided baseline information for both social and environmental issues. These DEIA's covered impact assessments for different project areas, and were prepared as separate components to facilitate technical review as requested by the GOM. The DEIA's were in accordance with Mongolian standards and while they incorporated World Bank and IFC guidelines, they were not intended to comprehensively address overarching IFC policies such as the IFC Policy on Social and Environmental Sustainability, or the EBRD Environmental and Social Policy. Following submission and approval of the initial DEIA's, the GOM requested that OT LLC prepare an updated, comprehensive ESIA whereby the discussion of impacts and mitigation measures was project-wide and based on the latest project design. The ESIA was also to address social issues, meet GOM (legal) requirements, and comply with current IFC good practice.

For the ESIA, the baseline information from the original DEIA's was updated with recent monitoring and survey data. In addition, a social analysis was completed through the commissioning of a Socio-Economic Baseline Study and the preparation of a Social Impact Assessment (SIA) for the project. The requested ESIA, completed in 2012, combines the DEIA's, the project SIA, and other studies and activities that have been prepared and undertaken by and for OT LLC.

1.9 Water Management

Due to low average annual precipitation in the project area, water management and conservation are given the highest priority in all aspects of project design.

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 is used for process water supply, dust suppression in the mining areas, and potable use. Another major component of the water management plan is the diversion of the Undai River to accommodate project facilities. Undai River water is not used by the mine; the diversion is to preserve this water in the environment.

OT LLC has affirmed it is committed to water conservation and has benchmarked its water conservation efforts against other mines by assessing factors such as quantified water consumption per tonne of concentrate produced. The current water budget is based on the use of 550 L/t and operating performance of the concentrator suggests this is a reasonable estimate. The water consumption compares favorably with other large operations in similar arid conditions.

The GOM awarded a water utilization contract to OT LLC until 2040, which may in turn be extended for 20-year periods beyond 2040, in accordance with the Law on Water. OT LLC is currently entitled to utilize water at a rate of 918 L/s.

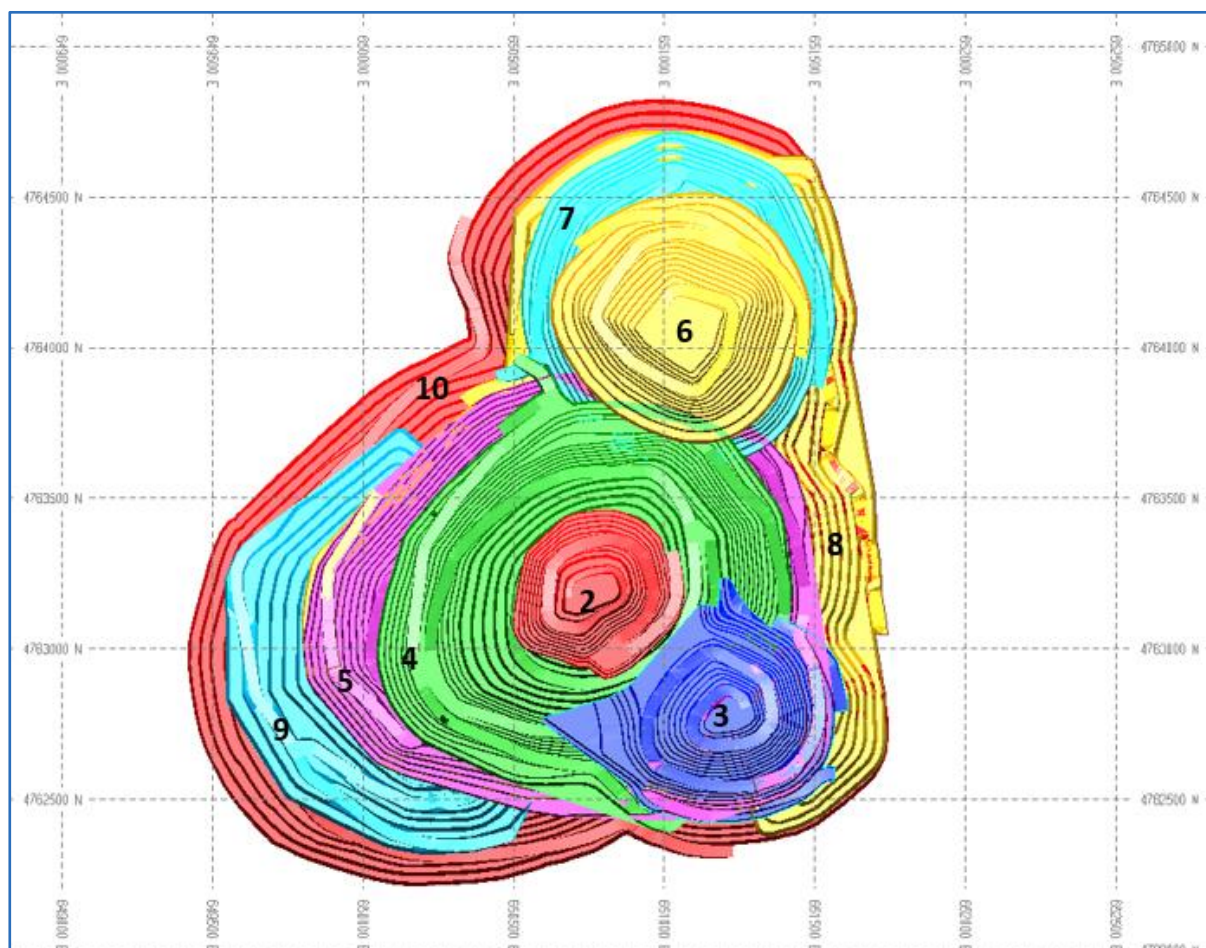
Updated hydrogeological modelling, completed in 2013, and based on all three hydrogeological investigation programmes, demonstrates that the Gunii Hooloi aquifer is capable of providing 1,475 L/s, based on the same time and drawdown conditions.

1.10 Open Pit Mining

The open pit mine at Oyu Tolgoi is a conventional shovel-truck operation. OT LLC's workforce carries out drilling, loading, hauling, and associated production support roles. The operation makes use of a mixed fleet of 34 m³ diesel hydraulic shovels and 56 m³ electric rope shovels working in tandem with 290 t haul trucks. Equipment maintenance is conducted under Service Agreements with selected original equipment manufacturers in-country dealers. A blasting contractor provides blasting products and down-the-hole services.

The ten phase pit designs were not revised from the work in OTFS14 for OTFS16. Mining in Phase 1 and 2 is complete and is currently progressing in Phases 3, 4 and 6. The Reserve was re-estimated for OTFS14 and was not revised for OTFS16 except for the mining depletion. A plan view Phases 2 to 10 is shown in Figure 1.9.

Figure 1.9 Open Pit Phase Designs



1.11 Underground Mining

The Hugo North, Hugo South, and Heruga orebodies are planned to be mined by underground panel caving methods. The first underground orebody to be mined is Hugo North, where two mining lifts are planned (Lift 1 first, then Lift 2). The first three panels of Hugo North Lift 1, the basis for OTFS16, contains the highest grade copper and gold and has the highest value.

The Hugo North Lift 1 underground construction formally re-commenced in July 2016. Development and construction activities will ramp up and continue through to the start of production in late 2019, defined as the point of commissioning the initial 30 kt/d production ore handling system. Production will ramp up to deliver an average of 95 kt/d of ore to the process plant during its peak production period from 2027 through 2035, ramping down to completion in 2039. The Hugo North Lift 1 reserves total 499 Mt at a grade of 1.66% Cu and 0.35 g/t Au.

To support mining of Hugo North Lift 1, two declines included in 203 km of lateral development, five shafts, 6.8 km of vertical raise-boring, and 115,000 m³ of mass excavations will be undertaken. The Lift 1 mining levels are approximately 1,300 m below surface. The orebody has average dimensions of 2,000 m long x 280 m wide. A total of 2,231 drawpoints are planned to be developed within the mining footprint, accessed from 52 extraction drives. Figure 1.10 illustrates the planned mine development projected onto the site layout showing the location of the twin declines relative to the processing plant and block cave. Figure 1.11 illustrates an isometric of the underground mine design. The major milestone dates are shown in Table 1.16.

The total conveying and hoisting capacity from the underground is planned to be approximately 140 kt/d (50 Mt/a). OT LLC has indicated that the primary advantage of the decline is that it will provide an upside opportunity to take advantage of potential improved productivity and increased production from Lift 1 or early access to a block cave on Hugo South or a future Hugo North Lift 2.

Figure 1.10 Hugo North Lift 1 Mine Design Projection

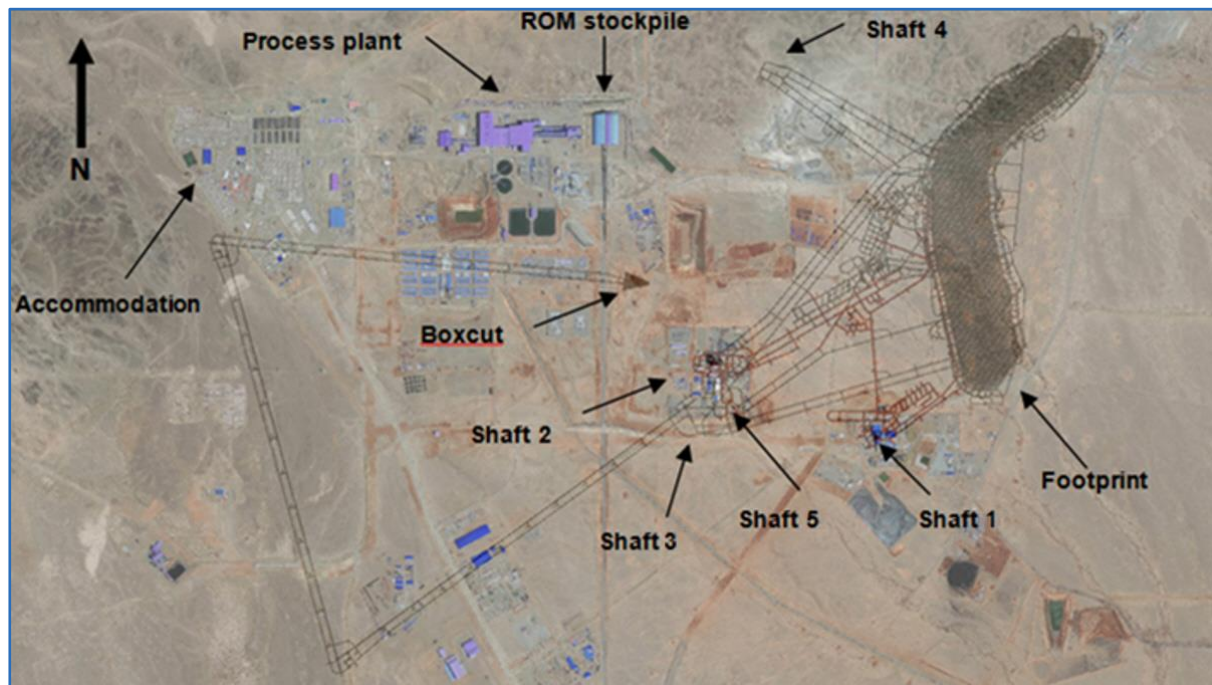


Figure 1.11 Isometric of Mine Design

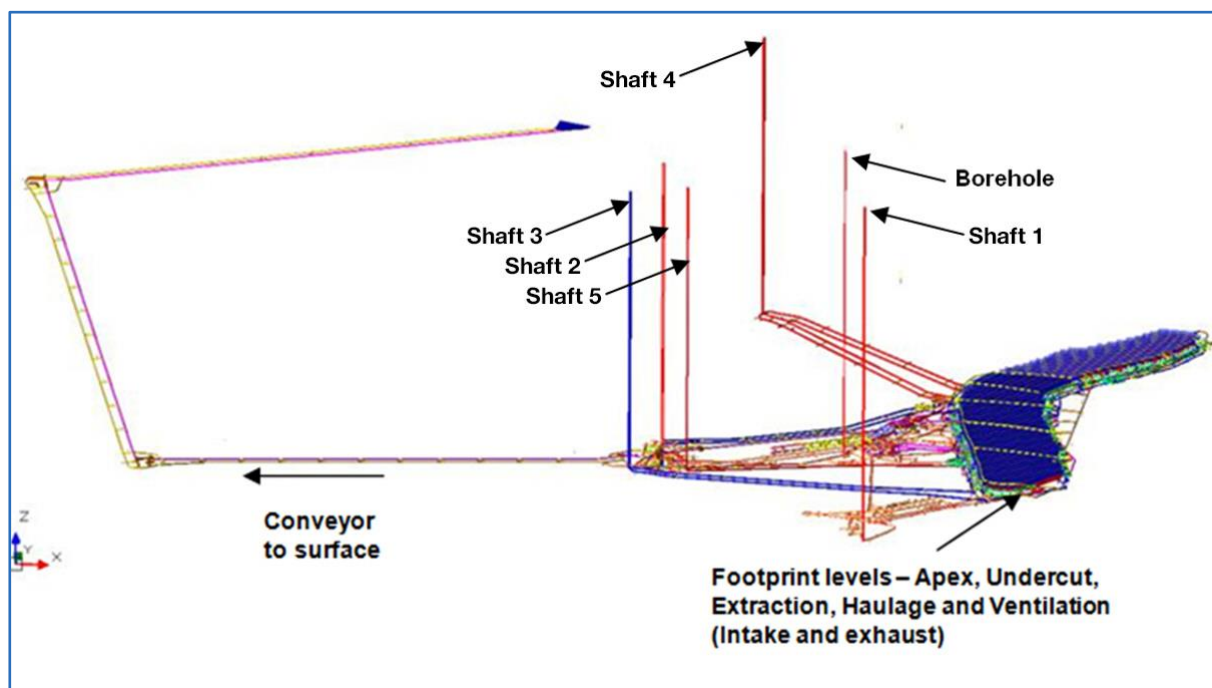


Table 1.16 Underground Major Milestones

Milestone	Date
Commission Shaft 2 Exhaust Fans	Late-2016
Commission Explosives Magazine	Late-2016
Commission Shaft 5	Late-2017
Commission Shaft 2 – Cage	Mid-2018
Commission Shaft 2 – Hoist, Loadout, Jaw Crusher	Mid-2018
Start Undercut Ring Drill and Blast*	Late-2019
Commission 30 kt/d Ore Handling System	Late-2019
First Drawbell Blasted*	Mid-2020
Production Ramp-up Commences	Early-2021
Conveyor to Surface Commissioned	Early-2022
Crusher 2 Commissioned	Mid-2022
Concentrator Upgrade Complete	Late-2022
Expansion / Development Capital Complete	Late-2022
Full Production Achieved (95 kt/d)	Early-2027

* Includes five months' schedule range contingency

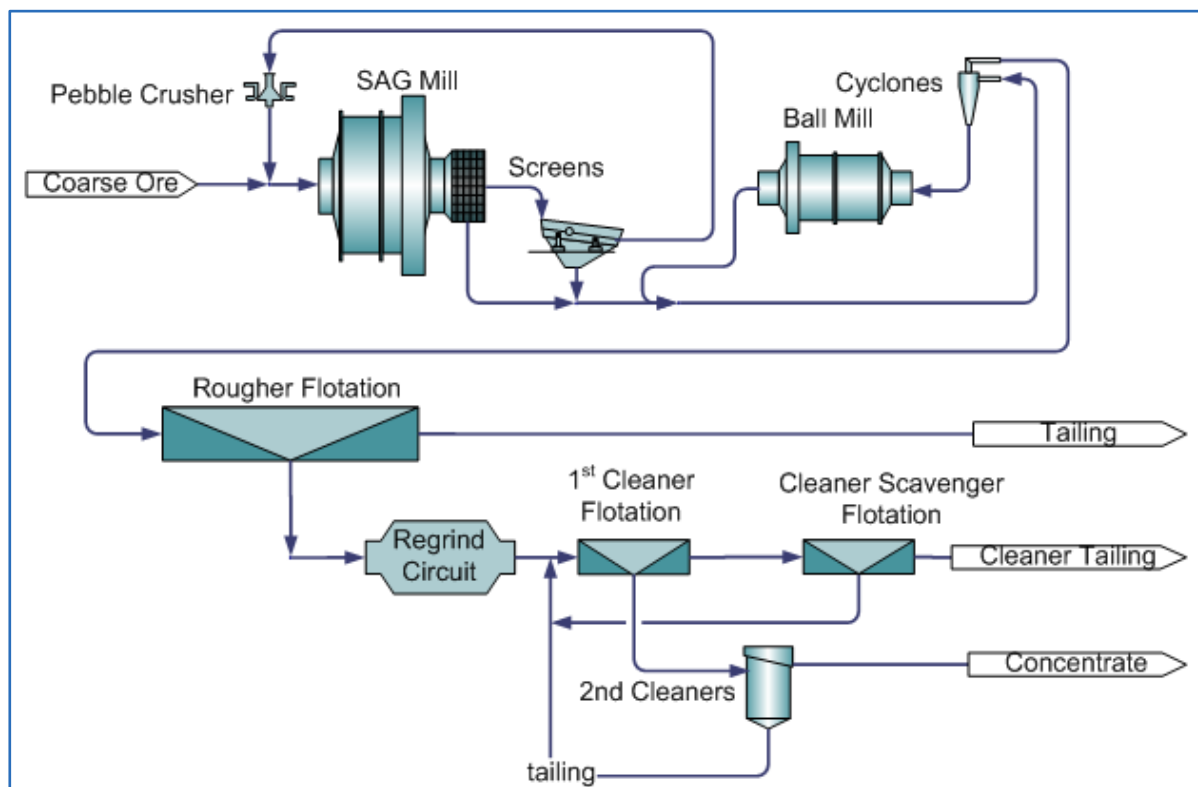
1.12 Exploration

OT LLC plans to continue exploration on the Oyu Tolgoi project mining licenses. The focus will be on resources that will increase the mine life and potentially defer development of deeper and lower grade resources. Smaller, incremental additions to the resource base are to be sought as greater knowledge of the orebodies at the known deposits, specifically geotechnical considerations, through infill drilling as part of a longer-term goal to convert resources into reserves. In addition to exploration drilling, further work will be conducted to improve confidence in inferred level resources at the Hugo Dummett deposits, especially around Lift 1 Panels 3–5, Lift 2, and Hugo South.

1.13 Concentrator

Oyu Tolgoi employs a conventional SAG mill / ball mill / grinding circuit (SABC) followed by flotation, as shown in the basic flowsheet (Figure 1.12).

Figure 1.12 Basic Oyu Tolgoi Flowsheet – Phase^o1



Phase 1, which commenced production in 2013, uses two grinding lines, each consisting of a SAG mill, two parallel ball mills, and associated downstream equipment to treat up to 100 kt/d the Southwest zone pit ore. Softer ore from the Central zone pit is to be processed at higher rates. Combined with Hugo North underground ore, concentrator feed rates for individual ore types will be as high as 121 kt/d, which represents the tailings handling capacity of the plant. In the production schedule for the 2016 Reserves Case the annual rate at full production is 110 kt/d (40 Mt/a). The Phase 2 scope is all the additional work required to process Hugo North Lift 1 production plus open pit ore to match Phase 1 SAG mill capacity, including:

- the addition of a fifth ball mill to achieve a finer primary grind P80 of 140–160 µm for a blend of Hugo North and open pit ores
- additional roughing and column flotation capacity to process the higher level of concentrate production when processing the higher grade Hugo North ore
- additional concentrate dewatering and bagging capacity.

OTFS16 noted that the scope may vary slightly depending on the throughput outcome of current and future concentrator improvement work for Phase 1.

1.14 Concentrate Shipment and Handling

Concentrate is transported in two-tonne bags. Mongolian custom clearance occurs at the marshalling yard at the Oyu Tolgoi site. Chinese customs clearance occurs at the bonded warehouse.

The current logistics model involves road transport from the mine to the bonded warehouse, located in China, storage at the warehouse for 14–21 days for customs clearance, and then a mix of road and rail from Ganqimaodao to the customer. Oyu Tolgoi is responsible for transportation up to the bonded warehouse, and customers are responsible for transportation.

At present, border operating hours and schedule are limited, and the border is subject to sporadic closure. Weather, communications, and energy failures all contribute to this situation. Efforts are underway on both sides of the border to increase its capacity and efficiency.

1.15 Alternative Production Cases

Oyu Tolgoi is a very large project that includes five separate deposits. The long-term development of Oyu Tolgoi would involve the resources in all deposits. Alternative Production Cases have been developed to provide early stage analysis of the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second lift of Hugo North). Development of these deposits will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi.

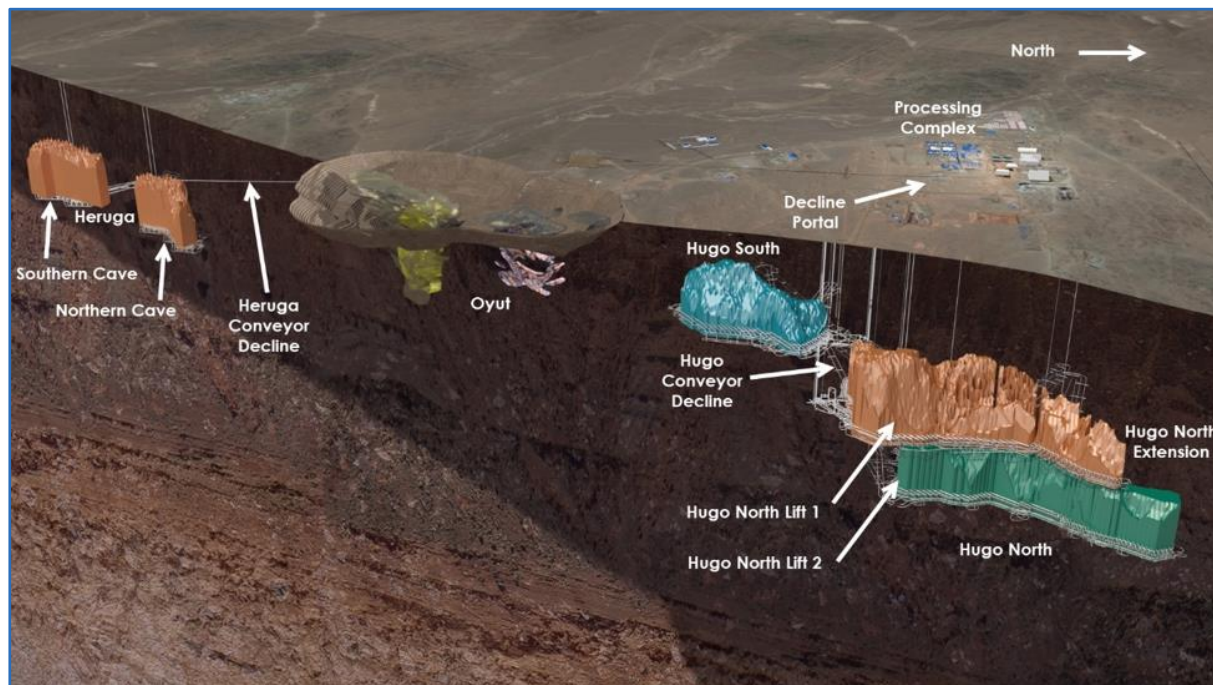
Accordingly, the analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

The 2016 Resources Case was prepared based on the OTFS16 Resources Case. This case has been submitted to the Mongolian authorities as part of the OTFS16 submission. The 2016 Resources Case, which is a baseline of the expansion analysis, assumes that the plant capacity remains at the planned OTFS16 average production rate of 110 kt/d (40 Mt/a).

The mine designs used for the Alternative Production Cases are shown schematically in Figure 1.13: The designs are:

• Oyut open pits	(2016 Mineral Reserves)
• Hugo North Lift 1 block cave (Panels 0–2)	(2016 Mineral Reserves)
• Hugo North Lift 1 block cave (Panels 3–5)	(Inferred)
• Hugo North Lift 2 block cave	(Inferred)
• Hugo South block cave	(Inferred)
• Heruga block cave	(Inferred)

Figure 1.13 Alternative Production Case Mine Designs



Three expansions to the 2016 Resources Case have been examined in order to confirm the long-term development strategy and timing. The analyses of the plant capacity expansions for Alternative Production Cases were prepared by OreWin with TRQ using the costs and assumptions from the 2016 Resources Case and from Phase 1 of Oyu Tolgoi. The costs include capital for the underground mines, plant and infrastructure. The plant capacities of these Alternative Production Cases are the production schedule averages not nominal capacity. The Alternative Production Cases and plant capacity assumptions are shown in Table 24.1.

Table 1.17 Alternative Production Cases and Plant Capacity Assumptions

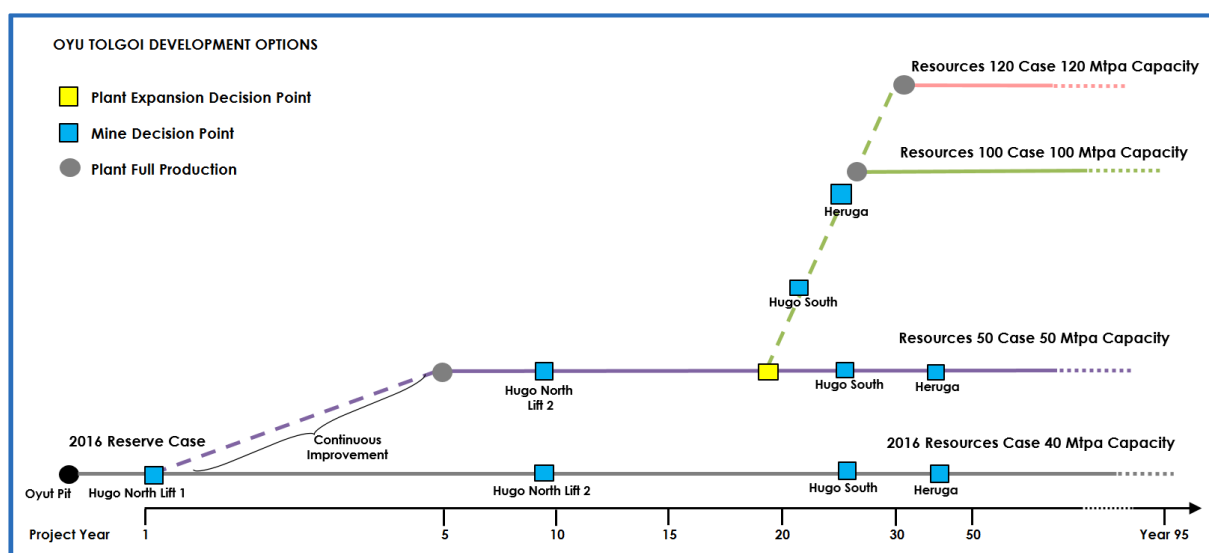
Alternative Production Case	Plant Capacity Assumptions
2016 Resources Case	Plant capacity 40 Mt/a for life.
Resources 50	Plant capacity 40 Mt/a with a 5% improvement in throughput capacity per year for five years to 125% of initial capacity. The average production is 50 Mt/a.
Resources 100	Resources 50 followed by an expansion to 100 Mt/a.
Resources 120	Resources 50 followed by an expansion to 120 Mt/a.

The 2016 Resources Case assumes that the plant capacity remains constant and that the Oyut open pit and Hugo North Lift 1 are followed by Hugo North Lift 2, Hugo South, and Heruga.

The Alternative Production Cases present development options shown in the potential decision tree in Figure 1.14. There are decision points for each of the mines and plant expansions. The development options shown represent possible scenarios. The actual decisions will consider the results of mine and plant performance over the intervening years and will need to identify the optimum timing and size of both mine and any plant expansions that may be applicable.

Following the commencement of development of Hugo North Lift 1, the next decision point for the Oyu Tolgoi project is the development of the Hugo North Lift 2 block cave shown in Figure 1.14 at Year-10. Optimization and utilization of the installed underground haulage capacity of 140 kt/d (50 Mt/a) will need to be considered along with the results of the Hugo North Lift 1 development and cave performance when determining the optimal Hugo North Lift 2 scenario. If the performance of Hugo North Lift 1 is such that it has a faster ramp-up and/or greater final production rate than predicted in OTFS16 and if that performance was to apply to Hugo North Lift 2, then the decision point for plant expansions (shown in Figure 1.14 at Year-20) would be brought forward.

Figure 1.14 Oyu Tolgoi Development Options



Separate capital and operating costs were prepared for the plant, infrastructure and individual mines for the OTFS16 Resources Cases.

For the Alternative Production Cases, separate capital costs estimates were prepared using the 2016 Reserves Case (base case) and OTFS16 Resources Case capital costs for equipment and installation of the plant and infrastructure, factored for each expansion. The mining costs for the underground caves in the Alternative Production Cases were taken from the OTFS16 Resources Case and shifted to suit the timing for the production schedule in each case. Underground capital and operating costs in the Alternative Production Cases are the same as for the OTFS16 Resources Case, except where the underground production rate in the Hugo Dummett deposits is greater than 140 kt/d, in which case costs for additional shafts were added to allow for the increase in the hoisting capacity. Open pit costs, based on the Alternative Production Cases open pit production, were prepared using the unit cost rates and a similar method as the base case. G&A and Operations Support costs were estimated

separately for each Alternative Production Case using the assumptions from the OTFS16 Resources Case and making allowances for changes in personnel numbers and other inputs.

A comparison was made of the 2016 Reserves Case (base case) with the Alternative Production Cases. Four cost sensitivity options were analyzed. Each sensitivity assumes an improvement in the costs and productivities. The improvements could be the result of optimization and efficiencies from the experience that will be gained over the years of developing and operating the plant and mines at Oyu Tolgoi. The cost assumptions are:

- Underground construction capital costs reduced by 30%.
- Operating costs reduced by 15%.
- G&A costs are assumed to reach a long-term average annual cost of US\$50M from Year-7. This cost is based on a review of costs from studies of other copper projects.
- Rail freight available to the project after 2020 and the concentrate freight cost is reduced to US\$25/t.

The after tax NPV8% results of the comparisons are shown in Table 1.18. Option 'A' compares the results using the base case assumptions and Options 'B' through 'E' show the results of applying the sensitivities cumulatively. The results indicate that for the Option 'A' (base case assumptions) there is an improvement in after tax NPV8% for the 2016 Resources Case and that the Resources 50 Case has the highest value. When each of the options is applied cumulatively there is an increase in value and the Resources 100 Case has the highest value for Options 'B' and 'C' and the Resources 120 Case has the highest value for Options 'D' and 'E'.

The after tax NPV8% results based on US\$3.50/lb copper and US\$1,400/oz gold is shown in Table 1.19. Detailed sensitivities for a range of metal prices are presented in Section 24.

The expansion capital costs and variation for each of the options are shown in Table 1.20. These costs include the expansion capital for each new mine and for plant and infrastructure.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

Table 1.18 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.00/lb Copper and US\$1,300/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	6.94	8.37	9.32	8.88	8.80
B	Underground Construction Capital Reduced by 30%	US\$b	7.85	9.64	10.57	10.59	10.51
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	8.97	10.20	11.86	12.00	11.98
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	9.14	10.43	12.20	12.50	12.57
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	9.62	11.02	13.15	13.58	13.69

Note: Based on US\$3.00/lb copper, US\$1,300/oz gold, US\$19.00/oz silver, and 8% discount rate.

Table 1.19 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.50/lb Copper and US\$1,400/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	10.80	12.95	14.36	14.59	14.69
B	Underground Construction Capital Reduced by 30%	US\$b	11.72	14.21	15.62	16.30	16.40
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	12.83	14.78	16.91	17.71	17.87
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	13.00	15.01	17.25	18.21	18.46
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	13.48	15.59	18.20	19.29	19.58

Note: Based on US\$3.50/lb copper, US\$1,400/oz gold, US\$19.00/oz silver, and 8% discount rate.

Table 1.20 2016 Reserves Case and Alternative Production Cases – Expansion Capital Costs

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	4.63	9.73	9.73	13.47	14.86
B	Underground Construction Capital Reduced by 30%	US\$b	4.13	7.69	7.69	11.43	12.82
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	4.13	7.69	7.69	11.43	12.82
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	4.13	7.69	7.69	11.43	12.82
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	4.13	7.69	7.69	11.43	12.82

Notes:

1. Expansion capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. In all cases total capital cost excludes capital costs for the year 2016. Expansion capital for 2016 excluded is US\$0.46b.

1.16 Future Work

Oyu Tolgoi is a very large project that includes five separate deposits. The long-term development of Oyu Tolgoi would involve the development of the resources on all deposits. Alternative Production Cases have been developed to provide early stage analysis of the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second Lift of Hugo North). Development of these deposits will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi.

The extensive resources at Oyu Tolgoi provide an opportunity to continually evaluate investment decisions as the mines progress, deposit knowledge increases and economic conditions change. Updates, revisions and reviews of the Alternative Production Cases will identify opportunities as they arise. This will include the development of the existing resources and exploration of new targets that may improve the cases. Specific areas of future work are described below.

1.16.1 Open Pit Mining

The OyuT open pit design consists of ten mining phases. Phases 1 and 2 are complete. The open pit reserves are mined over approximately 40 years, continuing while underground mining commences with the Hugo North Lift block cave. Various alternative operating scenarios for the open pit are currently being evaluated around pit phase design and sequencing. This evaluation shows strong potential for improving the value of the project, but the associated work streams for these improvements exceeded the time-frames available to finalise OTFS16. The areas of future work in the open pit are:

- Develop a scope of work to implement mining bottom benches.
- Develop a reconciliation of planned versus production actual according to conditions.
- Review the operator efficiency factors with actual performance and equipment productivity and cost rates.
- Review the equipment utilization factors and revise the effective hours and hours used for costing.
- Reconcile the current catch bench widths achieved versus design widths.
- Develop a contingency plan to manage risks due to potential geotechnical instability of slopes for open pit phases.
- Provide better communication of short- and medium-range planning results and planning cycles.
- Modify and evaluate the current configuration of trim blasts as a function of achievable mining of bench toes.
- Focus the mine planning process to exploit production on the gold core of the southwest zone.

1.16.2 Underground Mining

Underground mine design will proceed into detail design work, with the first priority being to support the restart plan. Detailed mine design will continue to be refined, focusing on final access, mass excavations and infrastructure excavation layouts, Panel 0 layout, and related boundaries associated with both geotechnical structures as well as panel interfaces. The following have been identified as key focus areas:

- Reviewing and managing geotechnical risks for the cave; benchmarking drawpoint loss and drawpoint availability, and further work on potential impacts for Oyu Tolgoi.
- Maximizing extraction drive and pillar stability to manage the risk of ground collapse and impact on production ramp-up and reserves. This will include a review of smaller-profile extraction drives and operating equipment and a detailed design review of ground support for on-footprint poor ground areas.
- Improving the understanding and incorporating any impacts of draw and dilution on production schedule. This will include a review of LHD draw strategies for pillar load shedding.
- Analyzing designs and schedules around crossing cave boundaries in detail. This will include a trade-off in parallel of alternate production sequencing to leaving a minable pillar at either end of Panel 0 and a review of Hazmap and identified poor ground areas against design and incorporate improvements.
- Review sequencing of footprint orepass development within cave stress shadow.
- Identify opportunities to de-risk and seek improvements in ramp-up and full production tonnage through increased drawbell construction rate and improved draw management.
- Continue to explore ways of exceeding development rates in critical path areas including access to and initial footprint panel development.
- Review and improve planning tools, systems, processes, and integration. Further work on simulation integration with development and production planning to progress detailed designs, sequencing, and scheduling. Focus on interaction and congestion management to seek improvement opportunities for preproduction and project ramp-up phases.

1.16.3 Metallurgical Process and Plant

Additional metallurgical work is required to advance designs for evolution of the process plant from Phase 1 to Phase 2. These studies will inform both short and long-term project planning. This will include:

- Additional metallurgical work: variability, comminution, tailings, concentrate, flotation, heap leach potential and gravity gold.
- Debottlenecking and incremental capacity studies.
- Concentrator conversion detailed design.
- Coarse flotation studies.

1.16.4 Infrastructure and Logistics

Further studies will continue to create a multi-channel and multi-transport mode solution. In particular, direct rail transport is considered a long-term transportation solution after this initial development period when other non-Oyu Tolgoi development projects are initiated in the region. OT LLC has advised that it expects revisions to the project infrastructure 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 further studies.

1.16.5 Tailings Storage Facility

Ongoing independent quality control and quality assurance will ensure that all earthworks, rockfill placement, and management and monitoring activities are undertaken in accordance with the design and related management and monitoring plans. OT LLC has an ongoing programme of studies to optimize the tailings storage facility (TSF) design. This optimization of the TSF will potentially provide cost savings over the assumptions in OTFS16.

1.16.6 Power Supply Determination

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. In terms of power, the IA includes an overarching commitment from the GOM and OT LLC to work together to determine the most optimal and reliable solutions for power supply. OT LLC will continue to work with the GOM and other stakeholders to evaluate and develop the power requirements for Oyu Tolgoi.

1.16.7 Health and Safety

Oyu Tolgoi has developed a comprehensive Health, Safety, Environment and Communities Management System (HSEC MS) that meets the requirements of the Rio Tinto HSEC Management System Standard and Health, Environment, Safety and Community Performance Standards. The management system is designed on the principles of continual improvement and adopts the methodology of Plan, Do, Check and Review, which comprises 17 discrete elements for implementation. The Oyu Tolgoi HSEC MS has been audited and is certified to ISO14001 and OHSAS18001. Continued focus on Health Safety and Environment

(HSE) is an important activity that will be required to minimize exposure of personnel and the project to risks while maximizing the overall project value.

1.16.8 Water Management

The GOM awarded a water utilization contract to OT LLC until 2040, which may in turn be extended for 20-year periods beyond 2040, in accordance with the Law on Water. OT LLC is currently entitled to utilize water at a rate of 918 L/s. Updated hydrogeological modelling, completed in 2013, and based on all three hydrogeological investigation programmes, demonstrates that the Gunii Hooloi aquifer is capable of providing 1,475 L/s, based on the same time and drawdown conditions.

Studies continue per defined ongoing monitoring and socio-economic programmes developed by OT LLC, specifically with regard to water resource management. 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, if any, on the concentrator expansion on water demand and to determine the need for obtaining approval from the GOM for any substantial increase in the approved water demand from the Gunii Hooloi aquifer.

1.16.9 Innovation and Technology Opportunities

OT LLC plans to investigate and implement projects in these areas: project monitoring, process technology, and underground technology. The innovations and possible applications outlined below are not exhaustive, nor definitive, but rather are currently viewed as having the most significant potential impact for Oyu Tolgoi.

OT LLC plans a longer-term view to developing its operations management capability to maximize performance through evaluation and implementation of advanced technologies. This approach would involve the strategic evaluation and collaboration with technology partners before further developing and implementing system capabilities in a phased and prioritized manner. Experience from a variety of industries has shown this approach to be crucial in achieving an integrated system that maximizes the potential of the various technologies and the benefit to Oyu Tolgoi. The innovation and technology opportunities that Rio Tinto is currently examining are:

- Project Monitoring and Optimization
- Process Technology
- Underground Technology
- Operating Systems and Technologies
- Data Management
- Geotechnical Research
- Extraction Level Construction
- Cave Production
- Cave Monitoring

1.16.10 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.

2 INTRODUCTION

2.1 Issuer for Whom Report Prepared

This report is titled the 2016 Technical Report (2016 OTTR) and has been prepared for Turquoise Hill Resources Ltd (TRQ).

2.2 Terms of Reference and Purpose of Report

The Oyu Tolgoi copper and gold project (Oyu Tolgoi) is located in the Southern Gobi region of Mongolia and is being developed by OT LLC. Oyu Tolgoi consists of a series of deposits containing copper, gold, silver, and molybdenum. The deposits lie in a structural corridor where mineralization has been discovered over 26 km strike length from Ulaan Khud in the north and Javkhlant in the south. The deposits stretch over 12 km, from the Hugo North deposit in the north through the adjacent Hugo South, down to the Oyut deposit and extending to the Heruga deposit in the south.

Independent Qualified Persons (QP), acting on behalf of TRQ, reviewed the available studies as part of the preparation for the 2016 OTTR and in conjunction with TRQ prepared the 2016 OTTR with costs to the end of 2016 as reported to TRQ by OT LLC. The 2016 OTTR used only Measured and Indicated Mineral Resources and is a complete study of all aspects of the project. The 2016 OTTR presents a reserve case (2016 Reserves Case) and is based on a feasibility quality level study complying with NI 43-101. The work of the 2016 OTTR meets the standards of US SEC Industry Guide 7 requirements for reporting Reserves.

2.3 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, October 2011, November 2012, and 28–31 January 2013, 2–14 December 2013, 18–19 March 2014, 27–29 October 2014, 23–27 August 2015, 8–10 December 2015, 23–25 February 2016 and 14–15 June 2016. Meetings were also attended in Ulaanbaatar with OT LLC (formerly IMMI) and Mongolian authorities to discuss the project from 2003–2016. Some of these meetings did not include site visits. Other visits were made to OT LLC offices in Mongolia, Australia, and China as part of work on Oyu Tolgoi.
- Sharron Sylvester visited the property from 28–31 January 2013, 12–14 December 2013, 27–29 October 2014, 23–27 August 2015, 8–10 December 2015 and 23–25 February 2016. Other visits were made to OT LLC offices in Mongolia and Australia as part of work on Oyu Tolgoi.

2.4 Units of Measure and Currency

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

2.5 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. The primary source of information for 2016 OTTR is the study Oyu Tolgoi Feasibility Study 2016 (OTFS16), which was prepared by OT LLC.

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 (report used to affirm the corporate structure and ownership of the licenses related to Oyu Tolgoi):

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 3.0 Ownership and Legal
 - 4.0 Government and Community Relations
- Emails from OT LLC Legal department, including edits to Sections, 1, 4, and 6 of the 2016 OTTR drafts.

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

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 3.0 Ownership and Legal
 - 5.0 Human Resources and Capability Development
 - 7.0 Environment
 - 8.0 Water Management

Reports used in Section 19 Market Studies and Contracts:

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 17.0 Marketing

Reports used in Section 20, Environmental Studies, Permitting, and Social or Community Impact:

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 3.0 Ownership and Legal
 - 7.0 Environment
 - 8.0 Water Management

Reports used in Section 22, Economic Analysis:

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 3.0 Ownership and Legal
 - 4.0 Government and Community Relations
 - 21.0 Risk Assessment

Reports used in Section 24.1.4, Risk Assessment:

- OT LLC 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016. Sections:
 - 21.0 Risk Assessment

4 PROPERTY DESCRIPTION AND LOCATION

References to Mongolian laws and regulations in this section are to the laws and regulations as they existed as at 22 April 2016.

4.1 Property Ownership and Boundaries

The project area comprises five contiguous licenses, as listed in Table 4.1.

Table 4.1 Contiguous Properties of the Project Area

License Number as at November 2010	Area (ha)	Legal Owner
MV-006708	4,533	OT LLC
MV-006709	8,490	OT LLC
MV-006710	1,763	OT LLC
MV-015225 (Javkhlant)	20,327 all under agreement	Entrée LLC (a subsidiary of Entrée Gold Inc.)
MV-015226 (Shivee Tolgoi)	42,592.58 all under agreement	Entrée LLC (a subsidiary of Entrée Gold Inc.)

OT LLC has an economic interest in MV-015225 and MV-015226 pursuant to an Equity Participation and Earn-in Agreement with Entrée (as amended). This agreement contemplates the establishment of a joint venture between OT LLC and Entrée that provides for OT LLC to hold legal title in MV-015225 and MV-015226, subject to the terms of the agreement and to OT LLC meeting prescribed earn-in expenditures.

The vast majority of the identified mineralization for the project occurs at the Hugo Dummett and Oyut porphyry deposits within the licenses held directly by OT LLC. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT). The northernmost extension of the Hugo North deposit crosses onto the Shivee Tolgoi Property (Hugo North Extension). The Heruga deposit lies almost entirely within the Javkhlant.

Property, with only the northern extent passing into MV-06709. There are numerous exploration targets across MV-006708, MV-006709, MV-015225, and MV-015226.

On 23 December 2003, OT LLC was granted 100% registered title to MV-006709 (OT License) in accordance with the Minerals Law of Mongolia for a term of 60 years, with an option to extend the license for an additional term of up to 40 years. In 2006 the Mongolian Parliament passed a new Minerals Law that changed the term of mining licenses to 30 years with two 20-year extensions. The IA is effective for an initial term of 30 years and an extension of a further 20 years.

The boundary coordinates of MV-006709 are defined by latitude / longitude and also by UTM coordinates (Table 4.2).

Table 4.2 Boundary Coordinates of Oyu Tolgoi Mining License MV-006709

Point	Latitude / Longitude, WGS-84		UTM, WGS-84, Zone 48N	
	Latitude	Longitude	Northing	Easting
1*	42°58'31" N	106°55'01" E	4,759,852.29	656,311.72
2	42°58'31" N	106°47'31" E	4,759,627.32	646,118.71
3	43°03'01" N	106°47'31" E	4,767,956.25	645,940.95
4	43°03'01" N	106°55'01" E	4,768,181.26	656,121.56

Note: * Point 1 corresponds with the south-east corner of the OT License, then moving clockwise.

An exploration license is valid for a 3-year period with three 3-year extensions, for a total of 12 years. Prior to expiry of the exploration license, application can be made for conversion to a mining license.

On 17 May 2006, an expert group established by the GOM recommended registration of the project's open pit reserves under Mongolian guidelines; as part of its conclusions, the expert group confirmed that Ivanhoe Mines Mongolia Incorporated (IMMI, precursor to OT LLC) had title to the Oyu Tolgoi license MV-006709.

On 1 July 2009, a new experts group and the Minerals Council recommended to the Mineral Resources Authority of Mongolia that the Oyu Tolgoi Commercial Minerals be registered. "Commercial Minerals" include Mongolian Mineral Resources and Mongolian Mineral Reserves and can only be reported in Mongolia by registration by the Mineral Resources Authority of Mongolia (MRAM). The Minerals Council recommended that the OT LLC licenses be acknowledged and that the Shivee Tolgoi 3148X and Javkhlant 3150X exploration licenses should be converted to mining licenses. This has been completed; the new license numbers are Shivee Tolgoi MV-015226 and Javkhlant MV-015225. The IA describes the exploration and mining licenses relating to the Oyu Tolgoi project and confirms OT LLC's interest in these licenses.

In early-2011, the GOM changed its official survey datum to WGS-84 Zone 48N. In accordance with the requirements of the change, Geomaster Co. Ltd. resurveyed the licenses and new license certificates reflecting the slight change from prior surveys were issued to OT LLC.

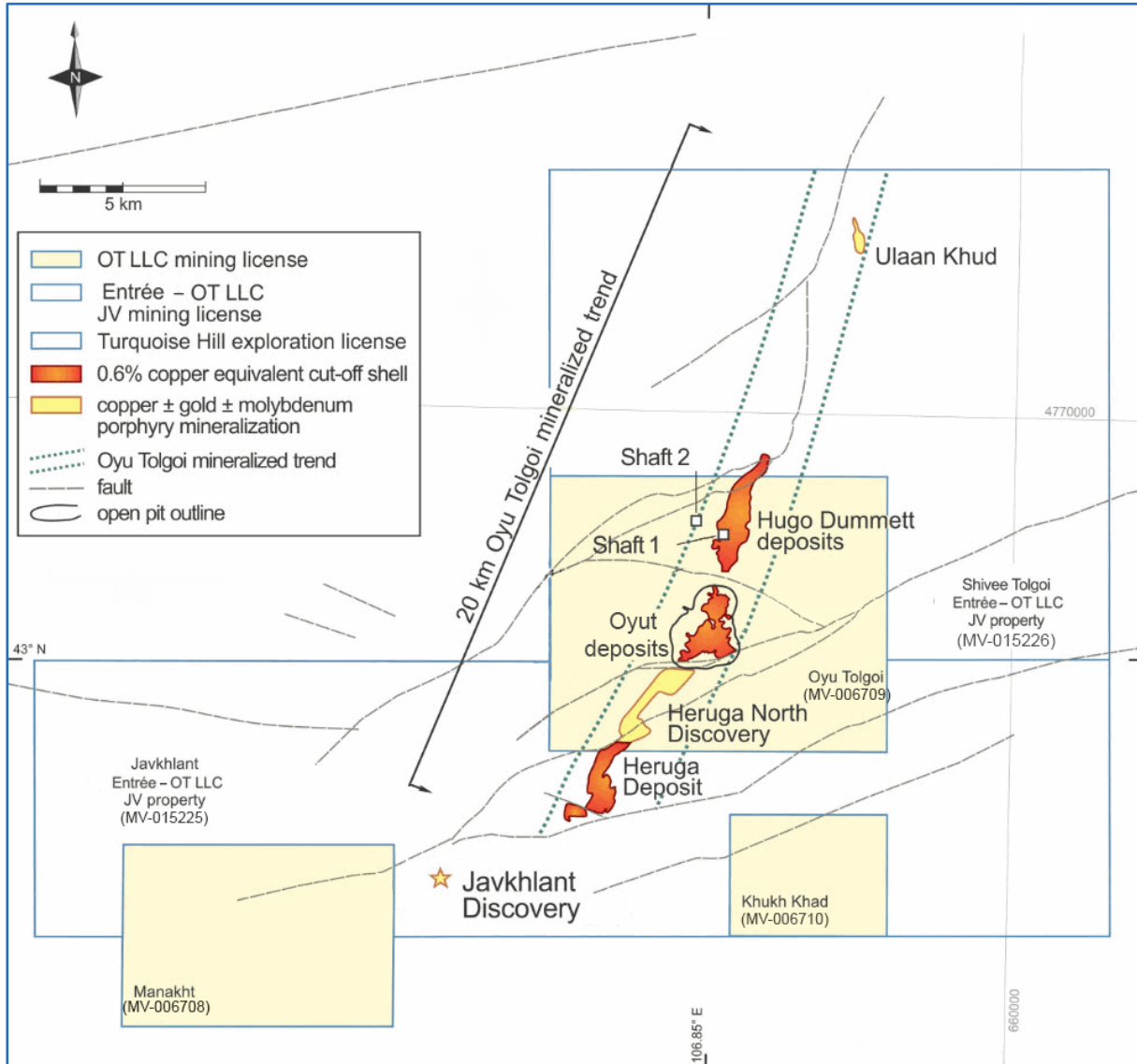
4.2 Entrée-OT LLC Joint Venture Property

In November 2004, an Equity Participation and Earn-In Agreement was finalized between Ivanhoe Mines Ltd (Ivanhoe) (subsequently renamed Turquoise Hill Resources Ltd (TRQ)) and Entrée, giving TRQ the right to earn an interest in a portion of Entrée's Shivee Tolgoi project property. The portions of Entrée's Shivee Tolgoi project property subject to the Equity Participation and Earn-In Agreement included:

- Javkhlant: specifically, the whole of the Javkhlant exploration concession (then identified as Claim No. 3150X, now identified as Mining License MV-015225), and
- Shivee Tolgoi: specifically, the portion of the Shivee Tolgoi exploration concession (then identified as Claim No. 3148X, now identified as Mining License MV-015226) that is located to the north and to the east of the OT License (MV-006709).

The Javkhlant and Shivee Tolgoi areas subject to the Equity Participation and Earn-In Agreement are shown in Figure 4.1.

Figure 4.1 Oyu Tolgoi Project Land Tenure



At the end of June 2008, TRQ notified Entrée that it had incurred sufficient expenditures (>US\$35 million) to earn an interest in the EJV property under the terms of the Agreement. While a formal joint venture agreement has not been entered into yet, the earn-in requirements have been met, and OT LLC's participating interest in the joint venture (including the licenses) will be:

- In respect of the proceeds from mining from the surface to 560 m below the surface, 70%, and
- In respect of the proceeds from mining from depths beneath 560 m, 80%.

In October 2009, Entrée received notice that its application for conversion of the Shivee Tolgoi and Javkhlant exploration concessions to mining licenses had been approved by the MRAM, a

condition precedent to the IA. It was a condition of the IA that the rights held by TRQ in respect of MV-015225 and MV-015226 were to be transferred to OT LLC; this was effected by an assignment agreement dated 1 March 2005 executed by TRQ in favor of OT LLC.

The property subject to the Equity Participation and Earn-In Agreement is now referred to as the Entrée–OT LLC Joint Venture (EJV) property. Details of the EJV licenses are provided in Table 4.3.

Table 4.3 Details of Javkhlant and Shivee Tolgoi EJV Property

Mining License	Mining License Name	Area (ha)	License Date	Date of License Extension
MV-015225	Javkhlant	20,327	27/10/2009	27/10/2039
MV-015226	Shivee Tolgoi	19,478	27/10/2009	27/10/2039
Total		39,805		

Between February and October 2013, the status of mining licenses MV-015225 and MV-015226 was subject to some uncertainty based on communications by the GOM. These licenses were considered to be under suspension for a temporary period by the MRAM, during which time rights to sell or transfer the licenses were restricted. This period coincided with GOM concerns about the nature of its economic and legal interest in these licenses through its 34% interest in OT LLC. The nature of the GOM's economic and legal interest in these licenses is the subject of discussions with Entrée.

On 1 October 2015, a License Fees Agreement was signed between OT LLC, Entrée Gold Inc., and Entrée LLC, under which the parties have agreed to negotiate in good faith to amend the EJV documents to include the western area of MV-015226, thereby giving OT LLC an economic interest in this area similar to its existing economic interest in MV-015226 and MV-015225. In return, OT LLC has agreed to meet the annual license fees (US\$15/ha) for the new area included in the OT LLC–Entrée arrangements.

4.3 Investment Agreement (IA)

In October 2009, Ivanhoe, OT LLC, and Rio Tinto International Holdings Limited signed an IA with the GOM. The IA defines the fiscal and regulatory environment under which the project will operate and stipulates that the GOM owns 34% equity of OT LLC, with the option to increase its equity by acquiring a further 16% after the IA is extended beyond its initial 30-year term, on terms agreed between the GOM and TRQ. At the time of signing the IA, there remained a number of conditions precedent to it becoming effective. On 31 March 2010, it was announced that these conditions had been met or waived, and this was confirmed by the GOM. The main conditions that were met or waived were as follows:

- The Mongolian feasibility study of the Oyu Tolgoi project has been considered and submitted in accordance with the laws and regulations of Mongolia.
- The balance of existing income tax losses, capitalized expenses, and outstanding tax liabilities or credits has been confirmed by the tax office.
- The balance of existing shareholder loans has been agreed upon.

- Company restructuring required to execute the agreement has been completed.
- A standing working committee has been established with members of the GOM and OT LLC to provide a means to expedite permits, customs clearance, or general GOM administration.
- TRQ's interests in mining licenses MV-015225 and MV-015226, held by Entrée Gold LLC, have been transferred to OT LLC. Although the licenses themselves have yet to be transferred to OT LLC, the condition precedent is fully satisfied and TRQ has transferred its existing economic interests in the licenses and its rights under the Equity Participation and Earn-in Agreement with Entrée, to OT LLC.

TRQ agreed with the GOM in connection with entering into the Underground Development and Financing Plan (UDP) signed on 18 May 2015 that it had no entitlement to receive any payment from OT LLC for the 2% NSR interest TRQ had acquired from BHP (now BHP Billiton) in November 2003. This NSR entitlement arose out of TRQ's purchase of BHP's original exploration license (MEL) and all future entitlements covering what is now MV-006709.

4.3.1 Funding and Taxation

The 2006 Minerals Law provided for a mining license holder investing US\$50M or more in the first five years of a project to enter into an investment agreement with the GOM that regulates aspects of the investment environment, particularly taxation rates. The IA gives OT LLC confidence in the terms the project will operate under. The IA has an initial term of 30 years and may be extended for a further period of 20 years, provided certain conditions are satisfied, including a provision that OT LLC makes capital cost expenditures of at least US\$9 billion prior to issuing an extension notice at least 12 months prior to the end of the initial 30-year term. The term of a mining license under the Minerals Law is for 30 years with two 20-year extensions.

In conjunction with the IA, a Shareholders' Agreement was entered into between OT LLC, the subsidiaries of TRQ that hold shares in OT LLC (TRQ Shareholders), and the GOM shareholder, Erdenes Mongol LLC (Erdenes), on 6 October 2009. The agreement was subsequently amended and restated (the Amended and Restated Shareholders' Agreement, or ARSHA) on 8 June 2011. The ARSHA outlines the rights and obligations between the shareholders of OT LLC, including obligations to provide funding to OT LLC to develop the project.

The GOM has the right to increase its shareholding in OT LLC to 50% if the IA is extended at the end of the initial 30-year term (as discussed above) on terms to be agreed with the TRQ Shareholders. This right is set out in Article 1.6 of the IA, which essentially provides that:

- The State will own 34% of the shares in Oyu Tolgoi.
- After the IA is extended for an additional 20 years the State will have an option to acquire a further 16% interest.

Accordingly, the GOM is only entitled to increase its stake in the Oyu Tolgoi project after having (first) agreed to the terms of the acquisition with TRQ.

During the period ending September 2016, TRQ subsidiaries provided for 100% of OT LLC's funding needs. In accordance with the terms of the UDP, TRQ has agreed that until 1 September 2021, it will fund the whole or part of any Erdenes OT LLC sum relating to capital or operating costs of the underground stage.

Until 1 September 2021 the TRQ shareholders have the right to determine whether funding provided to OT LLC is made in the form of shareholder debt or equity. However, Mongolian "thin capitalization" laws create adverse tax consequences if OT LLC's debt-to-equity ratio exceeds 3:1. The ARSHA also imposes certain restrictions upon the amount of equity funding that is contributed during the Funding Period.

In the case of shareholder debt, pre-IA loans made to OT LLC by shareholders attracted an effective annual interest rate of LIBOR plus 9.9% US CPI adjusted. Since 31 January 2011, the rate was decreased to LIBOR plus 6.5%.

The Project Financing package closure was completed in 2016 with 100% of project finance net proceeds and operating cash flow from the Oyut open pit used to fund underground development. Based on the study assumptions it is anticipated that additional supplemental debt, up to the senior debt cap of US\$6b, will be required to complete development. All project finance debt is forecast to be repaid by 2030.

The financing package includes a completion guarantee underwritten by Rio Tinto. For taking the risk of this completion guarantee, Rio Tinto will receive a fee based on the average outstanding annual debt until project completion, projected to be in the mid 2020's. This fee will be serviced from project cash flows.

The full US\$4.4b of project finance funds available were drawn down upon closing with net funds (post payment of fees) used to temporarily repay shareholder loans and deposited with Rio Tinto. All funds deposited will be invested back into the project to fund underground development as required. This recycling mechanism enables Oyu Tolgoi to reduce financing costs by repaying higher cost shareholder loans and replacing them with less-expensive project financing, thus avoiding paying commitment fees on any undrawn project finance facilities. In addition, the Rio Tinto guarantee fee will be waived for all amounts on deposit, reducing the effective rate of the guarantee.

Under the ARSHA, the TRQ Shareholders have appointed Rio Tinto OT Management Limited (RTOTM) to provide strategic and operational management. This appointment attracts a payment equal to 3% of total operating and capital costs prior to commencement of production (September 2013) and 6% of operating and capital costs during production. Under the terms of the UDP the rate to be applied to the capital costs of developing the Underground Stage shall be 3% instead of 6%.

The IA requires OT LLC to achieve commencement of production within seven years after 31 March 2011, which it did on 1 September 2013.

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 as they applied as at the date of the IA. The taxes and fees payable to the GOM and their rates, include:

- Corporate income tax 25%
- Mineral royalties 5% (sales value)
- Value added tax 10%
- Customs duties 5%
- Withholding tax 20%

In accordance with the Windfall Profits Tax (WPT) invalidating law, as from 1 January 2011, OT LLC will not be subject to the WPT.

OT LLC is also only subject to those taxes listed in the General Taxation Law as at the date of the IA and not taxes introduced at any future date.

In 2009, the GOM enacted amendments to the legislation governing the carry-forward of income tax losses. These terms are incorporated into the IA. The loss carry-forward period has been extended to eight years and, if sufficient, can be applied to offset 100% of taxable income. The IA also provides OT LLC with the benefit of a 10% tax credit for all capital investment made during the construction period.

By signing the UDP and related documents, OT LLC and its shareholders have agreed on key outstanding taxation matters including:

- (a) calculation and the amount of OT LLC's investment tax credits and losses carried forward;
- (b) the calculation of royalties (to be calculated on a gross rather than a net sales proceeds basis without deduction of costs for processing including treatment and refining charges), freight differentials, penalties, or payables; and
- (c) double taxation treaty stabilization and stabilized rate of the withholding tax.

Pursuant to the UDP, the OT LLC shareholders and the GOM have agreed that international market prices shall be used in determining sales values to calculate the mineral royalties' payable. The UDP sets out the applicable international markets for determining the sales value of copper, gold, and silver and makes provision if these market pricing mechanisms cease to be available.

4.3.2 Social and Environmental Impacts

The IA also addresses social and environmental impacts, mitigation, and management as it applies to OT LLC's "Core Operations," which are defined as comprising mineral exploration and mining activities in the "Contract Area" (which includes the area covered by mining licenses MV-006709, MV-006708, and MV-06710) and all other connected activities.

Regional Development

Clause 4 of the IA provides that the GOM will establish the South Gobi Regional Development Council (Council) and lead its activities. The Council will be governed by a board that will include representatives of the GOM, local governance organizations, private sector entities, civil society organizations, and donor and international financial organizations with activities directed toward the Southern Gobi region. OT LLC is a member of the Council's governing board and is obliged to support the Council and its activities.

In 2015, OT LLC signed a Cooperation Agreement with the Khanbogd soum, the Ömnögovi aimag, and other local communities, formalizing the contributions that OT LLC will make in the coming years to support local social and economic development. As well as allocating US\$5M/a for community development, the agreement sets out how the partners will work together to ensure that local people continue to benefit, sustainably over the long term, from Oyu Tolgoi.

Environment

Many of the environmental provisions contained in the IA track the requirements of the Air Pollution, Energy and Water Law, and OT LLC gives a contractual commitment to comply with these laws. OT LLC agrees to comply with the international environmental law treaties to which Mongolia is a party and the Minerals Law provisions including detailed environmental impact assessment reports in accordance with the Law on Environmental Impact Assessment.

OT LLC has the right to access and use its self-discovered water resources for purposes connected with the Mine Project, including to construct, commission, operate, and rehabilitate the Mine Project. Additionally, OT LLC will support the GOM in the supply of safe drinking (may require treatment) to the local soum communities affected by the Mine Project.

OT LLC will pay fees to the GOM for surface and underground water removed. The rate approved by Cabinet (MNT 959.04/m³) was the rate paid in 2015.

The GOM awarded a water utilization contract to OT LLC until 2040, which may in turn be extended for 20-year periods beyond 2040, in accordance with the Law on Water. OT LLC is currently entitled to utilize water at a rate of 918 L/s.

Infrastructure

The infrastructure provisions of the IA focus on power and transportation. In terms of power, the IA includes an overarching commitment from the GOM and OT LLC to work together to determine the most optimal and reliable solutions for power supply.

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. The IA includes an overarching commitment from the GOM and OT LLC to work together to determine the optimal and most reliable solutions for power supply.

Under the IA, OT LLC is required to secure its power requirements from within Mongolia within four years of commencement of production. However, this timeframe is currently suspended pursuant to the Southern Region Power Sector Cooperation Agreement entered into by OT LLC and the GOM on 14 August 2014 that proposes an independently funded and operated coal fired power plant at Tavan Tolgoi (TTPP Project).

So long as OT LLC continues to participate in the TTPP Project, the four-year timeframe for sourcing Mongolian power will be suspended. Upon a withdrawal from the TTPP Project by either OT LLC or the GOM, the four-year timeframe will be reinstated and recommence from the date of withdrawal. A request for proposals from potential investors in the TTPP coal fired power plant culminated in a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

OT LLC sources its present power under a four-year contract with a Chinese provider, the Inner Mongolia Power International Cooperation Company Ltd. (IMPIC) via the Mongolian National Power Transmission Grid (NPTG) authority. In May 2016 the parties agreed via a non-

binding Memorandum of Understanding (MoU), which captured key agreed principles of the new Power Purchase Agreement (PPA), to extend the power supply agreement to at least 2021. OT LLC and the GOM have agreed under the Power Sector Cooperation Agreement that the GOM will assume responsibility for securing the extension of the power import arrangements through its national grid company NPTG. The agreed MoU includes comparable power pricing to the current agreement. OT LLC is endeavoring to execute binding agreements with the GOM and IMPIC within 2016.

OT LLC has the right to construct, manage, and use an aerodrome in connection with the project, based on permits issued in accordance with Mongolian law. A permanent domestic airport, capable of servicing Boeing 737-800 series aircraft, has been constructed at Oyu Tolgoi to support the transportation of people and goods to the site from Ulaanbaatar. It further serves as the regional airport for Khanbogd soum.

The GOM may construct or facilitate the construction and management of a railway in the vicinity of the project to the China-Mongolia border. The GOM will consult with OT LLC on the location and route of the railway, and, if the railway is constructed, then it will be made available to OT LLC on commercial and non-discriminatory terms. Energy Resources is currently constructing a single-track heavy-haul rail from its Ukhua Khudag coal mine (approximately 120 km to the north-west of Oyu Tolgoi) to Gashuun Sukhait, ultimately to be interconnected with the Chinese rail network at Ganqimaodao on the Chinese side of the border. Once constructed, the South Gobi Rail alignment would pass within 10 km of the Oyu Tolgoi project area and therefore represents an opportunity for eventual connection of the mine to the rail network.

OT LLC also has the right to construct roads for the transport of its product. A gravel road has been constructed to the town of Khanbogd and is being maintained. OT LLC intends to construct a paved road from the mine site to the town of Khanbogd. A 105 km sealed road is being constructed to the Chinese border crossing at Gashuun Sukhait, with sealing of the entire road expected to be completed in 2017. On the Chinese side of the border a provincial road connects the border town of Ganqimaodao with the Jingzang Expressway via the towns of Hailiutu and Wuyuan. On 8 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 Oyu Tolgoi. These State special needs areas are to be used for infrastructure facilities necessary in order to implement the development of Oyu Tolgoi.

Most of the 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 government authorities in respect of such licenses. OT LLC has entered into certain consensual arrangements with some of the 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 terms of Resolution 175 and the Minerals Law (2006).

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.

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 but may be significant. The description of Resolution 175 has been provided by OT LLC and has been relied on under Item 3 of NI 43-101 Reliance on Other Experts.

In April 2015, the Standing Committee of the Parliament of Mongolia requested the GOM to modify Resolution 175 due to an alleged inconsistency between Resolution 175 and the Minerals Law and Land Law. OT LLC understands that the GOM supports the validity and justification for Resolution 175 and that Resolution 175 will not be modified or revoked.

The GOM has resolved to resume land to support infrastructure needs of the project under Government Resolution 175 and the amended Land Law. Despite this legislation, a former tenement holder has won a court case against the GOM regarding a resumed tenement. OT LLC understands that the GOM supports the validity and justification for Resolution 175 and is looking at potential solutions to resolve the matter without affecting OT LLC's interests.

Labor Relations, Employment, and Training

Many of the labor and employment provisions contained in the IA reflect current Mongolian laws, and OT LLC gives has undertaken under the IA to comply with relevant labor and employment laws of Mongolia. During the term of the IA, OT LLC, and the GOM will cooperate to ensure that a suitably qualified workforce is available to meet the timeframe of the Mine Project.

The GOM will provide support as requested by OT LLC to facilitate and expedite the granting of all permits necessary for the engagement of foreign nationals as part of the workforce.

4.4 Rio Tinto Agreements

Since 2006, Rio Tinto International Holdings Limited (Rio Tinto) has entered into a number of agreements to secure and consolidate joint ownership, funding and management control of the Oyu Tolgoi project development and operations. A summary of significant agreements is included in Table 4.4. Copies of the significant referenced agreements have been filed on SEDAR and can be accessed at www.sedar.com.

Table 4.4 Key Rio Tinto Agreements

Date	Agreement Description	Terms, Provisions and Outcomes
18-Oct-2006	Private Placement Agreements	• Rio Tinto to acquire 19.7% of TRQ • Subsequent warrants and purchases Rio Tinto increased in TRQ shareholding to approximately 29.6%. • Formation of a joint technical committee to oversee and approve Oyu Tolgoi project development
08-Dec-2010	Heads of Agreement	• Rio Tinto to provide US\$1.8b interim funding • Establishment of a joint "Operating Committee" • Potential Appointment of a Rio Tinto affiliate to manage the Oyu Tolgoi project development
17-Apr-2012	Memorandum of Agreement (MoA)	• Rio Tinto to provide US\$1.5b of bridge financing and standby commitment to US\$1.8b rights funding by TRQ • Rio Tinto to support funding including guarantees of certain TRQ funding obligations • Formation of a new 13 member TRQ board.
28-Jun-2013	Short-Term Bridge Funding Agreement	• Rio Tinto to make available to TRQ a convertible non-revolving term credit facility of up to US\$225M
7-Aug-2013; 23 Aug 2013	New Bridge Funding Agreement and 2013 MoA	• Rio Tinto and TRQ to amend the short-term bridge funding agreement to extend the Maturity Date to 28 August 2013 • Rio Tinto to provide TRQ a new secured bridge funding facility in the amount of US\$600M • If required, TRQ agreed to conduct a further rights offering prior to 30 December 2013 to cover interim funding and bridge facility amounts with a Rio Tinto standby commitment • TRQ agreed to enter into a general security agreement with Rio Tinto and reimburse Rio Tinto associated costs
15-Dec-2015	Project Finance Facility and Financing Support Agreements	• Syndicated US\$4.4b project finance facility intended for the development of the underground mine of the Oyu Tolgoi project with Rio Tinto completion support. • TRQ entered into a number of agreements in connection with the project financing, including: a financing support agreement with Rio Tinto, a financing support agreement with OT LLC and Rio Tinto, a cash management services agreement with affiliates of Rio Tinto, and a sponsor debt service undertaking agreement pursuant to which, among other things, Rio Tinto provided completion support to the project lenders and TRQ agreed to be subject to certain covenants with Rio Tinto.

4.5 Mongolian Legal Requirements

The construction- and operation-related laws of Mongolia of relevance to the Oyu Tolgoi project are summarized below.

Construction Law

The purpose of the Construction Law is to regulate relations arising out of the design development, construction, and maintenance of buildings and facilities; the production of construction materials; and the execution, use, and monitoring of construction work. It sets out general requirements for constructing all types of engineering-designed facilities including roads, concentrators, power, communications services, water facilities, aerodromes, shafts, open and underground mines, offices, and residential buildings.

Law on Sending Workforce Abroad and Receiving Foreign Workforce and Specialists from Abroad.

This law sets out the general requirements for the employment of foreign nationals in Mongolia. The major requirements are:

- obtaining a work permit from Centre for Employment Service, Ministry of Labor
- payment of workplace fees
- meeting the percentage of foreign workforce that may be employed each year in a business entity.

Energy Law of Mongolia

The Energy Law sets out requirements and procedures for obtaining energy-related licenses from the Energy Regulatory Committee, including generation, transmission, distribution, dispatching, supply, import, and export licenses, and licenses for construction of energy facilities. The same law also sets out the powers of the GOM, the Ministry of Energy, and Inspection Agencies to manage, supervise, and control energy-related activities, including the rate structure and tariffs.

Law on Auto Roads of Mongolia

The purpose of this law is to regulate construction, maintenance, financing, use, and monitoring of roads and road facilities. Powers of the GOM and inspection agencies for the management, supervision, and control over road-related activities are also involved under the Law on Auto Roads.

Occupational Health and Safety Rules

This incorporates specific mining related rules covering open pit and underground mining, mineral processing, storage and manufacture of explosives, temporary and permanent closure of mines, plus more general standards relating to labor safety. Further laws (Law of Mongolia on Health; Law of Mongolia on Hygiene) define national principles and standards.

Law of Mongolia on Nuclear Energy

This law regulates the use of radioactive minerals and nuclear energy for peaceful purposes, ensuring the safety of nuclear facilities and nuclear and radiation sources, and protecting individuals, society, and the environment against the detrimental effects of ionizing radiation within the territory of Mongolia.

Labor Law of Mongolia

The purpose of this law is to determine the rights and duties of employers and employees with regard to collective agreements, collective bargaining, collective and individual labor disputes, labor conditions, terms and conditions of work, liabilities for breach of the legislation, and equality of parties in employment relations.

Environmental Legal Requirements

Applicable national environmental laws and regulations for the project are listed below:

- Law of Mongolia on Environmental Protection
- Law of Mongolia on Special Protected Areas
- Law of Mongolia on Buffer Zones
- Law of Mongolia on Animal
- Law of Mongolia on Natural Plants
- Law of Mongolia on Forestry

Land and Land Use

Applicable laws include:

- The Constitution
- Law on Land
- Law of Mongolia on Subsoil
- Minerals Law

Law of Mongolia on Water

This revised law regulates the protection, use, and recovery of water and its basins. It sets out a regime for an inventory of water resources and outlines the powers bestowed to, and responsibilities required, across all levels of government within Mongolia with regard to water matters.

Air and Air Pollution Fee Laws

Applicable laws include:

- The Law on Air
- Law on Air Pollution Fee

Waste and Hazardous Substances Law

Applicable laws include:

- Law on Waste
- Law of Mongolia on Hazardous and Toxic Chemicals

Law of Mongolia on the Protection of the Cultural Heritage

The purpose of this law is to regulate the collection, registration, research, classification, evaluation, preservation, protection, promotion, restoration, ownership, possession, usage, and advertisement of cultural heritage.

Law of Mongolia on the Natural Resource Fees

Under this law, certain fees are required to be paid for the use of natural resources such as water, mineral water, timber and fuel wood, land, animals, and natural plants.

4.6 Mongolian Environmental Quality Standards

Environmental quality and health and safety standards (Mongolian National Standards or MNS) relating to the Oyu Tolgoi project are provided in the ESIA (July 2012). These standards govern compliance with Mongolian national requirements. The specific standards within these regulations, and the policies and procedures adopted by OT LLC to address and ensure compliance with them, are discussed further in the ESIA.

4.7 International Agreements

The GOM has promulgated specific laws implementing specific international agreements and has also incorporated provisions in national laws indicating that where the national law is inconsistent with international agreements to which Mongolia is a signatory, the international laws prevail. Accordingly, the terms of international agreements to which Mongolia is a party need to be understood in the context of the project.

4.7.1 Environmental Impact Assessment

UNECE Convention on Environmental Impact Assessment in the Trans-Boundary Context (Espoo Convention)

The Espoo Convention sets out the obligations of parties to assess the environmental impact of certain activities at an early-stage of planning. It also lays down the general obligation of States Parties to notify and consult each other on all major projects under consideration that are likely to have a significant adverse environmental impact across boundaries.

UNECE Kyiv SEA Protocol to the Espoo Convention

This protocol aims to ensure that environmental considerations are taken into account in an integrated manner to inform governments' strategic decision-making to support environmentally sound and sustainable development. This protocol also provides for extensive public participation in the governmental decision-making process.

4.7.2 Protection of Flora and Fauna

Convention on the Conservation of Migratory Species of Wild Animals (CMS)

This convention aims to conserve terrestrial, marine, and avian migratory species throughout their range. Under the framework of the CMS, a number of agreements and Memoranda of Understanding (MoU) have been entered into, focusing on specific endangered species. Mongolia has ratified the CMS and is signatory to a number of agreements and MoUs.

4.7.3 Biodiversity and Sustainable Development

International Convention to Combat Desertification

The objective of this convention is to combat desertification and mitigate the effects of drought in countries experiencing serious drought and/or desertification through effective action at all levels and supported by international cooperation.

Convention on Biological Diversity

This convention focuses on promoting sustainable development and the conservation of biological diversity. It requires the development of government strategies to incorporate principles of sustainable use of biological resources into national planning activities, but it does not impose any direct standards applicable to OT LLC.

4.7.4 Energy and Climate Change

United Nations Framework Convention on Climate Change (UNFCCC)

The convention provides a framework for negotiating specific international treaties (called protocols) that may set binding limits on greenhouse gases. Therefore, this convention itself has no impact on OT LLC.

Energy Charter Treaty

The treaty focuses on promoting transparency and efficiency in the operation of energy markets and includes provisions addressing the protection of foreign investment; non-discriminatory conditions for trade in energy; the resolution of disputes; and the promotion of energy efficiency and reduction of environmental impact resulting from energy production and use.

Energy Charter Protocol on Energy Efficiency and Related Environmental Impacts

This protocol defines policy principles to promote energy efficiency and reduce the adverse environmental impacts of energy systems. The provisions of this charter form part of the law of Mongolia.

4.7.5 Ozone Depleting Substances

Vienna Convention and Montreal Protocols

The Vienna Convention outlines States' responsibilities for protecting human health and the environment against the adverse effects of ozone depletion in the stratosphere and establishes the framework under which the Montreal Protocol was negotiated.

4.7.6 Hazardous Substances

Rotterdam Convention on the Prior Informed Consent Procedure for Certain Hazardous Chemicals and Pesticides in International Trade

The convention covers pesticides and industrial chemicals that have been banned or severely restricted for health or environmental reasons by parties to the convention and which have been notified by Parties for inclusion in the Prior Informed Consent Procedure.

Stockholm Convention on Persistent Organic Pollutants

The objective of this convention is to protect human health and the environment from chemicals that remain intact in the environment for long periods, become widely distributed geographically, and are bio-accumulative in humans and wildlife.

4.7.7 Waste

Convention on the Trans-Boundary Movement of Hazardous Wastes and their Disposal

The convention includes a number of principles, including environmentally sound management and the disposal of waste close to its source.

4.7.8 Noise

Convention on International Civil Aviation, Annex 16 – Aircraft Noise

Annex 16 to the 1944 Convention on International Civil Aviation (CICA) deals with the protection of the environment from the effects of aircraft noise.

4.7.9 Tangible and Intangible Cultural Heritage

Convention Concerning the Protection of World Culture and Natural Heritage

This convention focuses on the identification, protection, conservation, presentation, and transmission to future generations of cultural and natural heritage.

Convention on the Safeguarding of Intangible Cultural Heritage

This convention focuses on protecting intangible cultural heritage.

4.7.10 Livestock Production

Agreement for the Establishment of a Regional Animal Production and Health Commission for Asia and the Pacific

The purposes of the commission include the promotion of livestock development; building up national and regional livestock programmes; promoting livestock production as an industry; and raising the level of nutrition and standard of living of small farmers and rural communities.

4.8 Corporate Policies

In June 2015, OT LLC adopted the updated Rio Tinto Groups' Code of Business Conduct (The Way We Work). It sets out the values and principles by which the Rio Tinto Group, including OT LLC, conducts its operations including the health and safety of employees, best practice environmental management, contributing to sustainable communities, and always doing business with integrity, for the benefit of all the project shareholders and the people of Mongolia.

OT LLC's Code of Business Conduct and Ethics and the Corporate Securities Trading Policy reflect OT LLC's commitment to honesty, integrity, and accountability, and outline the basic principles and policies that all company employees are expected to abide by.

OT LLC has obtained certification to ISO 14001:2004 (Environmental Management System) and AS/NZS ISO OHSAS 18001:2007 (Occupation Health and Safety Management System) Standards. OT LLC undergoes annual internal and third party audits to maintain certification to these standards and to ensure continuous improvement of its health, safety, and environmental performance.

OT LLC has also adopted a comprehensive set of policies and procedures to address the treatment of employees in the OT LLC workforce, including aspects such as recruitment and employment, human rights, employee working conditions and benefits, performance management, drugs and alcohol, and employee training.

4.9 Import and Export Regulations

The GOM determines types and codes of “finished mining products.” Under the VAT Law of Mongolia (2006) and the VAT Law (2015), which becomes effective from 1 January 2016, VAT will be assessed at the rate of 0% (zero percent) for finished mining products.

4.10 Transportation

OT LLC has been constructing a road between the project and the Gashuun Sukhait border. Pursuant to the IA, roads and other transportation infrastructure funded or constructed by OT LLC for the implementation of the project shall be required to be constructed to a standard necessary to meet the specific requirements of the Oyu Tolgoi project only.

4.11 Foreign Investment Regulation

On 3 October 2013, the Parliament passed the new Investment Law that provides a comprehensive regulatory regime for foreign and local investments in Mongolia. This law was last amended on 14 May 2015.

Key features of the law include:

- The law grants a land possession right to foreign investors.
- A foreign investor may be entitled to a tax stabilization if its investment in Mongolia is more than MNT 30b. The maximum period of tax stabilization in the mining sector is 18 years. The period of an investment agreement may be longer than the period of tax stabilization.
- A foreign investor that invests more than MNT 500b may enter into an investment agreement with the Minister in charge of investment matters.

The Regulation on Establishing Investment Agreement was adopted by the GOM on 21 February 2014 in accordance with Article 20.6 of the Investment Law. The GOM has the right to suspend or terminate an investment agreement if an investor fails to meet obligations under such an agreement.

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 centred at approximately latitude 43°00'45"N, longitude 106°51'15"E or UTM coordinates 4,764,000 mN and 651,000 mE with datum set to WGS-84, Zone 48N. The Oyut 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 1,140–1,215 m above sea level (masl).

5.2 Property Access

5.2.1 Property Access – General

The Oyu Tolgoi Mine is located in the South Gobi region of Mongolia, approximately 550 km south of the capital city, Ulaanbaatar. Access to the property from the Mongolian capital, Ulaanbaatar, is possible either by:

- Driving an unpaved road, via Mandalgovi, which is a 12-hour drive under good conditions, or
- Flying, with in-air travel time of less than two hours.

A permanent domestic airport has been constructed at Oyu Tolgoi to support the transportation of people and goods to the site from Ulaanbaatar. It further serves as the regional airport for Khanbogd soum. Transportation routes, facilities, cities and communities in the region are shown in Figure 5.1.

The permanent airport is 11 km north of the Oyu Tolgoi camp area. It is a non-precision approach, visual flight rules (VFR) facility. The runway surface is concrete 3,250 m long x 45 m wide, with a concrete apron at the terminal building. The runway has been aligned to the prevailing north-west-south-east wind direction to minimize cross-wind conditions and facilitate optimal landing and take-off conditions. The design is set to service commercial aircraft up to the Boeing 737-800 series aircraft.

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. There is currently no access from the project site to the rail line within Mongolia except along a 330 km desert trail north-east to Sainshand.

A private Chinese rail operator, Shenhua, has recently expanded its rail network connecting Baotou to Gashuun Sukhait at the border. Shenhua has also concluded negotiations with

Mongolian authorities to extend the line into Mongolia, to the vicinity of Tsaagan Khad. On completion, this rail line will technically enable Mongolia to export minerals to seaborne markets via the Chinese Port of Tianjin. This would require exports to originate in and traverse China using the same transport mode (rail gauge) in order to be classified as bonded cargo and therefore exempt from 17% Chinese VAT.

The GOM may construct or facilitate the construction and management of a railway in the vicinity of the project to the China-Mongolia border. The GOM will consult with OT LLC on the location and route of the railway, and, if the railway is constructed, then it will be made available to OT LLC on commercial and non-discriminatory terms. Energy Resources is currently constructing a single-track heavy-haul rail from its Ukhua Khudag coal mine (approximately 120 km to the north-west of Oyu Tolgoi) to Gashuun Sukhait, ultimately to be interconnected with the Chinese rail network at Ganqimaodao on the Chinese side of the border. Once constructed, the South Gobi Rail alignment would pass within 10 km of the Oyu Tolgoi project area and therefore represents an opportunity for eventual connection of the mine to the rail network.

The Chinese Government has upgraded 226 km of road from Ganqimaodao to Wuyuan, providing a direct road link between the Mongolian border crossing at Gashuun Sukhait, 80 km south of Oyu Tolgoi, and the Trans-China Railway system. A 105 km sealed road is being constructed to the Mongolian-Chinese border crossing at Gashuun Sukhait. There is 23 km of road that remains to be sealed.

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 Ovoo.

Oyu Tolgoi will make use of the Chinese Port of Tianjin, some 150 km south-east of Beijing, to import freight from overseas. The port is open year-round and has no ice restrictions during winter. Subsequent road delivery will follow the extensive network of Chinese highways connecting Tianjin to Wuyuan, a distance of about 1,050 km, from there along a state highway to Hailiutu, about 60 km, and then on to the China-Mongolia border crossing at Ganqimaodao-Gashuun Sukhait. This will be the primary border crossing for both cargo and Chinese personnel immigration for the project. Baotou, just east of Wuyuan, will be the consolidation point for freight originating from China.

The Port of Tianjin is the largest port in northern China and one of the largest in the world. At the end of 2012, it covered more than 121 km² and had in excess of 32 km of quay shoreline and 159 production berths. As at 2012, the main channel had been dredged to a depth of 21 m, allowing 300,000 DWT ships at high-tide. The port is supervised and regulated by the Tianjin Municipality People's Government, which has set up a Port Services Office to coordinate port services. Given the port's ability to handle large tonnages and containers, it will not be a limiting factor in logistics planning for the project.

5.2.2 Property Access – Protected Areas

The Small Gobi Strictly Protected Area (SGSPA) is approximately 80 km south of the OT License 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.

Figure 5.1 Oyu Tolgoi Project Transport Routes



5.3 Regional Population Centres and Infrastructure

There are a number of communities in the South Gobi (Ömnögov) relatively near Oyu Tolgoi. The most prominent is Dalanzadgad, population 21,581 (as of the end of 2013), which is the centre of the Ömnögov aimag and is 220 km north-west of the project site. Facilities at Dalanzadgad include a regional hospital, tertiary technical colleges, a domestic airport, and a 6 MW capacity coal-fired power station. OT LLC sees Dalanzadgad as a suitable centre for regional recruiting and training and in 2010 signed a Memorandum of Understanding (MoU) with the Ministry of Education, Culture, and Science that included the conduction of a new mining-focused professional and technical training centre there.

In addition to new vocational centres at Dalanzadgad and Ulaanbaatar, OT LLC is investing in facilities and equipment upgrades at five vocational educational establishments in the country, and the Rio Tinto Group (including OT LLC) is entering partnerships with universities for specific programmes. In particular, the Rio Tinto Group will provide the National University of Mongolia (NUM) with educational resource materials, scholarships, and internships. In turn, NUM will support the company's public education and awareness initiatives and deliver short-term courses.

Other Ömnögov aimag communities near Oyu Tolgoi include Khanbogd, the centre of the Khanbogd soum, 45 km to the north-east and closest to the project site, with a population of approximately 4,712 (as of the end of 2013); Bayan Ovoo (population 1,600), 65 km to the

west; and Manlai (population 2,400), 150 km to the north. These communities could all conceivably increase in size as a result of the Oyu Tolgoi project. Farther north, Mandalgovi (population 13,500), the capital of the Dundgovi aimag, 310 km north of the project on the road to Ulaanbaatar, could also be subject to project-related effects.

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 project 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 Bayan Ovoo, approximately 65 km west of Oyu Tolgoi, and from two 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 Hailiutu, 200 km south-west, but the information from Bayan 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 -13°C. Data from Bayan Ovoo are shown in Table 5.1.

Table 5.1 Bayan Ovoo Monthly Temperatures

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Minimum (°C)	-34	-33	-25	-22	-13	0.4	4	3	-5	-20	-27	-33
Average (°C)	-13	-8	-0.4	9	18	23	25	23	17	7	-3	-10
Maximum (°C)	9	16	24	31	38	50	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	13	3	2	1	1	5	8	1	2	5	11
Average (%)	53	38	24	24	19	31	37	36	34	30	41	44
Maximum (%)	81	67	88	90	100	97	100	100	100	81	85	81

5.4.5 Ground Temperature

From the data available to date, the minimum recorded ground temperature is -22°C and the maximum is +40°C. Table 5.3 shows the design freezing depths at the site based on the Mongolian Climate Data and Geophysical Parameters.

Table 5.3 Design Soil Freezing Depths

Soil Type	Freezing Depth (m)
Clayey soil	1.7
Sandy soil	2.1
Gravelly soil	2.5

5.4.6 Solar Radiation

Solar radiation data have been collected at the Oyu Tolgoi site station since 2002. Solar radiation is measured in watts per square metre (W/m²) and fluctuates during the day, ranging from 0 W/m² at night and peaking soon after midday. The average daily maximum for the two 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 mm/a, 90% of which falls as rain and the rest as snow. Snowfall accumulations rarely exceed 50 mm. Maximum rainfall events of up to 44 mm/h 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 10–15 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 Bayan Ovoo data for 1975–2002.

Table 5.4 Rainfall Summary

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Maximum daily (mm)	2.1	3.8	4.4	10.4	19.0	16.2	29.5	102.0	19.2	4.0	4.3	1.5	–
Avg. monthly (mm)	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
Avg. rain days per month (days)	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

Return Interval Duration	Rainfall Intensity (mm/h)					
	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 2–8 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 analyze 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 (Open Water Bodies) (mm)	Total (mm)
January	22	82	104
February	41	101	142
March	–	142	142
April	–	256	256
May	–	439	439
June	–	378	378
July	–	382	382
August	–	285	285
September	–	192	192
October	–	132	132
November	53	11	64
December	27	88	115
Total	143	2,488	2,631

5.4.9.1 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 five to eight days each winter. Spring dust storms are far more frequent and can continue through June and July. The average storm duration is six to seven 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 one hour speeds recorded at Bayan Ovoo are shown in Table 5.7. The number of dust storms per month is shown in Table 5.8.

Table 5.7 Bayan Ovoo Maximum One-Hour Wind Speeds

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Maximum Wind Speed (m/s)	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 Gobi Desert Frequency of Dust Storms

	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

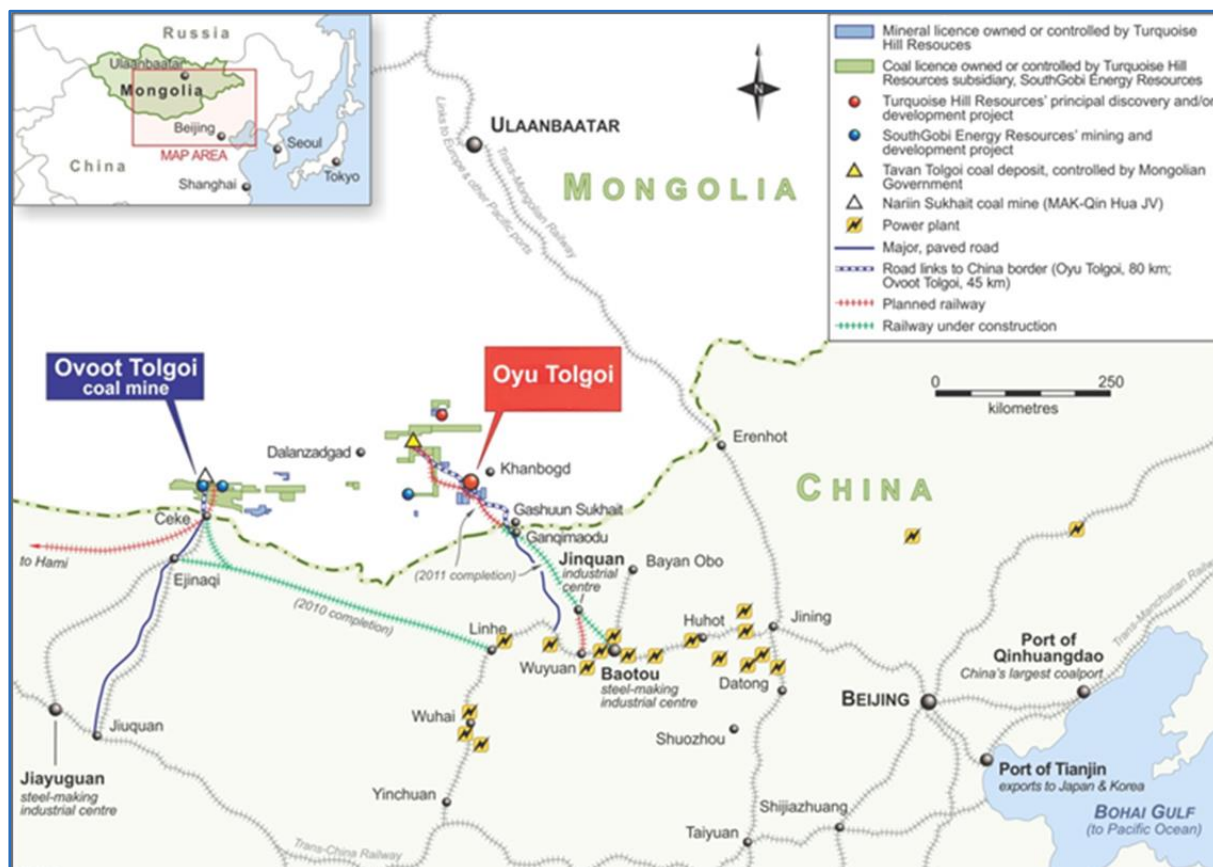
Under the IA, OT LLC is required to secure its power requirements from within Mongolia within four years of commencement of production. However, this timeframe is currently suspended pursuant to the Southern Region Power Sector Cooperation Agreement, entered into by OT LLC and the GOM on 14 August 2014, that proposes an independently funded and operated coal fired power plant at Tavan Tolgoi (TTPP Project).

So long as OT LLC continues to participate in the TTPP Project, the four-year timeframe for sourcing Mongolian power will be suspended. Upon a withdrawal from the TTPP Project by either OT LLC or the GOM, the four-year timeframe will be reinstated and recommence from the date of withdrawal. A request for proposals from potential investors in the TTPP coal fired power plant culminated in a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

OT LLC sources its present power under a four-year contract with a Chinese provider, the Inner Mongolia Power International Cooperation Company Ltd. (IMPIC) via the Mongolian National Power Transmission Grid (NPTG) authority. In May 2016 the parties agreed via a non-binding Memorandum of Understanding (MoU), which captured key agreed principles of the new Power Purchase Agreement (PPA), to extend the power supply agreement to at least 2021. OT LLC and the GOM have agreed under the Power Sector Cooperation Agreement that the GOM will assume responsibility for securing the extension of the power import arrangements through its national grid company NPTG. The agreed MoU includes comparable power pricing to the current agreement. OT LLC is endeavoring to execute binding agreements with the GOM and IMPIC within 2016.

The locations of power providers in Northern China is shown in Figure 5.2.

Figure 5.2 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 Undai 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 2002–2009. 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 programmes.

Potential impacts on surface water systems in the project area include local changes to natural flow paths and depletion of springs, ephemeral wetlands, and salt-marshes from project development and operational 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 used to avoid and mitigate potential impacts to air and surface water quality. These studies are 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 Oyu Tolgoi's estimated water demand. Groundwater investigations conducted in the mine license 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 brownfields project and extensive infrastructure has been constructed in addition to the concentrating facilities. The major initial infrastructure elements include:

- **Water Borefields**

Water is supplied from the Gunii Hooloi basin, which extends 35–75 km north of Oyu Tolgoi. Bores were 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 has been constructed to treat raw water from the Gunii Hooloi borefield to drinking (potable) and domestic water standards.

- Housing

Accommodation facilities have been 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 is located approximately 11 km north of Oyu Tolgoi and facilitates 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 centre, fire station, heating plant, fuel storage, and warehouse facilities, among others were constructed to support operational requirements over the life-of-mine.

- Railroads

Initially, the transport of bulk supplies and the delivery of copper concentrate to China is by access road to the railhead. However, direct rail transport is considered a long-term transportation solution after this initial development period when other non-OT development projects are initiated in the region.

5.5.4 Other

5.5.4.1 Land Use

The land surrounding the mine license 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 programme 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; trans-boundary matters, 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 is and will continue to 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 permits 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 2011 that a seismic hazard assessment of Oyu Tolgoi was completed. The seismicity of Oyu Tolgoi 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 50–100 km west of Oyu Tolgoi.

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

6 HISTORY

6.1 Oyu Tolgoi Project History

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 a joint Mongolian and Russian geochemical survey team identified a molybdenum anomaly over the Central zone. Evidence of alteration and copper mineralization at the South zone was first noted by Garamjav in 1983, during a regional reconnaissance of the area. In September 1996, Garamjav guided geologists from Magma Copper Company to the area. These geologists identified a porphyry copper leached cap over what is now known as the Central zone of the Oyu deposit and quickly moved to secure exploration tenements. Magma Copper Company was subsequently acquired by BHP, later BHP-Billiton (BHP). The mineralization target at Oyu Tolgoi was a large supergene-enriched porphyry.

Geophysical surveying at Oyu Tolgoi was first initiated by BHP in 1997. 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 using a single gradient array with a 2,000 m AB electrode spacing and a ground magnetometer survey. The first survey was conducted on north-south oriented lines and produced data that were difficult to reconcile to the then-known geology. A later survey by Ivanhoe Mines in 2001 was conducted on east-west oriented lines and therefore perpendicular to the structural trend. This immediately showed the close correlation between mineralization and chargeable response, which has proven to be highly successful in further exploration. Both IP datasets were surveyed by a local Mongolian surveying team at 250 m line spacing. The surveys covered the Southern zone, Southwest zone, Central zone, and North zone exploration targets but did not extend into the Far North region that ultimately became the Hugo Dummett deposits.

BHP carried out geological, geochemical (stream sediment and soil), and geophysical surveys and diamond drilling programmes (23 drillholes in total) in the Central and South zones in 1997 and 1998. Copper and gold values were encountered at depths from 20–70 m below surface, and a supergene-enriched, chalcocite blanket was encountered in one drillhole (OT-3). Based on the results of this drilling, BHP performed a Mineral Resources estimate in 1998, but the resulting tonnage and grade estimate was considered too small to meet BHP corporate objectives, and BHP elected to offer the property for joint venture. Ivanhoe Mines visited Oyu Tolgoi in May 1999 and agreed to acquire 100% interest in the property, subject to a 2% NSR royalty (IVN Royalty). In 2000, Ivanhoe, through its subsidiary Ivanhoe Mines Mongolia XXK (IMMI), completed 8,000 m of reverse circulation (RC) drilling, mainly at the Central zone, to explore the chalcocite blanket discovered earlier by BHP. Based on this drilling, IMMI updated the Mineral Resources estimates. (IMMI later became OT LLC.)

In 2001, IMMI continued RC drilling, mostly in the South zone area, to test for additional supergene copper mineralization, and then drilled three core holes to test the deep hypogene copper-gold potential. One of these holes, OTRCD150, drilled over Southwest zone, intersected 508 m of chalcopyrite mineralization from a depth of 70 m, grading 0.81% Cu and 1.17 g/t Au. This marked the discovery of the Oyu deposit. The Oyu deposit was formerly known as Southern Oyu Tolgoi (SOT).

These results encouraged Ivanhoe to mount a major follow-up drill programme. In late-2002, drilling in the far northern section of the property intersected 638 m of bornite–chalcopyrite-rich mineralization in drillhole OTD270, starting at a depth of 222 m. This hole marked the discovery of the Hugo Dummett deposits.

The first Mineral Resource was reported on the Oyut deposit in 2003. A first-time Mineral Resources estimate for the deposit was prepared in 2004 on Hugo South (formerly called Far North), and the Hugo Dummett Mineral Resources were updated in 2005 to include Hugo North. In 2007 and 2014 the Hugo North Mineral Resources were updated.

In 2004, a NI 43-101 Preliminary Economic Analysis (PEA) was completed on the economics of open pit mining the Oyut. The Integrated Development Plan 2005 (IDP05) was also a PEA. IDP05 presented open pit mining on the Oyut deposit, two block caves on Hugo North and one block cave on Hugo South, the plant capacity examined was 25.5 Mt/a with an expansion to 51 Mt/a. In 2006 a NI 43-101 Feasibility Study presented the open pit Oyut Mineral Reserves as an open pit only scenario.

The Shaft 1 headframe, hoisting plant, and associated infrastructure were completed in January 2006. The shaft had been sunk to a depth of 1,385 m by January 2008. Development from the shaft has enabled additional delineation drilling and rock characterization for proposed mining operations.

In 2009, the IA was agreed with the GOM, which thereby became a 34% shareholder in OT LLC (formerly IMMI) through the immediate issue of OT LLC's common shares to a shareholding company owned by the GOM. As part of the process of agreement, OT LLC presented a Mongolian Feasibility Study (MFS09) to the GOM. The MFS09 included mining scenarios of the open pit on the Oyut deposit and underground mining by block caving on Hugo North, Hugo South, and Heruga. The plant capacity examined was 36.5 Mt/a with an expansion to 58 Mt/a.

The Integrated Development Plan 2010 (IDP10) was a NI 43-101 Technical Report released in 2010 that included Mineral Reserves for open pit mining of the Oyut deposit and block caving of Hugo North Lift 1. The plant capacity examined was 36.5 Mt/a with an expansion to 58 Mt/a.

In 2011, OT LLC completed the Integrated Development and Operating Plan (IDOP) that updated IDP10 using the same production scenario. A NI 43-101 Technical Report was released on IDOP. Sinking of Shaft 2 commenced in 2011.

In 2012, the Detailed Integrated Development and Operating Plan (DIDOP) was prepared examining the project scenario of open pit mining on Oyut and underground block caving on Hugo North Lift 1 without a plant expansion. DIDOP was released in the NI 43-101 2013 Oyu Tolgoi Technical Report (2013 OTTR).

In August 2013, development of the underground mine was delayed to allow matters, including the tax dispute, approval of the project feasibility Study by Oyu Tolgoi's shareholders and acceptance by the Mongolian Minerals Council, agreement of a comprehensive funding plan including project finance, and receipt of all necessary permits, to be resolved between the parties to the IA (TRQ, Rio Tinto, and the GOM).

In 2014, OT LLC submitted the Oyu Tolgoi Feasibility Study 2014 (OTFS14) to the GOM. OTFS14

included a Reserves Case (open pit mining on Oyut and underground block caving on Hugo North Lift 1) and a Resources Case (open pit mining on Oyut and underground block caving on Hugo North Lift 1 and Lift 2, Hugo South and Heruga) both cases were at the plant rate of 36.5 Mt/a without expansion. The OTFS14 Reserves Case was released by TRQ in the 2014 Oyu Tolgoi Technical Report (2014 OTTR).

During the course of 2013–2014, many of the matters between the parties were resolved or progressed. The Mongolian Reserves and Resources in OTFS14 were submitted to the GOM to update the Mongolian State Reserves in 2014. Further submissions in the as a statutory feasibility study titled OTFS15 based on modifications to OTFS14, were made to the GOM and accepted by the Minerals Council. The UDP, signed on 18 May 2015, addressed the key outstanding shareholder matters and set out an agreed basis for the funding of the project. OTFS16 incorporates matters resolved between the shareholders and has been approved by the OT LLC board of directors and shareholders.

6.2 EJV Licenses

IMMI (now OT LLC) initiated exploration work on the Javkhlant and Shivee Tolgoi licenses in November 2004, following the signing of an Equity Participation and Earn-in Agreement with Entrée.

Before that time, Entrée had undertaken soil geochemical surveys, ground magnetics, Bouguer gravity and pole-dipole geophysical surveying, and geological mapping, but had failed to locate any mineralization of significance.

Starting at the northern boundary of the Oyu Tolgoi mining 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. Subsequent IP surveys covering smaller areas within the larger area were carried out with gradient arrays.

The IP surveys resulted in the delineation of a significant chargeability feature being traced for approximately 4 km north along strike of the Hugo North deposit. Additional IP chargeability targets were also revealed 2.5–3.0 km west of the Hugo North trend and are referred to as the Eagle anomalies.

Ivanhoe commenced drilling northward from the northern boundary of the Oyu Tolgoi mining license in 2005. A first-time resource estimate for the Hugo North Extension deposit was completed in 2006. Underground mining plans for Hugo North Extension were included in a Sensitivity Analysis Addendum to the DIDOP report.

In 2005 and 2006, IMMI (now OT LLC) conducted IP surveying on 100 m spaced, east–west lines across Entrée's Javkhlant license to the south of the Oyut Mineral Resource area. This resulted in the discovery of three significant chargeability IP anomalies subsequently named the Sparrow South, Castle Rock, and Southwest magnetic anomalies. Core drilling was initiated to test these IP anomalies in early-2007. A series of successful drillholes in the area supported a first-time Mineral Resources estimate over what is now known as the Heruga deposit (formerly the Sparrow South anomaly) in 2008.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

The information in this section is collated from geological and technical reports completed on the project during the period 2003–2011, the PhD thesis completed by Alan Wainwright on the Oyu Tolgoi deposit setting in 2008, and other sources as noted.

7.1.1 Deposit Model

The Oyu Tolgoi deposits display copper–gold porphyry and related high-sulphidation copper–gold deposit styles.

The following discussion of the typical nature of porphyry copper deposits is sourced from Sillitoe (2010), Singer et al. (2008), and Sinclair (2006).

7.1.1.1 Geological Setting

Porphyry copper systems commonly define linear belts, some many hundreds of kilometres long, and some occurring less commonly in apparent isolation. The systems are closely related to underlying composite plutons, at paleo-depths of 5–15 km, which represent the supply chambers for the magmas and fluids that formed the vertically elongate (>3 km) stocks or dyke swarms and associated mineralization.

Commonly, several discrete stocks are emplaced in and above the pluton roof zones, resulting in either clusters or structurally controlled alignments of porphyry copper systems. The rheology and composition of the host rocks may strongly influence the size, grade, and type of mineralization generated in porphyry copper systems. Individual systems have life spans of circa 100,000 years to several million years, whereas deposit clusters or alignments, as well as entire belts, may remain active for 10 million years (Ma) or longer.

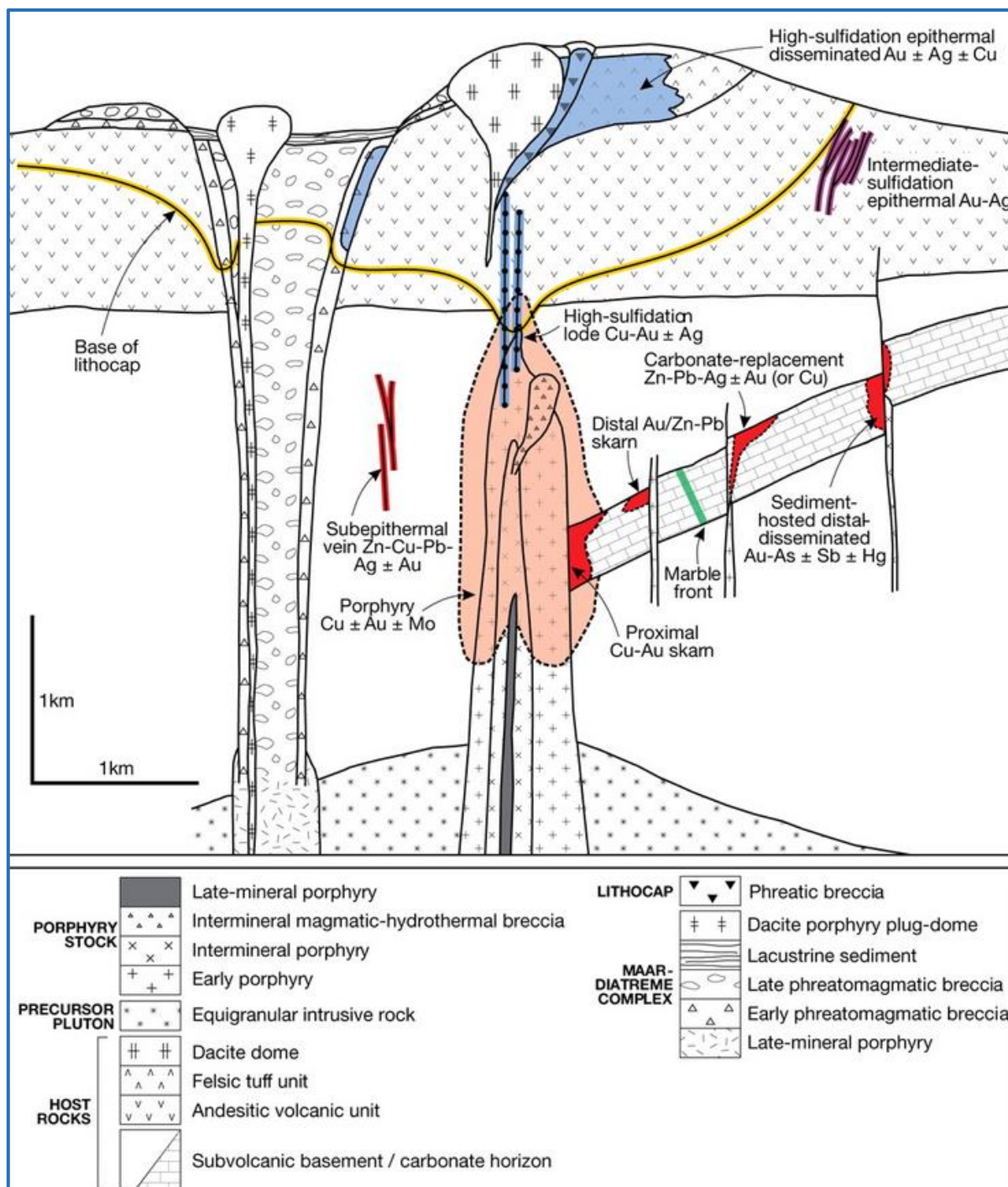
Deposits are typically semicircular to elliptical in plan view. In cross-section, ore-grade material in a deposit typically has the shape of an inverted cone with the altered, but low grade, interior of the cone referred to as the 'barren' core. In some systems, the barren core may be a late-stage intrusion.

The alteration and mineralization in porphyry copper systems are zoned outward from the stocks or dyke swarms, which typically comprize several generations of intermediate to felsic porphyry intrusions. Porphyry copper–gold–molybdenum deposits are centred on the intrusions, whereas carbonate wall rocks commonly host proximal copper–gold skarns and less commonly, distal base metal and gold skarn deposits. Beyond the skarn front, carbonate-replacement copper and/or base metal–gold deposits, and/or sediment-hosted (distal-disseminated) gold deposits can form. Peripheral mineralization is less conspicuous in non-carbonate wall rocks, but may include base metal or gold-bearing veins and mantos. Data compiled by Singer et al. (2008) indicate that the median size of the longest axis of alteration surrounding a porphyry copper deposit is 4–5 km, while the median size area of alteration is 7–8 km².

High-sulphidation epithermal deposits may occur in lithocaps above porphyry copper deposits, where massive sulphide lodes tend to develop in their deeper feeder structures, and precious metal-rich, disseminated deposits form within the uppermost 500 m.

Figure 7.1 shows a schematic section of a porphyry copper deposit, illustrating the relationships of the lithocap to the porphyry body and associated mineralization styles.

Figure 7.1 Schematic Section of Porphyry Copper Deposit



Note: Figure from Sillitoe (2010).

7.1.1.2 Mineralization

Porphyry copper mineralization occurs in a distinctive sequence of quartz-bearing veinlets as well as in disseminated forms in the altered rock between them. Magmatic-hydrothermal breccias may form during porphyry intrusion, with some breccias containing high-grade mineralization because of their intrinsic permeability. In contrast, most phreatomagmatic breccias, constituting maar–diatreme systems, are poorly mineralized at both the porphyry copper and lithocap levels, mainly because many such phreatomagmatic breccias formed late in the evolution of systems, and the explosive nature of their emplacement fails to trap mineralizing solutions.

Copper mineral assemblages are a function of the chemical composition of the fluid phase and the pressure and temperature conditions affecting the fluid. In primary, unoxidized or non-supergene-enriched ores, the most common sulphide assemblage is chalcopyrite ± bornite, with pyrite and minor amounts of molybdenite. In supergene-enriched ores, a typical assemblage can comprise chalcocite + covellite ± bornite, whereas in oxide ores a typical assemblage could include malachite + azurite + cuprite + chrysocolla, with minor amounts of minerals such as carbonates, sulphates, phosphates, and silicates. Typically, the principal copper sulphides consist of millimetre scale grains, but may be as large as 1–2 cm in diameter and, rarely, pegmatitic (larger than 2 cm).

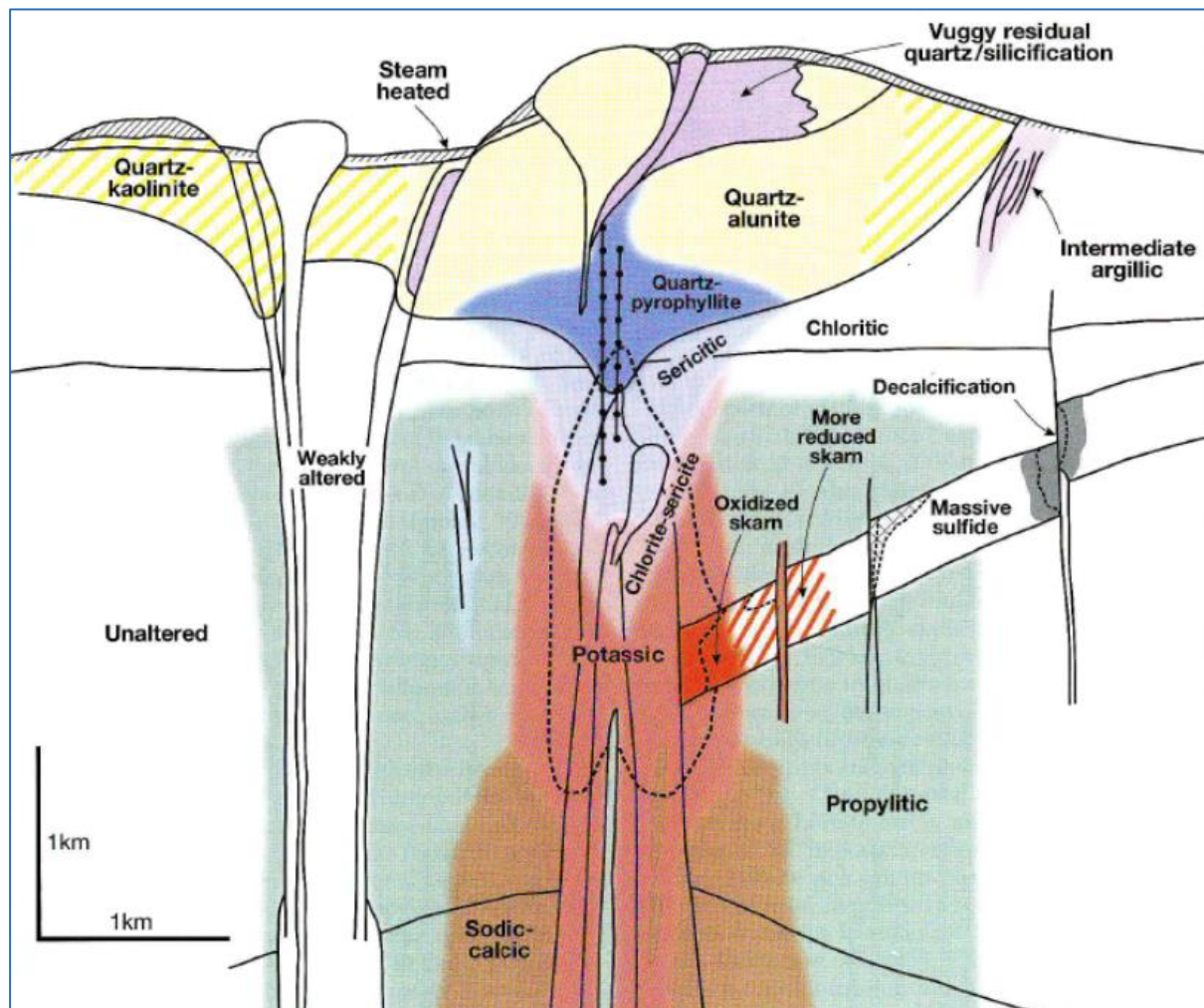
7.1.1.3 Alteration

Alteration zones in porphyry copper deposits are typically classified on the basis of mineral assemblages. In silicate-rich rocks, the most common alteration minerals are K-feldspar, biotite, muscovite (sericite), albite, anhydrite, chlorite, calcite, epidote, and kaolinite. In silicate-rich rocks that have been altered to advanced argillic assemblages, the most common minerals are quartz, alunite, pyrophyllite, dickite, diaspore, and zunyite. In carbonate rocks, the most common minerals are garnet, pyroxene, epidote, quartz, actinolite, chlorite, biotite, calcite, dolomite, K-feldspar, and wollastonite. Other alteration minerals commonly found in porphyry copper deposits are tourmaline, andalusite, and actinolite. Figure 7.2 shows the typical alteration assemblage of a porphyry copper system.

Porphyry copper systems are initiated by injection of oxidized magma saturated with sulphur- and metal-rich, aqueous fluids from cupolas on the tops of the subjacent parental plutons. The sequence of alteration and mineralization events is principally a consequence of progressive rock and fluid cooling, from >700°C to <250°C, caused by solidification of the underlying parental plutons and downward propagation of the lithostatic–hydrostatic transition. Once the plutonic magmas stagnate, the high temperature, generally two phase hyper-saline liquid and vapour responsible for the potassic alteration and contained mineralization at depth and early overlying advanced argillic alteration, respectively, gives way, at <350°C, to a single-phase, low-to-moderate-salinity liquid that causes the sericite–chlorite and sericitic alteration and associated mineralization. This same liquid also is a source for mineralization of the peripheral parts of systems, including the overlying lithocaps.

The progressive thermal decline of the systems combined with syn-mineralization paleo-surface degradation results in the characteristic overprinting (telescoping) and partial to total reconstitution of older by younger alteration and mineralization types. Meteoric water is not required for formation of this alteration and mineralization sequence, although its late ingress is commonplace.

Figure 7.2 Schematic Section Showing Typical Alteration Assemblages



Note: Figure from Sillitoe (2010).

7.1.1.4 Applicability of the Porphyry Copper Model to Oyu Tolgoi

Features that classify the Oyu Tolgoi deposits as porphyry copper-type deposits include:

- Mineralization is in or adjoining porphyritic intrusions of quartz monzodiorite composition.
- Multiple emplacements of successive intrusive phases and a variety of breccias are present.
- Mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks.
- Large zones of veining and stockwork mineralization, together with minor disseminated and replacement mineralization, occur throughout large areas of hydrothermally altered rock, commonly coincident wholly or in part with hydrothermal or intrusion breccias.
- Hydrothermal alteration is extensive and zoned, which is common to porphyry copper deposits. Major alteration minerals in the biotite–chlorite, intermediate argillic, sericite,

and K-feldspar alteration zones include quartz, chlorite, sericite, epidote, albite, biotite, hematite–magnetite, pyrophyllite, illite, and carbonate. Advanced argillic alteration zones can contain minerals such as kaolinite, zunyite, pyrophyllite, muscovite, illite, topaz, diaspore, andalusite, alunite, montmorillonite, dickite, tourmaline, and fluorite. In the leached cap, smectite and kao-smectite can also occur. The alteration assemblages are consistent with the physio-chemical conditions of a porphyry environment.

- Pyrite is the dominant sulphide, reflecting the typical high-sulphur content of porphyry copper deposits. The major ore minerals include chalcopyrite, bornite, chalcocite, covellite, and enargite. In some zones, minerals such as tennantite, tenorite, cubanite, and molybdenite have been identified. Gold typically occurs as inclusions in the sulphide minerals.
- Copper grades are typical of the range of porphyry copper grades (0.2% Cu to >1% Cu).

The Oyu Tolgoi porphyry copper deposits display a range of mineralization styles, alteration characteristics, and deposit morphologies that are likely to reflect differences in structural controls, host rock lithology, and depth of formation. For the most part, structural influences account for the differences in shape and distribution of mineralization within the deposits. The more typical copper–gold porphyry style alteration and mineralization tend to occur at deeper levels, predominantly within basalt and quartz monzodiorite.

High-sulphidation mineralization and associated advanced argillic alteration are most common within the wall rocks (basaltic tuffs and fragmental rocks) to the quartz monzodiorite, where it intrudes to levels high in the stratigraphic succession and in narrow structurally controlled zones. High-sulphidation mineralization often forms in steam condensate zones and then collapses back into the hypogene zone, causing overprinting and textural destruction.

The Hugo Dummett deposits have several features that are unusual when compared with typical porphyry copper systems, including:

- Anomalously high copper and gold grades, particularly in the northern part.
- An unusually weakly altered pre-mineralization volcano-sedimentary cover sequence that lies just above the porphyry system.
- Quartz + sulphide vein contents commonly exceeding 15%, and locally in excess of 90%, in the high-grade part of the deposit.
- A highly elongate, gently plunging tabular shape to the high-grade stockwork system.

The formation of the known, 800 m extent, high-grade portion of the Hugo Dummett deposits as a tabular, intensely veined, sub-vertical body contrasts markedly with most porphyry copper deposits, which tend to have steep, roughly cylindrical, or elongate forms. The unusual form of the Hugo Dummett deposits could be the result of emplacement within a structurally restricted zone. The lack of alteration in the overlying sequence is likely a reflection of the chemical inertness of the siltstone sequences.

The Heruga deposit is also slightly unusual in that, unlike the other Oyu Tolgoi deposits, it has distinctly higher grades of molybdenum, which form a molybdenum-rich carapace at higher elevations overlying gold–copper-rich mineralization at depth.

7.1.2 Regional Geology

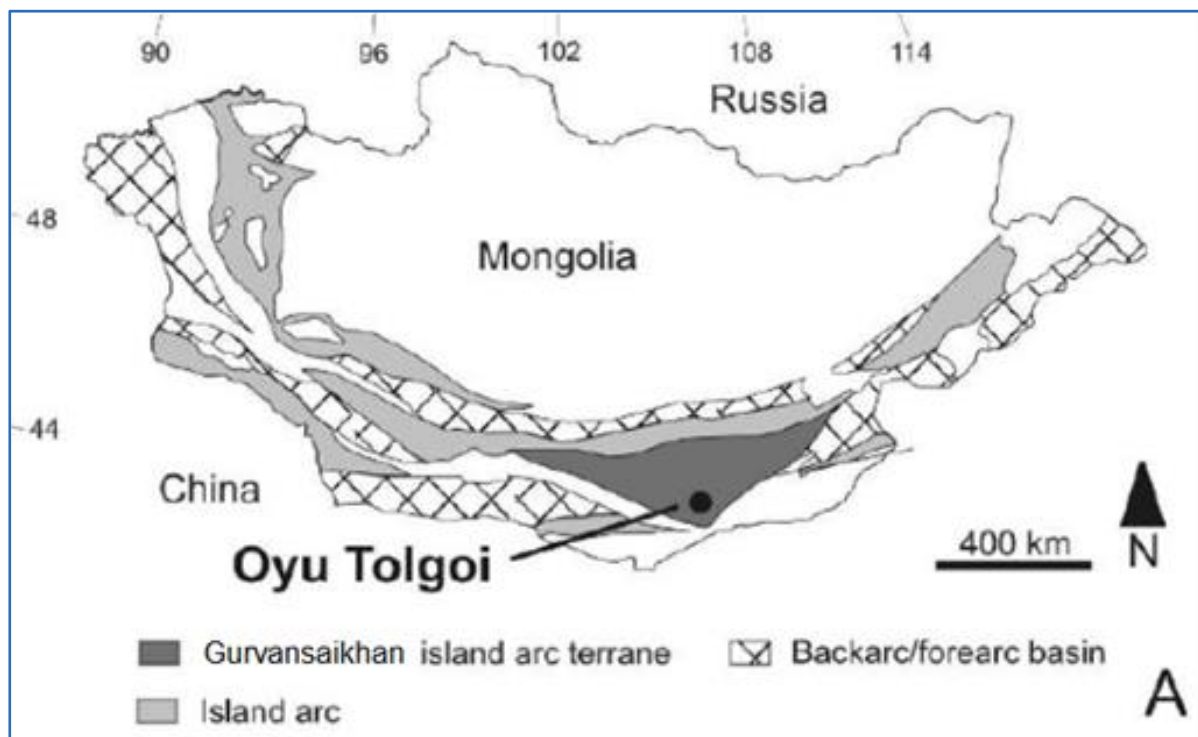
The Oyu Tolgoi porphyry deposits are hosted within the Gurvansaikhan Terrane, part of the Central Asian Orogenic Belt, rocks of which now comprize the South Gobi region of Mongolia (Figure 7.3 and Figure 7.4).

Development of the Central Asian Orogenic Belt consisted of Palaeozoic age accretionary episodes that assembled a number of island and continental margin magmatic arcs, rifted basins, accretionary wedges, and continental margins. Arc development ceased by about the Permian. During the Late Jurassic to Cretaceous, north–south extension occurred, accompanied by the intrusion of granitoid bodies, unroofing of metamorphic core complexes, and formation of extensional and transpressional sedimentary basins. North-east–south-west shortening is superimposed on the earlier units and is associated with major strike-slip faulting and folding within the Mesozoic sedimentary basins.

The Gurvansaikhan Terrane is interpreted to be a juvenile island arc assemblage that consists of highly deformed accretionary complexes and volcanic arc assemblages dominated by imbricate thrust sheets, dismembered blocks, mélanges, and high-strain zones. Lithologies identified to date in the Gurvansaikhan Terrane include Silurian to Carboniferous terrigenous sediments, volcanic-rich sediments, carbonates, and intermediate to felsic volcanic rocks. Sedimentary and volcanic units have been intruded by Devonian granitoids and Permo-Carboniferous diorite, monzodiorite, granite, granodiorite, and syenite bodies, which can range in size from dykes to batholiths.

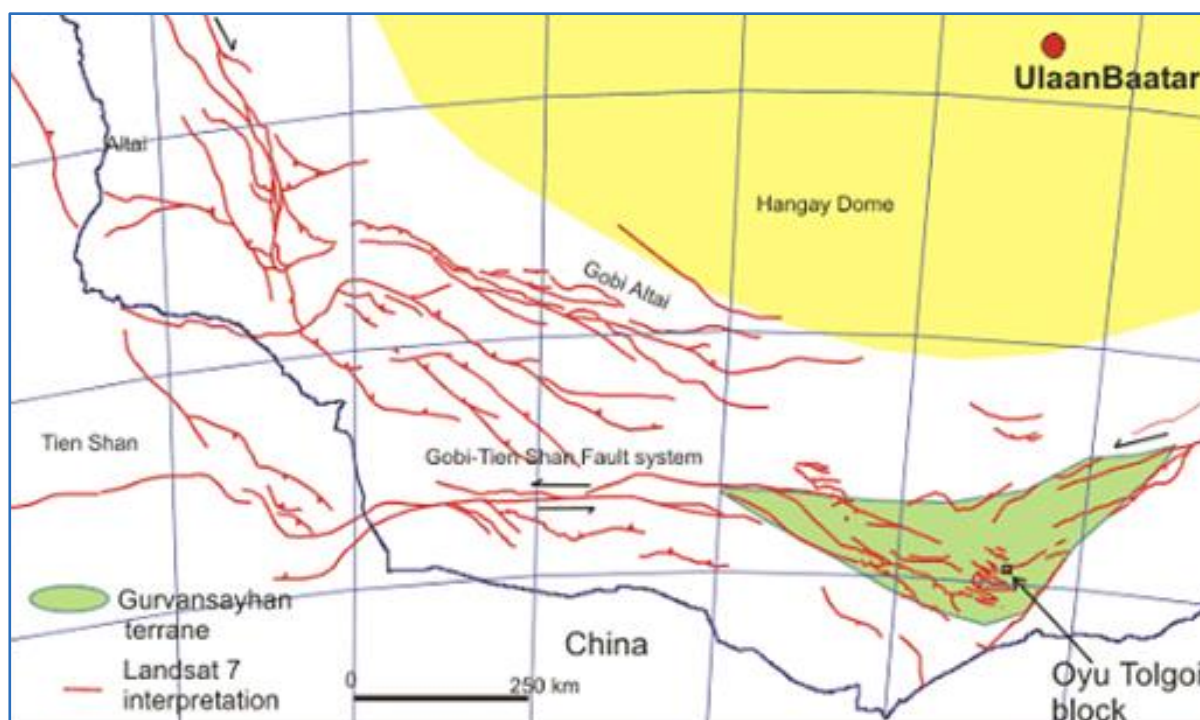
Major structures to the west of the Gurvansaikhan Terrane include the Gobi-Tien Shan sinistral strike-slip fault system that splits eastward into a number of splays in the Oyu Tolgoi area, and the Gobi Altai Fault system, which forms a complex zone of sedimentary basins over-thrust by basement blocks to the north and north-west of Oyu Tolgoi (refer to Figure 7.4). To the east of the Gurvansaikhan Terrane, regional structures are dominated by the north-east striking East Mongolian Fault Zone, which forms the south-east boundary of the 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-east elongate sedimentary basins along the fault trace.

Figure 7.3 Regional Setting, Gurvansaikhan Terrane



Note: Figure from Wainwright (2008).

Figure 7.4 Regional Structural Setting, Gurvansaikhan Terrane



7.1.3 District Geology

7.1.3.1 Overview

The Oyu Tolgoi copper–gold porphyry deposits are situated in a poorly exposed inlier of Devonian mafic to intermediate volcanic, volcanoclastic, and sedimentary rocks that have been intruded by Devonian to Permian felsic plutons. These rocks are unconformably overlain by poorly consolidated Cretaceous sedimentary rocks and younger unconsolidated sedimentary deposits. A district-wide stratigraphic column that shows the relative thicknesses of the various lithologies is presented in Figure 7.5.

Two major stratigraphic sequences are recognized in the project area:

- Tuffs, basaltic rocks, and sedimentary strata of probable island-arc affinity, assigned to the Upper Devonian Alagbayan Group.
- An overlying succession containing conglomerates, fossiliferous marine siltstones, sandstones, water-lain tuffs, and basaltic to andesitic flows and volcanoclastic 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 about 10–15 Ma.

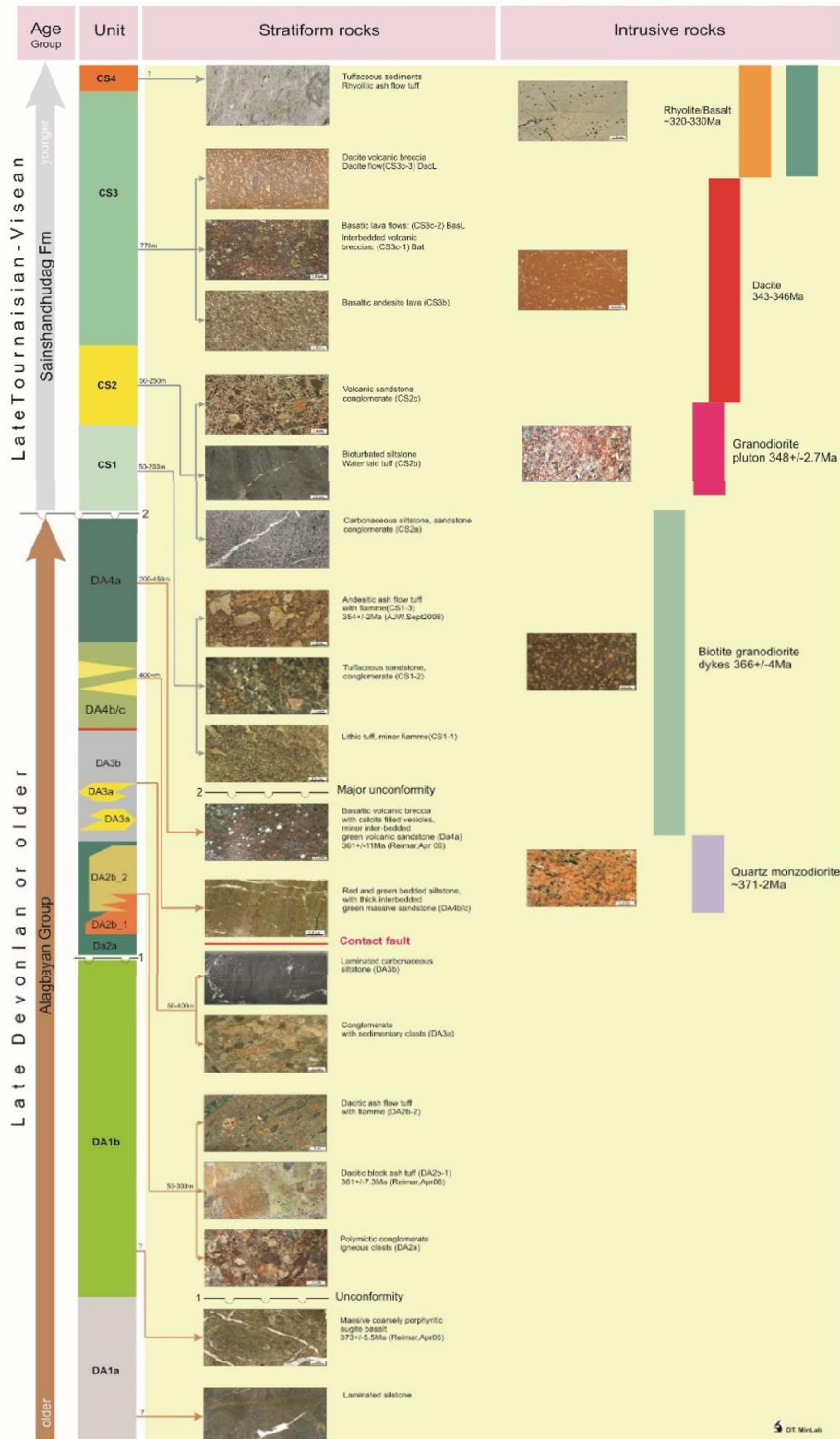
The volcanic and sedimentary rocks are cut by several phases of intrusive rocks ranging from batholithic intrusions to narrow discontinuous dykes and sills. Compositional and textural characteristics vary.

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

The Oyu Tolgoi area is underlain by complex networks of poorly exposed faults, folds, and shear zones. These structures influence the distribution of mineralization by both controlling the original position and form of mineralized bodies, and modifying them during post-mineralization deformation events. The district geology is shown in Figure 7.6.

Figure 7.5 Project Stratigraphic Column

STRATIGRAPHY OF THE OYU TOLGOI AREA



The map displays the Oyu Tolgoi region in Mongolia, with a coordinate grid ranging from 630,000 mE to 660,000 mE and 4,750,000 mN to 4,770,000 mN. A scale bar indicates 5 km, and a north arrow is present in the top left corner.

Geological Units and Mining Licenses:

- Ulaan Khuud Block** (Top center)
- Shivee Tolgoi Block** (Top left)
- Shivee Tolgoi** (Center left)
- Ulaan Khuud South** (Top right)
- Hugo North** (Center right)
- Hugo South** (Center right)
- Central Oyu** (Center right)
- SW Oyu** (Center right)
- South Oyu** (Center right)
- Heruga North** (Center right)
- Heruga** (Center right)
- Khukh-Khad Mining license** (Bottom right)
- Castle rock** (Bottom right)
- Raven** (Bottom center)
- Javkhilant** (Bottom left)
- Manakht Mining license** (Bottom left)
- Javkhilant Entree-OT LLC Earn-in Property** (Bottom left)
- Shivee Tolgoi Entree-OT LLC Earn-in property** (Top left)

Legend:

- Geological units:**
 - Quaternary
 - Tertiary
 - Cretaceous
 - Triassic
 - Permian
 - Carboniferous
 - Devonian
 - Silurian
 - Ordovician
 - Silurian
 - Ordovician
 - Devonian
 - Carboniferous
 - Permian
 - Triassic
 - Cretaceous
 - Tertiary
 - Quaternary
- Mineral resources:**
 - Gold
 - Copper
 - Molybdenum
 - Fluorine
 - Uranium
 - Vanadium
 - Chromium
 - Nickel
 - Cobalt
 - Platinum
 - Palladium
 - Rhodium
 - Ruthenium
 - Indium
 - Gallium
 - Germanium
 - Antimony
 - Bismuth
 - Lead
 - Zinc
 - Aluminum
 - Silica
 - Graphite
 - Asbestos
 - Clay
 - Sand
 - Gravel
 - Crushed stone
 - Crushed brick
 - Crushed concrete
 - Crushed asphalt
 - Crushed stone
 - Crushed brick
 - Crushed concrete
 - Crushed asphalt

Map Information:

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The Oyu Tolgoi copper–gold deposits currently comprise, from north to south:

- Hugo Dummett (includes the Hugo North Extension zone, which is the extension of the Hugo North deposit onto the EJV ground).
- Hugo South.
- Oyut (includes the Southwest, South, Wedge, Central, Bridge, Western, and Far South zones). The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).
- Heruga (includes Heruga North, which is the extension of the Heruga deposit onto the Oyu Tolgoi mining license).

The surface traces and surface projection of the distinct porphyry centres define a north–north-east trending mineralized corridor underlain by east dipping panels of Upper Devonian or older layered sequences intruded by quartz monzodiorite and granodiorite stocks and dykes (refer to Figure 7.5).

7.1.3.2 Sedimentary Lithologies

Four major lithological divisions are present within the Alagbayan Group, and each of these divisions consists of two or more mappable sub-units (Table 7.1).

The two lower units are commonly strongly altered and form important mineralization hosts, while the two upper units lack significant alteration and mineralization. Unit DA4 is separated from the underlying Alagbayan Group units by a contact-parallel fault, known as the Contact Fault.

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 (Table 7.2). The unit post-dates porphyry mineralization and is separated from the underlying Devonian rocks by a regional unconformity.

7.1.3.3 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 (Table 7.3).

Copper–gold porphyry mineralization is related to the oldest recognized intrusive suite, consisting of large Devonian quartz monzodiorite intrusions that occur in all of the deposit areas.

Table 7.1 Major Units of the Alagbayan Formation

Unit	Lithologies	Description
DA1	Basaltic flows and volcaniclastic rocks; several hundred metres in thickness.	Two sub-units: Lower: grey to green, finely-laminated, volcanogenic siltstone and interbedded fine sandstone (DA1a). Upper: dark green, massive porphyritic (augite) basalt. Overlies and partially intercalated with basal unit (DA1b).
DA2	Dacite tuff / volcaniclastic rocks; at least 200 m thick	Three sub-units: Lower: monolithic to slightly polyolithic basaltic lapilli tuff to volcaniclastic conglomerate/breccia. Underlies and partially intercalated with middle unit (DA2a) Middle: buff to dark green, dacite lapilli tuff. Overprinted by intense sericite and advanced argillic alteration (DA2b_1) Upper: weakly altered to unaltered polymictic block tuff to breccia, with lesser intercalated lapilli tuff (DA2b_2).
DA3	Clastic sedimentary sequence; approximately 100 m thick	Two sub-units: Polyolithic conglomerate, sandstone, and siltstone. Abundant in the South zone and parts of the Hugo South deposit (DA3a). Rhythmically interbedded carbonaceous siltstone and fine brown sandstone. Ubiquitous in drillholes in Hugo North and is also discontinuously distributed in the more southerly deposits (DA3b).
DA4	Basaltic flows / fragmental rocks, siltstone; approximately 600 m thick	Three sub-units: Dark green basaltic volcanic breccia with vesicular, fine-grained to coarsely porphyritic basaltic clasts is the dominant lithotype; interlain with volcanogenic sandstones and conglomerates (DA4a). Thinly interbedded red and green siltstone, which contain subordinate basalt layers in their lower levels (DA4b). Massive green to grey sandstone with rare siltstone interbeds (DA4c).

Table 7.2 Major Units of the Sainshandhudag Formation

Unit	Lithologies	Description
CS1	Andesitic lapilli tuff and volcaniclastic rocks; approximately 200 m thick	Andesitic lapilli tuff with abundant fiamme, and subordinate block tuff to breccia.
CS2	Conglomerate, sandstone, tuff, and coal; approximately 200 m thick	Typically shows a progression from a lower conglomerate-sandstone-siltstone dominant unit (CS2a) to an overlying siltstone-waterlain tuff unit (CS2b). Carbonaceous siltstone and coal beds occur in the lower part of the sequence.
CS3	Basaltic and andesite lava and volcaniclastic rocks; approximately 800 m thick	Four sub-units: Basal: thin volcanic sandstone (CS3a). Lower middle: discontinuous porphyritic basaltic andesitic lava sequence (CS3b). Upper middle: thick basaltic breccia-to-block tuff unit (CS3c_1). Upper: intercalated to overlying porphyritic basalt flow sequence (CS3c_2).

Table 7.3 Major Intrusive Rock Units

Unit	Lithologies	Age	Description
Intrusions	Quartz monzodiorite to monzodiorite	371 ± 2 Ma	Texturally and compositionally varied. Typically phenocryst-crowded, with >40% plagioclase phenocrysts up to 5 mm long, and 10%–15% biotite and hornblende. Abbreviated to Qmd.
Intrusion, dykes and sills	Biotite-granodiorite	366 ± 4 Ma	Contain large plagioclase phenocrysts with lesser small biotite phenocrysts, within a fine-grained to aphanitic brown groundmass. Intrusions are compositionally and texturally varied and probably include several intrusive phases. Forms a large stock at Hugo North (BiGd).
Intrusions	Syenite, granite, quartz monzonite, quartz diorite, and quartz syenite	348 ± 3 Ma	Large, polyphase granitic complex bounding Oyu Tolgoi to the north-west.
Dykes	Hornblende–biotite andesite and dacite	343 ± 3 Ma	Typically strongly porphyritic with feldspar, hornblende, and biotite. Quartz phenocrysts are common.
Dykes and sills	Rhyolite; range from metres to a few tens of metres wide	320 ± 10 Ma	Aphanitic and aphyric. Intrusive breccias are common along dyke contacts, commonly incorporating both rhyolitic and wall rock fragments within a flow-banded groundmass.
Dykes	Basalt / dolerite; in deposit area range from metres to a few tens of metres wide; in the south-west part of the project can occur as large, sill-like intrusive masses	Carboniferous	Intrude all stratified units. Typically aphanitic to fine-grained, locally vesicular, and contain variable amounts of plagioclase phenocrysts.
Intrusions	Alkaline granite	Permian	Large, circular intrusion exposed just east of Oyu Tolgoi that is defined by abundant pegmatite dykes.

Table 7.4 Major Structures

Structure	Setting	Description
Central Fault	West-north-west striking, moderately north dipping structure that lies between the Hugo South deposit and the Central zone of the Oyut deposit.	Fault consists of several splays and may have experienced multiple periods of displacement. Early fault displacement resulted in north side down apparent offset, followed by a later apparent reverse displacement of lesser magnitude. Visible as linear magnetic feature cutting the overlying Sainshandhudag Formation.
Contact Fault	Low-angle thrust fault generally parallel to bedding; occurs from Heruga deposit in the south to the Hugo Dummett deposits in the north.	Places overturned upper Devonian sedimentary and volcanic rocks belonging to the DA4 unit over upright Devonian sediments of unit DA3b. Does not truncate mineralization.
7100 Fault	North-west strike, steep dip	Offset of north side down, displacement of all rock units
Bogd Fault	East-west strike, steep dip	Oblique slip fault with dextral lateral displacement
Lower Fault	North-north-west striking, moderate dip	Deposit parallel fault, shear zone
110 Fault	East-west strike, moderate dip to North	Boundary between Hugo North and Hugo South
Axial Fault	Hypothetical, based on alignment of the Southwest and Central zones and the Hugo Dummett deposits, and the elongate form of the Hugo Dummett deposits.	Alignments suggest an underlying north-north-east striking fault or fault zone controlled emplacement of porphyry intrusions and related hydrothermal activity.
West Bat, East Bat Faults	North-north-east trending, bounding Hugo Dummett deposits.	Control the structural high, which hosts Hugo Dummett. Offsets of post-mineralization stratigraphic contacts measure at least 1 km (east side up) for the West Bat Fault, and 200–300 m (west side up) for the East Bat Fault.
East Bounding and West Bounding Faults	North-east to north-north-east trending; bounding the Southwest zone.	Form a primary control on the distribution of copper and gold mineralization. Presence of mineralized clasts within the fault zones implies faults were active post-mineralization. East bounding fault is a gently listric, steeply west dipping fault zone in the order of >50 m wide. The fault has been modelled as a series of segments displaced across newly interpreted north-west-south-east trending faults.

Structure	Setting	Description
Bor Tolgoi and Bor Tolgoi West Faults	North-east to north-north-east trending; bounding the Heruga deposit.	Display 300–500 m of west side down apparent offset of stratigraphic contacts.
Boundary Fault system	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. Faults dip steeply to the north or north-west, and occur as strongly-developed, foliated gouge and breccia zones ranging from tens of centimetres to several tens of metres wide.
North-west Shear Zone	Ductile shear zone that cuts across the far north-west corner of the Oyu Tolgoi area.	Wide shear zone with mylonitic to ultra-mylonitic rocks in the centre, grading outward over about 200 m to rocks lacking visible ductile strain. Marks the break between the Alagbayan and Sainshandhudag Formations and the Carboniferous granitic complex.
Solongo Fault	East to east-north-east striking, sub-vertical structure; cuts across Oyu Tolgoi just south of the Southwest and South zones.	Typically occurs as a strongly tectonized, foliated zone up to several tens of metres wide. Forms a major structural break; a minimum of approximately 1,600 m of south side down stratigraphic offset where it juxtaposes mineralized basalt (unit DA1) in the South zone against sediments correlated with the upper Alagbayan Formation (unit DA4) to the south.
North-west trending faults	Oyut	Sub-vertical to steeply north-east dipping faults associated with rhyolite dykes.
East-north-east striking faults	Regional bounding faults at Heruga deposit.	Form prominent features on both magnetic and satellite images. Geological mapping shows a 500 m apparent dextral displacement of dykes and stratigraphic contacts across the faults.

Variations in bedding attitude recorded in both oriented drill core and surface outcrops define two orientations of folds at Oyu Tolgoi: 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 Formation and Alagbayan Group. They may be present in stratified rocks throughout the property, but outcrop and drillhole data are insufficient to define them in many areas. There is no evidence of a penetrative fabric (e.g., cleavage) associated with folding.

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, including grading, scour and fill structures, load casts, and cross-bedding, are sporadically observed in drill core by Oyu Tolgoi geologists along the east flank of the Hugo Dummett deposits. These suggest that parts of the Alagbayan Group are overturned. However, no large isoclinal folds have been mapped from drill core. These folds are cut by dykes of the 366 Ma biotite-granodiorite suite and therefore were formed within the Late Devonian. Such folds and geopetal features are difficult to ascribe to regional tectonic events, and may simply be localized features of rapid facies changes in a proximal submarine volcanic environment.

When completed, a structural mapping programme currently underway at OT LLC may result in the revision of some elements of the interpretations. Preliminary results of this work are discussed in Section 9.4.

8 DEPOSIT TYPES

8.1 Mineral Deposits

The deposits that are incorporated in the current mine plan are the Oyut and Hugo North (Lift 1). The Hugo North (Lift 2), Hugo South, and Heruga deposits are currently outside the mine plan but are included in the Alternative Production Cases outlined in Section 24.

Figure 8.1 is a schematic long section showing the open pit and underground mineralized areas. Figure 8.2 shows the locations of the major deposit areas in relation to the Oyu Tolgoi and EJV license boundaries. The latter figure also indicates the locations of drillhole collars and the type of hole.

The Oyut deposit has historically been treated as a number of separate zones; however, for mining purposes, the one pit (or potential future underground beneath the pit) will extract all Oyut mineralization, and therefore the descriptors in this section have taken the approach that the orebody comprizes a number of mineralized zones within an overall single deposit framework.

Figure 8.1 Schematic Long Section (looking west)

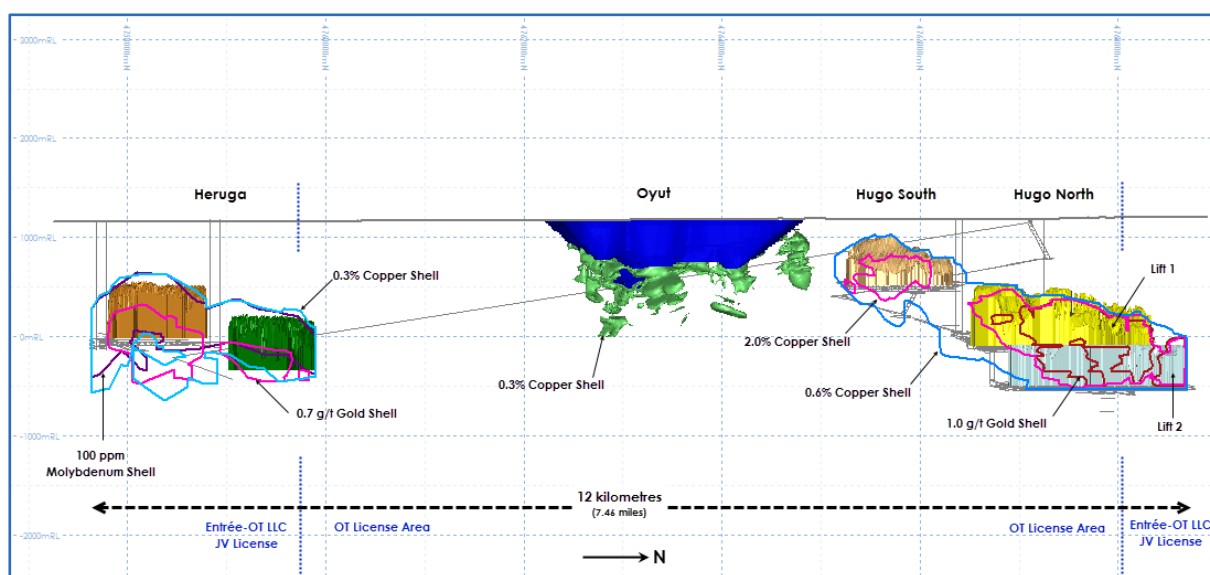
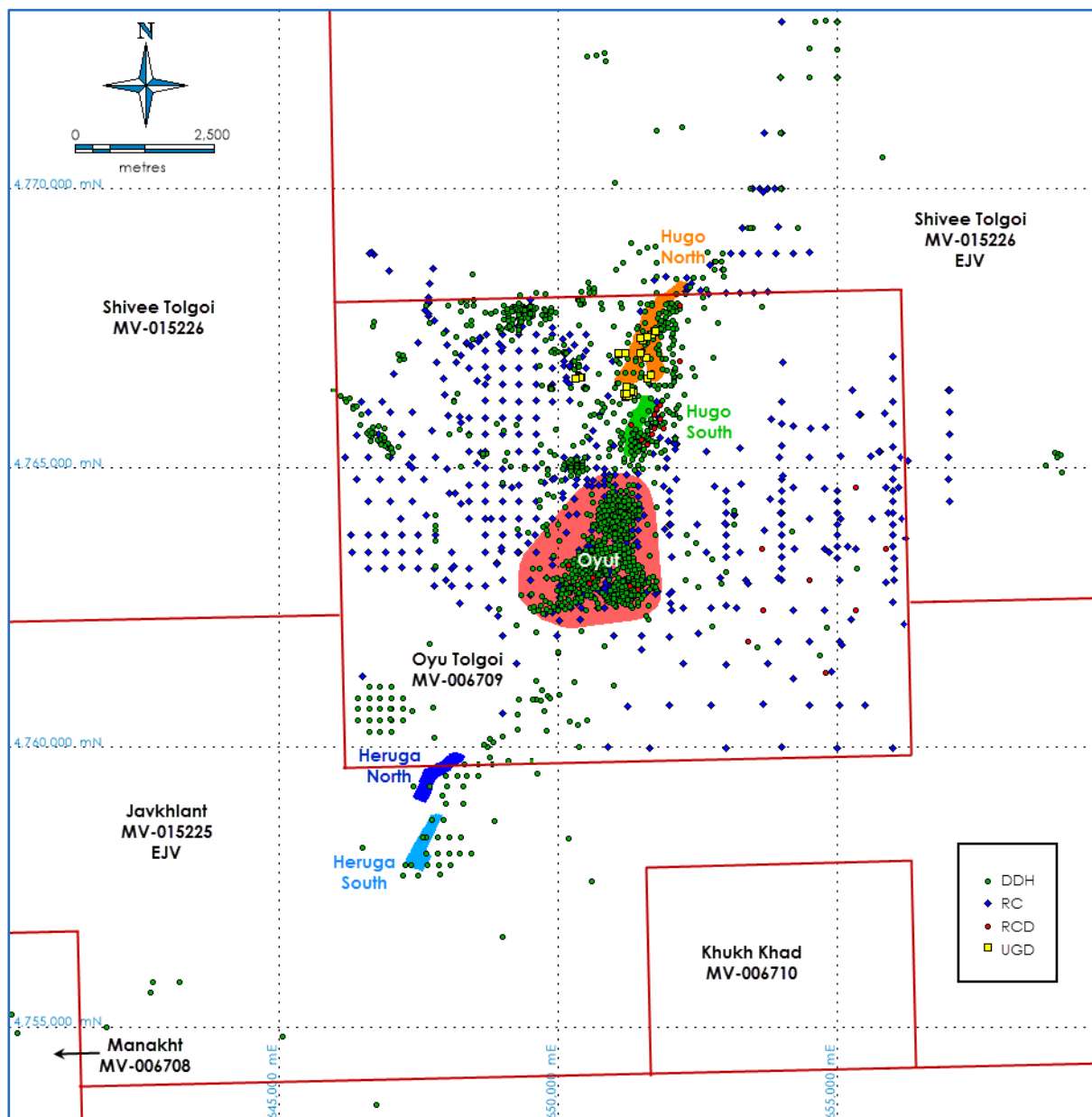


Figure 8.2 Deposit Layout Plan showing Drillhole Collar Locations and Types

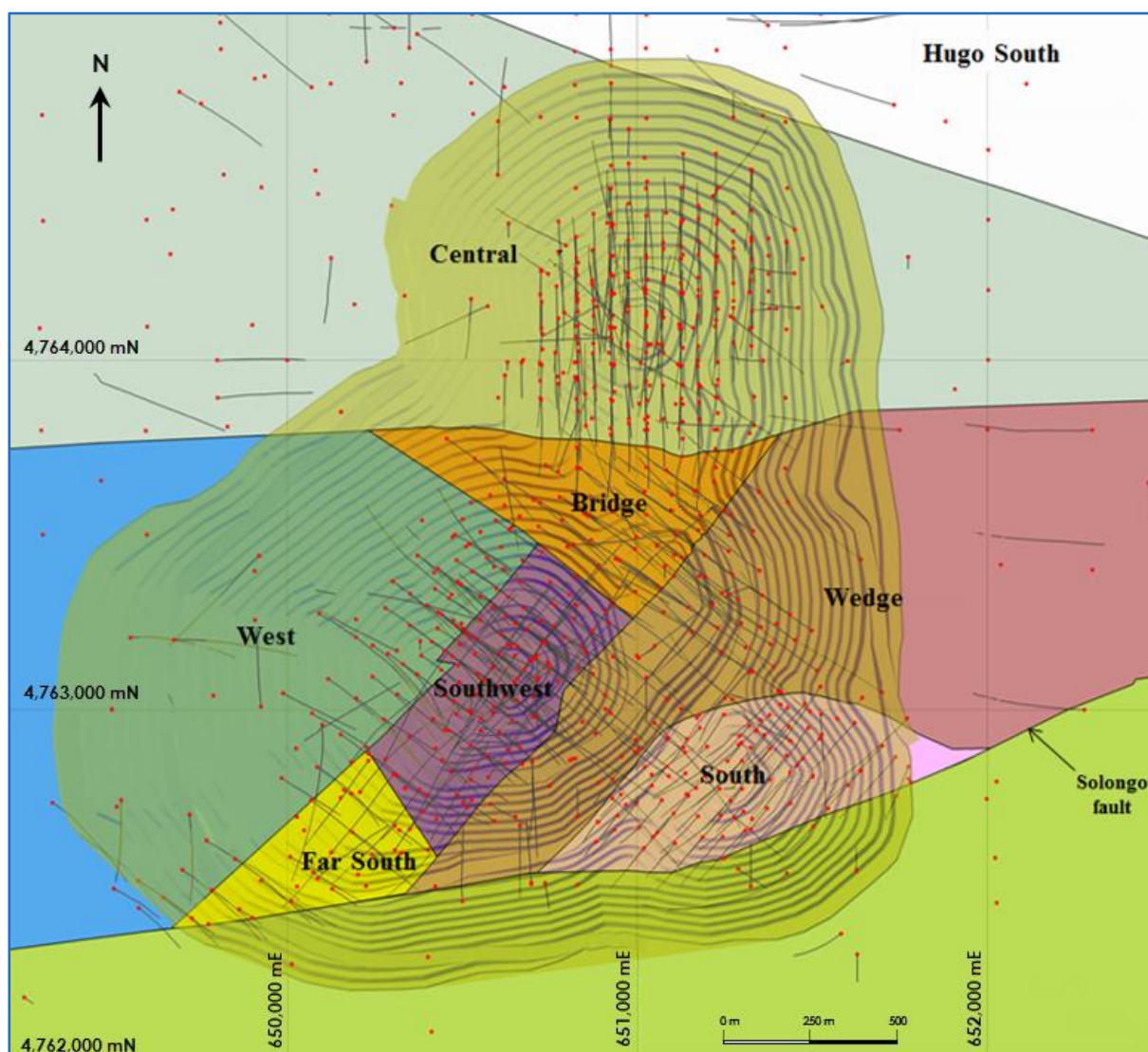


Note: DDH = diamond drillhole, RC = reverse circulation drillhole, RCD = combined RC and DDH drillhole, UGD = underground drillhole, PCD = polycrystalline drillhole (none within limits of image). Deposits represented by current mine plans.

8.1.1 Oyut Deposit

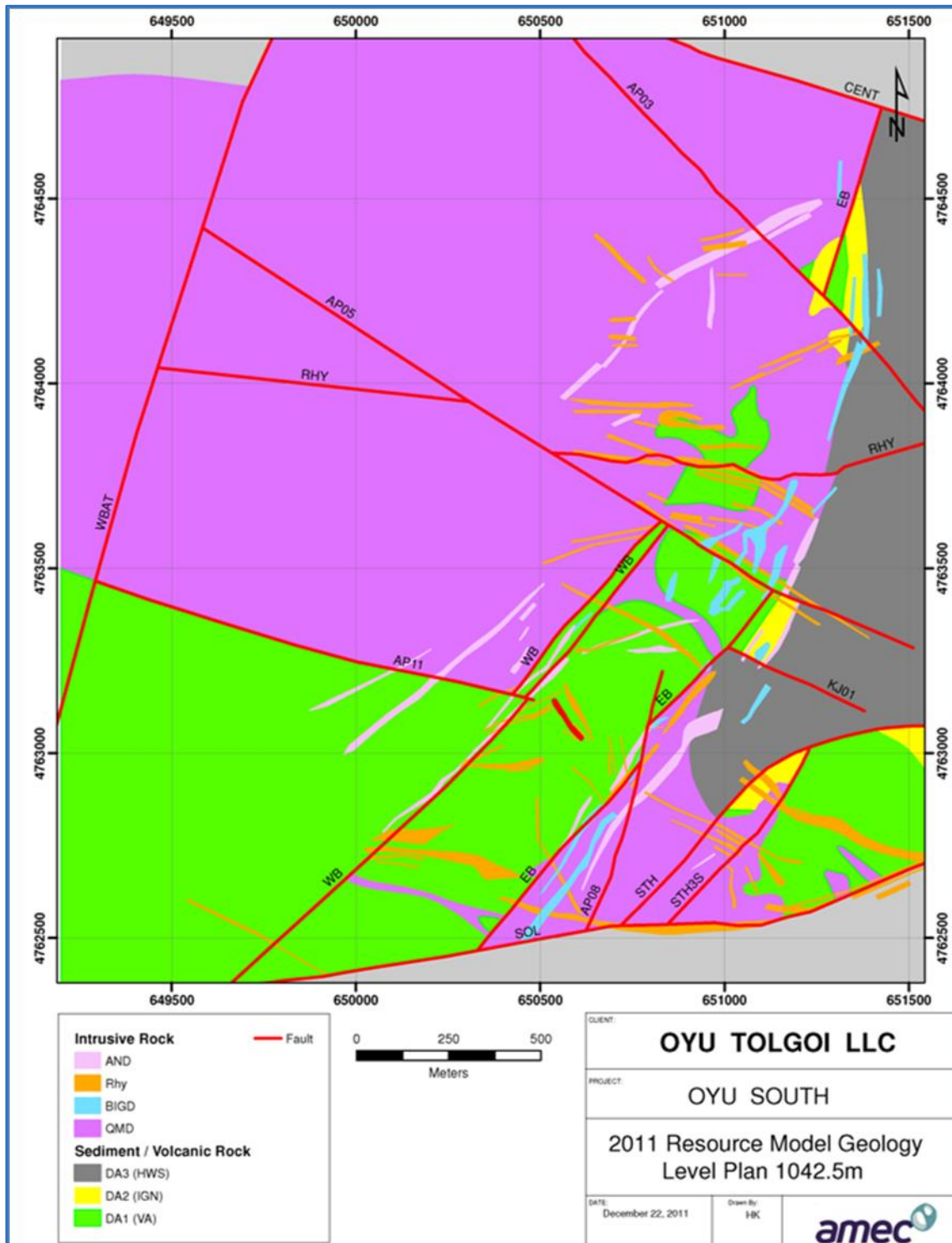
The Oyut deposit includes the main Southwest, South, Wedge, and Central zones (Figure 8.3) and a number of smaller, fault-bounded zones, described in the following subsections. The planned open pit will incorporate the majority of these zones. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

Figure 8.3 Schematic Plan of Oyut Deposit Showing Major Zones



The zones form contiguous sectors of mineralization representing multiple mineralizing centres, each with distinct styles of mineralization, alteration, and host rock lithology. The boundaries between the individual deposits and zones coincide with major faults. Faulting has resulted in different erosional histories for the zones, depending on the depth to which a zone has been down-faulted or uplifted relative to neighboring zones. A level plan showing the simplified structural geology of the Oyut area is included as Figure 8.4.

Figure 8.4 Geology Plan, Oyut Area



Note: WB – West Boundary; EB – East Boundary; WBAT – West Bat Fault; SOL – Solongo Fault; STH – South Fault; STH3S – South 3 Splay; RHY – Rhyolite; CENT – Central Fault. Faults with an AP prefix are from a 2010 structural interpretation by Alasdair Pope. Faults with a KJ prefix are from a 2011 structural interpretation by K. Jennings.

8.1.1.1 Southwest Zone

Dimensions – Southwest Zone

The Southwest zone is a gold-rich porphyry system characterized by a south-west plunging, pipe-like geometry that has a vertical extent of up to 700 m. The high-grade core of the zone is about 250 m diameter; the low-grade shell (0.3% Cu) surrounding the core may extend for distances as much as 600 m x 2 km.

Lithologies – Southwest Zone

Over 80% of the deposit is hosted by massive to fragmental porphyritic augite basalt of the Upper Devonian Alagbayan Group, with the remainder hosted by intra-mineralization, Late Devonian, quartz monzodiorite (Qmd) intrusions.

The quartz monzodiorite intrusions form irregular plugs and dykes related to several distinct phases:

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

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

Structures – Southwest Zone

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

These bounding faults consist of foliated cataclasite, gouge/breccia, and mylonitic bands that occur in zones ranging from a few metres to a few tens of metres wide. The cataclasite within the fault zones contains abundant quartz, quartz sulphide, and sulphide (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 sub-horizontal, sinistral movement on the bounding faults. Both faults have local sub-parallel splays. 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 Group rocks (augite basalt) hosting the deposit, while the West Bounding Fault is mainly intra-formational 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.

Structural Setting – Southwest Zone

Fault geometry and kinematics, vein orientations, and overall geometry at Southwest support a structural model invoking formation of 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 zone reflects the local stress geometry within this zone of dilation. The Southwest zone probably formed as a sub-vertical cylindrical body and attained its present west–south-west plunge during post-mineralization regional deformation. This post-mineralization rotation is consistent with the easterly stratigraphic dips of both pre and post-mineralization rocks in the area.

Mineralization – Southwest Zone

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

Alteration – Southwest Zone

Alteration within the basaltic rocks at Southwest zone 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-sulphidation mineralization in the quartz monzodiorite breccia.

8.1.1.2 Far South

The Far South is considered to be an extension of the Southwest zone, and is separated from it by a major north-west–south-east trending fault. The zone is approximately 100 m x 250 m in area.

8.1.1.3 South Zone

Dimensions – South Zone

The South zone is developed mainly in basaltic volcanics and related to small, strongly-sericite altered quartz monzodiorite dykes. Zone dimensions are about 400 m x 300 m in area, and mineralization extends to depths of more than 500 m.

Lithologies – South Zone

The South zone occurs within an east to north-east dipping sequence of Alagbayan Group 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, which was previously interpreted to be dacitic in composition. However, whole-rock geochemistry has shown that the basaltic tuff sequence is similar in composition to the underlying basalt and to have been affected by advanced argillic alteration to give it the appearance of a dacitic tuff.

To the north-east, the altered and mineralized rocks are overlain by mudstones and conglomerates of the upper Alagbayan Group, which pass up-section into the overlying basalt and sediment sequence and ultimately into rocks of the Sainshandhudag Formation.

The zone is cut by numerous barren dykes, most of which belong to the post-mineralization rhyolite and basalt intrusive suites. These dykes are typically only a few metres wide, with the exception of a major, east-west rhyolite dyke that cuts through the middle of the South zone 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 approximate west to west-north-west strikes and steep dips. In contrast, the basalt dykes have moderate north-east dips, which are sub-parallel to contacts within the stratified host rocks.

Structures – South Zone

The South zone lies within a faulted block that is bounded 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 of several strands over a width of up to 90 m that 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 zone. It forms a wide, strongly tectonized zone. Stratigraphic offset on the Solongo Fault is at least 1,600 m (south block down). Consequently, no significant mineralization has been identified on the south side of the fault at shallow (<1,000 m) levels.

Mineralization – South Zone

Copper mineralization in the South zone is associated with stockworks of thin (usually <10 cm) quartz–sulphide 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.

In the South zone, mineralization is hosted dominantly in quartz monzodiorite in the south-western part of the zone, in basalt throughout the central part, 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 zone. As a result, fracture-controlled sulphide veins are minor, and sulphides are present dominantly as disseminated chalcopyrite, bornite, and molybdenite. Chalcopyrite is the principal copper sulphide, but in higher grade areas bornite locally exceeds chalcopyrite in abundance. Magnetite occurs as disseminations and as veins; small zones with elevated gold values are found locally.

A small zone of high-sulphidation mineralization lies within a quartz monzodiorite breccia in the western part of the zone, 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 zone and consists of malachite, azurite, cuprite, chrysocolla, neotocite, and tenorite, hosted within basalt and quartz monzodiorite.

8.1.1.4 Wedge Zone

Zone Dimensions – Wedge Zone

The Wedge zone has an irregular, crescent-like shape. The zone is about 500–600 m at the widest point in the north, tapering to a width of about 200 m in the south, and approximately 1,400 m long. Wedge zone mineralization extends to depths of over 500 m.

Lithologies – Wedge Zone

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

Mineralized rocks in the Wedge zone are cut by abundant 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 zone and typically strike north-east, parallel to the East Bounding Fault. These intrusions were interpreted as sills, frequently intruding along the stratigraphic contact between the basaltic tuff and the overlying sedimentary strata. In 2011, most contacts were re-interpreted as steep-dipping, north-east trending features. Rhyolite dykes are common throughout the zone and typically have steeply dipping contacts but varied strike orientations.

Structures – Wedge Zone

The Wedge zone is 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 zone against older strata on the adjacent blocks containing the Southwest and South zones. Stratigraphic contacts are relatively continuous between the Wedge zone and the Central zone, implying that displacement on the East Bounding Fault is largely transferred to the Rhyolite Fault (between the Southwest and Central zones), leaving the Wedge and Central zones as a structurally intact block that has been displaced downward relative to the Southwest and South zones.

Fault disruption is common along the contact between the Alagbayan Group 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.

Mineralization – Wedge Zone

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

High-sulphidation mineralization consists of pyrite, chalcopryite, bornite, enargite, covellite, and primary chalcocite in advanced argillic-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 to the north.

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

Alteration – Wedge Zone

Basaltic tuff and breccia within the Wedge zone are 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 sector occurs marginal to the advanced argillic alteration and is progressively overprinted by advanced argillic alteration assemblages with increasing copper grades towards the centre of the zone. 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 zone, sericite and pyrite alteration is present within the quartz monzodiorite.

8.1.1.5 Central Zone

Dimensions – Central Zone

The Central zone is approximately 2,300 m wide and tapers from some 200 m in length to the east to more than 600 m to the west. Central zone mineralization extends to depths of over 500 m.

Lithologies – Central Zone

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

Several phases of intra-mineralization and late-mineralization quartz monzodiorite intrusions have been distinguished in the Central zone 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 zone. The quartz monzodiorite dykes terminate within the base of the sedimentary units of the upper Alagbayan Group.

Basalt flows and basaltic tuffs of the Alagbayan Group 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-mineralization dykes are common in the Central zone and comprise rhyolite, biotite-granodiorite, hornblende-biotite andesite, and dacite dykes. The rhyolite dykes are most abundant, with most being west and west-north-west striking bodies in the southern half and on the periphery of the zone. Biotite-granodiorite dykes along the eastern margin of the zone 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 zone.

Structures – Central Zone

The structural setting within the Central zone is still not well understood.

Drillholes show little evidence of significant post-mineralization faulting, and the mineralogical zoning, grade distribution, and continuity of contacts are consistent with the overall area, being contained in a structurally intact block. However, there is also evidence to suggest that a series of north-west trending faults both bound and displace some of the host augite basalt blocks. Given that the majority of the high-grade mineralization in the Central zone lies within the augite basalt units, some discontinuity of grade match is noted within and between the fault-bounded blocks. Additional work is required to better constrain the structural model for the Central zone.

Post-mineralization 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-mineralization or syn-mineralization faults, if present, are largely obscured by intrusive and hydrothermal overprinting.

The Central zone is overlain to the east by unmineralized conglomerate, mudstone, and siltstone of the hanging wall Alagbayan Group (DA4). Wide zones of breccia and foliated breccia lie 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 zone is juxtaposed against the Southwest zone area by an east–west striking fault that is now occupied by a rhyolite dyke swarm (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 zone show a range in orientations, the most common of which have south-west, west, and north-west dipping attitudes. Vein orientations are similar to those documented in the Southwest zone, although the degree of preferred orientation in the core is weaker in the Central zone by comparison. Similar preferred vein orientations in the Central and Southwest zones suggest that these two zones were formed in a similar structural regime.

However, the Central zone lacks the strong bounding fault control that is fundamental to the form and geometry of the Southwest zone; this may account for the more irregular form of the mineralized body in the Central zone.

Post-mineralization tilting of the Central zone is implied by bedding dips in the enclosing and overlying stratigraphic sequence. By rotating the structural data for the Central zone sufficiently to restore bedding to horizontal, it is indicated that there was a strong preference for sub-vertical veins within the zone at the time of formation.

Mineralization – Central Zone

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

Chalcopyrite–gold mineralization is dominant on the southern and western margins of the Central zone 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 are disseminated in the zone and are also found as fracture fillings. Hematite is pervasive and overprints magnetite.

The high-sulphidation part of the Central zone lacks significant gold and contains a mineral assemblage of pyrite, covellite, chalcocite/digenite, enargite, tennantite, cubanite, chalcopyrite, and molybdenite. The dominant host rocks are dacite tuff and quartz monzodiorite. Higher grade mineralization is associated with disseminated and coarse-grained fracture-filling sulphides 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 sulphide-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 sulphides.

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 are intergrown with, or replace, enargite, and molybdenite occurs locally with quartz.

A supergene enrichment zone overlies the high-sulphidation assemblage and underlies a 20– 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 sulphide and is present 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 the Central zone. Chrysocolla, malachite, and neotocite mineralization is found over a 400 m x 300 m area as a thin, two to four-metre-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 zone, where patchy malachite and neotocite occur.

Alteration – Central Zone

Alteration in the Central zone 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-sulphidation 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 are present 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 the Central zone, 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-sulphidation 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, kao-smectite, illite, and kaolinite are the dominant clay minerals in the leached cap.

8.1.1.6 Bridge Zone

Dimensions – Bridge Zone

The Bridge zone has a triangular shape, tapering from about 500 m wide in the north to about 30 m wide in the south, and is approximately 250 m in length. Bridge zone mineralization extends to depths of over 500 m.

Lithologies – Bridge Zone

The Bridge zone consists of copper-mineralized basalt and quartz monzodiorite between the Southwest and Central zones.

Mineralization and Alteration – Bridge Zone

Low-grade copper mineralization is characterized by lower vein densities than in surrounding zones and is hosted in chlorite and epidote-altered basalt and lesser sericite-altered and albite-altered quartz monzodiorite. Magnetite veinlets post-date the quartz veins but pre-date the main sulphide event.

Chalcopyrite, bornite, and pyrite are mainly disseminated, with fracture-controlled or vein-controlled sulphides being less prominent. There is no clear geological boundary distinguishing the disseminated mineralization from the adjacent peripheral zone mineralization.

8.1.1.7 West Zone

The definition of the West structural zone is based on a soil anomaly and fault interpretations. Drillholes testing the Southwest zone and Far South zone mineralization were collared in the West zone and drilled to the south-east to intercept the mineralization in these zones. To date, results have not been not encouraging.

8.1.2 Hugo Dummett Deposits

The Hugo Dummett deposits, Hugo North and Hugo South, contain porphyry-style mineralization associated with quartz monzodiorite intrusions, concealed beneath a 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. The dividing line between the two deposits is 4,766,300 mN, a location marked by the thinning and locally discontinuous nature of the high-grade copper mineralization (defined by greater than 2% copper). The line, which is broadly coincident with the east striking 110° Fault (refer to Figure 7.6 for the projections of the major faults in the area of the Hugo Dummett deposits), separates the gold-rich and copper-rich zone hosted in augite basalt and quartz monzodiorite of the Hugo North deposit from the more southerly, gold-poor, ignimbrite- and augite basalt-hosted mineralization at Hugo South.

Early technical reports filed by Ivanhoe on the project refer to the Far North zone; this was the initial name for the Hugo Dummett area, and its use has been discontinued. Part of the Hugo North deposit extends onto the Shivee Tolgoi mining license. This area is known as the Hugo North Extension and is referred to as the Copper Flats deposit in technical reports filed by Entrée.

8.1.2.1 Hugo South

Deposit Dimensions – Hugo South

The Hugo South deposit is separated from the Oyut deposit group by the Central Fault and from the Hugo North deposit by the 110° Fault. The deposit is tapered, being approximately 650 m wide and about 850–1,300 m long.

Lithologies – Hugo South

The Hugo South deposit is hosted by an easterly dipping sequence of volcanic strata correlated with the lower part of the Devonian Alagbayan Group and by quartz monzodiorite intrusive rocks. The 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 100–200 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 it has been affected by advanced argillic alteration to give it the appearance of a dacitic tuff. As such, the true boundary between the augite basalt and the fragmental rocks is difficult to determine, being texturally destroyed and diffuse in character.

Weakly altered to unaltered sedimentary and volcanic rocks of the upper Alagbayan Group and Sainshandhudag Formation structurally overlie the mineralized sequence along the eastern flank of the Hugo South deposit. The thickness of the non-mineralized Alagbayan Group sequence commonly exceeds 600 m, although structural thickening within the sequence may be possible. The Sainshandhudag Formation strata unconformably overlie, and are locally faulted against, the Alagbayan Group.

There are several phases of intrusive rocks 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, sub-parallel to contacts in the enclosing stratified rocks. The quartz monzodiorite is broadly contemporaneous with alteration and mineralization; 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, and underlying, the entire deposit.

Late to post-mineralization 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 sub-vertical orientations, or less commonly as easterly dipping sills emplaced along stratigraphic contacts. They are non-mineralized and appear to be volumetrically insignificant except locally in the deposit.

The Hugo South deposit lies within a north–north-easterly elongate block bounded to 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.

Structures – Hugo South

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 (e.g., 110 Fault and Central Fault). The 110 Fault defines the division between the Hugo North and Hugo South deposits, although mineralization is continuous across the fault. The Central Fault is a shallowly to moderately north dipping structure that lies beneath the southern region of the Hugo Dummett mineralization.
- Steep north–north-east-striking faults (e.g., East Fault, West Bat Fault, East Hugo Fault, and Axial Fault). The linear mineralized trend defined by the Central and Southwest zones of the Oyut deposit and the Hugo Dummett deposits likely reflects the presence of a deep, north–north-east striking crustal fault or fault zone controlling magma emplacement and mineralization, termed the Axial Fault. The West Bat Fault is a north–north-east striking, sub-vertical structure that extends along the west side of the Hugo Dummett deposits. 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 East Bat Fault is a north–north-east striking, sub-vertical structure along the east side of the Hugo Dummett deposits. The East Hugo Fault is a north to north–north-west striking, steeply east dipping zone of strong to intense brecciation and clay gouge along the east limb of the Hugo South and Hugo North deposits.
- North–north-east striking faults that dip moderately east, sub-parallel to lithological contacts (e.g., Contact and Lower). The Contact Fault has been interpreted as 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). This interpretation remains to be confirmed. The Lower Fault is as an intensely brecciated, clay gouge-rich zone within the middle or lower part of the mineralized body, typically 200–400 m below the Contact Fault.
- East–west striking, sub-vertical faults (e.g., East–West). The East–West Fault cuts across and displaces the northern end of the Hugo South deposit.

Two orientations of folds were identified 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 are also found in Carboniferous post-mineralization strata, indicating that both events post-date mineralization and may have modified the form of the deposit.

Mineralization – Hugo South

Copper mineralization at the Hugo South deposit is centred on a high-grade zone (typically >2% Cu) 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 basaltic fragmental units. The intense stockwork zone has an elongate tabular form, with a long axis that plunges gently to the north–north-west and an intermediate axis that plunges moderately to the east. Copper grades decrease gradually 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 lies below and to the west in basalt and quartz monzodiorite.

The dominant sulphide minerals at Hugo South are chalcopyrite, bornite, chalcocite, and pyrite, with minor molybdenite, enargite, tennantite, and covellite. Sphalerite and galena are less common. Sulphides are zoned, with bornite $\pm$ chalcopyrite, chalcocite, and tennantite comprising the highest grades ($>2.5\%$ Cu), grading outward to chalcopyrite ($1\%–2\%$ Cu). Pyrite–chalcopyrite $\pm$ enargite, tennantite, bornite, chalcocite, and (rarely) covellite occur in a lower grade zone ($<1\%$ Cu), mainly in advanced argillic-altered basaltic tuff.

Alteration – Hugo South

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 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 are present locally in the high-grade zone in the quartz monzodiorites.

8.1.2.2 Hugo North

Deposit Dimensions – Hugo North

The Hugo North deposit is virtually contiguous with the Hugo South deposit and lies within a similar geological setting. The two deposits are separated by a 110° -striking, $45^\circ–55^\circ$ north dipping fault that displaces Hugo North vertically down a modest distance from Hugo South. Hugo North extends from +500 masl to depths well below 400 masl, has a strike length in excess of 1,800 m, and is 500 m wide.

Lithologies – Hugo North

The host rocks at Hugo North are an easterly dipping sequence of volcanic and volcanoclastic strata correlated with the lower part of the Devonian Alagbayan Group and with quartz monzodiorite intrusive rocks that intrude the volcanic sequence.

The 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 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 Group and Sainshandhudag Formation structurally overlie the mineralized sequence along the eastern flank of the deposit.

Intrusive rocks at Hugo North are dominated by quartz monzodiorite bodies that underlie the entire deposit area and host a significant portion of the copper and gold mineralization. Intrusive contacts are typically irregular but overall show a preferred easterly dip, sub-parallel to the stratification in the overlying rocks. The sub-parallel position of the quartz monzodiorite to the overlying strata may suggest that the intrusion was a sill emplacement that became

tilted to the east, possibly due to the many intrusive events in the Carboniferous. More work is required to confirm this theory.

The quartz monzodiorite bodies are contemporaneous with alteration and mineralization. The quartz monzodiorite is considered to be the progenitor porphyry, and two zones are distinguished on the basis of alteration characteristics and position within the deposit.

A late to post-mineralization biotite-granodiorite intrusion forms a northerly striking dyke complex cutting across the western edge and deeper levels of the deposit. At higher levels, the biotite-granodiorite flares out considerably to form a voluminous body. Although this intrusion locally contains elevated copper grades adjacent to intrusive contacts or within xenolith-rich zones, it is essentially barren.

Based on correlations between drillhole intersections and measurements of individual contacts using oriented drill core, the positions and orientations of dyke contacts are reasonably well established in the Hugo North deposit area. Dominant dyke orientation varies with depth. At levels above approximately 250 mRL, where the biotite-granodiorite cuts through the non-mineralized hanging wall strata, it is present as a single intrusive mass with contacts dipping moderately to steeply to the west. The hanging wall sequence model should identify the nature of the contact between the hanging wall strata and the biotite-granodiorite and assist in modelling the subsidence zone. Below this level, the biotite-granodiorite is more complex, found as multiple and sub-parallel to anastomosing dykes that cut through the quartz monzodiorite intrusion and mineralized Alagbayan Group strata.

Structures – Hugo North

The Hugo North deposit lies 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 east–west and north-east striking faults near the northern boundary of Oyu Tolgoi. The structural geometry and deformation history of the Hugo North deposit generally similar to those of the Hugo South deposit.

Several iterations of the structural framework have been modelled from 2007 through 2014. The following paragraphs describe the structural setting of the Hugo North deposit as understood from the 2014 modelling. The structural interpretation was updated in 2014.

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 (e.g., 110 Fault). The 110 Fault defines the division between the Hugo North and Hugo South deposits.
- East–west striking, steeply dipping faults with locally varying dips between north and south (Bogd, Bumbat, Dugant, and Blacktail). These faults offset the lithology and mineralization in oblique slip fashion (dextral displacement).
- Steep, north–north-east-striking faults (e.g., East Bat Fault and West Bat Fault, 160 Fault, and Axial Fault). The linear mineralized trend defined by the Central and Southwest zones of the Oyu deposit and the Hugo Dummett deposits likely reflects the presence of a deep, north–north-east striking crustal fault or fault zone controlling magma emplacement and mineralization, termed the Axial Fault. The sub-vertical, north–north-east striking West Bat Fault runs along the west side of the Hugo North deposit and cuts the western edge of the northern part of the deposit.

- The north–north-east striking East Bat Fault follows the east flank of the Hugo Dummett deposits, well east of the known deposit extents. The 160 Fault can be traced through the southern part of the Hugo North deposit, where it cuts across stratigraphic contacts at moderate angles and forms a sharp break in alteration intensity and copper grade.
- North–north-east striking, moderately east dipping faults sub-parallel to lithological contacts (e.g., Contact, Lower, and Intermediate faults). The Contact Fault is a bedding-parallel detachment zone that lies at the contact between tectonized mudstones that stratigraphically overlie the deposit (unit DA3) and overlying basalt flows and volcaniclastic rocks (unit DA4). The Lower Fault at Hugo North is an intensely brecciated, locally foliated, clay-rich gouge zone within the middle or lower part of the high-grade mineralized body, typically at a level 200–400 m below the Contact Fault. The Intermediate Fault is sub-parallel to the augite basalt (Va) and ignimbrite (Ign) contact.
- East–north-east striking faults (e.g., Boundary Fault System, Kharaa Fault, Eroo Fault, and Rhyolite Fault). The Boundary Fault follows the intrusive contact of the granitic complex in the north-west part of the Oyu Tolgoi area and juxtaposes strongly mineralized rocks against post-mineralization Carboniferous strata near the northern property boundary. The North Boundary Fault juxtaposes Carboniferous granitic rocks against Carboniferous strata to the south. The sub-vertical, east–north-east striking Rhyolite Fault cuts across the southern part of the Hugo North deposit and coincides with a wide zone of rhyolite dykes. The Kharaa Fault and the Eroo Fault may be reactivated splays associated with the North Boundary Fault System.
- North-west striking faults that offset (oblique slip) lithologies and in some case mineralization (7100 Fault, Noyon Fault, Gobi Fault, Burged Fault, Javkhlant Fault). These faults may have an oblique slip component.

Fold styles and orientations in the Hugo North deposit are similar to those at Hugo South, with most folding restricted to the upper part of the Alagbayan Group and overlying Sainshandhudag Formation.

Mineralization – Hugo North

The highest grade copper mineralization in the Hugo North deposit is related to a zone of intensely stockworked to sheeted quartz veins known as the QV90 zone, so named because >90% of the rock has >15% quartz veining. The high-grade zone is centred on thin, east dipping quartz monzodiorite intrusions or within the apex 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 and has a high Au (ppm) to Cu (%) ratio of 0.5 : 1.

Bornite is dominant in the highest grade parts of the deposit (3%–5% Cu) and is zoned outward to chalcopyrite (2% Cu). At grades of <1% Cu, pyrite–chalcopyrite dominates. Within the upper levels where advanced argillically altered basaltic tuff is reported, the assemblage comprises pyrite–chalcopyrite ± enargite, tennantite, bornite, chalcocite, and more-rarely, covellite.

The high-grade bornite zone consists of relatively coarse bornite permeating quartz and disseminations in wall rocks, usually intergrown with subordinate chalcopyrite. Pyrite is rare to absent, except locally where the host rocks are advanced argillically altered. Although chalcocite is commonly found with bornite at Hugo South, it is less common at Hugo North.

High-grade bornite is associated with minor amounts of tennantite, sphalerite, hessite, clausenthalite, and gold that 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. The quartz monzodiorite in the lower zone exhibits a characteristic pink to buff colour, with a moderate intensity of quartz veining (5%–25% by volume), and is characterized by finely disseminated bornite and chalcopyrite. Sulphides are disseminated throughout the rock in the matrix as well as in quartz veins. The fine-grained bornite has a black sooty appearance. Red colouration of the rock type is attributed to fine hematite dusting, primarily associated with albite.

Alteration – Hugo North

The Hugo North deposit is characterized by copper–gold porphyry and related styles of alteration similar to those at Hugo South. These include zones of 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. 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 lie mainly along the outer and upper margins of the intrusive system.

Chlorite–illite marks the outer boundary of the advanced argillic zone, 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. As with elsewhere within the property, the advanced argillic alteration is texturally destructive and often obliterates the contact between the augite basalt and overlying basaltic tuff. As a result, a diffuse lithological and mineralization contact typically characterizes this zone.

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.

Hematite–siderite–illite–pyrophyllite–kaolinite–dickite (intermediate argillic) is an inward zonation from the advanced argillic zone. It 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.

8.1.2.3 Hugo North Extension

The Hugo North Extension is a term used to define that part of the Hugo North deposit that extends into the EJV ground. The zone extends north from the license boundary for approximately 700 m and appears to be closed off to the north; drilling on a section approximately 150 m north of the northernmost extent of the Hugo North Extension grade shell has indicated that mineralization is truncated by an east–west trending fault. North of this fault, the prospective stratigraphy has been down-dropped to depths greater than 2,000 m below the surface.

8.1.3 Heruga

The Heruga deposit is the most southerly of the currently known deposits at Oyu Tolgoi. Heruga is a copper–gold–molybdenum porphyry deposit and is zoned with a molybdenum-rich carapace at higher elevations overlying gold-rich mineralization at depth. The top of the mineralization starts 500–600 m below the present ground surface.

The deposit has been drilled over a 2.3 km length, is elongated in a north–north-east direction, and plunges to the north. Exploration of the down-plunge extension was active as at 31 March 2012. The northern boundary of the mineralization is assumed to be the Solongo Fault, which marks the southern boundary of the planned Oyut open pit.

Quartz monzodiorite intrusions intrude the Devonian augite basalts as elsewhere in the district, and again are considered to be the progenitors of mineralization and alteration. Within Heruga itself, quartz monzodiorite intrusions are small compared to the stocks present in the Hugo Dummett and Oyut areas, perhaps explaining the lower grade of the Heruga deposit. Unmineralized dykes, which make up about 15% of the volume of the deposit, cut all other rock types. However, the quartz monzonite body appears to flare to the east and forms a large stock within the Heruga North area of interest.

The deposit is transected by a series of north–north-east trending vertical fault structures that step down 200–300 m to the west and have divided the deposit into at least two structural blocks.

Mineralized veins have a much lower density at Heruga than in the more northerly Oyut and Hugo Dummett deposits. 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.

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. 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.

There is no oxide zone at Heruga. No high-sulphidation style mineralization has been identified to date.

8.1.4 Exploration Potential

The project has significant exploration prospectivity, and exploration activities are ongoing. The OT LLC emphasis has shifted from drilling to data acquisition, compilation, and interrogation to identify and prioritize a pipeline of exploration targets on the mine licenses.

The objective is to develop low-cost options with the potential to directly improve the value of the operations and to focus on deposits that could represent enhanced opportunities in line with the current development of the Oyu Tolgoi orebodies.

The Exploration Team is also continuing to build legacy datasets with a focus on ground magnetics, geophysical modelling, geochronological studies, core re-logging, and surface geological mapping programmes.

8.1.4.1 Heruga North

The Heruga North zone, which in earlier reports was referred to as the New Discovery zone, is the down-plunge extension of the Heruga mineralization. The top of Heruga North is approximately 1,100 m below surface and plunges gradually downward as it extends to the north. The Solongo Fault forms the projected northern limit of mineralization associated with Heruga.

An exploration drilling programme was completed at Heruga North in late-2012. Since that time the fundamental data have been validated and a preliminary structural and geological model has been created. The next step is to estimate the resources of the entire Heruga and Heruga North system.

8.1.4.2 Javkhlant

The Javkhlant target area is an area of interest within the current exploration work programme. The Javkhlant target was originally identified from regional IP surveying in 2005. EJD0035A, drilled in 2010, encountered an intercept of 30 m at 0.92% Cu from a depth of 1,422 m.

The Javkhlant target is being advanced through geophysical and geochemical analysis to determine prospectivity and identify potential drilling targets.

At present, this is the southernmost known mineralization on the Oyu Tolgoi trend.

8.1.4.3 Hugo West Shallow

Hugo West Shallow was identified in 2013 following a review of the current drilling data, mapping, and geophysics. It is located in the north part of the Oyut block that hosts the current open pit resources. The deposit is a small porphyritic intrusion on the footwall of the Central Fault. Drilling completed on this deposit in Q4'13 and Q1'14 identified low-grade mineralization. The mineralization is hosted in the Devonian host sequence and Qmd intrusives and is characteristic of the Oyut zones.

8.1.4.4 Hugo West

The Hugo West Deep target was identified in 2012 by a magnetotellurics (MagTell) survey and geological review. The area was drill tested in 2013. The first drillhole returned 502 m at 0.54% Cu and 0.32 g/t Au. The mineralization associated with the Hugo West Deep target is hosted entirely within the Qmd intrusives. The mineralization is currently thought to represent an extension of the grade shells associated with Hugo North and Hugo South. The drilling has identified a large, low-grade target that will require future work to identify potential resources. The target area is currently being studied to identify future drill testing requirements based on business needs.

8.1.4.5 Deep Targets

Three deep, high-grade targets have been identified by the exploration reviews: West of the West Bat Fault, North of the West Bat Fault, and Heruga North bornite. These three targets are being studied via various geological, geophysical, and geochemical methods. OT LLC exploration review will aim to identify future potential drilling requirements, if any.

8.1.4.6 Future Exploration Strategy

Exploration will continue on the Oyu Tolgoi mining licenses, scaled to meet the business strategy and in line with annual budgets. The focus will be on resources that could change the schedules in the Alternative Production Cases and defer development of deeper and lower grade resources. The work aims to identify smaller, incremental additions to the resource base, increase knowledge of the orebodies at the known deposits, and plan infill drilling as part of a longer term goal to convert resources into reserves.

9 EXPLORATION

OT LLC plans to continue exploration on the Oyu Tolgoi project mining licenses. The focus will be on resources that will increase the mine life and potentially defer development of deeper and lower grade resources. Smaller incremental additions to the resource base are to be sought as greater knowledge of the orebodies at the known deposits, specifically geotechnical considerations, through infill drilling as part of a longer-term goal to convert resources to reserves. In addition to exploration drilling, further work will be conducted to improve confidence in inferred-level resources at the Hugo Dummett deposits, particularly around Lift 1 Panels 3–5, Lift 2, and Hugo South.

9.1 Fundamental Data

9.1.1 Grids and Surveys

Coordinates used by IMMI, and subsequently by OT LLC, 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 licenses are defined by latitude and longitude coordinates. The official Mongolian survey datum was MSK42 using the Baltic mean sea level as the elevation datum. Ivanhoe noted the coordinates obtained using the MSK42 data were almost identical to those using WGS-84, Zone 48N.

Various topographic surveys have been completed on the project area, the most recent of which was in 2010 by Geomaster, covering a 10 km x 10 km area using an electronic total station instrument with an accuracy of 5 cm. The survey had a contour interval of 1 m.

In 2011, the governing authority in Mongolia changed the official survey datum to WGS-84, Zone 48N. As a consequence, there has been a small shift in the bounding coordinates of the licenses. Geomaster completed a new survey of the boundaries using the total station instrument on behalf of OT LLC.

OT LLC acquired a GeoEye satellite image in 2012 and used it to derive a new topographic map.

9.2 Imaging

In 2001, Ivanhoe commissioned Pacific Geomatics from Vancouver to produce 1 : 100,000 scale Landsat satellite images and a structural and alteration interpretation over a 1,500 km² area centred on the project. These data are integrated into a GIS database and have been used to aid in the structural interpretation of the project and for alteration mapping.

In 2003, Ivanhoe requested that Pacific Geomatics provide Quickbird imaging over the entire OT License.

In 2012, OT LLC engaged Fugro Spatial to acquire GeoEye imagery over the entire area of the mining licenses and to extend this coverage along key infrastructure corridors such as the Gunii Hooloi water borefield and the road to the China-Mongolia border. Resolution is approximately 0.5 m in the vertical and horizontal components.

9.3 Geological Mapping

9.3.1 Surface Mapping

Geological mapping programmes have been restricted by the paucity of outcrop in the project area.

Outcropping mineralized zones (Southwest, South, and Central) were mapped at 1 : 1,000 scale and the central part of the OT License at 1 : 5,000 scale in 2001. The entire OT License area was mapped at 1 : 10,000 scale in 2002. Additional geological and structural mapping was completed by Alan Wainwright during 2005–2008 as part of his PhD thesis research.

Mapping on the Shivee Tolgoi license consists of 1 : 20,000 and 1 : 10,000 scale regional mapping, with detailed prospect-scale mapping at 1 : 2,000 scale, undertaken between 2004 and 2008.

In 2011, a detailed 1 : 2,500 surface geological mapping programme was initiated across part of the Javkhlant area west and south-west of Heruga. This programme focused on determining stratigraphic relations that may indicate vectors to prospective stratigraphy.

The long-term aim is to complete a detailed geological map of the entire OT License area at a scale of 1 : 5,000.

9.3.2 Underground Mapping

Detailed geological mapping has been undertaken on exposed development faces on the 1300 Level in the Hugo North underground workings. The mapping was done initially on paper sheets, which were scanned, imported to Vulcan software, geo-referenced, and converted to lithological and structural strings for interpretation.

The mapping was used to help predict ground conditions in front of planned development and to validate the geology model interpreted from drillholes. The 2014 geology model update incorporates useful information from the underground mapping, such as the location and nature of contacts between the BiGd and Qmd units in areas of sparse drilling coverage. Underground mapping is planned for the underground development.

The following programmes will help define the relationships of the fault structures:

- In addition to the underground mapping data used in the updated 2014 Hugo North geology model, efforts to improve the understanding of the faults on the 1300 Level include re-logging of some underground core. Core photos were re-examined to look for fault zones that had not previously been recognized by the core loggers. It is planned that the underground geology mapping will be integrated with the Adamtech photos, geotechnical mapping data, and the previous underground mapping by Ivanhoe to create a detailed map of the 1300 Level underground.
- The hanging wall sequence (HWS) east of the Hugo North deposit (Carboniferous units adjacent to the BiGd) and west of the West Bat Fault still require modelling. This was not done in the past because modelling was focused on the orebody. This work will lead to a better understanding of the continuity of structures across the West Bat Fault and the relationship with the overlying Carboniferous sediments.

9.4 Structural Studies

9.4.1 Oyut

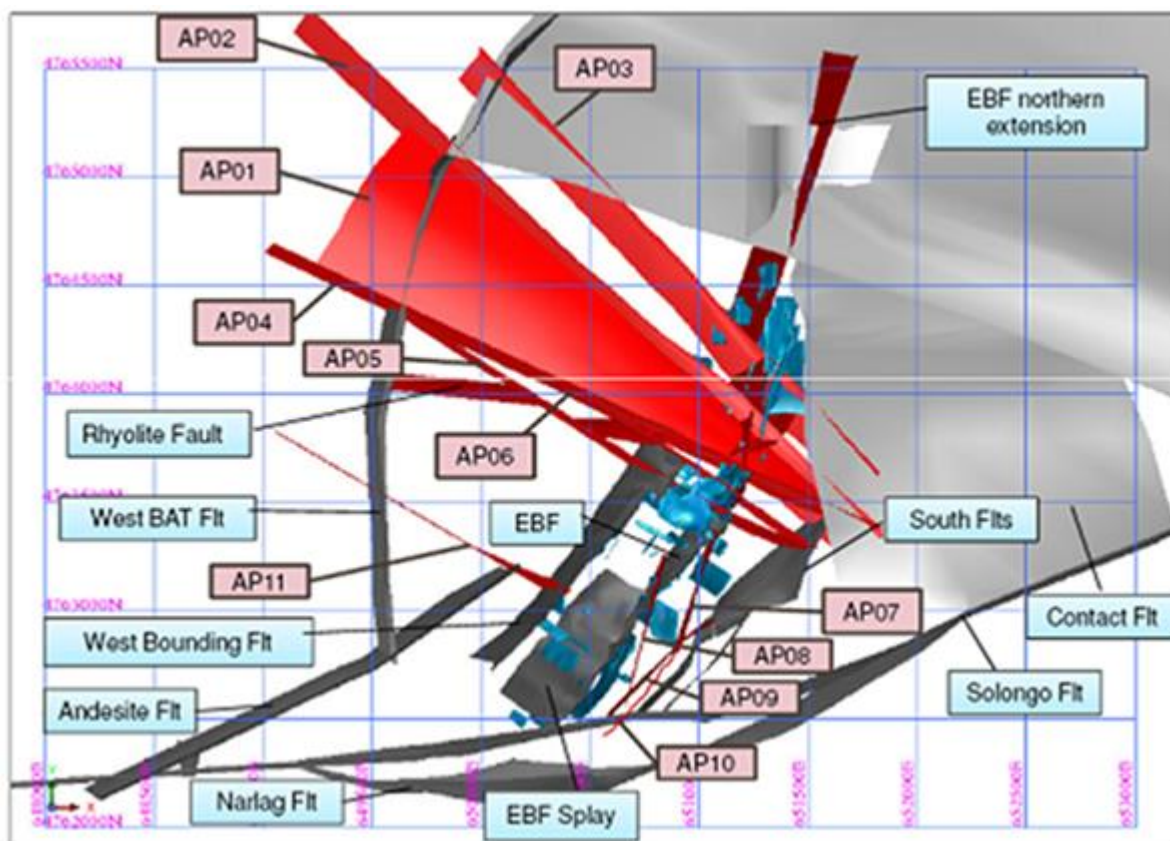
During 2010–2011, the structural and geological model for Oyut was updated with all then-currently available information and based on level plan interpretations. Major faults identified from this programme are incorporated into the 2011 Oyut resource block model.

A provisional fault hierarchy for the area was established:

- Early-stage: north–north-east trending faults (East Boundary Fault, West Boundary Fault, and Andesite Fault).
- Middle-stage: north–north-west trending faults (AP01 Fault, AP02 Fault, AP03 Fault, AP04 Fault, AP05 Fault, AP06 Fault, AP011 Fault, and Rhyolite Fault).
- Late-stage: east–north-east trending faults (Solongo Fault system and associated splays).

Figure 9.1 shows the current fault interpretation for Oyut.

Figure 9.1 Fault Locations and Orientations (2011 Interpretation)



Key elements from the work completed to date are as follows:

- 11 new faults were modelled from level plan interpretations.
- Faults in the southern part of the Hugo Dummett fault model were extruded to the margins of the model for convenience. The result is that the northern part of the Oyut open pit and Hugo South fault models overlap, but do not actually link together structurally or geologically. Rectification of this issue is planned and is critical to obtaining a district-wide coherent structural architecture.
- The East Boundary Fault was extended to the north, re-modelled as a steeply west dipping, gently listric fault, and interpreted as a >50 m wide fault zone characterized by fault gouge, lithological repetitions, and a swarm of BiGd dykes. The fault has been modelled as a series of segments displaced across newly interpreted north-west-south-east trending faults.
- The geologically unusually shallow-dipping BiGd intrusions in the Central Zone were re modelled as steeply dipping dykes, compatible with the BiGd intrusions associated with the well-constrained East Boundary Fault in the Southwest zone. The dykes are interpreted to be intimately associated with extensional faulting along the East Boundary Fault. The orientation of these dykes presents a potential geotechnical risk, and further work will be required to confirm the sub-vertical reinterpretation. This is currently being addressed through a programme of pit mapping and reconciliation to the resource geology model.
- The previously modelled east-west trending Rhyolite Fault between the Southwest and Central zones, which was the northernmost fault in the 2007 model, was retained but re-modelled into segments separated by the newly interpreted north-west-south-east trending faults.
- The north-north-west trending faults appear to be sub-vertical to steeply north-easterly dipping. Faults AP04, AP05, and AP06 are coincident with a swarm of presumably synchronously intruded rhyolite dykes (which have the same strike, but shallower, 60°–70° dips) between the Central and Southwest zones.

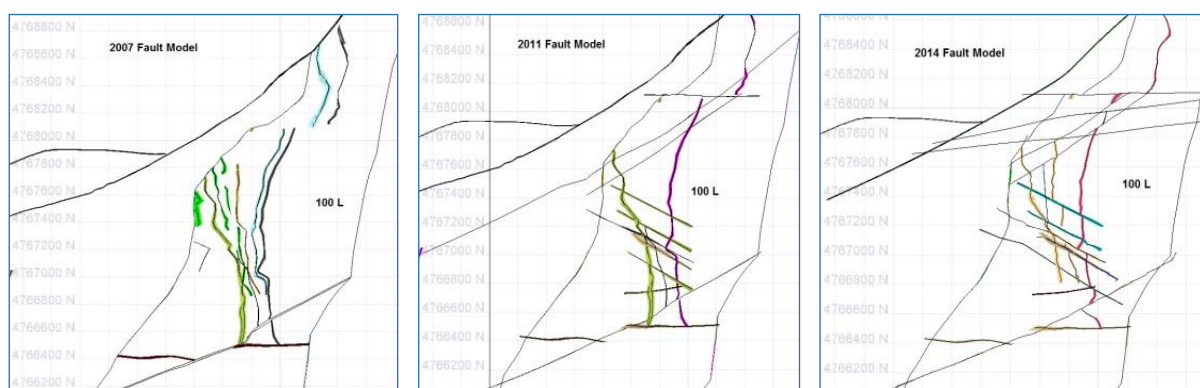
It was concluded that:

- The East Boundary Fault is likely to form a significant geotechnical and hydrogeological feature during mining operations.
- Changes in interpretations of the dip of the BiGd intrusions, when confirmed, could have significant benefits for slope design. In the previous model, BiGd dykes are not only parallel to the slope, but also immediately below the ramps on the east side of the pit, posing a major slope stability hazard. It was recommended that the BiGd units in the Central zone be re-modelled, with geotechnical holes sited to determine whether the BiGd units are favourably steeply dipping (as in the 2011 work), or unfavourably dipping to the west, parallel to the proposed Central zone pit slope (as previously modelled).

9.4.2 Hugo Dummett

The interpretation of the structural framework of Hugo North has evolved over time. OT LLC, performed an initial structural review of the faulting and fault models during 2009–2010 for the Hugo Dummett deposits area in support of the planned block cave mining operation. The result was a preliminary fault model based on apparent displacements of geological boundaries, coupled with interpretation and analysis of structural data collected from drill core. These interpretations were used as a guide for the 2011 and 2014 structural framework used to construct the Hugo North geology model (Figure 9.2).

Figure 9.2 Hugo North Progression of Fault Models 2007–2014



The 2011 geology model introduced several north-east and north-west trending faults to the Hugo North geology model. These structures were added based on faults in the drillholes and to accommodate offsets in lithology that were previously explained by folding and the thickening and thinning of the volcanic package. The cross-sectional and plan view interpretations in the 2014 model added an additional set of east–west faults in the north-east extension area to explain the sudden directional change of the strike of the orebody from north–south trending to north-east trending. These faults are steep-dipping, but dip direction may change locally from north to south. These east–west faults are interpreted as being late developments.

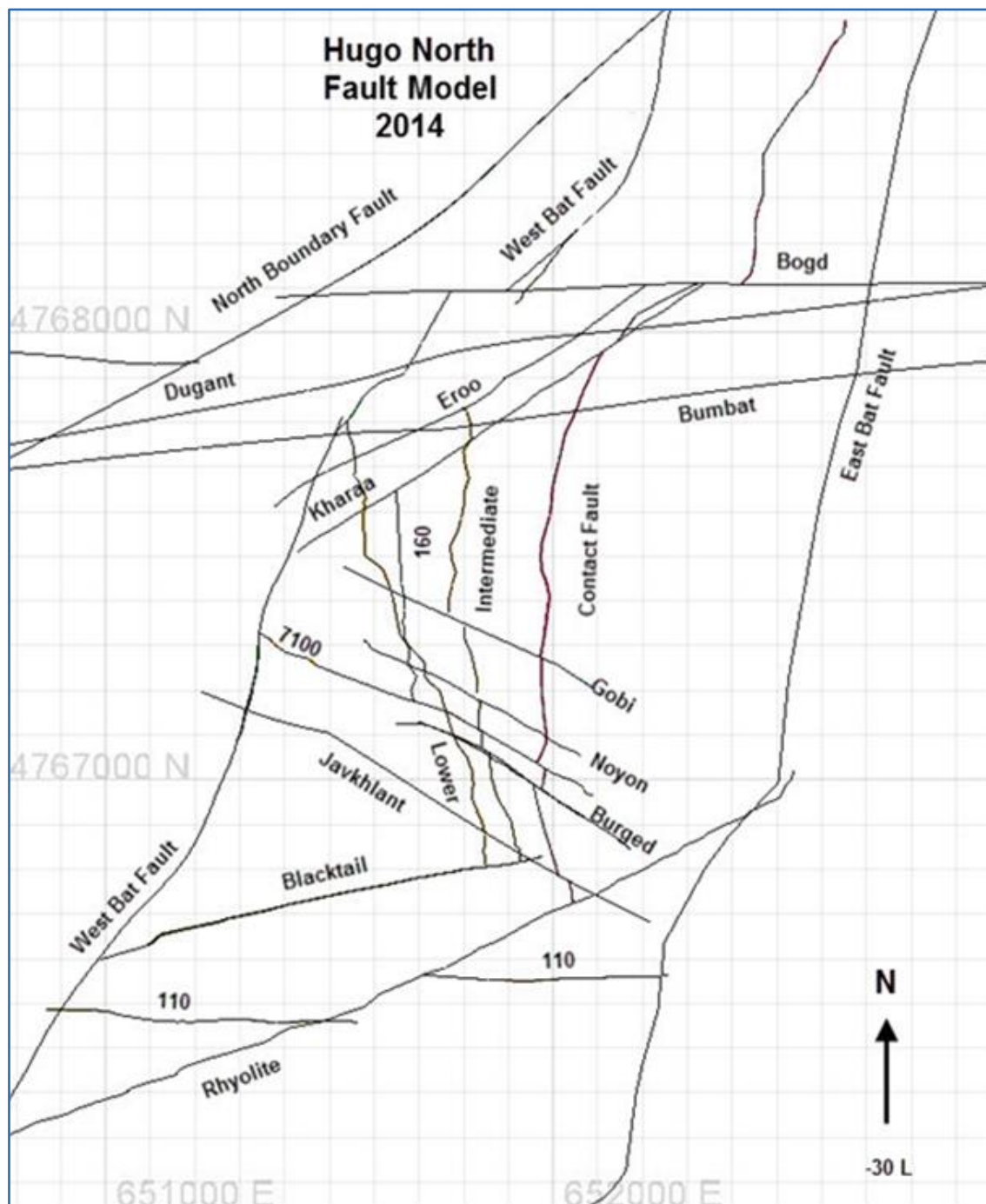
During 2010–2011, a significant amount of work was undertaken to review the geological and structural setting of the Oyu Tolgoi mineralization, particularly in the Hugo North area. This work formed the basis of the 2011 structural interpretation of Hugo North, which added a series of north-east and north-west faults and one east–west trending faults in addition to the legacy faults from the 2007 geology model. The 2014 geology model then added east–west faults in the north-east extension area to explain the directional change of the orebody from north trending to north-east trending as a series of dextral offsets.

The additional faults that were modelled in each year of the model updates are as follows (Figure 9.3):

- 2007: The Legacy faults include the Contact Fault, Lower Fault, Intermediate Fault, 160 Fault, 110 Fault, West Bat Fault, East Bat Fault, North Boundary Fault, Rhyolite Fault, and 7100 Fault.

- 2011: Kharaa Fault and Eroo Fault (north-east trend), Noyon Fault, Gobi Fault, Javkhlant Fault, Burged Fault, Kharaa-Suult Fault (north-west), and the Selenge Fault (east-west).
- 2014: Bogd Fault, Bumbat Fault, and Dugant Fault (east-west). Bogd Fault replaces the Selenge Fault.

Figure 9.3 Hugo North Fault Locations 2014



Key elements from the work completed to date are as follows:

- The north-west and north-east trending faults effectively broke apart and offset the deposit-parallel faults (Contact Fault, Intermediate Fault, Lower Fault).
- There is an offset in lithology and mineralization along the 7100 Fault, indicating an oblique slip nature.
- The Kharaa Fault and the Eroo Fault may be (reactivated?) splays from the North Boundary Fault.
- The Bogd Fault, Bumbat Fault, and Dugant Fault offset the mineralization and lithology and have an oblique slip component associated with them, including significant dextral movement across the Bogd Fault.
- If the deposit is turned on its side, then the north-west and north-east faults form a basin such that the volcanic units are close to horizontal. This theory is conceptual at present. The model of the hanging wall sequence should help in understanding this development in the ore deposition cycle.

An elementary fault hierarchy for Hugo North was established, although more work needs to be done to understand and confirm the fault hierarchy:

- Early-stage: The deposit-parallel faults (Contact Fault, Lower Fault, Intermediate Fault, and 160 Fault).
- Mid-stage: The north-west trending faults are earlier than the East Bat and West Bat faults. However, the 7100 Fault cuts the West Bat Fault and therefore may have been reactivated.
- Mid to late-stage: North Boundary Fault, Kharaa Fault, Eroo Fault, 110 Fault, and Rhyolite Fault.
- Late-stage: East-west trending oblique slip faults in the north-east extension area (Bogd Fault, Bumbat Fault, and Dugant Fault).

Future work to locate the faults in the hanging wall sequence (HWS) will target the BiGd and volcanic sequence contact above the block cave and model the faults on the west side of the West Bat Fault. The HWS model should help the understanding of whether or not certain faults cut across the West Bat Fault and if the BiGd / HWS contact is faulted.

9.4.3 Heruga

OT LLC updated the Heruga structural and geological model in 2013 and this work was reviewed by Alasdair Pope (Rio Tinto Principal Structural Geologist) in October 2013.

The surface map shows a north-east trending Carboniferous syncline axial trace directly above the Heruga deposit footprint. Level plans of 1 km down in the mineralized zone show a Devonian core with Carboniferous on the flanks, i.e. an anticline. The corresponding anticline axial trace at surface lies approximately 500 m to the east, suggesting that the axial surface dips in the order of approximately 60° to 70° to the west-north-west.

On level plans and cross-sections, the anticline looks like a positive flower structure (i.e. transpressive – a pop-up structure) developed in an east dipping homocline, but the surface map shows anticline and syncline fold closures. This may be a potentially important

difference in trying to understand the genesis and geometry of the deposit, and may indicate the presence of more mineralization in the immediate vicinity.

The current model has been built from cross-sections. Due to wide drillhole spacing on the flanks of the deposit, the model has an irregular, saw-tooth geometry when viewed in level plans. By concentrating on individual distinctive marker units and smoothing the interpretation to something that looks geologically reasonable, the geological setting of the deposit, in the faulted core of an anticline, becomes apparent.

Faults have been modelled as vertical. However, the easterly offset surface anticline axial trace and parallel related faults suggest that the north-east trending faults may also dip steeply west. This is also suggested by sequential level plans where the mineralized Qmd in the northern part of the deposit migrates across three fault panels while maintaining the same contact relationships and showing no sign of displacement. If faults seen in drillholes have been projected vertically to surface and to depth, then the modelling could conceivably have included more faults than are actually present, instead of correlating fewer, steeply west dipping faults.

There is a generally modest to poor correlation between faults in the Heruga model and those mapped on surface. It is important to link poorly drill-defined faults, and often barely more than conceptual faults, to the well-defined surface fault traces as part of the process of integrating the sub-surface geology with the surface in order to de-risk the interpretation. Interpreted faults that do not correspond with, or at least fit the pattern of, surface-mapped faults can be considered high-risk. The degree of non-correlation between modelled and surface-mapped faults is an indicator of the level of uncertainty and hence geological risk in the model.

9.5 Geochemical Surveys

Work completed on the EJV area includes trenching, soil and mobile metal ion (MMI) sampling, rock chip and grab sampling, and stream sediment and pan concentrate sampling. The total geochemical dataset is summarized in Table 9.1.

Although Ivanhoe and previous companies completed a great deal of prospecting and litho-geochemical sampling in the OT License area, these data have been superseded by drilling information in the Oyut and Hugo Dummett areas. This work is summarized in Table 9.2 for the soil samples collected between 1997 and 2008.

Table 9.1 Geochemical Sampling Totals, EJV Area

License	Year	Rock Chip Samples	Soil Samples	Stream Sediment Samples	Trench Samples
Shivee Tolgoi	2003–2003	75	2,140	–	450
	2004	–	–	–	1,363
Javkhlant	2002–2003	45	–	25	–
	2006–2007	43	314	–	–

Table 9.2 Soil Sampling

Area/Prospect	Year	Company
Southwest	1997–1999	BHP
Copper Flats	2002–2003	Entrée
Eastern Entrée	2003–2004	Entrée
Oortsog	2003–2004	Entrée
Southwest	2004	Entrée
West RAB	2004	Ivanhoe
Western Entrée	2004–2005	Entrée
Exotic Cu	2005	Ivanhoe
OT South	2005	Ivanhoe
Hugo South	2005–2006	Ivanhoe
West	2006	Ivanhoe
Gandulga	2006	Ivanhoe
Ulaan Khuud	2006	Ivanhoe
BHP3	2006	Ivanhoe
Heruga	2008	Entrée

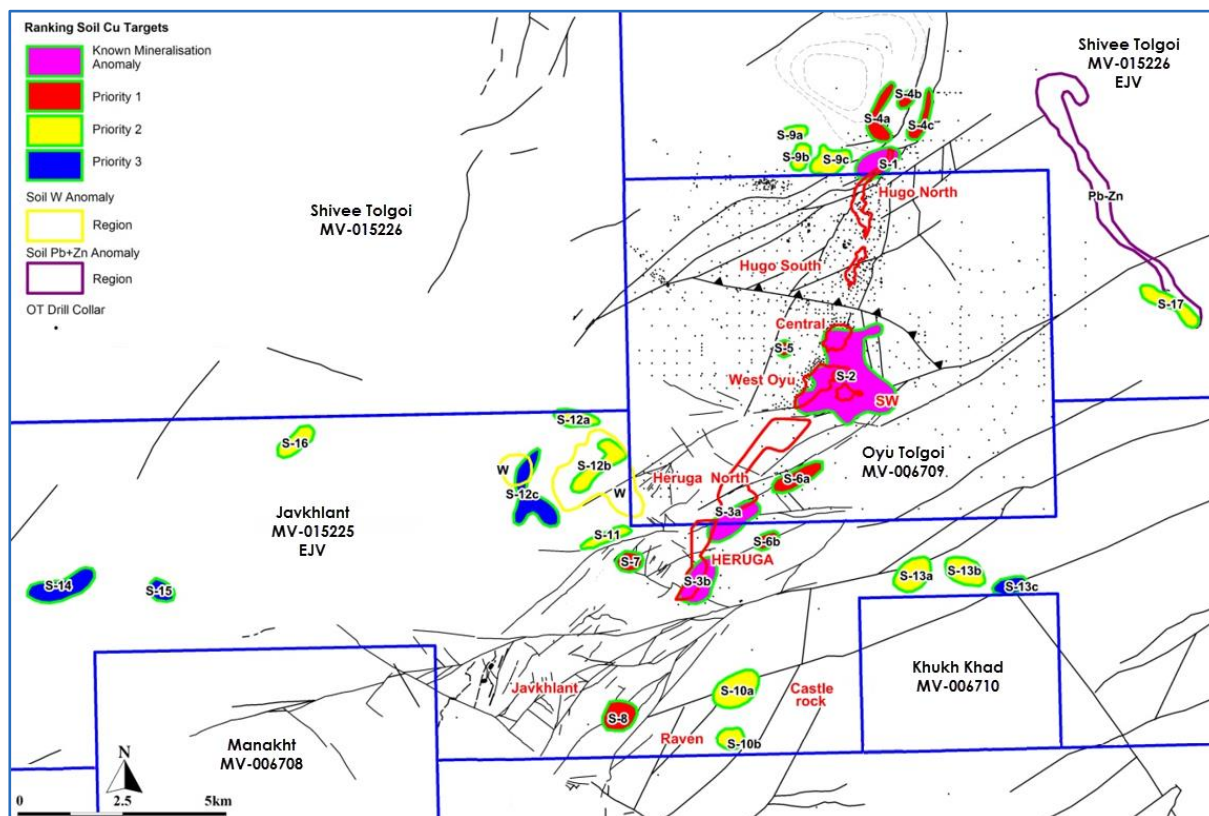
During 2011, all previous geochemical surveys completed in the Oyu Tolgoi area were reviewed (Sketchley, 2011). Survey data were levelled and compiled into a single dataset, and the anomalies were ranked according to location and type (Bell et al., 2012). Anomalous zones were compared to the rock chip and drillhole databases. Known anomalies are shown in Figure 9.4.

Results of the review:

- Anomalous areas are considered to be related to known and explored mineralization or are lithologically associated.
- Areas not previously covered by soil geochemistry are underlain by large intrusions, non-prospective rock exposures, or thick alluvial cover.
- Highly prospective areas have been extensively drilled, and thick cover sequences render buried mineralization undetectable by surface geochemical methods.

From this review it was concluded that no new or additional infill surface sampling was warranted in the license areas.

Figure 9.4 Summary Plan, Surface Copper Geochemical Anomalies



9.6 Geophysics

9.6.1 Oyu Tolgoi License

The initial geophysical surveys conducted by BHP in 1997 and 1998 consisted of airborne magnetics, ground magnetics, and gradient-array IP. The airborne magnetic survey was flown on 300 m spaced east–west lines with approximately 100 m average terrain clearance. The ground magnetic survey and IP survey were completed on 250 m line spacings; the latter showed chargeability anomalies over the Central, South, and Southwest zones of Oyu.

In 2001, Ivanhoe contracted Delta Geoscience of B.C., Canada, to conduct gradient-array IP on 100 m spaced north–south lines over the 3 km x 4 km core block of Oyu Tolgoi. Using multiple current (AB) electrode spacing's ranging from 1,000–3,600 m, the sulphide assemblages in the Southwest, South, and Central zones were clearly defined on all of the AB plans, indicating significant vertical depths for the mineralization in all zones. Delta Geoscience re-oriented the IP survey lines to east–west and resurveyed the core block of Oyu Tolgoi on 100 m spaced lines using multiple current AB electrode spacing's. This survey resulted in an entirely different chargeability signature that reflected a continuous zone of sulphide mineralization extending north–north-easterly from the south-western end of the Southwest zone through to the northernmost extent of the property, for a total strike length of approximately 5 km. Detailed total field, ground magnetic surveys, reading 25 m x 5 m and

50 m x 10 m centres, were completed over the full OT License. The data were merged to produce a high-quality magnetic image of the block.

In 2002, the geophysical programme 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 centres over the core of the concession and 100 m centres over the extremities. The Bouguer map was reduced to residual gravity for contouring.

In 2005, telluric electromagnetic (TEM) surveying was also 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 the fault, and form reservoirs for extensive water resources.

At Oyu Tolgoi, 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. Moreover, given the relative lack of supergene alteration of the known deposits and a paucity of evidence of unroofing of the porphyries, the potential for a large-scale exotic copper deposit is considered unlikely.

A Zeus-based geophysical survey operation started at Oyu Tolgoi on 19 June 2009 and terminated on 11 November 2009. Survey work continued during 2010 outside the Ivanhoe ground in the Shivee Tolgoi license. The Zeus IP survey used east-west lines, which resulted in the generation of north-south trending anomalies.

The Zeus System is a high-powered, low signal-to-noise ratio induced polarization / resistivity instrumentation platform. The Zeus system is based on the use of multiple signal measuring systems with broadly spaced electrode configurations. The owners claim that post-acquisition processing then allows images of conductive and resistive blocks to depths below 3 km. However, no inversion is carried out on the data, and independent reviews have identified this as a potential issue. To address this, the owners have recently pursued inversion processing of the data. However, progress has been slow because of the complexity of inverting gradient array-type three-dimensional (3D) IP data.

During 2011, Fugro was contracted to complete a district-wide magnetotellurics (MagTell) survey. Experience with MagTell has shown that the data can detect and delineate isolated conductors at substantial depths, and reliable 3D models of conductivity can be derived that are readily integrated with geology. The Fugro field crew collected 1,006 individual stations covering the main mineralized zones from Ulaan Khuud in the north extending down to Javkhilant. Approximately 30 planned stations around the Oyut area and to the west of the Hugo deposits were omitted because of mine construction activities. Approximately 10% of the stations were repeated because of high noise levels that were attributed to cultural activity. Results are currently being interpreted in the context of new target generation activities within the district.

9.6.2 Joint Venture Licenses

An initial IP survey (and detailed magnetics), using gradient-array with 11,000 m electrode spacing, commenced north of Oyu Tolgoi but was eventually extended through 2005 to cover all of the ground within the Shivee Tolgoi license and to include the Javkhlant license area. Three north–south trending chargeability anomalies were defined.

In late-2008, a 26.6 km² detailed magnetometer survey was undertaken in the Hugo North Extension area. Lines were oriented east–west at 25 m spacing with continuous readings. Two large magnetic features were found in the survey area.

At the same time, a 26.6 km² detailed magnetometer survey was undertaken in the Heruga area to obtain a more detailed view of the geology and structure. Lines were oriented east–west at 25 m spacing, with continuous readings.

During mid-2011, a ground-magnetic survey was undertaken to the east and west of Javkhlant and Heruga, extending eastward to the edge of the Khukh Khaad mining license to the east and westward into the Manakht license:

- Manakht license – 1,138 line-km over 161 lines with 25 m line spacing oriented east–west, with continuous readings.
- Khukh Khaad license – 1,007 line-km over 221 lines with 25 m line spacing oriented east–west, with continuous readings.

A GEM GSM 19W (v7) Proton Precession equipment unit was used for this work.

9.7 Trenching

During 2002, two trenches were completed over surface exposures of the Southwest zone. Both trenches were approximately 60 m in length and provided early-stage geological and assay information about the area.

During 2003 and 2004, an extensive 8,000 m trenching programme was carried out over the South and Southwest zones, plus an additional 20 km of trenches were completed in various other locations throughout the license. Trenches at the South zone ranged in length from 280–1,177 m and averaged around 600 m in length. Trenches were generally excavated 25 m apart, sampled over 2 m intervals, and assayed for Cu, Au, Mo, As, and Ag.

A number of trenches were excavated in 2009–2011 to support construction activities. These were reviewed by site geologists to confirm mapping across the license areas. The vast majority of the excavations were within Cretaceous clays, however, and no sampling was carried out.

9.8 Petrology, Mineralogy, and Other Research Studies

Several research, petrological, and mineralogical studies have been undertaken. These include age dating of key lithological units, detailed stratigraphic reviews, petrographic and spectral analysis of alteration products and minerals, and detailed structural reviews, particularly in the areas proposed for the block caving operation at Hugo Dummett.

Ivanhoe established, and OT LLC maintains, an in-house petrology laboratory in the Oyu Tolgoi Geosciences Department. Equipment for making polished mineral specimen blanks and polished thin sections is currently housed therein.

Alteration minerals are determined by short-wave infrared spectrometry (short-wave infrared (SWIR) or portable infrared mineral analyzer (PIMA) analysis) on typical specimens from a number of alteration zones in each drillhole.

A programme of preparing mineralization samples and making metallurgical index estimates from all of the Oyu Tolgoi deposits was undertaken between 2002 and 2006.

9.8.1 Research Studies

A number of research theses have been completed on the project area and are listed below in alphabetical order by author surname:

- Ayush, O., 2006. Stratigraphy, geochemical characteristics and tectonic interpretation of Middle to Late Paleozoic arc sequences from the Oyu Tolgoi porphyry Cu-Au deposit: MSc thesis (in Mongolian), Mongolian Univ. Science and Technology, Ulan Bator, Mongolia, 80 p.
- Jargaljav, G., 2009. Mineralization and metasomatic alteration of Central Oyu copper-gold deposit: PhD thesis (in Russian), Irkutsk Technical University, Irkutsk, 129 p.
- Khashgerel, B., 2010. Geology, whole-rock geochemistry, mineralogy and stable isotopes (O, H and S) of sericitic and AA alteration zones, Oyu Tolgoi porphyry Cu-Au deposits, Mongolia: PhD thesis, Univ. of Tsukuba, Japan, 114 p.
- Myagmarsuren, S., 2007. Sulphide mineral paragenesis at the Hugo Dummett porphyry Cu-Au deposit, Oyu Tolgoi, Mongolia: MSc thesis, Tohoku University, Japan, 93 p.
- Oyunchimeg, R., 2008. Sulphide mineralogy and gold mineralization at Hugo Dummett porphyry Cu-Au deposit, Oyu Tolgoi mineral district, Mongolia: PhD thesis (in Mongolian), Mongolian Univ. Science and Technology, Ulan Bator, Mongolia, 116 p.
- Savage, N., 2010. Origin of clasts, mineralization and alteration within the DA2a conglomerate, Heruga porphyry Cu-Au-Mo deposit, Oyu Tolgoi, Mongolia; evidence for an older porphyry system or part of the early Oyu Tolgoi paragenesis?: M.Sc Mining Geology Dissertation, Camborne School of Mines UK. 119 p.
- Wainwright, A. J., 2008. Volcanostratigraphic framework and magmatic evolution of the Oyu Tolgoi porphyry Cu-Au district, South Mongolia: PhD: Univ. British Columbia, Vancouver, 263 p.

10 DRILLING

10.1 Drill Programmes

Diamond core drillholes (DDH) are the principal source of geological and grade data for Oyu Tolgoi. A small percentage of the total drilling comes from reverse circulation (RC) or combined RC / DDH drilling (RCD – RC at the collar and DDH at depth). Most of the RC holes were drilled in the early days of exploration at the Oyut deposit. RCD holes make up a small percentage (<2%) of the total number of holes on the project. Fifty-two polycrystalline (PCD) holes were also drilled, but these are peripheral to the mine area.

The first drilling was completed by BHP in 1997 and 1998, when 23 diamond core holes (3,902 m) were drilled at the deposit now known as Oyut (previously Southern Oyu Tolgoi (SOT)). Ivanhoe completed approximately 109 holes (8,828 m) of RC drilling in 2000, mainly at Central zone, to explore the chalcocite blanket discovered earlier by BHP.

In 2001, Ivanhoe continued RC drilling (16 holes totalling approximately 2,091 m), mostly in the South zone area; however, an RCD method was tested for hole number OTRCD149. Ivanhoe drilled two additional holes using RCD method (OTRCD50 and OTRCD52), along with seven additional RC holes totalling 801.5 m (up to drillhole OTRC158), before switching to diamond core drilling methods for all of its exploration.

As at 31 December 2015, a total of approximately 1,115,706 m of drilling in 2,603 holes has been completed on the project. Of this, 1,022,254 m was diamond core drilling in 1,807 holes and 72,696 m was completed in 713 RC holes. The drilling has been spread mostly over the Hugo Dummett, Oyut, and Heruga deposits. These totals include approximately 525 holes (75,427 m) drilled as part of a condemnation programme to assist in the determination of suitable sites for the proposed plant, infrastructure, and dumps, and for water and geotechnical purposes.

Table 10.1 is a summary of all drillholes. The near-mine drillhole collar locations and types are shown in Figure 8.2.

10.2 Drill Orientations

The drillholes are drilled at a wide range of azimuths and dips depending on the orientation of the mineralization, but an east to west orientation is dominant throughout the project area. Drilling is normally oriented perpendicular to the strike of the mineralization. Depending on the dip of the drillhole and the dip of the mineralization, drill intercept widths are typically greater than true widths.

Average drillhole lengths at the Hugo Dummett and Oyut deposits range from 316 m (South zone) to 894 m (Hugo North) and the average overall is approximately 525 m.

The drill spacing is a nominal 70 m on and between drill sections in the Oyut zones. Drill spacing at Hugo North is on approximate 125 x 75 m centres. Drill spacing typically widens toward the margins of the deposits.

10.3 Drill Contractors

Diamond core drilling on the project has been almost continuous since 2001, using a variety of different contractors. Most of the more recent drilling was contracted to Major Pontil Pty Ltd. (Major), based out of Australia, which has used a variety of rigs, some with depth capabilities in excess of 2,500 m; these include UDR 1000, UDR 1500, and UDR 5000, LM90, and Schramm rigs.

Other drilling campaigns have been completed by Gobi Drilling, Can Asia, Mongolia Drilling Services, Australian Independent Diamond Drillers, and Soil Trade.

10.4 Diamond Core Diameters

The vast majority of diamond core drilling diameters at the Oyut and Hugo Dummett deposits are either 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 diamond core holes drilled at Hugo North were collared with PQ core and then reduced to HQ at depths of around 500 m prior to entering the mineralized zone. A few diamond core holes have continued to depths of about 1,300 m using PQ diameter. The depth of size reduction in any given hole varies depending on drilling conditions.

Many of the deeper holes, especially those at Hugo North (including Copper Flats), include multiple daughter holes (wedges) drilled from a PQ-diameter parent drillhole. Daughter holes are achieved by making a deliberate bend in the parent hole at the location where the planned daughter holes are to branch off by means of a Navi-Drill (Navi) bit, which is lowered down the hole to the desired depth and aligned along the azimuth of the desired bend. As the Navi bit advances, a bend is achieved at the rate of 1° every three metres. No core is recovered from the Navi-drilled interval, and the subsequent core diameter is reduced, generally to HQ size.

Most diamond core has been collected using Ball Mark or Ace oriented core marking systems to assist with geological and structural interpretations and for geotechnical purposes. More recently a Reflex ACT II Rapid Descent tool has been used for core orientations.

10.5 Diamond Core Transport

At the drill rig, the drillers remove the diamond core from the core barrel and place it directly in wooden or plastic core boxes. Individual drill runs are identified with small wooden or plastic blocks, where the depth (m) and drillhole number are recorded. Unsourced core is never left unattended at the rig; boxes are transported to the OT LLC core logging facility at the main camp twice a day under a geologist's or technician's supervision. Core is transported in open boxes in the back of a truck. Those holes drilled specifically for geotechnical purposes typically use triple tube methods, which are pumped out at the rig, transferred to a steel V-rail, and logged on-site before transport back to the core shed.

Table 10.1 Drillhole Summary Table

Deposit	Surface DDH Count	Length of Surface DDH (m)	RC Holes	Length of RC (m)	RCD Holes	Length of RCD (m)	UG DDH Count	Length of UG DDH (m)	PCD Holes	Length of PCD (m)	All Holes	Total Length (m)
Hugo Dummett Deposits												
Hugo South	143	100,109	45	3,263	12	8,886	–	–	–	–	200	103,787
Hugo North	383	294,067	8	2,737	4	2,417	150	29,977	–	–	545	329,198
Total Hugo Dummett	526	394,176	53	6,000	16	11,303	150	29,977	–	–	745	441,456
Oyut Zones												
Southwest	233	130,641	31	6,668	3	2,092	–	–	–	–	267	139,401
Central	240	90,511	71	6,694	–	–	–	–	–	–	311	97,205
South	90	33,542	30	4,405	2	891	–	–	–	–	122	38,838
Wedge	47	27,186	12	1,337	–	–	–	–	–	–	59	28,523
West	47	17,614	115	4,929	–	–	–	–	–	–	162	22,543
Total Oyut	657	299,494	259	24,033	5	2,983	–	–	–	–	921	326,510
'Other' Drilling												
Shaft Farm (exploration and geotechnical)	25	19,499	28	847	–	–	–	–	–	–	53	20,346
X-Grid	6	571	–	–	–	–	–	–	–	–	6	571
East Side License	18	3,901	164	21,780	8	2,400	–	–	–	–	190	28,081
'Other'	189	32,174	120	11,160	–	–	–	–	–	–	309	43,334
Total 'Other' Drilling	238	56,145	312	33,787	8	2,400	–	–	–	–	558	92,332
EJV Drilling												
Far South	68	73,170	14	3,272	–	–	–	–	–	–	82	76,442
Shivee Tolgoi: Hugo North Extension	114	96,844	75	5,604	2	736	–	–	52	3,335	243	106,519
Javkhlant: Heruga Total	54	72,447	–	–	–	–	–	–	–	–	54	72,447
Total EJV Drilling	236	242,461	89	8,876	2	736	–	–	52	3,335	379	255,408
Grand Total (All Drilling)	1,657	992,277	713	72,696	31	17,422	150	29,977	52	3,335	2,603	1,115,706

Note: Includes all holes drilled to 12 May 2011. Not all of this drilling is relevant to the Mineral Resource estimate.
Some of these holes were collared in the Shivee Tolgoi lease and drilled back into the Oyut Tolgoi lease.
Includes holes drilled for geotechnical, water, and condemnation purposes.
Approximately 79% of the combined RC/core drilling is by core methods.

10.6 Geological Logging

Diamond core logging facilities are indoors. Core logging takes place on sturdy steel racks, each of which is capable of holding upwards of 25 or more core boxes. Upon arrival at the core shed, the core is subject to the following procedures:

- Quick review.
- Box labelling check: The core boxes are checked to ensure they are appropriately identified with the drillhole number, metres from and to, and box number written with an indelible 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, rock quality designation (RQD), and maximum length, structural data, and oriented core data. Orientated core measurements were logged as interval data using standardised codes for structural and vein data only; the orientated core measurement did not usually begin until the hole was within the mineralized zone.
- Geological logging: Until August 2010 this was completed on paper logging forms. Subsequently, OT LLC implemented a digital logging data capture system, using commercially available (acquire) software, which uses standardised templates and validated logging codes that must be filled out prior to log completion. The logging is entered directly into laptops at the core shed and is wirelessly synchronized with the geological database. The template includes header information, lithology description and lithology code, graphic log, coded mineralization, and alteration.
- The geologist marks a single, unbiased cutting line along the entire length of the core for further processing.

The RC logging involves capture of geological, alteration, and mineralization data on paper logging forms.

10.7 Recoveries and Rock Quality Designation

OT LLC's geological staff measure the following core recovery and rock quality designation (RQD) parameters at the core logging area:

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

The methods used for measuring recovery is standard industry practice.

In general, OT LLC 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 occurs in a very small percentage of the overall mineralized zones. In addition, OT LLC notes decreased recoveries near surface in overlying non-mineralized Cretaceous clays and to a lesser extent in some of the oxidized rocks (generally above 100 m depth below surface), owing to the lower competencies of these units.

Recovery data were not collected for the RC drilling programmes.

Table 10.2 shows the recovery averages per year from 1998–2013.

Table 10.2 Summary of Average Drilling Recoveries

Year	All Drilling		Drilling >100 m Below Surface	
	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
2010	99.3	12,312	99.9	11,741
2011	99.5	22,117	99.8	21,524
2012	99.5	15,832	99.8	15,384
2013	99.4	9,598	99.7	9,390

10.8 Collar Surveys

Collar survey methods were similar for diamond core and RC drillholes.

Upon completion of a drillhole, the collar and anchor rods are removed, and a PVC pipe is inserted into the hole. The drillhole collar is marked by a cement block inscribed with the drillhole number (e.g., OTD663). Proposed drillhole collars are surveyed by a hand-held GPS unit for preliminary interpretations. After the hole is completed, a Nikon theodolite or Differential GPS (DGPS) instrument is used for final survey pickup. The two collar readings are compared, and if any significant differences are noted the collar is re-surveyed; otherwise the final survey is adopted as the final collar reading.

10.9 Downhole Surveys

OT LLC uses downhole survey instruments to collect the azimuth and inclination at specific depths of the drillholes for most of the diamond core programmes. The principal types of survey method used over the duration of the drilling programmes include Eastman Kodak, Pontil, Flexit, Ranger, gyro, and north-seeking gyro.

No downhole survey data were collected for the first 149 holes drilled on the project, including the initial diamond core programme by BHP in 1998 and the 125 RC holes completed by Ivanhoe in 2001 and 2002.

The first surveys initiated by Ivanhoe were for holes OTRCD149, OTRCD150, and OTRCD152, which were surveyed by the Eastman Kodak method. Ivanhoe used this method interchangeably with gyro and Ranger as the principal means of measuring deviations until approximately drillhole OTD397, after which gyro, north seeking gyro, Flexit, and Ranger methods were 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 Oyu Tolgoi.

Since January 2006, the procedure has been to measure deviations initially using a Flexit instrument along 30–60 m intervals to monitor the drillhole progress. At completion, all holes are re-surveyed with a north seeking gyro or SRG gyro instrument at approximately 5–20 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.

OT LLC has a detailed validation programme built into the database to reveal any moderate kinks or deviations in the downhole data. All of these are checked and adjusted, if required, before finalising the database.

RC drillholes were typically not surveyed downhole. Where no downhole survey exists, RC drillholes are assumed to be without deviation from the collar survey. In general, most RC holes are less than 100 m in depth and therefore considered unlikely to experience excessive deviations in the drill trace.

10.10 Core Storage

All 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 approximately 1 m (15 boxes per pallet). Each pallet is covered with a canvas tarpaulin, which is labelled with drillhole identification and the interval stacked in the pallet.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Methods

11.1.1 Geochemical Sampling

Ivanhoe's sampling programmes at Oyu Tolgoi included stream sediment, soil, trench, and rock chip samples. All of the sampling was carried out by Ivanhoe personnel or contractors.

Sampling performed by Entrée and Ivanhoe personnel on the EJV Shivee Tolgoi license also included stream sediment, soil, trench, and rock chip samples.

Because all of these early-stage sampling methods have been superseded by drill data, which form the basis of the Mineral Resources estimates, the early-stage sampling methods are not discussed further.

11.1.2 Core Sampling

The core cutting protocols at the now decommissioned Oyu Tolgoi Camp core shed for core drilling in both the Oyu Tolgoi and EJV areas were as follows:

- The core cutting protocols at the now decommissioned OT Camp core shed for core drilling in both the Oyu Tolgoi and Shivee Tolgoi JV areas were as follows:
- Core is photographed.
- The uncovered core boxes are transferred from the logging area to the cutting shed (approximately 50 m) 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 regularly flushed with fresh water.
- Both halves of the core are returned to the box in their original orientation.
- The uncovered core boxes are transferred from the cutting shed to the sampling area (approximately 50 m) by a forklift carrying several boxes on a wooden pallet:
 - Constant two-metre sample intervals are measured and marked on both the core and the core box with a permanent marker.
 - A sample tag is stapled to the box at the end of each two metre sample interval.
 - Sample numbers are pre-determined and account for the insertion of quality assurance and quality control (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 marked numbers on the outside. Samples are regularly transferred to a sample preparation facility operated by SGS Mongolia LLC (SGS Mongolia) approximately 50 m from the sample bagging area.

The core cutting and sampling procedures in the new Crane-Kavalieris core shed have been modified slightly to the following.

- After being photographed, a pallet jack transfers the core boxes on pallets to the core cutting room.
- Long pieces of core are broken into smaller segments with a hammer.
- The core is placed in a core cradle and cut in half using automated feed Almonte core saws.
- Half the core is placed directly into a pre-numbered sample bag and half the core is returned to the core tray.

The unsampled half of the core remains in the box, in its original orientation, as a permanent record. Where additional sampling is required (e.g., for metallurgical testwork), a skeleton core is left. In some cases, however, the additional testwork has consumed the entire core, and only photographic records remain. Core boxes are subsequently transferred to the on-site core storage area.

Non-mineralized dykes that extend more than 10 m along the core length are generally not sampled.

11.1.3 Dry Bulk Density Determinations

Ivanhoe, and later OT LLC, collected an extensive database of bulk density measurements for the Oyut and Hugo Dummett deposits from diamond core samples dating back to 2002. Currently, there are 49,365 bulk density determinations in the database relating to the deposits as shown Table 11.1.

Table 11.1 Number of Bulk Density Measurements for Each Deposit

Prospect	Number of Measurements	Average Bulk Density (g/cm ³)
HGN	17,962	2.75
HGS	8,600	2.76
Oyut	19,905	2.74
Heruga	2,898	2.82
Total	49,365	2.77

Prior to March 2012, 10 cm samples of full or halved diamond core were taken at approximately 10 m intervals per drillhole for bulk density determination.

The bulk density for non-porous samples (the most common type) is calculated using the weights of representative samples in water (W_{water}) and in air (W_{air}). The bulk density is calculated by the formula:

$$\text{Bulk Density} = W_{\text{dried in air}} / (W_{\text{dried in air}} - W_{\text{water}})$$

In March 2012, the immersion method was improved slightly to account for minor porosity in the rocks. The sample size was increased to 20 cm lengths of full core. The samples are weighed and then oven dried for 12 hours at 105°C. The dry weight is then measured.

The sample is then lowered into a basket submerged in water beneath the scales and the immersed weight is measured. The sample is removed from the suspended basket, excess water is brushed off, and the saturated weight is measured. Bulk density is then calculated:

$$\text{Bulk Density} = W_{\text{dried in oven}} / (W_{\text{saturated}} - W_{\text{water}})$$

Using the saturated weight is an improvement on the previous method because it accounts for the water absorbed by the sample during immersion to give a more accurate measure of displacement.

Less commonly, 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 bulk density.

In March 2012, a calliper method was introduced as a quality assurance check on the immersion method. Because the calliper method requires a cylinder of core, the procedure changed to taking a representative rock sample every 20 m. The samples are 20 cm long and cut perpendicular to the core axis using a core saw to create a cylinder for measurement. Where substantial chipping occurred when cutting the ends, the samples were rejected. The samples are weighed and then oven dried for 12 hours at 105°C. The dry weight is measured and the sample is then measured using a digital calliper. Three measurements are taken of the diameter of the cylinder and two measurements of the length. These values are then averaged.

The formula for the calliper method is:

$$\text{Bulk Density} = W_{\text{dried in oven}} / (\pi \times ((d_1 + d_2 + d_3) / 3)^2 \times (l_1 + l_2) / 2)$$

Where:

$W_{\text{dried in oven}}$ = Weight of sample dried in oven

D_{1-3} = Diameter of the core in three positions

L_{1-2} = Length of the core in perpendicular position

Table 11.2 summarizes the bulk density values for the key lithologies.

Table 11.2 Bulk Density Values by Lithology

Lithology	Lithology Code	Average Density Value	Lithology	Lithology Code	Average Density Value
Andesitic ignimbrite (CS1)	Andi	2.70	Basaltic volcanoclastic	Vat	2.85
Andesitic lava (CS3b)	AndL	2.69	Fine laminated tuff (DA1a)	VatI	2.81
Gold-rich Qmd	Auqmd	2.75	Dacitic block-ash tuff (DA2b)	Vbx	2.71
Basalt	Ba	2.72	Andesitic-dacitic volcanic breccia (DA2b)	Vbx2	2.73
Brown augite basalt lava (CS3c_2)	BasL	2.68	Porphyritic basalt	Vp	2.77
Basaltic lapillic tuff (CS3c_1)	Bat	2.67	Tuffaceous sandstone	VSst	2.75
Bi granodiorite	BiGd	2.70	Xenolithic biotite-granodiorite	xBiGd	2.72
Conglomerate	Cong	2.73	Xenolithic porphyritic andesite	xPan	2.75
Cretaceous clay	Cret	2.04	Xenolithic quartz monzodiorite	xQmd	2.74
Carbonaceous shale	CSh	2.50	Blank (used for control)	(blank)	2.29
Porphyritic dacite	Dac	2.64	Khanbogd Granite	Kgte	2.66
Dacitic flow	DacL	2.63	Coarse volcanoclastic with sedimentary clasts	Vcx	2.67
Dolerite dykes	Dol	2.79	Rhyolitic Ignimbrite	Rhyl	2.67
Early Qmd	Eqmd	2.81	Hornfels	Hfs	2.67
Fault zone	Fz	2.71	Fine-grained granodiorite	Gd	2.65
Globular ignimbrite (DA2a)	Glob	2.79	Intrusive breccia	Qmdx	2.71
Hornblende-Biotite-Granodiorite	HbBi	2.68	Dacite-basalt breccia	DacB	2.68
Hydrothermal breccia	Hbx	2.76	Diorite	Dio	2.75
Ignimbrite - dacitic to andesitic ash flow tuff / lapilli tuff	Ign	2.83	Trachyandesite lava	Tand	2.66
Basaltic- tuffs and flows	L	2.79	OT quartz monzodiorite	OT-Q	2.74
Late Qmd	Lqmd	2.78	Quartz monzonite	Qm	2.74
Quartz monzodiorite	Qmd	2.75	Andesitic tuff	AndT	2.72
Quartz vein percentage >90% with a vein percentage >15%	Qv90	2.78	Quaternary cover	Qco	2.18
Rhyolite	Rhy	2.62	Porphyritic augite basalt (DA1b)	Va	2.84
Fine grained sandstone-siltstone; tuffaceous, carbonaceous	Sst	2.73	-	-	-
Average All Lithologies					2.75

11.2 Analytical Laboratories

Until September 2011, all routine sample preparation and analyses of the Oyu Tolgoi samples were carried out by SGS Mongolia, which operates an independent sample preparation facility at the Oyu Tolgoi site and an analytical laboratory in Ulaanbaatar. SGS Mongolia, part of the global SGS Group, and predecessors have maintained a full service laboratory in Ulaanbaatar since the late 1990s. This laboratory was recognized as having ISO 9001:2000 accreditation and conforms to the requirements of ISO/IEC 17025 for specific registered tests. The laboratory performs all fire assay analyses.

Since September 2011, a second pulp has also been sent to the ALS Chemex facility in Vancouver, Canada, for inductively-coupled plasma and LECO analyses. ALS also acts as the check assay lab for SGS and vice versa. Since 2005, ALS Chemex has held ISO/IEC 17025 accreditation.

During 2002 and 2003, the on-site sample preparation facility and analytical laboratory were operated under the name Analabs Co. Ltd. (Analabs). Analabs is an Australian-based company controlled by Scientific Services Limited, which was bought by the SGS Group in 2001. SGS is an internationally recognized organization that operates more than 320 laboratories worldwide, many of which have ISO 9002 certification. The operating name of the Mongolian subsidiary was changed to SGS Mongolia LLC (SGS Mongolia) in 2004.

Until May 2005 (OTD900), SGS Welshpool in Perth, Australia, was designated as the secondary (check) laboratory. This laboratory currently has ISO 17025 accreditation, but whether it did at the time of the analyses is unknown.

After May 2005, the secondary laboratory was changed to Genalysis Laboratory Services Pty Ltd. (Genalysis), also in Perth. The National Association of Testing Authorities Australia has accredited Genalysis to operate in accordance with ISO/IEC: 17025 (1999), which includes the management requirements of ISO 9002:1994.

Check assays were also performed by ActLabs Asia LLC, part of the global ActLabs Group, which has maintained a full service laboratory in Ulaanbaatar since 2006. The laboratory has sample preparation, weighing, fire assaying, wet laboratory, and instrumentation sections.

It maintains an ISO 17025 accreditation and participates in CANMET and Geostats Proficiency Testing Programmes.

Check assays in the early phases of project drilling were performed by Bondar Clegg and Chemex laboratories. It is not known what certification these laboratories held at the time of the check assay programmes.

11.3 Sample Preparation

All Ivanhoe rock and drill samples since 2002, and subsequently all OT LLC samples since 2010 have been submitted to the same sample preparation and analytical laboratory that was operated by either Analabs or SGS.

The preparation laboratory was installed in 2002 as a dedicated facility for Oyu Tolgoi during exploration and resource definition stages. The laboratory was operating continuously up to the end of 2008, when it was put on care and maintenance during a slowdown in drilling

operations. It re-opened sporadically during 2009, and resumed continuous operations in mid-2010, when drilling operations increased. Although the facility has mostly dealt with samples from the project, it also has, on occasion, prepared some samples from other Ivanhoe projects in Mongolia. In March 2014 the facility was again put under care and maintenance as drilling operations ceased.

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

All sample preparation procedures and QA/QC protocols were established by Ivanhoe in consultation with SGS Mongolia and have been continued by OT LLC. The maximum sample preparation capacity has been demonstrated to be around 600 samples per day when the sample preparation facility is fully staffed.

The sample preparation facility has one large drying oven, two Terminator jaw crushers, and three LM2 pulverizers. The crushers and pulverizers have forced air extraction and compressed air for cleaning.

The sample preparation protocol for Oyu Tolgoi samples is 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 one 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 five minutes to achieve nominal 90% at 75 µm (200-mesh). A 150 g sample is collected from the pulverizer and sealed in a Kraft envelope. The pulp rejects are stored on-site.
- The pulps are put back into the custody of OT LLC personnel, and standard reference materials (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 Mongolia personnel and taken to the analytical laboratory. SGS Mongolia staff confirm 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 the project office in Ulaanbaatar for storage.

Between sample processing, all equipment is flushed with barren material and blasted with compressed air. Screen tests are done on crushed and pulverized material from one sample taken from the processed samples that make up 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.

11.4 Analytical Methods

SGS Mongolia routinely assayed all samples submitted for gold, copper, iron, molybdenum, arsenic, and silver on two-metre composite intervals.

Up to September 2011, copper and molybdenum were determined by acid digestion of a sub-sample, followed by an AAS finish. Samples were 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. Routine assays up to 2% Cu used a sub-sample size of 0.5 g, whereas a sub-sample size of 0.25 g was used for samples expected to be over-range, or >2% Cu. The detection limits of the copper and molybdenum methods were 0.001% and 10 ppm, respectively.

Gold was determined using a 30 g fire assay fusion cupelled to obtain a bead and digested with aqua regia, followed by an atomic absorption spectroscopy (AAS) finish, with a detection limit of 0.01 g/t. The same acid digestion process used for copper and molybdenum was also used for analyses of silver and arsenic with detection limits of 1 ppm and 100 ppm, respectively.

A trace elements composites (TEC) programme was undertaken in addition to routine analyses. Ten-metre composites of equal weight were made up from routine sample pulp reject material. The composites were subject to multi-element analyses comprising a suite of 47 elements determined by inductively-coupled plasma optical emission spectroscopy/mass spectrometry (ICP-OES/MS) after four-acid digestion. Additional element analyses included mercury by cold vapour AAS, fluorine by potassium hydroxide (KOH) fusion / specific ion electrode, and carbon/sulphur by LECO furnace. Results from the TEC programme were used for deleterious element modelling.

During 2011, an audit of assay techniques was instigated on the restricted suite of Cu, Au, Fe, Mo, Ag, and As. The audit suggested that high detection limits for As, Ag, and Mo restricted the usefulness of the information gained from these elements to only well-mineralized areas. The most serious example of high-detection thresholds was for As, where 99% of the As assays were found to be below the threshold for repeatable data, the threshold being equal to five times the lower detection limit. The result is that all elements analyzed, apart from Cu and Fe, are considered to have been compromised for exploration purposes by the detection limit. Since Au is assayed by a separate fire assay method, the assays are deemed not to have been compromised.

Similarly, gold concentrations were historically analyzed using a 30 g fusion with an atomic absorption (AA) determination. This gives an accurate range from 0.01–10 g/t Au; however, by using inductively-coupled plasma/atomic emission spectroscopy (ICP-AES) to analyze the gold in solution concentration, a ten times decrease in detection limit is possible with a similar upper detection limit. The current drill database shows 50% of the Au analyses are below five times the current detection limit.

As a consequence, a shift to high-resolution ICP-mass spectrometry (MS) for routine samples was implemented in September 2011. Given the relative complexity of ICP-MS equipment and the tendency for laboratories to centralize them globally to assist with operation and maintenance, this has necessitated a shift to an offshore laboratory for analysis of all resource and exploration samples.

As a result, the following actions were taken:

- SGS continued to manage the on-site sample preparation facility.
- SGS in Ulaanbaatar was appointed the primary laboratory for Au and F to ensure rapid turnaround of gold values.
- ALS (Vancouver) was appointed the primary laboratory for the high-resolution multi-element ICP-MS based suite (42 elements) and LECO sulphur and carbon analyses.
- ALS and SGS were to act as the secondary laboratories for each other, reinstating the secondary laboratory checks systematically in resource and exploration drilling. The check sample rate was at a nominal check rate ratio of one sample in 20.

The intended outcome for this was to:

- Identify grade and mineralization type (Cu, Au).
- Identify new mineralization from pathfinder elements (As, Bi, Pb, Zn, etc.).
- Determine the distribution of potential credit elements (Ag, Mo).
- Determine deleterious elements and allow mitigation procedures to be prepared (S, As, F, Cl, Se, and Ti).
- Support the mapping of deleterious alteration or rock types to allow mitigation procedures to be prepared (Si, K, Na, and Ca).
- Support the mapping of rock types for appropriate logging of litho-types.

Run-of-mine samples from the open pit and concentrator are subject to a separate analytical flowchart at the mine laboratory situated within the concentrator complex on-site.

11.5 Quality Assurance and Quality Control Methods

Geological aspects of the QA/QC programme were set up during 2001 by Charles Forster, who was Ivanhoe's manager for Oyu Tolgoi at the time. Simple analytical quality control procedures were followed until March 2002 when a formal programme was set up under the direction of Dr Barry Smee, P. Geo., an independent quality control consultant. This work included development of procedural guidelines, laboratory audits, and preparation of reference materials, with initial on-site monitoring conducted by designated Ivanhoe staff and later OT LLC staff.

All sampling and QA/QC work before 2007 was overseen on behalf of Ivanhoe by its QA/QC Manager Dale Sketchley, M.Sc., P. Geo. QA/QC reviews were intermittent in the period 2007 to end-2010.

Ivanhoe had also retained independent geologist/geochemist Dr Barry Smee to conduct semi-annual audits of both the preparation and analytical facilities from March 2002 through 2008 (Smee, 2008). Dr Smee's reports over this period are available through OT LLC.

The most recent audit of QA/QC data was completed on behalf of Ivanhoe by Dale Sketchley in 2011.

11.5.1 QA/QC Programme Outline

Samples were initially assembled into groups of 15 or 16 samples, and then four or five quality control samples were interspersed to make up a batch of 20. The quality control samples inserted by Ivanhoe consisted of one duplicate split core sample, one uncrushed field blank, a reject or pulp preparation duplicate, and one or two standard reference material (SRM) samples (<2% Cu and >2% Cu if higher grade mineralization was present based on visual estimates). The two copper SRMs were necessary because SGS Mongolia used a different analytical protocol to assay all samples >2% Cu. The SRMs were matrix-matched to ensure consistency with routine analytical samples. OT LLC has continued this procedure.

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 programme to ensure the material is barren of any of the grade elements before the blank samples are used for monitoring contamination.

From January 2006 (OTD930/EGD53) to mid-2007, the check assay programme was in abeyance based on recommendations from Smee (2006).

Check sampling was re-instated in mid-2007 and continued through to mid-2009, when it was again discontinued due to a slowdown in drilling activities.

In September 2011, when the change in laboratory protocols was initiated, check assays once again became routine in the programme. Insufficient data are currently available from the re-introduced programme to draw firm conclusions; however, no evidence of bias is apparent from the small dataset that is available.

11.5.2 Standard Reference Materials

The standard reference materials (SRMs) routinely used at Oyu Tolgoi are matrix-matched and developed from drill core crushed rejects. Materials are pulverized, screened to minus 75 µm, homogenized and tested for homogeneity, and then sets of randomly selected samples are sent to international laboratories for round-robin testing.

Tolerance limits for SRMs were set at two and three standard deviations from final round-robin mean value of the reference material. A single batch failed when SRM assays were beyond the three-standard deviation limit, and any two consecutively assayed batches failed when SRM assays were beyond the two-standard deviation limit on the same side of the mean.

The performance of the SRM samples was monitored as the assay results arrived at site in the period 2002–2007. The ability of the laboratories to return assay values in the prescribed SRM ranges steadily improved to more than 99% by end-2007. All samples were given a 'fail' flag as a default entry in the project database. Each sample was re-assigned a date-based 'pass' flag when the assays passed acceptance criteria.

In August 2007, a commercial molybdenum SRM was introduced to monitor molybdenum assays from Heruga. Because other deposits at Oyu Tolgoi have low molybdenum values, close to detection limit, a molybdenum standard had not previously been required. In August and September 2007, the failure rate of the SRMs increased to 9% and 5%, respectively. The higher failure rate was attributed to the commercial SRM not being matrix-matched to core samples and therefore producing a significantly low bias in the molybdenum assays. In October 2007 the commercial SRM was replaced with a matrix matched molybdenum SRM, and by January 2008 the failure rate had fallen to 1%.

The 2011 Sketchley review noted that for SRMs analyzed between 1 January 2008 and 1 November 2010, the laboratory has a slight upward drift for copper, resulting in an operating range for some SRMs that partly overlaps the two standard deviation tolerance limit established for SRM data.

11.5.3 Blanks

Barren material was procured from a local site and tested to ensure its barren nature for use as field blanks. Tolerance limits for field blanks were set at 0.06 g/t Au, 0.06% Cu, and 10 ppm Mo. Batches are automatically failed and re-assayed if these tolerance limits are exceeded, unless values are extremely low, in which case a barren override is applied in the database, and the batch remains as is.

Evaluation of the blank samples submitted to the laboratory in the period 2002–2007 indicated a low incidence of contamination for the analytical programmes for the Oyut and Hugo Dummett deposits. A few cases of sample mix-ups were identified during the review of the blank performance, which were investigated at site and corrected.

No evidence of systematic contamination was noted for the review of data from 1 January 2008 to 1 November 2010 (Sketchley, 2011).

11.5.4 Duplicate Samples

Duplicates routinely used at Oyu Tolgoi include core, coarsely- crushed rejects, and pulps. Core duplicates are taken in the field from one-half of core that has been split along a continuous line marked along the middle of the core, parallel to the long axis. Coarsely crushed rejects and pulp duplicates are taken in the laboratory by using a riffle splitter. Assays of each type follow the parent sample in a batch.

Laboratory check pulp samples sent to an umpire laboratory were only used up to the end of 2005 for the Oyut and Hugo Dummett deposits. Other duplicate sample types employed in the QA/QC programme were core, coarse reject, and pulp.

In the period 2002–2007 copper generally performed very well with absolute relative difference results well within expected limits; gold absolute relative difference results were higher than copper but considered acceptable. Core duplicates for both copper and gold were above the ideal arbitrary absolute relative difference value of 30%, which was related to an uneven distribution of mineralization between core halves as typically caused by quartz vein and fracture-controlled mineralization.

At Hugo North, percentage differences for gold for the coarse reject and pulp duplicate samples were about the same because of the finer reject crushing size. Although the reject precision was within the ideal threshold, the pulp duplicates tended to be higher, probably because most gold values lay near the detection limit where precision was poorer. This is further supported by an improvement in precision at higher grades, although there is also a possibility of gold liberation during pulverization. For copper, both coarse reject and pulp duplicates were also similar because of the finer reject crushing size, with both being well within the ideal limits.

The review of the 2008–2010 data noted a strong bias for several gold duplicate samples, which is most likely related to sample mix-ups, as that pattern is present for core, coarsely crushed, and pulp samples. The remaining data display normal distribution patterns, and the precision is deemed acceptable for the types of material and mineralization being examined (Sketchley, 2011).

11.5.5 Sample Security

Sample security relies on the fact that the samples were always attended to or locked in a sample dispatch facility. Sample collection and transportation were always undertaken by company or laboratory personnel using company vehicles. Chain-of-custody procedures included filling out sample submittal forms that were sent to the laboratory with the sample shipments to ensure that the laboratory received all the samples.

11.6 Databases

Before August 2010, all geological and geotechnical drillhole data were entered into an MS Access relational database that had been developed in-house. Data were exported from the main database to meet end-user requirements.

In August 2010, OT LLC elected to migrate the MS Access database to a full Oracle content (OCDB) acQuire database with links to the end-user software programmes. The database is read-only for these programmes, preventing accidental overwriting and ensuring up-to-date live and centralized data, rather than distributed databases.

Before August 2010, all drillhole data were initially manually recorded in the field or in the core logging shed on paper logging sheets. The logging geologist then introduced logging information into the MS Access database, which had a series of embedded checking programmes to look for obvious errors. Formational names were subsequently assigned according to the accepted geological interpretation and position within the stratigraphic column.

With the move to the acQuire database, which instituted direct digital data capture, the design stubs for the logging sheets do not permit any invalid data. No drillhole can be completed and entered into the database until the logging is correctly entered.

SGS Mongolia reports the results digitally by email and submits signed paper certificates. General turn-around is approximately seven days. All hard copy assay certificates are stored in a well-organized manner in a secure location on-site.

Before August 2010, the digital assay results were imported to the MS Access files once the assay data had been received from the laboratory in Ulaanbaatar. With the subsequent direct import to the acQuire database, none of the assay data are entered manually. Project personnel visually check each assay on the signed paper certificate against the assay entry in the digital database.

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

Data are presented in up-to-date 50–100 m spaced drill sections in two directions (north-west and north-east) and reconciled to 50 m spaced level plans to ensure that domains (solids) were properly constructed and interpretations were sound. Sections and levels are reviewed regularly to ensure that all holes crossed the target where planned and that data density is sufficient to make an appropriate interpretation.

The solids of all lithologies and mineralization types are present on the interpretations, and if significant deviations are noted in the holes, or they appear to miss their targets, additional holes are planned to infill untested areas in the model.

OT LLC intends to develop a comprehensive and coherent geological (geometric) model based on sound and accurate geological information as the basis for future resource estimates.

Digital data is backed up regularly.

12 DATA VERIFICATION

12.1 External Reviews 2002–2012

A number of data reviews have been undertaken by independent third-party consultants as part of preparation of technical reports on the project, including the following:

- Roscoe Postle Associates, 2002 – Review of exploration information from earlier work by BHP and Ivanhoe and visited the project site in Mongolia and the Analabs assay laboratory in Ulaanbaatar. A suite of independent core samples was collected and assayed. Duplicate analytical datasets were examined. No biases or errors were noted that would affect Mineral Resource estimates.
- AMEC and AMEC Minproc, 2002–2007 and 2012 – Review of QA/QC data and databases in support of Mineral Resource estimates undertaken in 2002, 2003, 2005, 2006, 2007 and 2012, and independent core check sampling. QA/QC reviews showed acceptable analytical precision, low contamination, and a small number of sample mix-up errors. The database iterations reviewed were considered sufficiently error-free to support Mineral Resource estimation.
- Barry Smee, 2002–2008 – Review of sample preparation, analytical, and QA/QC data. Inspections and reports were completed in 2002, 2003, 2004, 2005, 2006, and 2008. No significant biases or errors were noted that would affect Mineral Resource estimates.
- Quantitative Geoscience, 2007–2008, 2010 – Data verification of previous AMEC estimates, review of on-site sample preparation facility, independent sampling, and review of geology, mineralization, core sampling, sample preparation, QA/QC, and Mineral Resource modelling for the Heruga and Heruga North (New Discovery) areas and for geotechnical drilling underway at Hugo North. No biases or errors were noted that would affect Mineral Resource estimates.

Other than the issues noted above, all reviewers have concluded the Oyu Tolgoi drillhole dataset was sufficiently free of errors, reasonably accurate, precise, and free of contamination, and suitable for use in estimating Mineral Resources.

12.2 TRQ Reviews 2011–2012

In 2011, TRQ (formerly Ivanhoe) carried out a review of the QA/QC system. The review covered laboratory audits, quality assurance procedures, quality control monitoring, and database improvements at Oyu Tolgoi for the period 2008 to 2010. Recommendations arising from the review included:

- QA/QC improvements at site:
 - Updating SRM inventory and sample storage
 - Re-designing batch layouts
 - Re-instating bias charts, failures table, load statistics, failure rates
 - Upgrading analytical suites
 - Improving density measuring techniques
 - Rectifying SRM failures and duplicates mix-ups

- Completing check assaying
- Completing regular progress reports
- Preparation laboratory improvements:
 - Minimizing fine particle extraction biases
 - Using correct pulverising amounts
 - Rectifying safety issues
 - Using correct sizing test amounts
 - Proper archiving
- Database improvements:
 - Improving functionality
 - Correcting integrity issues
 - Fixing operational issues

Implementation of the recommendations by OT LLC continues, with a number of the recommendations either already addressed, such as changes in analytical methods for multi-element exploration suite, or under advisement.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

Oyu Tolgoi currently has two deposits scheduled for production: the Oyut deposit, which includes the Southwest and Central zones, and the Hugo North deposit. Two additional deposits (Hugo South, and Heruga) remain in the assessment phase and are scheduled for a later stage of development. The Southwest and Central zones are now being mined by an open pit method. Hugo North will be mined by underground block caving.

A substantial amount of metallurgical testwork has been conducted over the years, which was presented in the previous studies of Oyu Tolgoi. The work has focused on verifying assumptions made during design with actual operation experience gained from the start of commissioning and operation of the concentrator. In addition, further flotation variability testwork has been conducted on Hugo North, Central zone, and blends of Southwest zone and Hugo North. This work was used by OT LLC to confirm the assumptions for metallurgical performance and throughput estimation that were prepared for OTFS14. The OTFS14 assumptions for defining the Mineral Reserves were prepared in Base Data Template 31 (BDT31) and were not changed for OTFS16. For the preparation of the OTFS16 production schedule, the plant throughput volumetric limit was changed from 5.5 kt/h to 5.0 kt/h and the SAG capacity was increased by 2.2%.

On completion of the variability flotation test work on the individual deposits a series of locked cycle tests were conducted on further composites representing chronological blends of ore planned to feed the mill in the mine plan.

Southwest zone and Hugo North blend locked cycle tests, not only responded as well, but better than either the Hugo North or Southwest zone composites individually. The composite with 25% Hugo North and composites containing 50% Hugo North achieved 92.8% recovery to a 30.2% copper grade and 92.5% recovery to a 27.3% copper concentrate respectively.

The Central zone covellite–chalcocite composite yielded a concentrate grade of 26.2% with 58.6% copper recovery and 22.5% gold recovery, which is significantly higher than in the batch tests. Gold losses were high in the cleaner tails, but not economically significant on this ore zone, containing only 0.08 g/t Au in feed. There was only sufficient material to perform a single cleaner test on the Central zone covellite–chalcocite composite, which gave a copper recovery of 70.9% to a copper grade of 25.5%. Subsequent in-house testing programmes on fresh core in advance of Central / Southwest ore treatment in H2'16 have improved on these results, and concentrate grade targets and marketing specifications have been lowered to maximize recoveries with Central / Southwest ore blends.

13.2 Evaluation of Testwork and Process Modelling

13.2.1 Grinding Capacity and Flotation Feed Size Modelling

For Phase 1, Minnovex Minerals Services derived two generic equations to describe the capacity and the flotation feed sizing expected from Southwest zone ore. Both equations use the same comminution parameters as developed for use in its Comminution Economic Evaluation Tool (CEET):

- SAG Power Index (SPI), (in minutes) – closed-circuit small-scale dry grinding test conducted on –12.7 mm ore.
- Modified Bond Index (MBI or BWI, kWh/t) – a short form of the Bond Ball Mill Index test, which is calibrated or validated by several full Bond tests.
- Minnovex Crushing Index (Ci) – developed from the sample preparation process for SPI, which is a predictor for the fraction of material already finer than SAG discharge closing screen size.

These parameters were used to model a large number of conventional SAG mill / ball mill (SABC) circuits, with successful prediction of capacity (or throughput (TPUT) the instantaneous tonnage per hour (t/h) achievable through grinding 8,059.2 h/a) and P_{80} (the 80% passing size of grinding circuit product). The Phase 1 plant has achieved and exceeded design production rates with primary grind P_{80} in-line, or better than predicted by the model.

The exercise was not repeated for Hugo North ore, since the range of SPI and MBI values in Hugo North fall well within the range of values encountered in the Southwest zone.

Because the equations developed are generic, good agreement is to be expected on any ore fed to the same circuit configuration, which is within the range of comminution indices presented by the majority of Southwest zone ore. Figure 13.1 shows the correlation between SPI and MBI, the two strongest determinants of capacity, for Central zone comminution samples, and Figure 13.2 shows the range of all 336 comminution samples as a cumulative frequency distribution of SPI and MBI. Also plotted are the distributions of 137 samples from Hugo North testing in 2007, a further 82 samples from the more northerly set tested in 2011, and 74 Central zone samples – these data are discussed in more detail in Section 13.3. All the Hugo North samples fall within the Southwest zone range, while half of Central zone ore samples fall within the Southwest zone SPI data range.

These correlations effectively describe the capacity and the product sizing from the Phase 1 circuit and will not be improved upon unless operational experience indicates that fine-tuning of the constants with actual operating data is required. It is an empirical, power-based method that allows capacities and product sizings with different ores to be estimated from the same constants and exponents applied to new comminution parameters, as long as the SAG circuit configuration and closing screen aperture are kept constant. These equations assume that the SAG mill can be kept near full loading on soft ore (e.g. with a high percentage of Central zone ore).

The softer Central zone ore allows more tonnage to pass the grinding circuits. The volumetric capacity limit is only reached when the proportion of Central zone chalcocite and covellite ores exceeds 40% of the feed and the balance is primarily Hugo North ore.

With the planned feed change to softer and higher grade underground Hugo North ore, the mill volumetric constraint becomes one of concentrate handling and tailings handling capacities. The volumetric capacity limit in OTFS14 was 5.5 kt/h (44.3 Mt/a, 121 kt/d). After a review of the volumetric capacity in OTFS16, this was reduced to 5.0 kt/h (40 Mt/a, 110 kt/d). Further elevation and revision of the limit is quite likely as de-bottlenecking and optimization of the plant continues. The OTFS16 limit has already been reached and may be exceeded as the Central zone ore is treated.

During the years 2020–2036 it is projected that the flotation feed will be slightly above the optimum P_{80} for Hugo North. However, the flotation test results indicate little sensitivity for recovery in the expected range of grind sizes. The effect of a change in SAG : ball mill power ratio has been estimated by taking the original flotation feed P_{80} predicted by the Minnovex equation and adding the kWh/t change in ball mill energy applied to the tonnage processed, resulting in a finer flotation feed P_{80} than for the reference case. Recent operating experience treating softer zones of Southwest ore near both the SAG capacity limit and the volumetric limit has suggested that the Minnovex P_{80} formula is conservative, i.e. finer grinds are being achieved than predicted by the indices, at the SAG tonnage predicted accurately by the same indices.

Figure 13.1 Correlation between SPI and MBI for Central Zone Comminution Samples

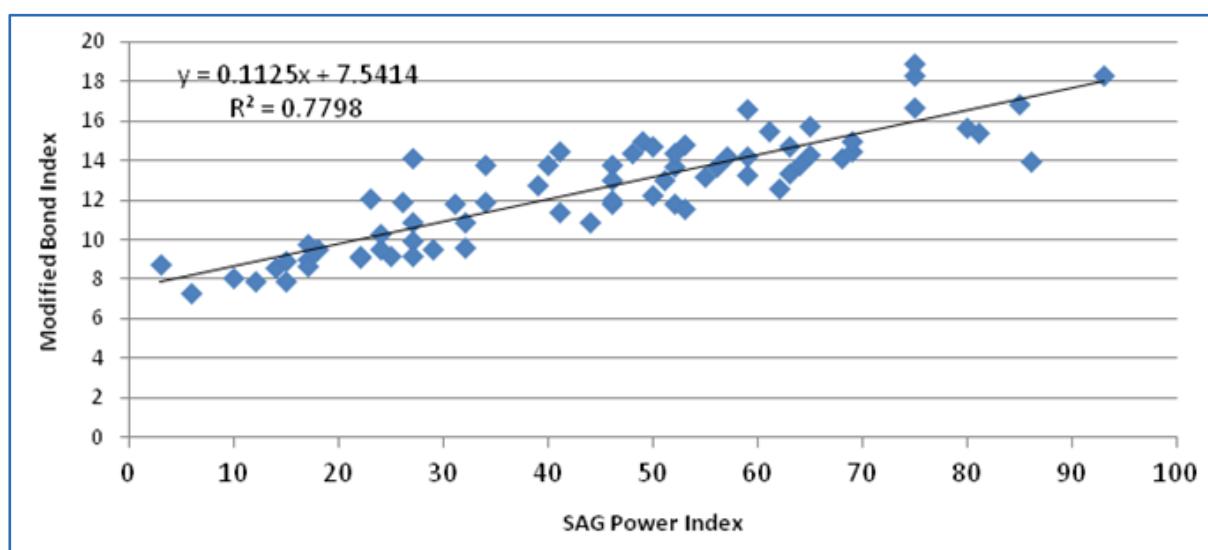
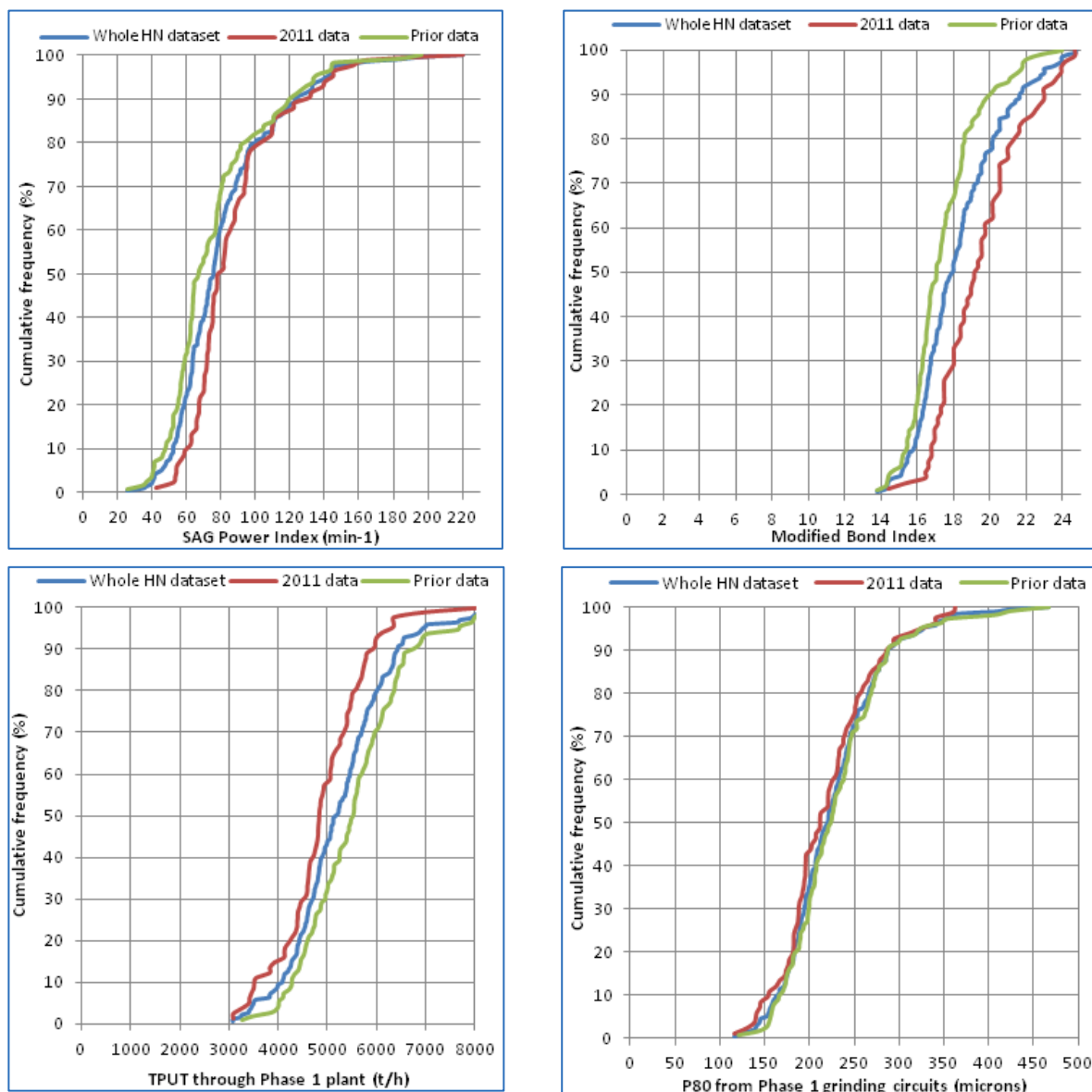


Figure 13.2 Cumulative Frequency Distributions of SAG Power Index, Modified Bond Index, TPUT, and P₈₀ of Flotation Feed at 100% through Phase 1 Circuits



13.2.2 Validation of the Minnovex Comminution Predictions

The prediction of throughput and transfer size is fundamental to the mine planning process and is the basic determinant of annual production capacity. It is also fundamental to predicting the capacities of the operation in a variable ore source environment. However, it is necessary to validate the predictions in early operation so that they can either continue to be used with increased confidence; or if the predictive power is poor, be replaced by a better system.

Plant surveys were carried out in November 2013 and survey samples were submitted for comminution testing. This allowed correlation of plant capacity against orebody characteristics. Besides SPI, MBI, and crushing index (Ci) measurements, other tests performed on the samples included the Julius Kruttschnitt Mineral Research Centre (JK) drop-weight tests to evaluate potential alternative predictive methods.

It was concluded that the actual SAG mill capacity in Surveys 1 and 2 was in excess of the generic model by about 10%, when corrected for charge level. In addition, the SAG mill appeared to be producing more fines than anticipated, leading to a finer P_{80} in flotation feed than expected. The surveys recorded P_{80} values of 130–150 μm on relatively hard ore with a work index of 22.6 kWh/t, Ci of 19.5, and SPI of 117.3. These parameters are at the 40th percentile for Southwest zone ore SPI, but at the 80th percentile for Hugo North SPI and above the 90th percentile for both deposits for MBI. With the same material, the generic model used in the mine plan would have predicted a P_{80} of 218 μm .

Due to the difficulties in representative sampling of coarse SAG mill feed and the impact of belt cuts on survey stability, these results must still be considered indicative, but encouraging, for Phase 2 performance. Sensitivity analysis to JK drop-weight parameters around Survey 2 was also carried out by simulation. When the survey hardness parameters were replaced with values representing the softest and hardest Southwest zone ores, SAG capacity increased by 19% and decreased by 15.5%, respectively, while achieving product P_{80} values of 130–134 μm . This is in line with the capacities indicated by the generic capacity prediction model, although P_{80} appears to be more conservatively estimated by the Minnovex model.

The latest daily correlations between plant capacity and Minnovex indices show good correlation between the model and plant performance, with the plant outperforming the model by only 1%–2%. There is a slight skew in the model, with the plant outperforming the model by more than the average on harder ores, and underperforming the model on softer ores. This result is normal in that SAG ball charge and SAG grate and closing screen openings are optimized for the harder ores, and will be changed gradually as a higher percentage of softer ores are processed as underground ore tonnage is ramped up.

13.3 Sample Spatial Representation and Selection Criteria

The number of samples and tests for each orebody is listed in Table 13.1. The major recent additions are as follows:

- Twenty variability composites from 72 holes throughout the Hugo North block cave, for abrasion index and crusher work index. Sub-samples were taken for mineralogy, head grade, and rougher flotation testing.

- Thirteen variability samples from Central zone, selected based on copper mineralogy, pyrite grade, and location, for Bond ball mill, Bond rod mill, and crusher work index. A blended sample of covellite ores was taken for JK tests, and flotation tests were carried out on samples categorized as chalcocite, covellite and chalcopyrite, based on dominant copper speciation.
- Seven composites from the Southwest zone, designed to represent the first seven years of the mine's life, subjected to mineralogy and flotation testing. Two master composites represent the years when Southwest zone ore is processed individually and then as a blend with Hugo North.

Table 13.1 Number of Comminution Samples

Deposit / Zone	No. of Holes Sampled	No. of Samples Tested
Southwest Zone	77	204
Central Zone	25	74
Hugo North	79	239
Hugo South	6	15

13.3.1 Southwest Zone

The Phase 1 design has been based on Southwest zone ore with up to 90% mixture of early Hugo North ore. The following testwork has been carried out Southwest zone since 2009:

- QEMSCAN on overall Southwest zone composites to provide simulation inputs for regrind optimization for fluorine rejection to inform the final number of Phase 1 regrind mills.
- Roughing and cleaning tests to produce representative tailings materials for humidity cell testing on Southwest zone, Hugo North, and Central zone ore types and blends.
- Physical simulation of process water and measurement of process water quality by taking raw water from the borefield and applying an equivalent number of grinding and flotation water exchanges as applied at the full-scale water balance.
- Determining the impact of simulated process water on flotation performance.
- Bulk solids handling testwork on Southwest zone concentrate to inform the Phase 1 and Phase 2 bagging plant designs.
- Production of Southwest zone composites representing early ore to assist in commissioning.
- Production of Southwest zone composites representing 25% and 50% Hugo North admixture for locked cycle tests.

13.3.2 Comminution Sampling

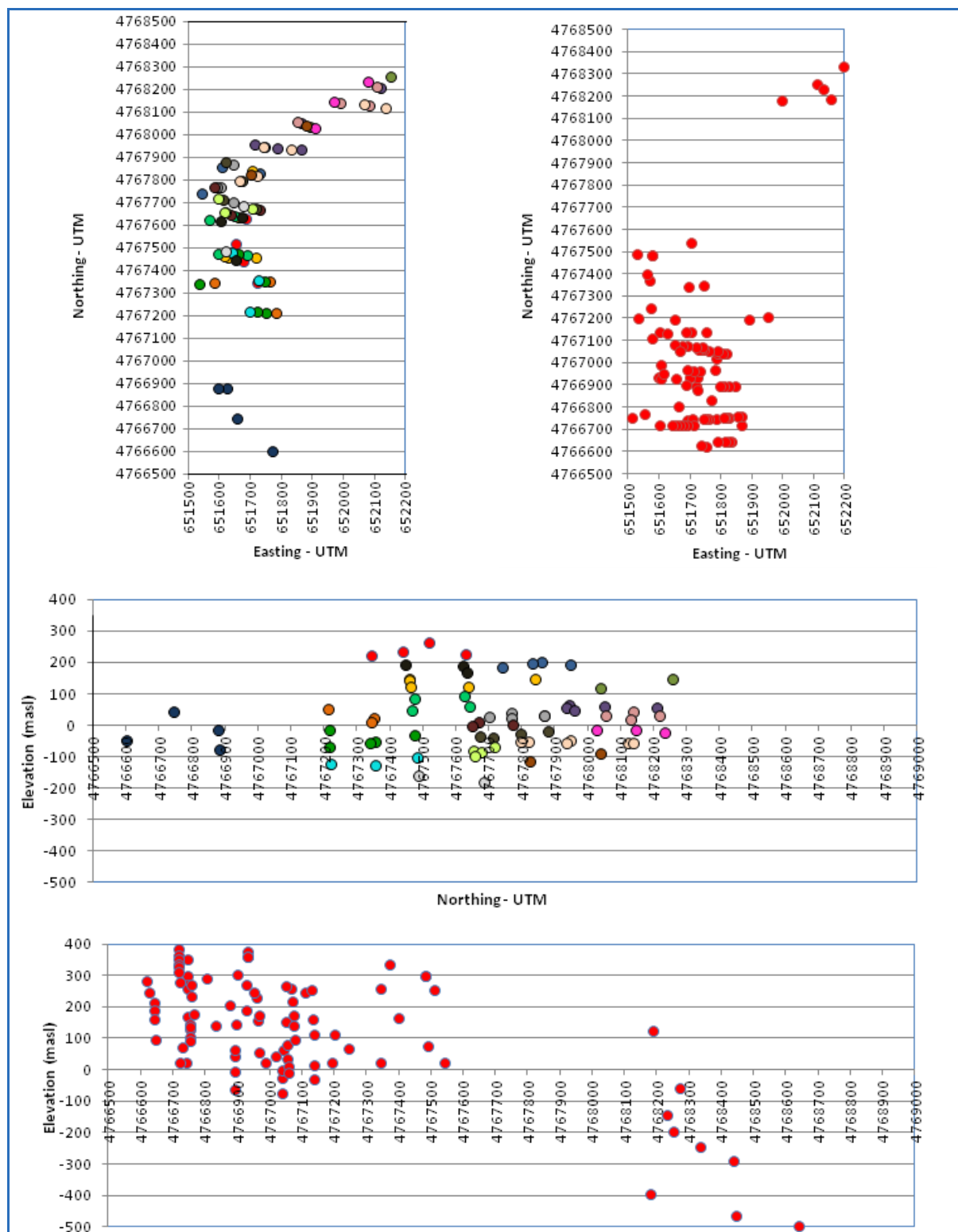
With the completion of the Phase 1 design for Southwest zone ore, the primary design focus for sample selection was better definition of the northern third of the Hugo North Lift 1 block cave envelope. The later sample locations are shown in red in Figure 13.3. Cave initiation is in the vicinity of 4,767,700 mN and 651,650 mE. Ore located south of 4,766,800 mN would not report to the current block cave drawpoints, but would be mined as a separate Hugo South block cave. Comminution test sampling quantities are given in Table 13.2.

The sample quantity was doubled in the Hugo North block cave compared to the open pit zones because of the inability to re-sample a block cave by drilling once fragmentation has commenced. Sample density in the open pit zones is 2–4 times higher in the early production years than for the Reserve as a whole, and as such meets the criteria normally applied in the SPI methodology.

Table 13.2 Minnovex Comminution Test Result Quantities

Orebody	SPI Tests	Ci Tests	Modified Bond Tests	BWI Tests	SPI Quantity (tests per Mt)
Southwest Zone	219	106	209	31	0.31
Central Zone	73	13	71	5	0.28
Hugo North	239	218	237	18	0.55
Total	514	206	480	54	0.37

Figure 13.3 Locations of Hugo North Comminution Samples



13.3.2.1 Central Zone

A secondary focus of the study was better definition of the Central zone ore, which will be processed with Hugo North and Southwest zone ores. A plan view and sections of Central zone comminution composite sample locations are presented in Figure 13.4.

Little additional comminution work was done on Central zone ore in this phase of study because prior work had shown that the Central zone indices were uniformly low for both SPI and MBI, resulting in volumetric limitation of the flotation circuits at the plant design tonnage before the SAG mill or flotation feed sizing limits were met. There is a trend of increasing hardness with depth. Owing to the reasonably good correlation between SPI and MBI for Central zone ore, as shown in Figure 13.5, there is potential to use the grind times in flotation testwork to infer hardness in SPI terms.

Figure 13.4 Locations of Central Zone Comminution Samples

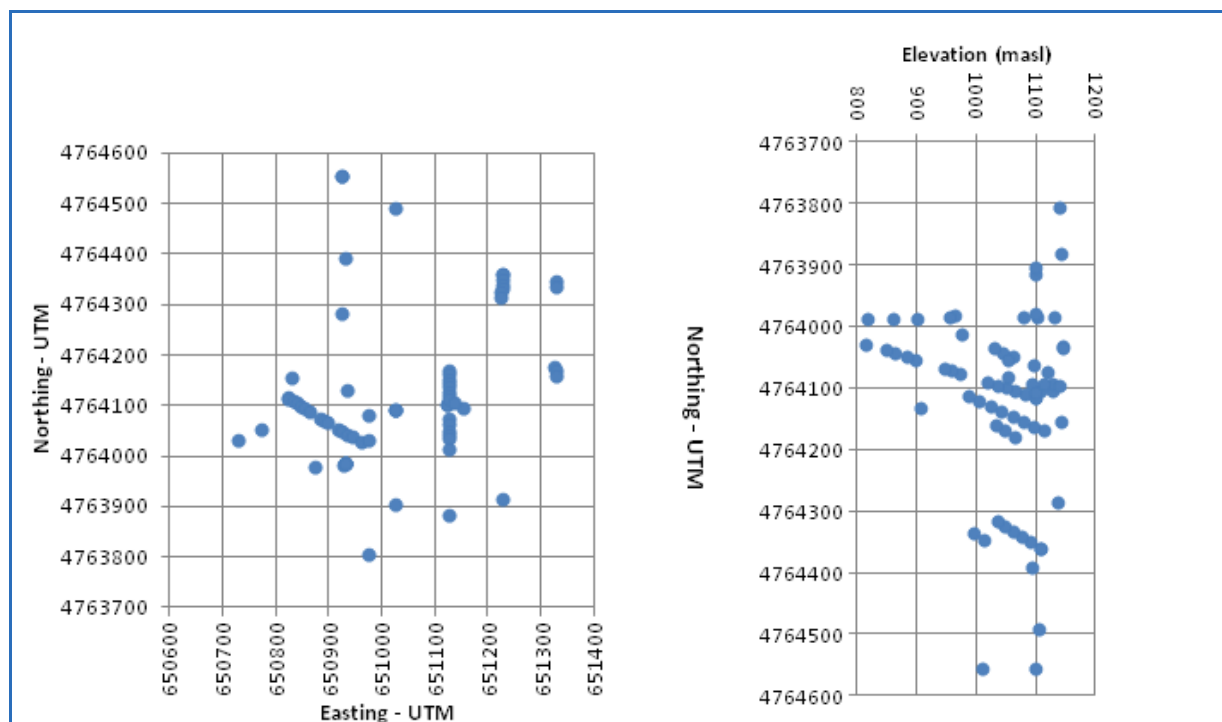
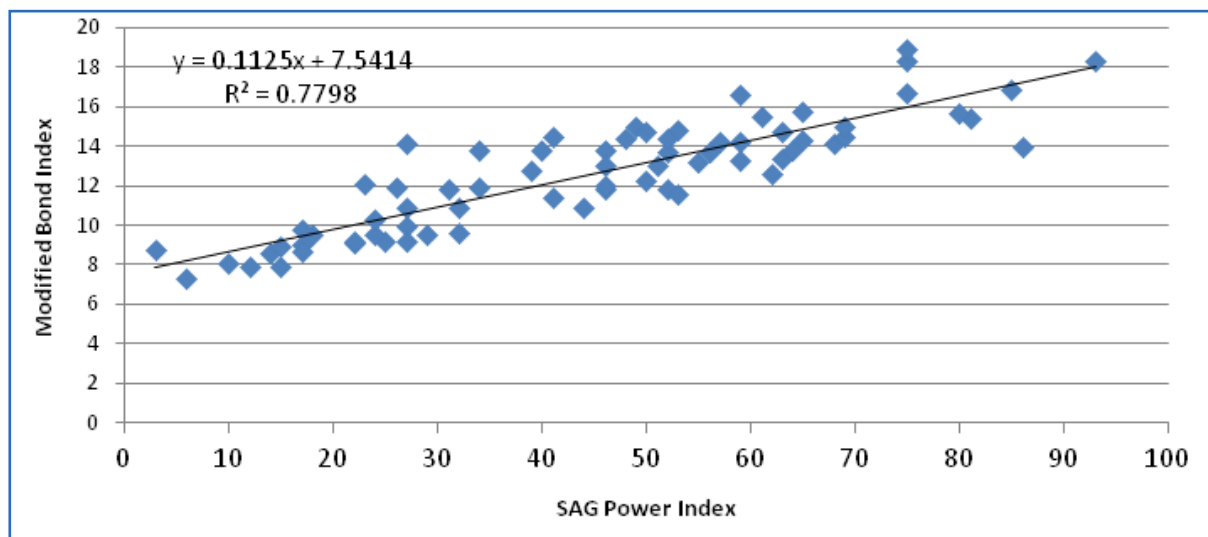


Figure 13.5 Correlation between SPI and MBI for Central Zone Comminution Samples



13.3.3 Geostatistical Analysis of Hugo North Comminution Dataset

The earlier Hugo North comminution dataset was populated more densely to the south of the prior block cave envelope. The cave initiation point subsequently moved north to the orebody inflexion point to access a high gold core that has recently accounted for more value at higher long-term gold price projections (Table 13.3).

Table 13.3 Comparison of Mean Values for Hugo North Comminution Indices

Dataset	SPI (min ⁻¹)	MBI	Ci	TPUT (t/h in Phase 1)	P ₈₀ (Phase 1 P ₈₀ in µm)
2011 dataset	88.1	16.1	19.5	4,906	219
Prior dataset	76.2	19.6	17.4	5,557	231
Combined dataset	81.4	18.1	18.3	5,279	226

The 2011 dataset is compared to the prior dataset and the combined dataset in Figure 13.2. TPUT and P₈₀ are derived from the generic Minnovex formulae and reflect the hypothetical situation where Lines 1–2 are fed with 100% Hugo North ore.

Comparison of the combined dataset with the previous dataset indicates a 5% reduction in the predicted capacity to 5.3 kt/h from 5.6 kt/h, compared to that currently attributed in the block model. This potential bias was corrected by inclusion of the 82 sample results in the Hugo North block model.

Minnovex MBI results were checked against the Standard Bond Index test on 18 samples, with generally good agreement, moderate scatter, and no evidence of bias. This indicated that the MBI results can be used to populate the block model and wherever else Standard Bond index results may be required, as in the calculation of incremental ball milling requirements.

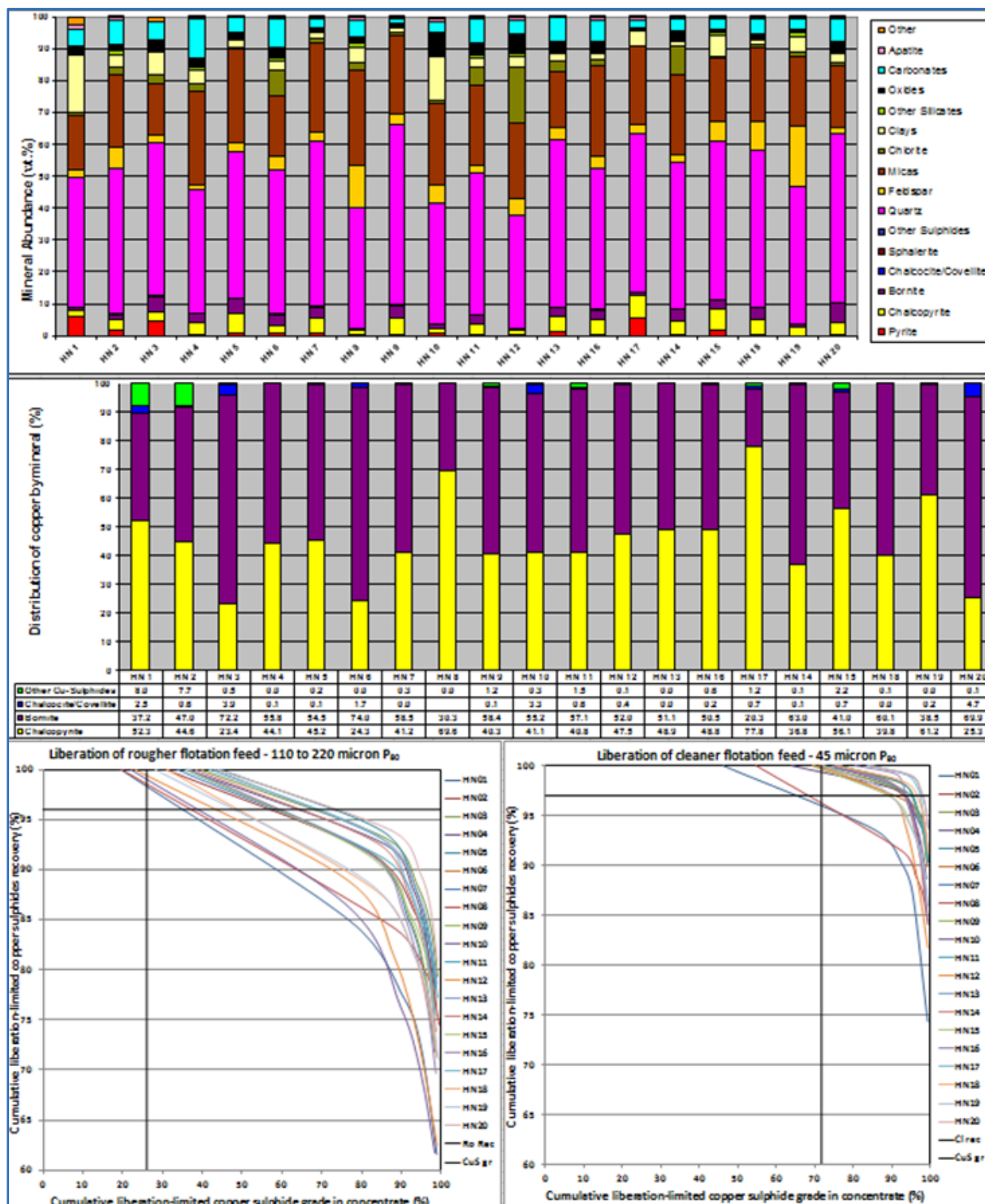
13.4 Mineralogy

A large number of direct and indirect mineralogical assessments have been carried out on ore and flotation products, in the following categories:

- Routine thin sections on intervals of core in conjunction with logging to qualitatively assess the nature of the copper mineral and gangue mineral assemblage.
- Routine semi-quantitative clay mineral measurements by infrared spectroscopy to assist in alteration classification and to potentially identify rheology-modifying species that could be problematic in processing.
- Visual logging of all core with respect to estimated sulphide mineral totals.
- Mineralogical assessment of ore sections from all deposits by Terra Mineralogical Services (TMS), including analysis of gold association, fluorine deportment in ore and concentrate, copper mineral associations in tailing, and leach residues (49 reports and memos from 2002–2005).
- The production by TMS of a spatial 'metallurgical index' block model of metallurgical degree of difficulty, primarily for the Southwest and Central zones, but also with some coverage of Hugo North (Far North Extension as then known).
- Diagnostic leach work on oxide and secondary copper zones to distinguish between chalcocite, chalcopyrite, and covellite.
- QEMSCAN on particulate Southwest and Hugo North composites (flotation feed and rougher concentrates).
- Full QEMSCAN analysis on all 20 flotation feed composites from Hugo North and Central zone testwork programmes (Blue Coast/SGS).
- X-ray diffraction and QEMSCAN on composites of flotation tailings produced for NAF/PAF characterization.
- Mineralogy inferable from the 48-element ICP assays on 24,000 intervals over all deposits.
- Liberation analysis by conventional particle counts on Heruga.

A graphical summary of QEMSCAN results for the 20 Hugo North composites, are given in Figure 13.6.

Figure 13.6 Presentation of QEMSCAN Results for 20 Hugo North Composites



The first graph in Figure 13.6 displays mineral abundance by weight in the feed, summing to 100%. Sulphides are at the bottom in the stacked chart, with pyrite in red. Pyrite is only present at significant levels in three of 20 composites and when present is usually accompanied by higher-than-average levels of copper sulphides, which leads to easier separation. Little dilution of concentrate by pyrite has been observed in previous flotation work, as expected from this mineralogy.

Copper sulphides plus pyrite rarely form more than 10% of the total weight, with chalcopyrite, bornite, and chalcocite/covellite present at 3.9%, 2.7%, and 0.04%, respectively by weight. Quartz is the dominant rock-forming mineral (46% on average), followed by sericite mica (24%), chlorite (3%), and feldspar (5%). Clays account for 1%–18% of the mineral components in the composites, but average less than 5% overall. The broad footprint of the cave is likely to minimize daily variation in clay content to very manageable levels in the grinding and flotation circuits.

Oxides, primarily of iron (magnetite, hematite, and goethite), average only 2.8% and carbonates 5.4%. The former are too low to provide much benefit from magnetite recovery, while the latter present useful buffering capacity to minimize acid mine drainage from tailings. Apatite is present at 0.6 wt%, and is moderately variable. It can locally form a significant source of fluorine in feed and thus, by entrainment, in concentrate. Overall, the previous work has indicated less fluorine contributed by apatite, than by sericite and fluorite.

The second graph shows the relative contributions of chalcopyrite, bornite, and chalcocite / covellite to the total copper content of the feed. Due to its high stoichiometric grade, on average bornite contributes 52.3% of the copper, followed by 45.5% from chalcopyrite, only 1.1% from chalcocite/covellite, and 1.2% from other copper sulphides. The latter will also include the sulfosalts tennantite and, to a much lesser degree, tetrahedrite. The former is the predominant arsenic source for Hugo North and is difficult to depress, even at high pH. In Central zone ore, arsenic reports to concentrate at 78% of the copper recovery and is likely to be similarly related in Hugo North. The high bornite content implies a limiting average grade of 46% copper in concentrate. The metallurgical correlations from flotation testwork include the dilution contributed by pyrite flotation, by entrained free gangue minerals, and by incomplete liberation of both minerals from the copper sulphides. This results in an average 35% reduction in copper grade below the theoretical limit established by quantitative mineralogy.

Incomplete liberation also results in incomplete copper sulphide recovery, as indicated by the lowest pair of graphs. The left graph shows the rougher feed cumulative liberation yield (CLY) profile. A copper-sulphide mineral grade versus incremental recovery plot is obtained by including progressively less-liberated particles from the lower right to the upper left, until all copper sulphide containing particles have been included, at the 100% recovery axis. At roughing sizes from P_{80} 110–220 μm , there is a fair degree of variation in liberation level, which is only partially independent of the P_{80} variation from the different sample work indices. Hugo North composites HN1 and HN8 are softer, finer, but less well-liberated, while HN18 and HN19 are harder, coarser, and also less well-liberated. The average Hugo North rougher flotation grade-recovery point (96% to 12% Cu, or 26% copper sulphides) is included for reference, at the intersection of the horizontal and vertical target lines. It is comfortably to the left of any of the 20 CLY curves, allowing room to include significant dilution resulting from the 10% mass yield to rougher concentrate that occurs naturally from gangue entrained in water in froth after 30 minutes of continuous froth removal.

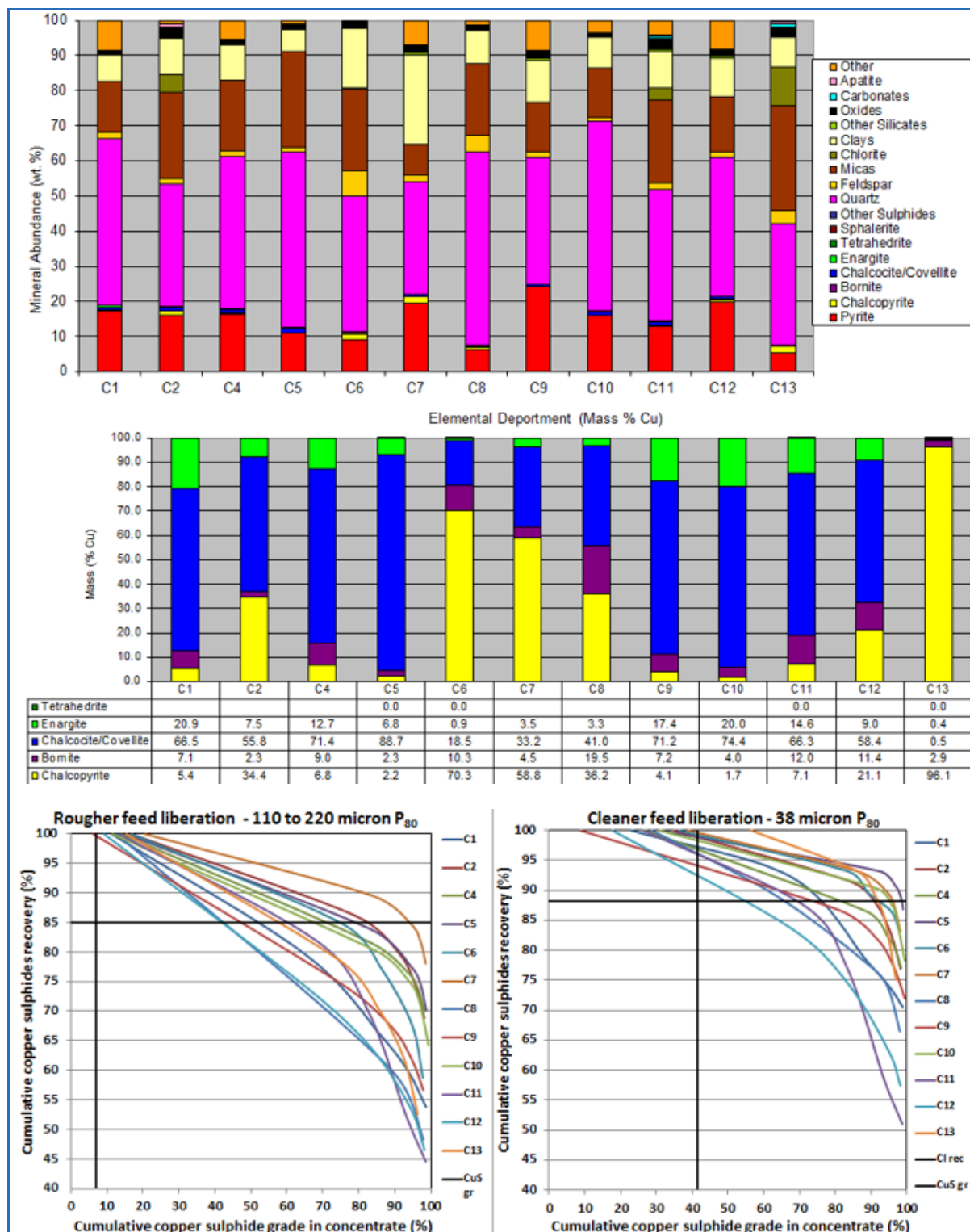
All but four of the 20 composites intersect the 100% recovery axis between 30%–45% copper sulphides, which is at the bottom of the normal range of liberation for porphyry copper processing. Lower plant recoveries are to be expected for the other four composites, with only 20%–25% copper sulphides when 100% of the copper distribution is included. The data also demonstrates the importance of regrinding, compared to other, better-liberated porphyry copper deposits, where regrind circuits are sometimes considered an optional extra. If a 90% liberation level is considered necessary for the production of marketable concentrate, as is usual, then overall copper recovery in roughing would have to be restricted to 80%–85%, if the regrind circuit were shut down, compared to 93% overall with regrind. This assumes that the normal offset between theoretical CLY distribution and actual plant recovery performance applies.

The right-hand graph shows the same data for the cleaner feed size distribution, at a P_{80} of 45 μm . This is constructed from a weighted average of the data for separate $-38 \mu\text{m}$ and $-106+38 \mu\text{m}$ fractions. The degree of liberation after regrinding to the target Hugo North size distribution is much higher than in roughing and the variability much reduced. All but two of the 20 composites meet the 100% copper distribution axis at 65%–85% copper sulphides and the concentrate is 90%–96% liberated at the 97% cleaner recovery target. The two composites that have liberation challenges are HN1 and HN8, which also showed sub-normal liberation in roughing. Referring to Table 13.4, the copper assays for both composites are below 1%, so that not much copper distribution is at risk. They are unlikely to present more than 5% of the draw at any given time.

The mineralogy of the Central zone composites, Figure 13.7, shows more complex sulphide speciation, significant levels of copper present as arsenic-bearing enargite, much higher pyrite content (8%–24%) and much finer and more variable grain sizes at rougher and cleaner feed sizes. Bornite is minor in all zones and chalcopyrite is predominant in the Central chalcopyrite zone, represented by composite C13 and also by composites C6 and C7. In the other composites, most of the copper is present as covellite, followed by chalcocite, then by chalcopyrite, enargite and bornite.

It is possible to produce Central zone concentrate of 25% grade and 75% recovery by use of very high pH in cleaning. These levels are shown in copper sulphide grade and cleaner stage recovery terms in the cleaner feed liberation graph. However, the use of lime to pH 12 at 25 μm to depress/liberate pyrite is very expensive and is untested in blends with majority Hugo North ore to reflect the conversion case, further high pH is expected to depress copper and final gold recoveries for Hugo North ores and so Central zone ores will be processed instead at Hugo North / Southwest ore conditions. In recognition of this, concentrate grade and recovery predictions for Central zone ore chalcocite and covellite were reduced in Base Data Template 31 (BDT31) compared to prior predictions, to allow for less rejection of pyrite and less-effective collection of copper minerals when flotation conditions were optimized for the economically-dominant Hugo North ore.

Figure 13.7 Presentation of QEMSCAN Results for 12 Central Zone Composites



13.4.1 Availability and Volume of Testwork Conducted

The samples selected in the latest confirmatory programme of work on the northern area of Hugo North are shown in Table 13.4 and Table 13.5. The confirmatory work better investigated low pH conditions for Central zone ore flotation of composites that were both spatially and mineralogically discrete in terms of sulphide speciation. It also generated flotation results for Hugo North composites, which displayed the full range of copper head grades, gangue mineralogy / alteration, and for which comminution characteristics had been defined in SPI / MBI / Ci terms.

Table 13.4 New Flotation Composite Selections for Hugo North

Designator	Grade		Alteration
	Cu (%)	Au (g/t)	
HN1	0.99	0.07	Intermediate Argillitic
HN2	1.85	0.43	Mainly sericitic (SER)
HN3	4.18	0.38	Mix of IA, chloritic (CHL) & SER
HN4	2.34	0.38	IA
HN5	3.16	0.75	Mainly SER
HN6	2.69	1.18	Mix of IA, CHL & SER
HN7	3.15	1.26	SER
HN8	0.81	0.15	SER
HN9	4.04	1.82	SER
HN10	1.30	0.24	Mainly IA
HN11	2.68	1.05	Mainly CHL
HN12	0.79	0.11	Mix of SER & CHL
HN13	3.15	1.11	Mix of IA, CHL & SER
HN14	3.04	0.95	Mainly SER
HN15	3.09	0.50	Mainly SER
HN16	2.49	0.57	Mainly SER
HN17	2.57	0.37	SER
HN18	3.25	1.25	SER
HN19	1.43	0.37	SER
HN20	3.79	0.44	Mainly IA

Table 13.5 New Flotation Composite Selections for Central Zone

Composite Descriptor	Composite No.	Weight (kg)	Cu (%)	Au (g/t)	Approximate Ratio of Pyrite to Copper Mineral Content
High Pyrite 1	C1	30.1	0.75	0.15	80 : 19
North East High Pyrite	C2	45.1	0.85	0.40	80 : 19
Central- central	C3	41.5	0.66	0.16	80 : 20
High Pyrite 2	C4	27.7	0.46	0.10	85 : 14
Low Pyrite 1	C5	21.1	0.71	0.05	59 : 40
Low Pyrite 2	C6	21.1	0.74	0.38	52 : 47
South	C7	19.7	0.93	0.15	75 : 24
South East	C8	18.4	0.72	0.08	72 : 27
North East Low Pyrite	C9	39.6	0.47	0.18	68 : 31
Deep	C10	36.0	0.68	0.21	74 : 25
Chalcocite 1	C11	45.1	0.98	0.13	32 : 67
Chalcocite 2	C12	49.9	0.85	0.16	81 : 18
Chalcopyrite	C13	51.9	0.59	0.82	58 : 41

13.4.1.1 Variability Testwork

The locations of the additional flotation samples are aligned with the Hugo North comminution samples. Spatial variability composites for flotation were generated from three to seven interleaved sub-samples of the core intervals selected for the comminution samples. These are designated by points of a common colour on Table 13.4. The new Hugo North flotation samples are described in Table 13.4. Selection criteria for compositing were primarily spatial, with fairly tight groupings that could be tracked via similar height-of-draw. However, the process managed to differentiate a wide range of head grades for head grade-recovery relationship development and also managed to classify partly by alteration type.

Plan views and sections of Central zone flotation composite sample locations are presented in Figure 13.8 and Figure 13.9, respectively. These samples were selected solely on the basis of flotation characteristics rather than comminution characteristics, since these are likely to be a greater constraint in operation than the latter. Central zone flotation samples were selected for compositing for variability testwork, as shown in Table 13.5, and were differentiated by dominant copper mineralogy and pyrite content to characterize the major performance drivers. Sample selection also retained some spatial separation to allow future refinement of the block model performance predictions by zone.

Figure 13.8 Sample Locations for Central Zone Flotation Composites – Plan

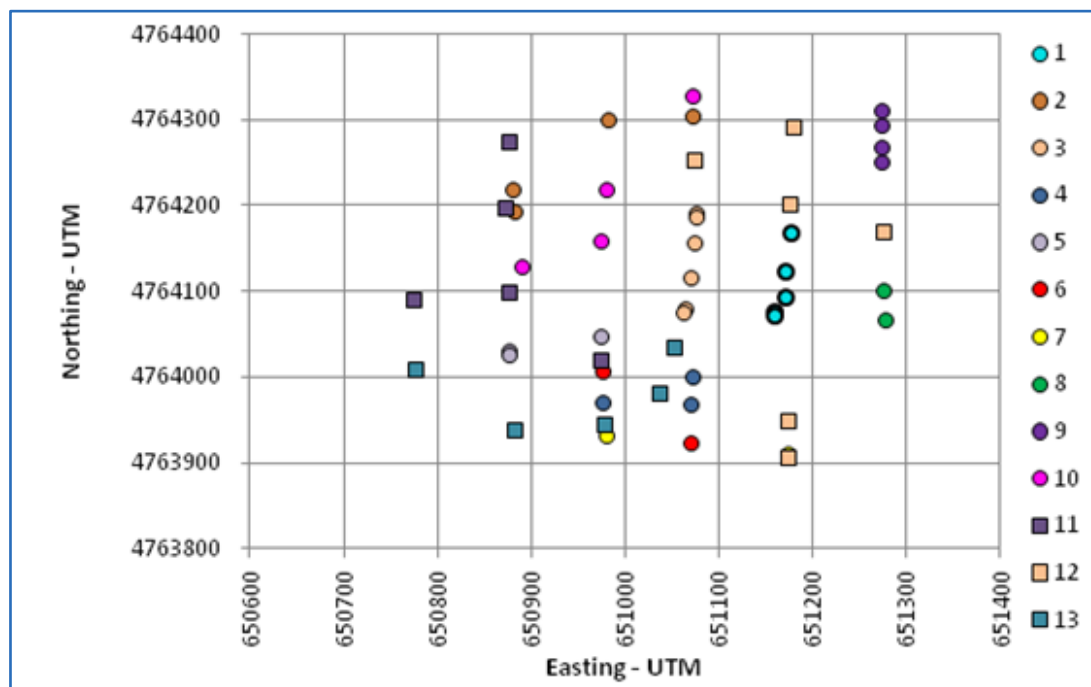
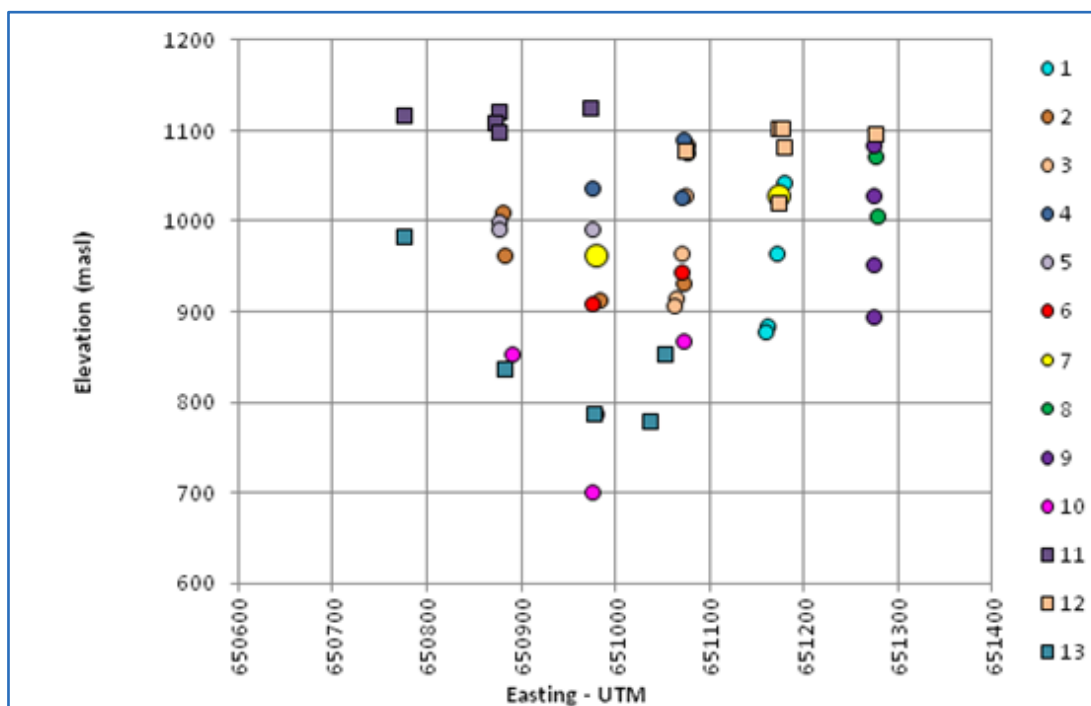


Figure 13.9 Sample Elevations for Central Zone Flotation Composites – Section



13.4.1.2 Effect of Processing Variables on Flotation

Rougher Flotation Feed P_{80}

Flotation feed sizing in the block model outputs is established by the SAG mill / ball mill power split and the ratio between SPI and MBI for Southwest zone, Central zone, and Hugo North. As discussed previously, a correction is required for the addition of Ball Mill 5. The economic optimum flotation feed sizes are summarized in Table 13.6. These values have been approached quite closely by the grinding circuit design and production schedule predictions via the hardness parameters in the block model, which allows the continued use of the Integrated Development Plan 2005 (IDP05) metallurgical predictions for Hugo North and Southwest zone with minor corrections of the latter for recent operating experience. The size-by-size Aminpro grind–recovery optimization approach is described in Section 13.4.1.3 on the flotation capacity modelling.

Table 13.6 Optimum Primary Grind Size for Each Ore Type (P_{80} , μm)

Deposit/Composite	IDP05	Aminpro 2007
Southwest Zone	180	180
Hugo South	150	–
Hugo North	140	116
Central Zone	138	179

Cleaner Flotation Feed P_{80}

In the absence of penalty element liberation problems, the coarsest regrind sizing that achieves 90% liberation of copper sulphides in cleaner feed is generally considered a good estimate of the optimal regrind level in plant operation. Hugo North ore has showed uniformly lower fluorine levels than Southwest zone ore in concentrate from locked-cycle testwork. In testwork, one-third of the Hugo North concentrates would exceed the 300 ppm fluorine penalty level. Penalties between the 300 ppm penalty threshold and the 1,000 ppm rejection level are insufficient to repay much investment.

Rougher and Cleaner Flotation Conditions

The Aminpro work also used Southwest, Hugo North, and Entrée kinetic flotation work by PRA in Vancouver to develop flotation simulation models in roughing and in cleaning that could be calibrated against the kinetic work and used to simulate the effects of ore type, copper head grade, primary grind level, rougher pH, regrind level, and cleaner pH.

In general, the following trends were observed:

- With sufficient collector adjustment, copper recovery is insensitive to pH within broad ranges (pH 7–11).
- Gold recovery is adversely affected by lime addition (both pH and Ca^{++} concentration above pH 9) and is not as responsive to additional collector. This has influenced a slower

ramp-up of Central zone open pit development in OTFS16 until high-gold Hugo North ore has been processed (2022–2024). Gravity gold recovery is a possible contingency to recover slow-floating gold, while unit cell operation on regrind cyclone underflow is a possible means of preventing as much gold from becoming slow-floating by reducing over grinding.

- Additional collector and retention time is required at high copper head grades (feed forward strategy required to link collector addition to copper metal units in flotation feed).
- Better copper grade-recovery response and pyrite rejection are typically achieved with dithiophosphinate collector (3418A) than with any single xanthate (isopropyl, isobutyl, or amyl). However, xanthate storage and mixing facilities have been provided for potential synergistic addition with secondary gold collectors. The Blue Coast testwork completed in 2012 indicated a slight advantage in copper and gold recovery with potassium amyl xanthate. The results were not conclusive, however, against the comparative 3418A tests conducted at higher rougher concentrate grades.
- Additional cleaner collector is required at finer regrinds and higher pH values.
- There is a benefit from staged addition of collector.
- Rougher flotation kinetics might be slower at low pulp potential (eH). In recent confirmatory testwork, rougher flotation response was delayed until the flotation pulp potential (absolute) was above 0 mV. This trend was exacerbated by even modest lime additions, because increasing pH reduces eH. It is possible that this observation is a reflection of batch testwork and not representative of a continuous flow system. The cyclone overflow eH in almost all concentrators (except those treating ores with extremely high pyrite content, or an active pyrite or pyrrhotite content) is routinely in the range of 0–50 mV, with no specific chemical interventions or additional aeration in the grinding circuit. Even the most-pyritic Oyu Tolgoi ores have less than 15% pyrite content, which is not chemically active. If low eH is encountered, then additional aeration may be warranted. There is space to retrofit conditioners or aeration devices of a few minutes' capacity on the ball mill floor below the cyclones. In this event, it might also be necessary to retrofit an additional pumping stage, for which the grinding basement has sufficient space.

Water Quality

Testwork to assess the effect of water quality on flotation has been conducted since completion of IDOP. A bore water composite was collected from the Gunii Hooloi bore field as a simple average of samples from individual wells. Testwork was conducted at SGS, with Vancouver tap water used as a control. The testwork indicated that, for Southwest zone ore, recycle water was favourable to the copper grade-recovery curve at lower grades (possibly due to collector recycle), but unfavourable to copper recovery at higher grades. One year of experience at Oyu Tolgoi has shown no ill-effects from using process water.

13.4.1.3 Flotation Capacity Modelling

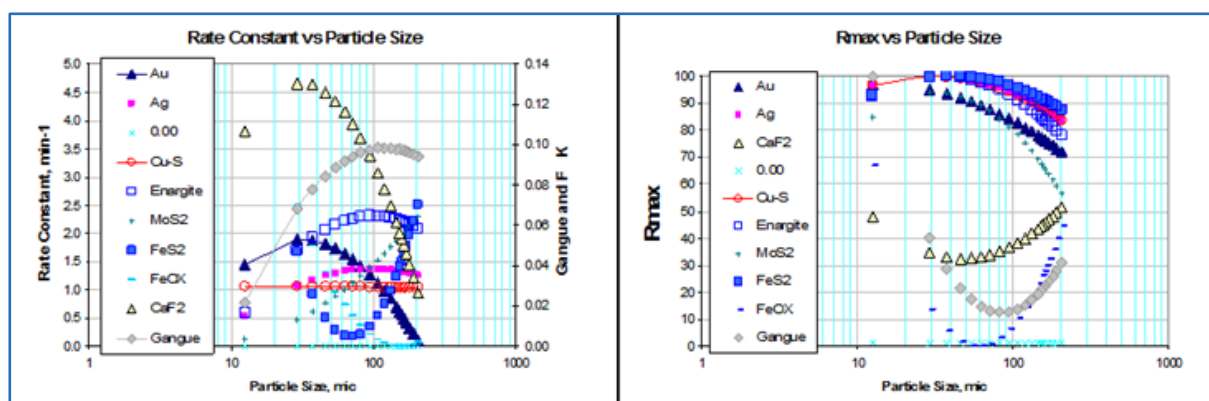
The selection of flotation design criteria for mechanical cells in the concentrator conversion has taken account of the following information:

- The laboratory bench kinetic testwork at Ammtec in roughing and cleaning, while achieving the rougher and cleaner overall stage recoveries required by the mass balance.
- The review of flotation kinetics by Aminpro and the results of the Minemaster model for Hugo North and Central zone ores. Column cell and mechanical requirements were confirmed at both 30 μm and 40 μm grinds by Aminpro simulations around results from PRA kinetic flotation test programmes carried out in Vancouver.
- Comparison with cell capacity allocations for Lines 1–2 in Phase 1, before and after an additional rougher bank.

Aminpro evaluated the kinetic tests carried out at PRA in Vancouver to determine rate constants (k) and maximum recoveries (R_{max}). These values formed the basis of the detailed design of the Phase 1 flotation circuit design. Two examples of the output for Southwest zone ore rougher flotation are given in Figure 13.10 and reflect FLEET methodology (FKT and VSKT tests). The curves shown represent fitting of rougher timed recovery results for four particle size fractions for each mineral (+150 μm , –150+75 μm , –75+25 μm , and 25 μm). The rougher work was carried out on Southwest, Central, Hugo North, and Entrée composites. Similar results are available at +32 μm , –32+25 μm , –25+20 μm , and –20 μm in cleaning for Entrée ore (Hugo North extension).

The mineral contents are developed from indicator assays (Au, Ag, F, Cu, Mo, and Fe, As, S) and balanced to 100%.

Figure 13.10 Aminpro Modelled Size / Recovery Relationships for Southwest Zone Ore



Changing the flotation P_{80} changes the weighting in each of the size fractions, which in turn alters grade-recovery curves. As in the Ammtec evaluation, the additional operating and capital costs associated with increased grinding were set against the additional revenue from copper and gold recovery. This was extended to the cleaner circuit and repeated for each ore type. This exercise was not repeated in the feasibility study for the concentrator conversion design, but rather retention times were specified from laboratory tests and compared with the capacities and retention times specified for Phase 1.

After the addition of the extra rougher bank for the conversion, rougher retention times and froth carrying capacities will be near those used in Phase 1. Currently the mechanical cleaners are handling a higher-than-expected flow due to low column stage recovery (20% versus 60% design). The mechanical cleaners, which are not being expanded, will have slightly shorter retention time and increased froth loading compared to Phase 1. The high recirculation of column cleaner tails observed in Phase 1 is not projected to persist when treating the high-grade Hugo North ore with 10 column cells in place of four in Phase 1.

The initial selection of column cell capacities for the expansion was factored from the Phase 1 design and the Minemaster modelling. The Phase 1 columns are currently operating at a copper stage recovery near 20%. It is projected that when treating Hugo North ore, the recoveries will be above the Phase 1 design of 40% due to coarser regrind (45 µm versus 35 µm) and lower upgrade ratios. The column cell expansion was determined by froth-carrying capacity rather than retention time. The Phase 1 column cell dimensions were retained for the six additional concentrator conversion cells.

13.4.1.4 Thickening and Filtration Capacity

Testwork has not focused on generating large volumes of concentrate and tailings for thickening and filtration testwork, as was carried out for Southwest Oyu ore. To allow for a conservative design, the same unit thickener capacities have been used for concentrate thickening as in Phase 1, despite the coarser regrind targets. The same is true in the final tailings area, where the dewatering duty for blended Southwest, Central, and Hugo North tailings is similar to Phase 1.

Phase 1 thickener optimization is still in its infancy, but significant reductions in flocculant addition (from 30–20 g/t) have recently been achieved, while improving underflow density.

Conservative design margins for the thickener unit area was adopted in Phase 1. This can be further enhanced by higher flocculant addition. In a recent operating period in which one thickener was out of service, the remaining thickener was operated at rates to 75 kt/d. It is unlikely that the conversion will push the existing thickeners to capacity. As a contingency measure in detailed design, more thickener area is available at locations that the original Fluor design had reserved for two additional tailings thickeners. Such additions would be motivated by operating experience on Southwest zone ore, but tonnages of 139 kt/d (5.8 kt/h) have been averaged for six days on soft ore and handled well by the thickeners when running two tailings lines to the closer discharge points. An additional booster pump stage is required at greater tailings dam elevations and pump bases and electrical supply was allowed for in the Fluor Phase 1 design. It is included in the 2016–2017 capital plan.

Phase 1 concentrate filtration performance has performed to specification. Filtration has been trouble-free, allowing a very straightforward scale-up for the conversion, which includes the addition of two more identical filters. Industrial experience indicates that cake formation rates will increase by 14% due to the envisaged coarser Phase 2 regrind (45 µm vs. 35 µm). A location for a fifth pressure filter has been reserved in the layout as a contingency against a further 20% increase in peak filtration duty.

It is recommended that further Hugo North tailings thickening and concentrate pressure filtration testwork at 0.1 m² scale be conducted before detailed design, but after underground development has progressed, to allow lower cost acquisition of larger-diameter core samples from a greater number of access points. Phase 1 operating performance will continue to be monitored.

13.5 Metallurgical Predictions

13.5.1 Metal Recoveries and Throughput

Each month, the OT LLC concentrator staff carry out a metal balance for the period and back calculate the concentrator feed based on the copper concentrate produced during the month and the tonnage and quality of concentrator reject reporting to the tailings. Using these data, the recovery of the various metals in the feed delivered from the mine, into the saleable concentrate, can be estimated. This work was used by OT LLC to confirm the assumptions for metallurgical performance and throughput estimation that were prepared for OTFS14 remained valid for OTFS16 as the copper grade of concentrate, and the copper and gold recovery into concentrate, are consistent with the ore feed make-up. The models are presented in Table 13.7 to Table 13.10.

The parameters used for Hugo North were also applied to the Entrée ore, which is a continuation of the same orebody beyond the Oyu Tolgoi lease boundary. Additional testwork conducted in 2012 on 20 Hugo North composites did not materially affect comminution or recovery estimates, given the greater volume of earlier work and the use of fresh core. The Hugo North core for the Blue Coast testwork at SGS was stored outside (at ambient conditions, but kept dry) for six years before selection, crushing, and freezer storage at SGS. Similarly, the Central zone core was stored outside for eight to ten years before testing. While local conditions are relatively non-conducive to oxidation, it is possible that flotation response suffered as a result. Fresh core from the 23 drillholes from the 2013 infill drilling programme have been preserved in frozen storage as a precaution.

Table 13.7 Base Data Template 31 – Copper Recovery

All Ores				
$a \times [(b \times \text{Cu}\%)/(1+b \times \text{Cu}\%)] \times [1-\exp(-b \times \text{Cu}\%)]$				
Southwest Zone a = 98 b = 14.5	Hugo North a = 95 b = 15	Central Zone Covellite a = 80 b = 15	Central Zone Chalcocite a = 72 b = 15	Central Zone Chalcopyrite a = 88 b = 12.2

Table 13.8 Base Data Template 31 – Gold Recovery

All Ores				
$c \times (d \times \text{Cu Recovery})$				
Southwest Zone c = 4.8 d = 0.80	Hugo North c = 9.8 d = 0.80	Central Zone Covellite c = 4.8 d = 0.65	Central Zone Chalcocite c = 4.8 d = 0.70	Central Zone Chalcopyrite c = 4.8 d = 0.80

Table 13.9 Base Data Template 31 – Silver Recovery

All Ores
$13 + 0.8 \times (\text{Cu Recovery})$

Table 13.10 Base Data Template 31 – Copper Assay in Concentrate

Southwest Zone	Hugo North	Central Zone Covellite and Chalcocite	Central Zone Chalcopyrite
$-3.6 \times (\text{Cu} : \text{S})^2 + (12.8 \times \text{Cu} : \text{S}) + 22.5$	$2.9 \times (\text{Cu}) + (11.4 \times \text{Cu} : \text{S}) + 15.3$	20	$-3.6 \times (\text{Cu} : \text{S})^2 + (12.8 \times \text{Cu} : \text{S}) + 21$

The plant throughput formula is shown in Table 13.11. The volumetric capacity limit in BDT31 that was used in OTFS14 was 5.5 kt/h (121 kt/d, 44.3 Mt/a). After a review of the volumetric capacity in OTFS16, this was reduced to 5.0 kt/h (110 kt/d, 40 Mt/a). For the preparation of the OTFS16 production schedule the plant throughput volumetric limit was changed from 5.5 kt/h to 5.0 kt/h and the instantaneous throughput was increased by 2.2%. Further elevation and revision of the limit is quite likely as de-bottlenecking and optimization of the plant continues. The OTFS16 limit has already been reached and may be exceeded as the Central zone ore is treated.

Table 13.11 Plant Grinding Throughput Rates

All Ores
Flotation feed $P_{80} = 113 \times \text{Ci}^{0.26} \times \text{SPI}^{-0.60} \times \text{BM}^{0.88}$ Maximum P_{80} guideline = 220 μm
Throughput in t/h (instantaneous) = $29,320 \times \text{Ci}^{0.19} \times \text{SPI}^{-0.36} \times \text{BM}^{-0.24}$
Maximum throughput = 5.5 kt/h (hydraulic limitation)

Note: For the preparation of the OTFS16 production schedule the plant throughput volumetric limit was changed from 5.5 kt/h to 5.0 kt/h and the SAG capacity was increased by 2.2%.

13.5.2 Penalty Element Mineralogy, Control and Economic Impact

Arsenic and fluorine are the only penalty elements that have been identified in the Reserves Case deposits. Enargite is the primary arsenic carrier in all deposits, although tennantite is locally important.

High flotation pH is the primary mineral processing control on arsenic recovery, but it is only partially effective because of the difficulty in depressing enargite and the related copper losses. In addition, high pH has an adverse impact on gold recovery and is therefore not expected to be used often.

Fluorine distribution in concentrates is more variable, being locally present as coarser-grained fluorite or finely intergrown topaz in some high-fluorine areas, but with a background level distributed as 0.6%–2% fluorine in sericite, which itself represents 15%–30% of the weight of the deposits. Regrind level and the degree of entrained gangue removal are the primary control mechanisms for fluorine.

As long as concentrator feed is managed such that rejection levels are avoided, the modest impact of fluorine and arsenic penalties averaging less than US\$5/t of concentrate. To handle production peaks while maintaining a base load for contract, a certain amount of the Oyu Tolgoi concentrate production has been considered for sale to traders for subsequent blending. This could be an avenue for disposal of high-penalty element concentrates.

For arsenic in copper concentrate, the production model assigns a rate of US\$2/t/1,000 ppm above a 3,000 ppm threshold up to the rejection level of 5,000 ppm. For fluorine, the production model assigns a rate of US\$2/t/100 ppm above a 300 ppm threshold up to the rejection level of 1,000 ppm. The penalties are in line with terms from custom smelters. However, it has been reported that no fluorine penalties have been applied under the contract terms in operation since sales commenced in late-2013, so some conservatism is inherent in the Net Smelter Return (NSR) estimates.

13.5.2.1 Penalty Element Predictions – Fluorine

Previous analyses of Hugo North and Southwest zone ore data from locked-cycle test results are shown in Figure 13.2, where the blue line describes the formula used for predicting fluorine in concentrate for all ore types in the IDP10, IDOP, OTFS14, and now OTFS16 (Table 13.12). The testwork results support the fluorine content of concentrates from the Central zone and Hugo North deposits. For Central zone ore, the fluorine assay in concentrate from 82 batch cleaner variability tests was predicted to be 0.153 (adjusted to pass through origin) of the fluorine assay in feed (green line).

Fluorine and arsenic predictions for all ore types are as shown in Table 13.12.

The fluorine grade in final concentrate from Southwest zone ore has been almost twice what would have been projected from the relationship above based on batch test and locked-cycle results. It is suspected that the especially good fluorine rejection in the lab work is a function of a generally finer P_{80} than currently targeted in plant operation (P_{80} of 25 μm vs. 35 μm) and partly because the bead mill used in batch mode in Ammtec laboratory work had a very steep size distribution, with most of the top 20 wt% very close in size to the P_{80} . In the past, size-by-size fluorine analysis of the final Southwest zone concentrate has indicated a rapid increase in fluorine at coarser sizes.

As a result of this trend, and the use of tower mills and the coarser regrind in future, it was decided to increase the factor from 0.15 to 0.3 for all ore types to account for the plant response. This has not presented a problem with rejection limit in the production schedules.

Figure 13.11 Fluorine Recovery and Mass Yield to Concentrate – Hugo North and Southwest Zone Locked-Cycle Correlation vs. Central Zone Ore Batch Test

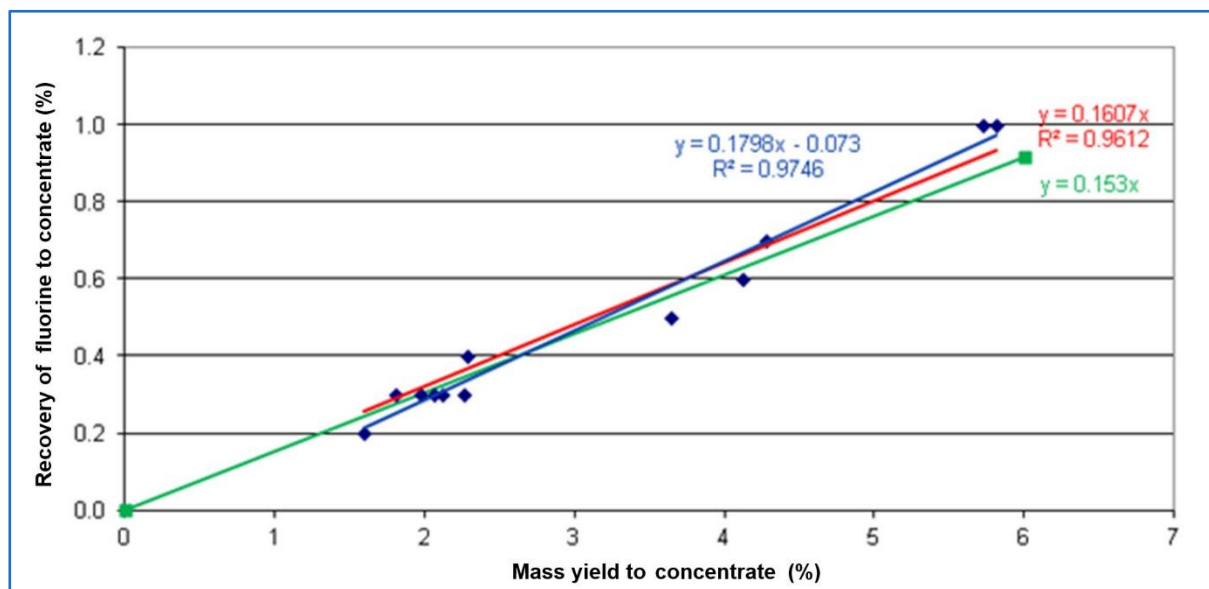


Table 13.12 Base Data Template 31 – Arsenic and Fluorine in Concentrate

Arsenic in Concentrate (ppm) [m x ConCu % x As (ppm)] / Cu%	Fluorine in Concentrate (ppm)
For Southwest zone ores: m = 0.125 For all Other ores: m = 0.780	0.3 x Fluorine in feed (ppm) for all ores

13.5.2.2 Penalty Element Predictions – Arsenic

Given its less-variable mineralogy and positive association with copper minerals, arsenic in concentrate should be predictable with greater precision than fluorine. The relationship in Table 13.12 was derived from locked-cycle tests for Hugo North and Southwest zone ores and was used for all ores in IDOP. It has been retained for OTFS16 and relates arsenic in concentrate directly to arsenic in feed, with a negligible intercept at normal arsenic levels. Because the mineralogy has indicated that arsenic is largely contained in copper sulphosalts (primarily enargite), which recover almost as well as the primary copper minerals, this result also requires that the As : Cu ratio in concentrate differs from the ratio in feed only by the ratio of arsenic to copper recovery.

A simpler correlation was originally developed on Central zone ore for 49 batch cleaner variability tests from chalcocite, covellite, and chalcopyrite zones (Figure 13.12). The correlation is of acceptable quality but has much greater scatter than for the Hugo North and Southwest zone data and has a gradient of 24 rather than 21. The updated hypothesis

described above was tested by correlating As : Cu ratios in feed and concentrate for the same data in Figure 13.13. A much better correlation was obtained. The gradient of 0.78 infers that the arsenic recovery predicted for Central zone ore is 78% of the copper recovery.

Although this relationship is expected to hold for all ores at Oyu Tolgoi, with enargite being the predominant arsenic carrier, there is a persistent over-estimation of the arsenic in final concentrate from current Southwest ore. As a result, the recovery ratio constant was reduced to 0.125 for Southwest zone ore to match the current arsenic levels in concentrate.

Figure 13.12 Arsenic in Feed and Concentrate – Central Zone Ore

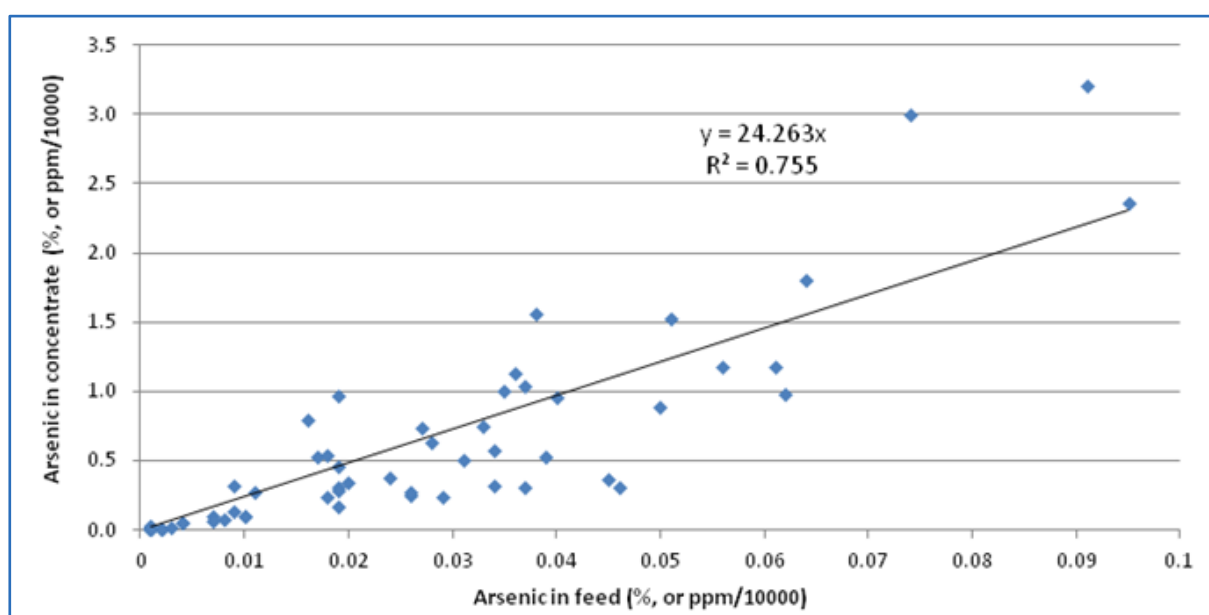
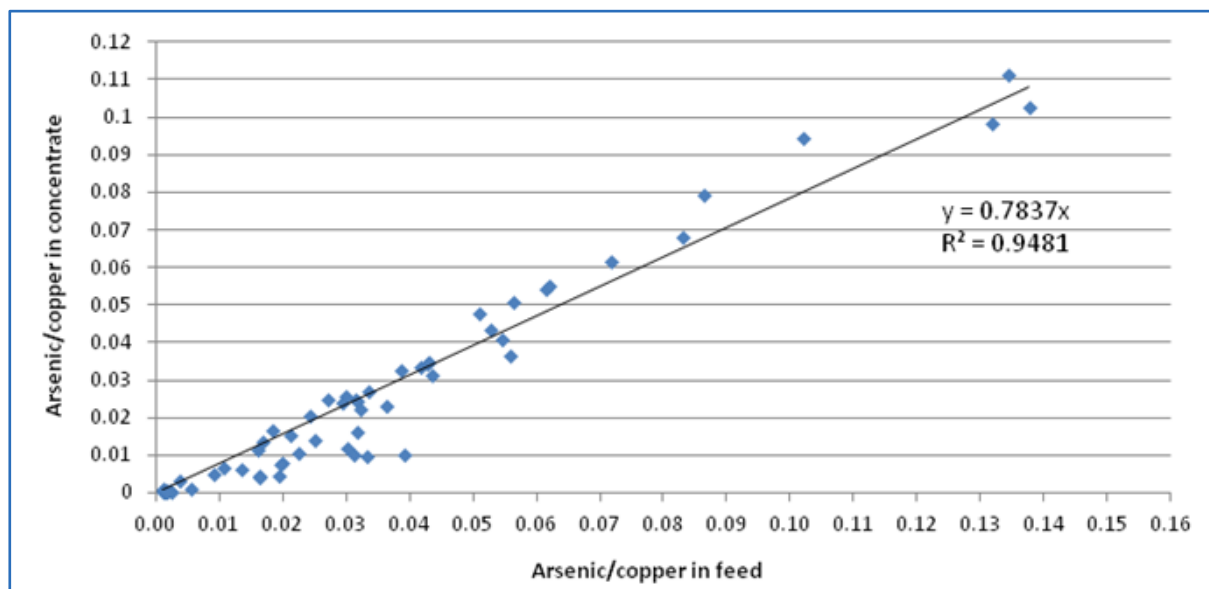


Figure 13.13 Arsenic to Copper Ratios in Feed and Concentrate – Central Zone Ore



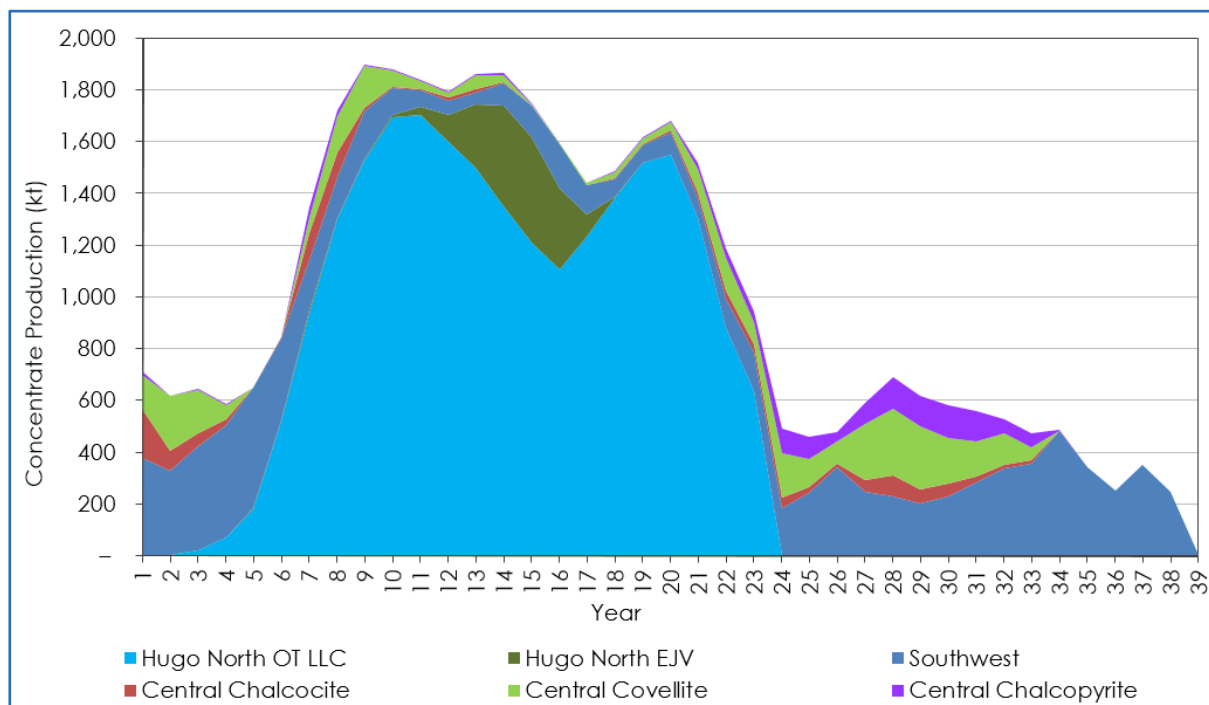
13.5.3 Concentrate Production, Payable Penalty and Minor Elements

Concentrate tonnages for sale, as predicted by the production model, are presented in Figure 13.14. Molybdenum content is insufficient in the 2016 Reserves Case to justify production of a separate concentrate, but does have value for Heruga in the Alternative Production Cases. Concentrate production volumes are dominated by the increasing contribution from Hugo North from 2020 onwards.

Copper assay varies with higher grade Hugo North production and increased bornite content early in the block cave. The peak grades from underground bornite are moderated by simultaneous treatment of large amounts of Central zone ore in 2022–2026. High copper content, especially high Cu : S ratio, is attractive to most smelters as it provides high copper yield while not taxing acid recovery and handling systems. The peak anticipated grades of 30%–35% Cu are projected from 2022 through 2030. The 2016 Reserves Case average after concentrator conversion is competitive with other imports to the Chinese market at 28% Cu. The minimum annual production grades of 23%–24% Cu in the last few years are less attractive, but are on a par with Erdenet product and represent small volumes far in the future.

Gold grade is much more variable, with peaks coinciding with extraction of the high-gold core to the maximum depth for successive phases of the Southwest zone pit. With open pit ore ramping up again from 2037, gold assay increases to three peaks over 2050–2055. Silver represents a much lower percentage of value and is elevated in the final years by virtue of a higher Ag : Cu ratio in feed. The significant variability in precious metals content may require shifts in concentrate allocations to smelters. Some smelters are better set up for precious metals recovery than others, thus making better margins relative to the amount of gold paid for.

Figure 13.14 Concentrate Production – 2016 Reserves Case



Arsenic and fluorine are penalty elements, but the terms have relatively little economic impact. At high levels in concentrate, smelters are unable to deal acceptably with arsenic and fluorine and, rather than a penalty, their presence becomes a basis for rejecting the concentrate. The Chinese State inspection agency also monitors quality and enforces national limits. Consequently, the primary concern is staying well clear of the rejection limits, and retaining the ability to respond to a potential decrease in the rejection limit if environmental standards become more stringent.

For each element, the annual mean level and the maximum level expected in a 5 kt shipment is estimated. Due to the differing sources of variation and measures available to control it, maximum fluorine is assessed at 1.2 times the annual average level, while maximum arsenic is assessed at 1.3 times the annual average. The fluorine variation allowed is based on an analysis of variation in Southwest production to date.

Average concentrate production is usually in a possible penalty position with regard to fluorine, if typical terms were applied. However, current shipments are not attracting a penalty and peak shipment levels still retain a minimum 10%–20% margin below the rejection level (1,000 ppm).

Average concentrate production will occasionally attract arsenic penalties when Central zone ores forms a significant fraction of feed. Arsenic maintains a minimum 30% margin to the rejection level.

Both fluorine and arsenic are modelled in the mine plan and neither element is expected to present significant long-term marketing difficulties. However, the primary control over fluorine rejection is in the hands of the concentrator, while the primary control over arsenic is by long-term planning and short-term grade control at the open pit mine. In Phase 1 and Phase 2, sufficient blending capacity exists in the concentrate slurry storage tanks (5–10 kt) and in the load-out shed (25 kt) to mitigate most process upsets affecting fluorine in a 5 kt smelter shipment. Such upsets would include loss of regrind efficiency or capacity or loss of control over column cleaner operation. Longer-term excursions in arsenic content in feed could be managed by maintaining a larger-than-usual inventory of higher arsenic as bagged product at the site and scheduling its release over a longer time.

Depression of arsenic by elevated pH in cleaning is not particularly effective and would affect gold recovery from Southwest zone and Hugo North ores. It would be a less-expensive measure whenever Central zone chalcocite and covellite ores predominate, when gold grades are much lower. Optimized cleaning schemes for Southwest zone and Central zone ores are possible when processed sequentially, but this is not possible in a mine plan optimized around aligning the grinding and volumetric capacity limits.

In addition to conventional payable and penalty elements, smelters are also interested in non-payable elements from which they may derive by-product credits (rhenium, mercury, selenium). There are also components that may be penalized in certain cases depending on other sources of smelter feed and their levels (bismuth, thallium). Other critical, non-penalty elements not tracked by the Oyu Tolgoi production model are also of importance in assessing a smelter's productive capacity (sulphur via the acid plant) or its operating costs and slag chemistry (Al, Ca, Mg, SiO₂, Fe). Such elements can be assayed directly in production year composites, or their overall variation inferred from other indicator assays or mineralogy. Finally, the particle size and the moisture of the concentrate are required to assess the dusting and bulk handling characteristics in the feed preparation and gas handling areas.

The expected means and ranges of these parameters are presented in Table 13.13. None of the parameters listed would appear to give smelters cause for concern. The ranges are necessarily wide to reflect the assay results from a variety of ore types treated over an extended mine life. They also vary due to the uncertainty in their recoveries to concentrate.

Final concentrate locked-cycle test concentrate assays were generated under conditions that follow those applied in Phase 1. Minor elements that were non-payable and non-penalty in nature were taken directly from the ranges observed in those tests. The major payable metal (Cu, Au) and penalty element (As, F) assay trends are best determined by applying the metallurgical prediction formulae for recoveries and final concentrate copper grade to the head grades predicted by the open pit and underground mining plans, block models, and dilution and mixing models.

Product specification will generally become more attractive and volumes will increase as the tonnage of high-grade Hugo North ore increases rapidly from 2020 onwards.

The high levels of arsenic in early Central zone ore will need to be managed by blending with the low-arsenic Hugo North ore, as has already been discussed. The arsenic content in final concentrate is a fairly direct function of As : Cu ratio in feed, and this parameter is one of the constraints in the mine production schedule.

As Hugo North ore production ramps down after 2036, arsenic levels are projected to increase significantly, but the metallurgical models predict a peak level of only 3,500 ppm; substantially lower than the current rejection limit of 5,000 ppm. The open pit mine plan has used a lower internal limit of 3,000 ppm from near-term production as a monthly average to avoid approaching rejection levels on a shipment-by-shipment basis. Contracts have been drafted so that payables and penalty elements are assessed on the weighted average of all lot assays in a 5 kt shipment.

Fluorine is projected to be above the usual penalty level but below the rejection limit throughout the mine life. Penalties are not always applied, but the 1,000 ppm rejection limit is legally enforceable. Unlike arsenic, control of fluorine is primarily within the scope of processing rather than pit grade control. The spread between peak shipment assays and annual average levels is based on variation observed in the first year of operation.

Major constituent non-payable, non-penalty components such as Fe, S, silica, and alumina are important for smelter metal and mass balances and are predictable from the mineralogy of ore and concentrate. The balance of less-significant concentrate components (minor elements) that are non-payable and non-penalty elements each form less than 1% of the total weight. Typical values and expected ranges are reported in Table 13.13. Ranges have been predicted from the full elemental assays for concentrate from each ore type, based on achieving a 100% mineral and/or metal balance in final concentrate with the predicted mineralogy and the average minor element assays.

Unlike the payable and penalty grades, major and minor non-penalty / non-payable components are stated as 'typical' values and are not expected to be a source of contract dispute, although moisture ranges should be respected, even with bagged product, to minimize freight costs either to seller's account (in Mongolia) or buyer's account (in China). Allowances have been made for the greater variation to be expected in a 5 kt shipment (representing one day's production at the Phase 2 peak) than in a monthly or annual average.

Table 13.13 Non-payable, Non-penalty Concentrate Analyses

Component	Unit	Combined Long-Term Typical Range (5 kt lots)
Al	ppm	4,000–15,000
Ba	ppm	20–100
Be	ppm	<0.1
Bi	ppm	<10
Ca	ppm	500–3,000
Cd	ppm	5–80
Cl	ppm	20–150
Co	ppm	50–200
Cr	ppm	15–100
Fe	%	22–36
Ge	ppm	0.5–3.0
Hg	ppm	0.2–5.0
K	ppm	1,500–3,500
Li	ppm	<5
Mg	ppm	500–4,000
Mn	ppm	50–400
Mo	ppm	500–4,000
Na	ppm	300–1,500
Ni	ppm	50–150
P	ppm	<100
Pb	ppm	100–1,000
Pd	ppm	0.05–0.30
Pt	ppm	0.02–0.15
Re	ppm	0.02–0.40
S	%	26–36
Sb	ppm	5–400
Se	ppm	150–500
SiO ₂	%	3–10
Sn	ppm	1–8
Sr	ppm	15–300
Te	ppm	4–60
Ti	ppm	500–1,600
Tl	ppm	<0.5
V	ppm	20–100
Y	ppm	2–10
Zn	ppm	200–3,000
Zr	ppm	200–600
Moisture	%	7–9
D ₈₀	microns	25–50

13.6 Future Work

Additional work is required in the following areas to advance the design of the process plant:

- Concentrator conversion feasibility study update.
- Phase 2 detailed engineering.
- Ongoing metallurgical testing programme for Hugo North and Central zone ores.
- Prefeasibility programme for additional resources (Heruga, Hugo South).
- Smelter studies.
- Heap leach studies.
- Magnetite recovery.
- Gravity or flotation separation of gold from the regrind circuit.
- Other improvements to gold recovery.
- Enhanced tailings treatment to reduce water retention.

14 MINERAL RESOURCE ESTIMATES

14.1 Mineral Resource Estimation

The following subsections describe the methods used for and results of the mineral resource estimation for the Oyut, Hugo Dummett, and Heruga deposits.

14.1.1 Databases

Database close-off dates for the Mineral Resource estimates are summarized in Table 14.1. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

Table 14.1 Database Close-off Dates

Deposit	Data Close-off Date
Hugo South	1 November 2003
Heruga	21 June 2009
Oyut	12 May 2011
Hugo North	14 February 2014
Hugo North Extension	14 February 2014

14.1.2 Geological and Grade Shell Models

OT LLC produced 3D geological models of the major structures and lithological units based on the structural and geological information outlined in the geological discussion in this report. The geological shapes for the deposits are listed in Table 14.2 and Table 14.3 for each deposit, appropriate copper and gold shells at various cut-off grades (Table 14.4) were also defined. These shapes were then edited on plan and section views to be consistent with the structural and lithological models and the drill assay data.

Checks on the structural, lithological, and grade shell models indicated that the shapes honoured the drillhole data and interpreted geology.

The lithological shapes and faults, together with copper and gold grade shells and deposit zones, constrain the grade analysis and interpolation. Typically, the faults form the first order of hard boundaries constraining the lithological interpretation.

The solids and surfaces were used to code the drillhole data. Sets of plans and cross-sections that displayed colour-coded drillholes were plotted and inspected to ensure the proper assignment of domains to drillholes.

Table 14.2 Surfaces and Lithology Solids used in Geological Modelling

Model Component	Comment
Surfaces – General	
Topography	Project-wide
Base of Quaternary cover	Project-wide
Base of Cretaceous clays and gravels	Project-wide
Base of oxidation	Project-wide, but relevant only for Oyut
Base of supergene alteration	Project-wide, but relevant only for Oyut
Solids/Surfaces – Lithology	
Top of quartz monzodiorite (Qmd)	Hugo South only
Quartz monzodiorite (Qmd) solid	Hugo North, Hugo North Extension, Oyut
Augite basalt (Va) D1	Oyut
Augite basalt (Va) D1 solid	Hugo North
Ignimbrite (Ign) DA2	Oyut
Ignimbrite (Ign) DA2 solid	Hugo North
Hanging Wall Sequence DA3, solid	Hugo North
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
Hanging wall sequence (HWS), DA3, DA4, CS1–CS4	Oyut zones
Biotite-granodiorite (BiGd) dykes	Project-wide, most important in Hugo deposits, unmineralized unit
Biotite-granodiorite (BiGd) dykes solid	Hugo North, unmineralized unit
Rhyolite (Rhy) dykes	Project-wide, most important in Oyut zones, unmineralized unit
Rhyolite (Rhy) dykes, solid	Hugo North, unmineralized unit
Hornblende–biotite granodiorite, solid	Hugo North, unmineralized unit
Hornblende–biotite andesites, dacites (And) dykes; HbBiAnd, Dac	Oyut zones

Table 14.3 Fault Surfaces used in Geological Modelling

Model Component	Comment
Surfaces – Faults	
East Bat Fault	Hugo area: used to define Hugo North eastern limit
West Bat Fault	Hugo area: used to define Hugo North, Central and West zones western limits
Contact Fault	Hugo North: defines post volcanic sequence, sub-parallel to lithological contacts
7100 Fault	Hugo North, north-west trending fault
Lower and Intermediate Faults	Hugo North, north trending faults sub-parallel to lithological contacts
Bogd Fault	Hugo North, east–west fault in Hugo North Extension
Khar Suult Fault	Hugo North, east–west fault in Southern area
Kharaa and Eroo Faults	Hugo North, north-east trend fault in Northern area
Bumbat and Dugant Faults	Hugo North, east–west fault in Hugo North Extension
Burged, Noyon, Gobi, Javkhlant Faults	Hugo North, north-west trending series of faults
160 Fault	Hugo North, north trending fault
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
Central Fault	Hugo South area: forms boundary between Hugo South deposit and Oyut Central zone
East Bounding Fault	Oyut area: forms eastern boundary to the Southwest zone and western boundary to the Wedge zone
West Bounding Fault	Oyut area: forms informal western boundary to the Southwest zone (generally marks contact between unmineralized Qmd and mineralized Va)
Rhyolite Fault	Oyut area: marks boundary between Southwest and Central zones, Hugo North
South Fault (includes South Splay 3)	Oyut area: marks boundary between Wedge and South zones
Solongo Fault	Oyut area: defines the southern edge of the South zone
AP and KJ fault series	Oyut area: Internal faults in the area of estimated Mineral Resources

Table 14.4 Grade Shell Construction Parameters

Deposit / Zone	Grade Shell Lower Cut-off		
	Au (g/t)	Cu (%)	Mo (ppm)
Oyut / Southwest Zone	0.3 0.7	0.3	–
Oyut / Central Zone	0.3 0.7	0.3	–
Oyut / South Zone	0.3 0.7	0.3	–
Oyut / Bridge Zone	0.3	0.3	–
Oyut / Wedge Zone	0.3	0.3	–
Hugo South Deposit	–	0.6 1.0 2.0	–
Hugo North Deposit	0.3 1.0	0.6 2.0 qtz veining 15% by vol.	–
Hugo North Extension	0.3 1.0	0.6 2.0 qtz veining 15% by vol.	–
Heruga Deposit	0.3 0.7	0.3	100

Domains were established using the codes outlined in Table 14.5, where the domain variable used in grade estimation was a four-digit integer code composed from the following fields in the composite database: deposit (DPOSIT), grade shell (GS_CU or GS_AU), lithology (FLAG), and supergene (SUPERG).

A third digit in the code was originally intended to accommodate a greater number of lithology codes, but currently remains unused.

Table 14.5 Domain Codes

DPOSIT (1st digit)		GS_CU or GS_AU (2nd digit)		FLAG (4th digit)	
Value	Code	Value	Code	Value	Code
Default / None	0	Outside grade shells	1	VA	1
Southwest Zone (SW)	1	Inside 0.3 (% or g/t) grade shell	2	Ign	2
Central Zone (CO)	2	Inside 0.7 (g/t) Au grade shell	3	Qmd	3
South Zone (SO)	3	–	–	HWS	4
Far South (FS)	4	–	–	BiGd	5
Bridge Zone (BZ)	5	–	–	And	6
Wedge Zone (WZ)	6	–	–	Rhy	7
West Zone (WO)	7	–	–	Clay	8
South of the Solongo Fault	8	–	–	–	–
Supergene	9	–	–	–	–

14.1.3 Grade Capping and Evaluation of Outlier/Extreme Grades

Extreme (outlier) copper and gold grades were evaluated using histograms, probability plots, and cumulative distribution function plots.

14.1.3.1 Oyut

No cap was applied to the copper for Oyut estimation. For most domains, an outlier restriction, rather than grade capping, was applied to gold, silver, molybdenum, and arsenic during grade estimation. A 50 m isotropic outlier sample search distance was used during estimation for the main lithological units: Va, Ign, Qmd, and HWS. A 20 m x 20 m x 15 m isotropic outlier sample search distance was used during estimation for dykes.

Top-cuts were applied to composites before applying the outlier search restriction to further limit the risk of outlier Au values in four domains, and consisted of 3 g/t Au in two domains and 8 g/t Au in the other two domains. The predicted metal removed for each element by capping, excluding blocks above the oxide surface, is as follows:

- Au = 3.5%
- Ag = 3.7%
- As = 0.4%
- Mo = 2.7%

Grade caps for the Oyut zones are summarized in Table 14.6.

Table 14.6 Outlier Restrictions / Grade Caps – Oyut

Domain	Outlier Search Restriction Threshold				Global Capping Threshold Au (g/t)
	As (g/t)	Mo (ppm)	Au (g/t)	Ag (g/t)	
1101	110	230	0.6	2.9	–
1103	115	85	0.6	2.5	–
1201	500	260	2.0	2.9	–
1203	500	260	1.0	2.9	–
1301	n/a	n/a	6.0	4.5	8.0
1303	n/a	n/a	8.0	4.5	–
2101	200	80	0.4	2.1	–
2102	900	260	0.7	2.1	–
2103	105	150	0.7	2.9	–
2104	100	12	0.15	1.0	–
2201	800	250	0.9	3.5	–
2202	1,100	240	0.9	3.5	–
2203	1,100	250	0.9	3.5	–
2204	–	–	n/a	n/a	–
2301	n/a	n/a	2.3	3.5	–
2302	n/a	n/a	0.7	3.5	–
2303	n/a	n/a	0.7	3.5	–
3101	400	70	0.55	3.6	3.0
3102	401	70	0.32	3.6	–
3103	500	85	0.55	4.6	3.0
3104	–	–	0.1	–	–
3201	1,050	85	3.0	8.5	–
3202	800	130	3.0	8.5	–
3203	1,050	85	3.0	8.5	–
3204	–	–	n/a	8.5	–
3301	n/a	n/a	3.0	8.5	–
3303	n/a	n/a	3.0	8.5	–
4101	110	230	0.6	2.9	–
4103	115	–	0.6	2.5	–
4201	500	350	2.0	2.9	–
4203	500	350	1.0	2.9	–
4301	n/a	n/a	6.0	4.5	8.0
4303	n/a	n/a	8.0	4.5	–
5101	110	82	0.6	2.9	–

Domain	Outlier Search Restriction Threshold				Global Capping Threshold Au (g/t)
	As (g/t)	Mo (ppm)	Au (g/t)	Ag (g/t)	
5102	110	82	–	–	–
5103	115	85	0.6	2.5	–
5104	–	–	–	–	–
5201	500	130	2.0	2.9	–
5202	–	–	n/a	n/a	–
5203	500	130	1.0	2.9	–
6101	900	130	0.28	1.6	–
6102	900	130	0.42	3.0	–
6103	900	130	0.42	3.0	–
6104	105	5.5	0.05	0.5	–
6201	500	160	1.5	4.0	–
6202	1,100	490	1.5	4.0	–
6203	1,100	160	1.5	4.0	–
6204	900	–	n/a	n/a	–
6301	n/a	n/a	1.5	4.0	–
6303	n/a	n/a	1.5	4.0	–
7101	300	150	0.45	2.9	–
7103	105	30	0.31	2.9	–
7201	105	160	1.3	4.0	–
7203	105	160	1.3	–	–
7301	n/a	n/a	1.3	4.0	–
7303	n/a	n/a	1.3	–	–
9100	–	150	0.4	3.8	–
9200	–	245	1.0	–	–
BiGd	–	–	0.35	–	–
And	–	–	0.35	–	–
Rhy	–	–	0.35	–	–

14.1.3.2 Hugo North and Hugo North Extension

A combination of outlier restriction and grade capping was applied during grade estimation for the Hugo North area (Hugo North and Hugo North Extension). In most cases, 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 over-representation of high-grade estimates owing to the disproportional numbers of high-grade to lower grade composites. In outlier-restricted kriging, outliers (i.e. values above

the specified cut-off) are restricted 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 outliers are used at their full value. The outlier thresholds applied at Hugo North and Hugo North Extension were defined at the 99th percentile of their respective population. The thresholds for restrictions are shown in Table 14.7 (copper) and Table 14.8 (gold).

Table 14.7 Grade Caps Applied to Cu, Au, and Ag Grade Domains – Hugo North

Grade Domain	Cu (%)	Au (g/t)	Ag (g/t)
101	1.0	1.2	2.5
102	–	0.4	8
103	1.5	2.0	–
104	–	n/a	–
105	–	2.0	10.5
201+202+203+204	5.5	2.5	17
205	n/a	no cap	n/a
301+303	9.5	3.5	–
302	3.5	no cap	n/a
304	n/a	–	n/a
305	n/a	6.0	2.5

Table 14.8 Outlier Restrictions (High Yield Restrictions) Applied to Cu, Au, and Ag Grade Domains – Hugo North

Grade Domain	Cu (%)	Au (g/t)	Ag (g/t)
102	2.5	–	–
103	–	–	10.5
104	–	–	1.5
105	3.0	–	–
301+303	–	–	21
101, 21+202+203+204, 205, 302, 304, 305	–	–	–

14.1.3.3 Hugo South

The grade caps on outlier grades employed at Hugo South are summarized in Table 14.9.

Table 14.9 Outlier Restrictions / Grade Caps – Hugo South

Grade Domain	Cu (%)	Au (g/t)	Mo (ppm)
2% Cu shell	11	2.0	600
1% Cu shell	5	2.0	1,100
0.6% Cu shell	3	2.0	1,100
Background	3	1.5	1,100
No. of assays capped	18	21	14

14.1.3.4 Heruga

As well as top-cutting of extreme grades, some outlier restriction was also applied for the Heruga deposit, particularly in the background domains. 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. The grade caps on outlier grades employed at Heruga are shown in Table 14.10

Table 14.10 Outlier Restrictions / Grade Caps – Heruga

Domain	Metal	Domain	Cap	Distance	Outlier Cap
Background	Au	1,000–4,000	3 g/t	50 m	1 g/t
Background	Au	5,000	3 g/t	50 m	0.3 g/t
Background	Mo	All	1,000 ppm	100 m	500 ppm
0.3 g/t Au shell	Au	2,000	3 g/t	–	–
0.3 g/t Au shell	Au	4,000	5 g/t	–	–
0.7 g/t Au shell	Au	2,000	10 g/t	–	–
100 ppm Mo shell	Mo	All	3,000 ppm	–	–

14.1.4 Composites

The drillhole assays were composited into downhole composites of a length that 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 for all deposits except in the Hugo North models. The domains used in compositing were derived from a combination of the grade shells and lithological domains. Composite lengths of 8 m (approximately half the selective mining unit (SMU) size of 15 m) were used for Oyut, and 5 m lengths were used for all other deposits.

Intervals of less than the fixed length (8 m or 5 m) represented individual residual composites from end-of-hole or end-of-domain intervals. Composites that had a length of less than 1.5 m (Hugo North) or 2 m (Heruga and Oyut) were excluded from the dataset used in interpolation.

For Oyut, the following default values were applied to any unsampled intervals during compositing:

- Cu 0.005%
- Au 0.005 g/t
- Ag 0.5 g/t
- As 25 ppm
- Mo 2.5 ppm

At Hugo North, the composites included any post-mineralization dyke intervals that were deemed too small to be part of a dyke geology model. Any unsampled intervals included in the composites dataset for Hugo North were set to:

- Cu 0.001%
- Au 0.01 g/t

For the Heruga deposit, the composites included any post-mineralization dyke material intervals that were deemed too small to be part of a dyke geology model. Any unsampled intervals included in the composites dataset for Heruga were set to:

- Cu 0.001%
- Au 0.01 g/t
- Mo 10 ppm

14.1.5 Exploratory Data Analysis

The lithological, structural, and mineralized domains for Hugo North and Oyut were reviewed 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.

The data analyses were conducted on composited assay data, typically using either 8 m or 5 m downhole composites. Descriptive statistics, histograms and cumulative probability plots, box plots, contact plots, and scatter plots were completed for copper and gold in each deposit area.

Results obtained were used to guide the construction of the block model and the development of estimation plans.

14.1.5.1 Oyut

Copper grades inside the grade shells display generally similar means and coefficients of variation (CVs) between zones, and low CVs of between 0.6 and 0.75. The Va, Ign, and Qmd units were able to be combined inside the grade shells due to similar mean values seen in univariate statistics. Outside the copper grade shells, and in Va or in the combined Va+Qmd units in the South zone, CVs are similar to those inside the grade shells. Other copper domains outside the grade shells typically display lower mean grades and higher CVs when compared to Va outside the grade shells or Va+Qmd in the South zone.

Gold grades inside the 0.3 g/t Au grade shells of the significant gold-bearing domains (Southwest, Far South, Bridge, and Central zones) display generally similar means and low CVs (<1). Samples inside the 0.7 g/t Au shell are dominantly in the Va unit of the Southwest zone, are similar in mean and CV to the Qmd inside the 0.7 g/t Au shell of Southwest, and show low CVs (approximately 0.7). The few samples inside the 0.7 g/t Au shell of the Central zone show lower mean grades and higher CVs than Southwest inside the 0.7 g/t Au shell.

Outside the gold grade shells in the Va unit, CVs are similar to inside the grade shells, with the exception of the combined Va+Qmd unit in the South zone, which has a very high CV due to some extreme outlier grades. Other gold domains outside the shells typically display lower mean grades and higher CVs when compared to the Va units outside shells.

The correlation coefficients between Cu–As, Cu–Mo, and Au–Ag, although varying by domain, were generally weak, ranging from 0.2–0.4, 0.2–0.4, and 0.20–0.25, respectively.

14.1.5.2 Hugo North and Hugo North Extension

Copper grades in the mineralized units (Va, Ign, and Qmd) show single lognormal to near-normal distributions inside each domain (0.6% and 2% Cu Shells). Coefficients of variation values are low at 0.3 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 Ign.

The cumulative distribution function patterns of copper data for all domains show evidence of three populations:

- a higher grade population (above a copper threshold value of 2.0%–2.5% Cu),
- a lower grade zone (threshold value of 0.4% Cu 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 Ign. Coefficients of variation values for the host lithologies are moderate, varying from 0.6 to 0.9.

The cumulative distribution function 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 0.2–0.3 g/t Au), and
- a background lowest grade domain.

The pattern supports the construction of the 1 g/t Au and 0.3 g/t Au grade shells.

At Hugo North, the gold : copper relationships that were identified in 2005 are poorer. Generally, two trends may be present. The more common is a low-gold trend that outlines a Au : Cu ratio of about 1 : 10 in the mineralized volcanic units. The Qmd unit also displays the 1 : 10 Au : Cu ratio trend but also shows a more gold enriched Au : Cu ratio at about 1 : 2.

14.1.5.3 Hugo South

Copper grade behaved as expected between grade shells. No significant 'within shell' variations due to lithology were observed. Gold grade distributions showed typical lognormal trends in all domains. Molybdenum grades were generally low, but Hugo South was observed to have higher molybdenum grades than Hugo North.

14.1.5.4 Heruga

Copper grades within the 0.3% Cu shell generally displayed single distributions with some evidence for a lower grade population resulting from the presence of unmineralized post-mineralization dykes that had not been captured by wireframes. CVs were relatively low at 0.5 to 0.6. The cumulative distribution function plot for the entire population supported the construction of a grade shell in the 0.3%–0.4% Cu range.

Gold grades were observed to display a moderate positive skew and multiple populations with evidence of lower grade populations in the range of 0.2–0.3 g/t Au.

Molybdenum grades within the 100 ppm Mo shell display a low-to-moderate positive skew and a single population distribution.

14.1.6 Estimation Domains

A strategy of soft, firm, and hard (SFH) boundaries was implemented to account for domain boundary uncertainty (dilution) and to reproduce the input grade sample distribution in the block model. Soft boundaries allowed full sharing of composites between domains during grade estimation; firm boundaries allowed sharing of composites from within a certain distance of the boundary; and hard boundaries allowed no composite sharing between domains.

Contact plots and visual inspection of grade distributions were also used in cases where results were unclear or were contrary to geological interpretations.

14.1.6.1 Oyut

The following major faults were considered as hard boundaries in all cases: Central, Solongo, East Boundary, West Boundary, Rhyolite, and South (near the mineralized zones). The exceptions are the boundaries between the Southwest and Bridge zones, and the Southwest and Far South zones, which were determined as soft, firm, or hard on a case-by-case basis from statistical relationships.

Boundaries between elements, lithologies, and grade shells were based on compilation of detailed SFH matrices. Typically, the following boundaries were broadly applied; however, actual boundaries as indicated in the SFH matrices are complex:

- Silver uses the same SFH relationships as gold.
- Arsenic and molybdenum use the same SFH relationships as copper.
- For copper, the contact between the Qmd and Va units is typically a soft boundary, but can be firm or hard. The Ign boundary with the Qmd and Va units is more often than not a soft boundary, but can on rare occasions be hard.
- Gold relationships across changing lithologies are more complex, and no general observations can be drawn.
- The Central, West, Wedge, and South zone boundaries are each hard to all other zones. The boundaries between the Southwest, Bridge, and Far South zones are soft in most cases.
- Grade shell boundaries are typically firm; most commonly sharing samples within 40 m of the boundary, but occasionally a more restricted distance is used, where a boundary can share samples within 20 m of the boundary.

14.1.6.2 Hugo South

Grades for blocks within the three copper grade shells were estimated with a hard boundary between the shells; only composites within the shell were used to estimate blocks within the shell.

14.1.6.3 Hugo North and Hugo North Extension

Different boundary designations of soft, firm, or hard can be used for the different lithologies, depending on the grade shell. The intra-domain contact boundaries are summarized in the matrix in Table 14.11 for copper and in Table 14.12 for gold. The various copper and gold grade shells used to constrain the selection of composites and blocks during the interpolation of block grades at Hugo North and Hugo North Extension are illustrated in Figure 14.1.

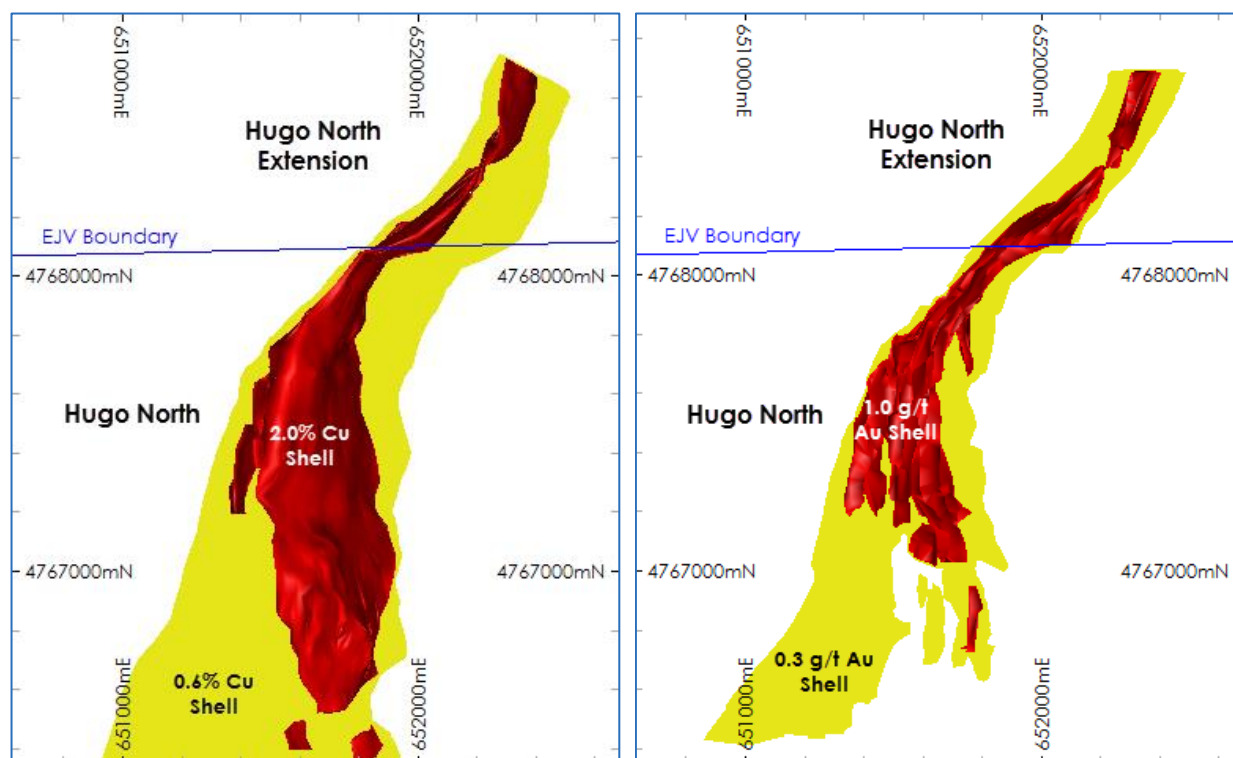
Table 14.11 Hugo North Intra-Domain Boundary Contacts – Copper

Shell	Va	Qmd	Ign	xBiGd
Background				
Va	Soft	Hard	Firm	Firm
Qmd	Hard	Soft	Firm	Firm
Ign	Firm	Firm	Soft	Firm
HWS	Firm	Firm	Firm	Soft
0.6% Cu Shell				
Va	Soft	Firm	Firm	Firm
Qmd	Firm	Soft	Firm	Firm
Ign	Firm	Firm	Soft	Firm
HWS	Firm	Firm	Firm	Soft
xBiGd	Hard	Hard	Hard	Hard
Qtz Vein (2% Cu) Shell				
Va	Firm	Firm	Hard	Hard
Qmd	Firm	Soft	Hard	Hard
Ign	Firm	Hard	Soft	Hard
HWS	Hard	Hard	Hard	Soft
xBiGd	Hard	Hard	Hard	Hard

Table 14.12 Hugo North Intra-Domain Boundary Contacts – Gold

Shell	Va	Qmd	Ign	xBiGd
Background				
Va	Soft	Firm	Soft	Firm
Qmd	Hard	Soft	Firm	Firm
Ign	Soft	Firm	Soft	Soft
xBiGd	Hard	Firm	Soft	Soft
HWS	–	–	–	–
0.3 g/t Au Shell				
Va	Soft	Firm	Firm	Hard
Qmd	Firm	Soft	Hard	Firm
Ign	Soft	Firm	Soft	Hard
xBiGd	Hard	Hard	Hard	Hard
HWS	Firm	Firm	Soft	Soft
1 g/t Au Shell				
Va	Soft	Firm	Hard	Hard
Qmd	Firm	Soft	Hard	Hard
Ign	Hard	Soft	Soft	Hard
xBiGd	Hard	Hard	Hard	–
HWS	–	–	–	–

Figure 14.1 Copper Grade Shells and Gold Grade Shells – Hugo North, and Hugo North Extension



14.1.6.4 Heruga

Data analysis showed no discernible difference between the two main host lithologies, augite basalt and quartz monzodiorite at Heruga. Therefore, for estimation purposes the two lithologies were able to be grouped into a single lithology domain. The post-mineralization lithologies (Lqmd, BiGd, HbBiAnd) were assigned zero grade. Within each structural domain, the cells in the model were therefore coded according to whether or not they were mineralized or unmineralized, and which grade shell they fell within.

14.1.7 Variography

14.1.7.1 Oyut

The quality of the variogram model fits for copper correlograms were excellent for the Southwest zone inside copper grade shells and good for the Central zone inside the copper grade shells. The quality of copper model fits in other domains ranged from moderate to good. Model fits for gold correlograms were less robust than for the copper correlograms. Gold grade shells are typically smaller than the copper grade shells, and are also divided into 0.3 g/t Au and 0.7 g/t Au datasets, resulting in less data being available to support correlogram construction for each grade shell domain.

The correlogram-model fits for copper and gold are considered to be of acceptable quality for global and local grade estimation at the Indicated level of confidence, or at the measured level of confidence for well-drilled domains.

Model fits for silver, arsenic, and molybdenum were moderate-to-good for the larger domains inside grade shells. For other domains, fits were typically interpretable.

Table 14.13 summarizes the results of the copper correlograms by domain, and Table 14.14 shows the results for gold by domain.

Table 14.13 Copper Correlogram Model Parameters by Domain

Domain	Structure	Gamma	Bearing (RotZ)	Plunge (RotY)	Dip (RotX)	Range X' (m)	Range Y' (m)	Range Z' (m)
1101	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.55	–13	5	–21	77.8	18.9	165
	Exp2	0.15	–13	5	–21	101.6	915	473
1103	Nugget	0.25	–	–	–	–	–	–
	Exp1	0.57	–29	12	7	154.5	102.3	321
	Exp2	0.18	–29	12	7	27.1	14.1	25
1201, 1203	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.36	230	–70	0	31.1	23.2	70.3
	Exp2	0.49	230	–70	0	391.9	347.3	256
2101 – 2103	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.5	–5	2	12	23.9	34.8	13.6
	Exp2	0.4	–5	2	12	79.7	141	259
2104	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.752	90	–45	0	35.7	80	58.1
	Exp2	0.098	90	–45	0	588.6	400	400
2201 – 2204	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.55	31	11	12	49.9	51.4	105
	Exp2	0.25	31	11	12	913	346.2	134
3101, 3103	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.6	60	–75	0	31.7	16.4	46.5
	Exp2	0.2	60	–75	0	197.9	176.1	86.6
3102	Nugget	0.15	–	–	–	–	–	–
	Sph1	0.5	–46	13	–10	125	90	40
	Sph2	0.35	16	28	–32	160	120	200
3104	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.752	90	–45	0	35.7	80	58.1
	Exp2	0.098	90	–45	0	588.6	400	400
3201 – 3204	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.652	320	0	–36	25.7	153.4	56.7
	Exp2	0.248	320	0	–36	1,145.1	109	343

Domain	Structure	Gamma	Bearing (RotZ)	Plunge (RotY)	Dip (RotX)	Range X' (m)	Range Y' (m)	Range Z' (m)
4101, 4103	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.55	–8	4	4	118.2	16.2	42
	Exp2	0.15	–8	4	4	91.3	397.3	1805
4201, 4203	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.65	50	–5	58	50	60	20
	Exp2	0.2	50	–5	58	75	25	30
5101 – 5103	Nugget	0.05	–	–	–	–	–	–
	Exp1	0.6	0	0	0	30	30	30
	Exp2	0.35	0	0	0	180	180	180
5104	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.752	90	–45	0	35.7	80	58.1
	Exp2	0.098	90	–45	0	588.6	400	400
5201 – 5203	Nugget	0.13	–	–	–	–	–	–
	Exp1	0.35	1	3	15	100	40	200
	Exp2	0.52	1	3	15	300	70	40
6101 – 6103	Nugget	0.35	–	–	–	–	–	–
	Exp1	0.5	38	–9	–19	220	116.2	53.8
	Exp2	0.15	38	–9	–19	700	279.3	1,000
6104	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.752	90	–45	0	35.7	80	58.1
	Exp2	0.098	90	–45	0	588.6	400	400
6201–6204	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.6	40	–11	–61	256.4	58	53.8
	Exp2	0.2	40	–11	–61	188.7	1,089.3	126
7101	Nugget	0.34	–	–	–	–	–	–
	Exp1	0.54	0	0	0	190	190	190
	Exp2	0.12	0	0	0	1,000	1,000	1,000
7103	Nugget	0.329	–	–	–	–	–	–
	Exp1	0.3	33	–6	3	48.9	9.3	296
	Exp2	0.371	33	–6	3	121.3	96.3	226
7201, 7203	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.58	0	0	0	23	23	23
	Exp2	0.12	0	0	0	22	22	22
9101 – 9203	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.9	0	0	0	100	100	100

Table 14.14 Gold Correlogram Model Parameters by Domain

Domain	Structure	Gamma	Bearing (RotZ)	Plunge (RotY)	Dip (RotX)	Range X (m)	Range Y (m)	Range Z (m)
1101	Nugget	0.4	–	–	–	–	–	–
	Exp1	0.46	24	–27	–96	86.6	32	105
	Exp2	0.14	–46	16	–9	107.6	258.1	1,896
1103	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.34	12	–65	–93	146.7	9.5	123
	Exp2	0.36	–3	–8	8	69.8	139.2	691
1201, 1203	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.6	43	–28	–66	28.7	27.1	10.1
	Exp2	0.2	43	–28	–66	230	285.8	93.7
1301	Nugget	0.21	–	–	–	–	–	–
	Exp1	0.49	140	0	–55	97.4	76.3	120
	Exp2	0.3	140	0	–55	533.3	414.3	182
1303	Nugget	0.33	–	–	–	–	–	–
	Exp1	0.42	37	88	–66	31	130.1	85
	Exp2	0.25	37	88	–66	62	407.9	61.2
2101	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.6	8	–37	–132	552.5	180.9	272
	Exp2	0.2	–103	13	14	98	8.3	469
2102–2103	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.55	–10	23	13	22.6	67.1	29.7
	Exp2	0.35	–10	23	13	129.8	285	186
2104, 3104, 5104, 6104	Nugget	0.05	–	–	–	–	–	–
	Exp1	0.757	4	22	–19	28.6	29.9	28.3
	Exp2	0.193	4	22	–19	223.3	220.6	221
2201–2303	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.45	56	13	53	97.3	20	32.4
	Exp2	0.4	56	13	53	122.4	299.4	122
3101	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.625	–46	–1	0	16.1	26.1	18.6
	Exp2	0.275	–46	–1	0	316.2	312.5	319
3102	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.585	–13	23	41	30.3	30.2	30.9
	Exp2	0.315	–13	23	41	294.8	289.3	291
3103	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.722	–22	5	–8	22.2	22.7	23.1
	Exp2	0.178	–22	5	–8	138.2	137.7	138
3201–3303	Nugget	0.207	–	–	–	–	–	–
	Exp1	0.376	104	3	14	211.6	41.7	5.1
	Exp2	0.417	3	53	–69	34	248.1	14.2

Domain	Structure	Gamma	Bearing (RotZ)	Plunge (RotY)	Dip (RotX)	Range X (m)	Range Y (m)	Range Z (m)
4101, 4103	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.65	0	0	0	21	21	21
	Exp2	0.25	0	0	0	435	435	435
4201–4303	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.55	0	–15	0	130	35	62
	Exp2	0.15	–33	–10	15	359.1	487.1	2,126
5101	Nugget	0.35	–	–	–	–	–	–
	Exp1	0.45	–18	–34	–154	45.4	8.9	143
	Exp2	0.2	–31	34	–32	159.2	496.5	2,300
5102–5103	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.6	10	40	3	60	100	30
	Exp2	0.3	10	40	3	100	700	130
5201–5203	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.6	43	–28	–66	28.7	27.1	10.1
	Exp2	0.2	43	–28	–66	230	285.8	93.7
6101–6103	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.6	44	–30	–23	108.7	8.2	47.6
	Exp2	0.25	48	4	–64	90.2	278.5	480
6201–6303	Nugget	0.1	–	–	–	–	–	–
	Exp1	0.4	0	0	0	135	135	135
	Exp2	0.5	0	0	0	135	135	135
7101	Nugget	0.15	–	–	–	–	–	–
	Exp1	0.35	49	–81	118	161.1	44.6	27.2
	Exp2	0.35	–4	2	2	64.9	410.5	19
7103	Nugget	0.3	–	–	–	–	–	–
	Exp1	0.54	55	–45	19	175.2	55.7	8.7
	Exp2	0.15	18	115	40	26.9	41.5	408
7201–7303	Nugget	0.4	–	–	–	–	–	–
	Exp1	0.45	0	0	0	15	15	15
	Exp2	0.15	0	0	0	140	140	140
9101–9203	Nugget	0.2	–	–	–	–	–	–
	Exp1	0.16	0	0	0	30	30	30
	Exp2	0.64	0	0	0	130	130	130

14.1.7.2 Hugo North and Hugo North Extension

Data in some shells were subdivided into north and south sectors for the variographic analysis to take into account the flexure in direction of the deposit that occurs near the 4,767,600 mN coordinate.

The mineralization controls observed were considered to be related to the intrusive history and structural geology (faults). The patterns of anisotropy demonstrated by the various correlograms tended to be consistent with geological interpretations, particularly to any bounding structural features (faults and lithological contacts) and quartz + sulphide vein orientation data.

The nugget variance tended to be low-to-moderate in all of the domains assessed. Copper variograms generally had nugget variances of between 15%–20% (relative) of the total variance, except in BiGd, where the nugget is 38% of total variance. The nugget variance for gold variograms varied from 5%–25%.

Both copper and gold displayed short ranges for the first variogram structure and moderate-to-long ranges for the second variogram structure (where modelled).

The model parameters for copper outside the 0.6% Cu grade shell are summarized in Table 14.15; for copper inside the 0.6% Cu grade shell and outside the 1% Cu grade shell in Table 14.16; for copper within the 1% Cu grade shell in Table 14.17; and for copper within the BiGd in Table 14.18.

The model correlogram parameters for gold outside the 0.3 g/t Au grade shell are summarized in Table 14.19; for gold inside the 0.3 g/t Au grade shell and outside the 1 g/t Au grade shell in Table 14.20; for gold within the 1 g/t Au grade shell in Table 14.21; and for gold within the BiGd in Table 14.22.

14.1.7.3 Hugo South

Correlograms indicated a north-easterly trend at Hugo South. The deposit displayed a consistent steep easterly dip with a flat plunge. Ranges were longest along strike of the respective trend for copper and a mixture of along-trend and down-dip of the trend for gold. Ranges tended to be less than 75 m for the first variogram structure in all metals and less than 200 m for the second variogram structure.

14.1.7.4 Heruga

Although data are limited, an attempt was made to model directional variograms for gold, copper, and molybdenum. Copper and gold showed relatively low nuggets of 25%–35% (relative) of the total variance, whereas molybdenum was moderate-to-high at 40% of the sill. All three metals showed relatively short first variogram structures and long second variogram structures of 250–300 m.

Table 14.15 Copper Correlogram Parameters, Outside 0.6% Cu Grade Shell, Hugo North

Estimation ID	Au Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
1011_1	Outside 0.6% Cu Grade Shell	1	Va	1	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1012_1				2	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1013_1				3	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1014_1				4	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1015_1				5	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1016_1				6	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1017_1				7	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1021_1		2	Ign	1	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1022_1				2	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1023_1				3	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1024_1				4	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1025_1				5	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1026_1				6	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1027_1				7	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1031_1		3	Qmd	1	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1032_1				2	0.1	2	Spherical	0.4	45	-15	-90	118	123	87	Spherical	0.5	45	-15	-90	675	382	99
1033_1				3	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1034_1				4	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1035_1				5	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1036_1				6	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110
1037_1				7	0.17	2	Spherical	0.39	270	60	0	88	16	69	Spherical	0.44	270	60	0	507	539	110

Table 14.16 Copper Correlogram Parameters, Inside 0.6% Cu Grade Shell and Outside 1% Cu Grade Shell, Hugo North

Estimation ID	Cu Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
2011_1	Inside 0.6% Cu Grade Shell and Outside 1% Cu Grade Shell	1	Va	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2012_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2013_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2014_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2015_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2016_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2017_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2021_1		2	Ign	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2022_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2023_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2024_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2025_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2026_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2027_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2031_1		3	Qmd	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2032_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2033_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2034_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2035_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2036_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2037_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2041_1		4	HWS	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2042_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
2043_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2044_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2045_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2046_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
2047_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104

Table 14.17 Copper Correlogram Parameters, Inside 1% Cu Grade Shell, Hugo North

Estimation ID	Cu Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
3011_1	Inside 1% Cu Grade Shell	1	Va	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3012_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3013_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3014_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3015_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3016_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3017_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3021_1		2	Ign	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3022_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3023_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3024_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3025_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3026_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3027_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3031_1		3	Qmd	1	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3032_1				2	0.06	2	Spherical	0.4	5.854	56.774	-61.813	47	104	20	Spherical	0.54	5.854	56.774	-61.813	224	356	40
3033_1				3	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3034_1				4	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3035_1				5	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3036_1				6	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104
3037_1				7	0.11	2	Spherical	0.59	181.102	25.659	56.31	175	140	103	Spherical	0.3	181.102	25.659	56.31	654	192	104

Table 14.18 Copper Correlogram Parameters, Inside BiGd, Hugo North

Estimation ID	Domain	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
1051_1	BiGd High Grade Domain	5	BiGd	1	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1052_1				2	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1053_1				3	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1054_1				4	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1055_1				5	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1056_1				6	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1057_1				7	0.08	2	Spherical	0.1	345	0	−65	94	90	164	Spherical	0.82	345	0	−65	319	214	205
1051_I2	BiGd Outside of Grade Shell			1	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1052_I2				2	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1053_I2				3	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1054_I2				4	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1055_I2				5	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1056_I2				6	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58
1057_I2				7	0.38	2	Spherical	0.19	356.384	−19.683	79.372	42	25	12	Spherical	0.43	356.384	−19.683	79.372	133	129	58

Table 14.19 Gold Correlogram Parameters, Outside 0.3 g/t Au Grade Shell, Hugo North

Estimation ID	Au Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
1011_1	Outside of 0.3 g/t Au Grade Shell	1	Va	1	0.23	2	Spherical	0.32	45	−75	−90	179	174	98	Spherical	0.45	45	−75	−90	619	442	139
1012_1				2	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300
1013_1				3	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300
1021_1		2	Ign	1	0.23	2	Spherical	0.32	45	−75	−90	179	174	98	Spherical	0.45	45	−75	−90	619	442	139
1022_1				2	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300
1023_1				3	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300
1031_1		3	Qmd	1	0.23	2	Spherical	0.32	45	−75	−90	179	174	98	Spherical	0.45	45	−75	−90	619	442	139
1032_1				2	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300
1033_1				3	0.23	2	Spherical	0.36	355.729	−10.545	−44.007	150	125	200	Spherical	0.41	355.729	−10.545	−44.007	762	300	300

Table 14.20 Gold Correlogram Parameters, Inside 0.3 g/t Au Grade Shell and Outside 1 g/t Grade Shell, Hugo North

Estimation ID	Au Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
2011_1	Inside 0.3 g/t Au Grade Shell and Outside 1 g/t Au Grade Shell	1	Va	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
2012_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
2013_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0
2021_1		2	Ign	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
2022_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
2023_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0
2031_1		3	Qmd	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
2032_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
2033_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0
2041_1		4	HWS	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
2042_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
2043_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0

Table 14.21 Gold Correlogram Parameters, Inside 1 g/t Au Grade Shell, Hugo North

Estimation ID	Au Grade Shell	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
3011_1	Inside 1 g/t Au Grade Shell	1	Va	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
3012_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
3013_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0
3021_1		2	Ign	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
3022_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
3023_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0
3031_1		3	Qmd	1	0.18	2	Spherical	0.19	45	0	120	61	63	26	Spherical	0.63	45	0	120	170	104	52
3032_1				2	0.22	2	Spherical	0.35	345	0	90	94	32	57	Spherical	0.43	345	0	90	599	373	176
3033_1				3	0.15	1	Spherical	0.85	281.31	-25.659	-16.102	36	89	106	Spherical	0	0	0	0	0	0	0

Table 14.22 Gold Correlogram Parameters, Inside BiGd, Hugo North

Estimation ID	Domain	Nlith	Rock Type	Zone Code	Variance Nugget	Variogram Structure Count	Structure 1								Structure 2							
							Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	Model Type	Sill	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis
1051h1	BiGd High-grade Domain	5	BiGd	1	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241
1052h1				2	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241
1053h1				3	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241
1051_1	BiGd Outside of Grade Shell			1	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241
1052_1				2	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241
1053_1				3	0.05	2	Spherical	0.78	345	−45	0	44	69	237	Spherical	0.17	345	−45	0	298	235	241

Note: 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.

14.1.8 Model Setup

14.1.8.1 General

The block size selected for the Oyut block model was based on mining selectivity considerations for open pit mining. It was assumed the smallest block size that could be selectively mined as mill feed or waste, referred to the SMU, was approximately 20 m east x 20 m north x 15 m high. A sub-cell model was used for resource estimation, where the 'parent' (maximum) block dimensions were equal to those of the ultimate resource block model (20 m x 20 m x 15 m) and the minimum sub-cell dimensions down to 5 m x 5 m x 5 m. The actual sub-cell sizes vary as necessary to fit the specified boundaries of the wireframes used to tag the block model. Grade variables were regularized to the tonnage-weighted (volume x density) mean of the like-domained sub-cell source grade values enclosed in the parent blocks.

For Hugo North, mining selectivity was less of an issue because the mining method envisioned, block cave mining, and does not allow for consideration of selectivity. A sub-celled model was used for resource estimation that has parent block dimensions equal to 15 m x 15 m x 15 m and minimum sub-block dimensions down to 5 m x 5 m x 5 m. Like the Oyut model, the actual sub-block sizes in the Hugo North model vary as necessary to fit the specified boundaries of the wireframes used to tag the block model.

Bulk density data were assigned to a unique assay database file. These data were composited into 8 m and 5 m fixed-length downhole values for the Oyut and Heruga models, respectively.

Various domain coding was done on the block models in preparation for grade interpolation. The block models were coded according to zone, lithological domain, and grade shell. Post-mineralization dykes were considered as potentially selectively mineable. For Hugo North, sub-celling was used to honour lithology, grade, and structural contacts.

Blocks above topography were removed from the block model. Non-mineralized units were flagged using a lithology code and were excluded during the interpolation process.

Blocks in the Hugo North model were assigned an estimation domain using a combination of grade shells or alteration and lithology.

With the exception of Central zone, only the hypogene mineralization was estimated. The base of the interpreted sulphide oxidation surface defined the top of the hypogene mineralization in the Oyut zones.

14.1.8.2 Oyut

Resource modelling at Oyut consisted of grade interpolation by ordinary kriging (OK). Nearest neighbour (NN) grades were also interpolated for validation purposes.

Search ellipsoid orientations were based upon the geometry of macro-scale grade trends, typically reflected in the grade shell boundaries. A three-pass kriging strategy was used to estimate the block grades. The first estimation pass kriging neighbourhood corresponds approximately to blocks expected to satisfy Measured and Indicated classification criteria. The kriging neighbourhood was expanded and relaxed with each successive pass while maintaining the same axial ratios for samples searches as in the first pass.

Estimations were run one domain at a time, with three passes per domain. The following exceptions represent groups of domains that were permitted to be estimated together:

- All supergene outside grade shells (three passes).
- All supergene inside grade shells (three passes).
- All BiGd dykes (one pass).
- All Andesite dykes (one pass).
- All Rhyolite dykes (one pass).

The selection of sample search ellipsoids was largely based on macro-scale grade controls seen in the average grade shell orientation within a given domain. In some domains, such as Southwest, search ellipsoid selection was influenced by grade trends that were identifiable within a grade shell by using robust anisotropic variogram models and/or visual identification of grade trends seen in drillholes.

The ranges and the rotation angles for the various search ellipsoids used in estimation at Oyut are shown in Table 14.23 (copper, arsenic, and molybdenum) and Table 14.24 (gold and silver).

A block discretization of 4 x 4 x 2 was used when estimating block grades. Pass 2 was executed on blocks that did not receive an interpolated grade in Pass 1, and Pass 3 was executed on blocks that did not receive an interpolated grade in Pass 1 or Pass 2. For all elements, a minimum of six composites and maximum of nine composites, as well as a maximum of three composites per drillhole, were required for the first and second estimation passes. For the third pass, a minimum of two composites and a maximum of eight composites were required, as well as a maximum of five composites per drillhole. A single estimation pass was used to estimate dyke blocks, requiring a minimum of three composites, a maximum of eight composites, and a maximum of five composites per drillhole.

During grade estimation of the sub-celled model, the estimated grade of the parent cells was assigned to each sub-cell. Grade composites flagged as less than zero grade were excluded from sample selection. Grade composites less than 2 m in length were also excluded. The composites were length-weighted during estimation. Composites were weighted by ordinary kriging according to variogram parameters, with the exception of dyke grades, which were estimated using inverse distance weighting to the second power (ID2) interpolation.

The bulk density estimation domains were based on three attributes: deposit, oxidation plane (above or below the lowest modelled limit of oxidation), and lithology. ID2 was used to estimate the bulk density values in the sub-blocked block model. The bulk density estimation domain boundaries were treated as hard boundaries; bulk density composites from one domain could not be used to inform blocks of another domain. Only bulk density composites from specific domains were used to inform target blocks of the same domain. The estimation search neighbourhood used an isotropic search of 100 m as well as conditions on sample counts: a minimum of two and a maximum of 12 composites within the search radius, and a maximum of two samples from any individual drillhole to inform the block estimate. A total of 12 bulk density estimation domains lacked samples in the composite database. Default bulk density values from nearby, lithologically similar domains were assigned to each of the domains lacking composites.

Table 14.23 Search Parameters for Cu, As, and Mo Estimations – Oyut

Oyut Domain	Bearing (RotZ)	Plunge (RotY')	Dip (RotX'')	Pass	X-Axis (m)	Y-Axis (m)	Z-Axis (m)	Long Axis Azimuth	Long Axis Dip	Intermed. Axis Azimuth	Intermed. Axis Dip
Southwest Zone, South Zone, Bridge Zone, Far South, Wedge Zone, and West Zone	50	0	-70	1	75	60	40	050°	0°	140°	-70°
				2	105	90	60	050°	0°	140°	-70°
				3	400	300	200	050°	0°	140°	-70°
Central Zone	0	0	0	1	60	60	75	000°	-90°	n/a	n/a
				2	90	90	115	000°	-90°	n/a	n/a
				3	300	300	375	000°	-90°	n/a	n/a
Supergene	0	0	0	1	60	60	30	090°	0°	000°	0°
				2	90	90	45	090°	0°	000°	0°
				3	240	240	120	090°	0°	000°	0°
Dykes	0	0	0	1	300	300	300	n/a	n/a	n/a	n/a

Table 14.24 Search Parameters for Au and Ag Estimations – Oyut

Oyut Zone	Bearing (RotZ)	Plunge (RotY')	Dip (RotX'')	Pass	X-Axis (m)	Y-Axis (m)	Z-Axis (m)	Long Axis Azimuth	Long Axis Dip	Intermed. Axis Azimuth	Intermed. Axis Dip
Southwest Zone, Bridge Zone, Far South, Wedge Zone, and West Zone	50	0	-70	1	75	60	40	050°	0°	140°	-70°
				2	125	90	60	050°	0°	140°	-70°
				3	375	300	200	050°	0°	140°	-70°
Central Zone	50	0	-90	1	75	60	40	050°	0°	000°	-90°
				2	125	90	60	050°	0°	000°	-90°
				3	375	300	200	050°	0°	000°	-90°
South Zone	130	0	0	1	60	40	60	130°	0°	000°	-90°
				2	90	60	90	130°	0°	000°	-90°
				3	240	160	240	130°	0°	000°	-90°
Supergene	0	0	0	1	60	60	30	090°	0°	000°	0°
				2	90	90	45	090°	0°	000°	0°
				3	240	240	120	090°	0°	000°	0°
Dykes	0	0	0	1	300	300	300	n/a	n/a	n/a	n/a

14.1.8.3 Hugo North and Hugo North Extension

Interpolation was limited to the mineralized lithological units (Va, Ign, Qmd, and xBiGd). Only composites belonging to those units were used. Grades and metal values within blocks belonging to all other units (post-mineralization dykes and sediments) were set to zero.

Modelling consisted of grade interpolation by OK, except for bulk density, which was interpolated using a combination of simple kriging and inverse distance weighting to the third power (ID3). Restricted and unrestricted grades were interpolated to allow calculation of the metal removed by outlier restriction. Grades were also interpolated using NN methods for validation purposes. Blocks and composites were matched on estimation domain.

The search ellipsoids were oriented preferentially to the general orientation of each estimation domain. The search strategy employed concentric expanding search ellipsoids. The first pass used a relatively short search ellipse relative to the long axis of the correlogram ellipsoid. For the second pass, the search ellipse was increased by 50% (up to the full range of the correlogram) to allow interpolation of grade into those blocks not estimated by the first pass. A final, third, pass was performed using a larger search ellipsoid.

To ensure that at least three drillholes were used estimate blocks in Pass 1, the number of composites from a single drillhole that could be used was restricted to three. Similarly, Pass 2 required a minimum of two drillholes to generate an estimate. The number of composites allowed from a single hole was restricted to three.

The search parameters for copper outside the 0.6% Cu grade shell are shown in Table 14.25; for copper within the 0.6% Cu grade shell and outside the 1% Cu grade shell in Table 14.26; for copper within the 1% Cu grade shell in Table 14.27; and for copper within BiGd in Table 14.28.

The search parameters for gold outside the 0.3 g/t Au grade shell are shown in Table 14.29; for gold inside the 0.3 g/t Au grade shell and outside the 0.1 g/t Au grade shell in Table 14.30; for gold inside the 0.1% Au grade shell in Table 14.31; and for gold inside BiGd in Table 14.32.

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 numbers of composites were adjusted to incorporate an appropriate amount of grade smoothing.

Estimation of sub-cells at the boundary of grade or lithology domains was based on assigning the parent cell grade to the sub-cells; the end result being that all like-flagged sub-cells within the larger parent cell contain the same grade.

For both copper and gold, a combination of outlier restriction and grade capping was used to control the effects of high-grade samples within the domains. This is discussed in Section 14.1.3.

Grade variables were regularized to the tonnage-weighted (volume x density) mean of the sub-cell source grade values enclosed in the parent blocks before they were provided for use in detailed engineering and tabulation of Mineral Resources.

Table 14.25 Copper Search Parameters, Outside 0.6% Cu Grade Shell, Hugo North

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
1011_1	Pass 1	Outside 0.6% Cu Grade Shell	1	Va	1	20	-20	-80	120	80	40	0	50	50	50	9	15	1	3
1012_1					2	45	-20	-90	120	80	40	0	50	50	50	9	15	1	3
1013_1					3	0	-15	-85	120	80	40	0	50	50	50	9	15	1	3
1014_1					4	-5	-20	-65	120	80	40	0	50	50	50	9	15	1	3
1015_1					5	15	-20	-75	120	80	40	0	50	50	50	9	15	1	3
1016_1					6	-15	-20	-65	120	80	40	0	50	50	50	9	15	1	3
1017_1					7	-30	-30	-50	120	80	40	0	50	50	50	9	15	1	3
1021_1			2	Ign	1	20	-20	-80	120	80	40	2.5	50	50	50	9	15	1	3
1022_1					2	45	-20	-90	120	80	40	2.5	50	50	50	9	15	1	3
1023_1					3	0	-15	-85	120	80	40	2.5	50	50	50	9	15	1	3
1024_1					4	-5	-20	-65	120	80	40	2.5	50	50	50	9	15	1	3
1025_1					5	15	-20	-75	120	80	40	2.5	50	50	50	9	15	1	3
1026_1					6	-15	-20	-65	120	80	40	2.5	50	50	50	9	15	1	3
1027_1					7	-30	-30	-50	120	80	40	2.5	50	50	50	9	15	1	3
1031_1			3	Qmd	1	20	-20	-80	120	80	40	2.5	100	50	100	9	15	1	3
1032_1					2	45	-20	-90	120	80	40	2.5	100	50	100	9	15	1	3
1033_1					3	0	-15	-85	120	80	40	2.5	100	50	100	9	15	1	3
1034_1					4	-5	-20	-65	120	80	40	2.5	100	50	100	9	15	1	3
1035_1					5	15	-20	-75	120	80	40	2.5	100	50	100	9	15	1	3
1036_1					6	-15	-20	-65	120	80	40	2.5	100	50	100	9	15	1	3
1037_1					7	-30	-30	-50	120	80	40	2.5	100	50	100	9	15	1	3
1011_2	Pass 2	Outside 0.6% Cu Grade shell	1	Va	1	20	-20	-80	180	120	60	0	50	50	15	6	12	1	3
1012_2					2	45	-20	-90	180	120	60	0	50	50	50	6	12	1	3
1013_2					3	0	-15	-85	180	120	60	0	50	50	50	6	12	1	3
1014_2					4	-5	-20	-65	180	120	60	0	50	50	50	6	12	1	3
1015_2					5	15	-20	-75	180	120	60	0	50	50	50	6	12	1	3
1016_2					6	-15	-20	-65	180	120	60	0	50	50	50	6	12	1	3
1017_2					7	-30	-30	-50	180	120	60	0	50	50	50	6	12	1	3
1021_2			2	Ign	1	20	-20	-80	180	120	60	2.5	50	50	50	6	12	1	3
1022_2					2	45	-20	-90	180	120	60	2.5	50	50	50	6	12	1	3
1023_2					3	0	-15	-85	180	120	60	2.5	50	50	50	6	12	1	3
1024_2					4	-5	-20	-65	180	120	60	2.5	50	50	50	6	12	1	3
1025_2					5	15	-20	-75	180	120	60	2.5	50	50	50	6	12	1	3
1026_2					6	-15	-20	-65	180	120	60	2.5	50	50	50	6	12	1	3
1027_2					7	-30	-30	-50	180	120	60	2.5	50	50	50	6	12	1	3

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
1031_2			3	Qmd	1	20	-20	-80	180	120	60	2.5	100	50	100	6	12	1	3
1032_2					2	45	-20	-90	180	120	60	2.5	100	50	100	6	12	1	3
1033_2					3	0	-15	-85	180	120	60	2.5	100	50	100	6	12	1	3
1034_2					4	-5	-20	-65	180	120	60	2.5	100	50	100	6	12	1	3
1035_2					5	15	-20	-75	180	120	60	2.5	100	50	100	6	12	1	3
1036_2					6	-15	-20	-65	180	120	60	2.5	100	50	100	6	12	1	3
1037_2					7	-30	-30	-50	180	120	60	2.5	100	50	100	6	12	1	3
1011_3	Pass 3	Outside 0.6% Cu Grade shell	1	Va	1	20	-20	-80	360	240	120	0	50	50	15	3	9	1	3
1012_3					2	45	-20	-90	360	240	120	0	50	50	50	3	9	1	3
1013_3					3	0	-15	-85	360	240	120	0	50	50	50	3	9	1	3
1014_3					4	-5	-20	-65	360	240	120	2.5	75	75	75	3	9	1	3
1015_3					5	15	-20	-75	360	240	120	0	50	50	50	3	9	1	3
1016_3					6	-15	-20	-65	360	240	120	0	50	50	50	3	9	1	3
1017_3					7	-30	-30	-50	360	240	120	0	50	50	50	3	9	1	3
1021_3			2	Ign	1	20	-20	-80	360	240	120	2.5	50	50	50	3	9	1	3
1022_3					2	45	-20	-90	360	240	120	2.5	50	50	50	3	9	1	3
1023_3					3	0	-15	-85	360	240	120	2.5	50	50	50	3	9	1	3
1024_3					4	-5	-20	-65	360	240	120	2.5	50	50	50	3	9	1	3
1025_3					5	15	-20	-75	360	240	120	2.5	50	50	50	3	9	1	3
1026_3					6	-15	-20	-65	360	240	120	2.5	50	50	50	3	9	1	3
1027_3					7	-30	-30	-50	360	240	120	2.5	50	50	50	3	9	1	3
1031_3			3	Qmd	1	20	-20	-80	360	240	120	2.5	100	50	100	3	9	1	3
1032_3					2	45	-20	-90	360	240	120	2.5	100	50	100	3	9	1	3
1033_3					3	0	-15	-85	360	240	120	2.5	100	50	100	3	9	1	3
1034_3					4	-5	-20	-65	360	240	120	2.5	100	50	100	3	9	1	3
1035_3					5	15	-20	-75	360	240	120	2.5	100	50	100	3	9	1	3
1036_3					6	-15	-20	-65	360	240	120	2.5	100	50	100	3	9	1	3
1037_3					7	-30	-30	-50	360	240	120	2.5	100	50	100	3	9	1	3

Table 14.26 Copper Search Parameters, Inside 0.6% Cu Grade Shell and Outside 1% Cu Grade Shell, Hugo North

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
2011_1	Pass 1	Inside 0.6% Cu Grade Shell	1	Va	1	20	-20	-80	120	80	40	1.1	50	50	50	9	15	1	3
2012_1					2	45	-20	-90	120	80	40	1.1	50	50	50	9	15	1	3
2013_1					3	0	-15	-85	120	80	40	1.1	50	50	50	9	15	1	3
2014_1					4	-5	-20	-65	120	80	40	1.1	50	50	50	9	15	1	3
2015_1					5	15	-20	-75	120	80	40	1.1	50	50	50	9	15	1	3
2016_1					6	-15	-20	-65	120	80	40	1.1	50	50	50	9	15	1	3
2017_1					7	-30	-30	-50	120	80	40	1.1	50	50	50	9	15	1	3
2021_1			2	Ign	1	20	-20	-80	120	80	40	1.1	50	50	50	9	15	1	3
2022_1					2	45	-20	-90	120	80	40	1.1	50	50	50	9	15	1	3
2023_1					3	0	-15	-85	120	80	40	1.1	50	50	50	9	15	1	3
2024_1					4	-5	-20	-65	120	80	40	1.1	50	50	50	9	15	1	3
2025_1					5	15	-20	-75	120	80	40	1.1	50	50	50	9	15	1	3
2026_1					6	-15	-20	-65	120	80	40	1.1	50	50	50	9	15	1	3
2027_1					7	-30	-30	-50	120	80	40	1.1	50	50	50	9	15	1	3
2031_1			3	Qmd	1	20	-20	-80	120	80	40	1.1	50	50	50	9	15	1	3
2032_1					2	45	-20	-90	120	80	40	1.1	50	50	50	9	15	1	3
2033_1					3	0	-15	-85	120	80	40	0.8	50	50	50	9	15	1	3
2034_1					4	-5	-20	-65	120	80	40	0.8	50	50	50	9	15	1	3
2035_1					5	15	-20	-75	120	80	40	0.8	50	50	50	9	15	1	3
2036_1					6	-15	-20	-65	120	80	40	0.8	50	50	50	9	15	1	3
2037_1					7	-30	-30	-50	120	80	40	0.8	50	50	50	9	15	1	3
2041_1			4	HWS	1	20	-20	-80	120	80	40	0.8	50	50	50	9	15	1	3
2042_1					2	45	-20	-90	120	80	40	1.2	50	50	50	9	15	1	3
2043_1					3	0	-15	-85	120	80	40	1.2	50	50	50	9	15	1	3
2044_1					4	-5	-20	-65	120	80	40	0	50	50	50	9	15	1	3
2045_1					5	15	-20	-75	120	80	40	1.2	50	50	50	9	15	1	3
2046_1					6	-15	-20	-65	120	80	40	1.2	50	50	50	9	15	1	3
2047_1					7	-30	-30	-50	120	80	40	0	50	50	50	9	15	1	3
2011_2	Pass 2	Inside 0.6% Cu Grade Shell	1	Va	1	20	-20	-80	180	120	60	1.1	50	50	50	6	12	1	3
2012_2					2	45	-20	-90	180	120	60	1.1	50	50	50	6	12	1	3
2013_2					3	0	-15	-85	180	120	60	1.1	50	50	50	6	12	1	3
2014_2					4	-5	-20	-65	180	120	60	1.1	50	50	50	6	12	1	3
2015_2					5	15	-20	-75	180	120	60	1.1	50	50	50	6	12	1	3
2016_2					6	-15	-20	-65	180	120	60	1.1	50	50	50	6	12	1	3
2017_2					7	-30	-30	-50	180	120	60	1.1	50	50	50	6	12	1	3
2021_2			2	Ign	1	20	-20	-80	180	120	60	1.1	50	50	50	6	12	1	3
2022_2					2	45	-20	-90	180	120	60	1.1	50	50	50	6	12	1	3
2023_2					3	0	-15	-85	180	120	60	1.1	50	50	50	6	12	1	3
2024_2					4	-5	-20	-65	180	120	60	1.1	50	50	50	6	12	1	3
2025_2					5	15	-20	-75	180	120	60	1.1	50	50	50	6	12	1	3
2026_2					6	-15	-20	-65	180	120	60	1.1	50	50	50	6	12	1	3
2027_2					7	-30	-30	-50	180	120	60	1.1	50	50	50	6	12	1	3

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
2031_2			3	Qmd	1	20	-20	-80	180	120	60	1.1	50	50	50	6	12	1	3
2032_2					2	45	-20	-90	180	120	60	1.1	50	50	50	6	12	1	3
2033_2					3	0	-15	-85	180	120	60	0.8	50	50	50	6	12	1	3
2034_2					4	-5	-20	-65	180	120	60	0.8	50	50	50	6	12	1	3
2035_2					5	15	-20	-75	180	120	60	0.8	50	50	50	6	12	1	3
2036_2					6	-15	-20	-65	180	120	60	0.8	50	50	50	6	12	1	3
2037_2					7	-30	-30	-50	180	120	60	0.8	50	50	50	6	12	1	3
2041_2			4	HWS	1	20	-20	-80	180	120	60	0.8	50	50	50	6	12	1	3
2042_2					2	45	-20	-90	180	120	60	1.2	50	50	50	6	12	1	3
2043_2					3	0	-15	-85	180	120	60	1.2	50	50	50	6	12	1	3
2044_2					4	-5	-20	-65	180	120	60	0	50	50	50	6	12	1	3
2045_2					5	15	-20	-75	180	120	60	1.2	50	50	50	6	12	1	3
2046_2					6	-15	-20	-65	180	120	60	1.2	50	50	50	6	12	1	3
2047_2					7	-30	-30	-50	180	120	60	0	50	50	50	6	12	1	3
2011_3	Pass 3	Inside 0.6% Cu Grade Shell	1	Va	1	20	-20	-80	360	240	120	1.1	50	50	50	3	9	1	3
2012_3					2	45	-20	-90	360	240	120	1.1	50	50	50	3	9	1	3
2013_3					3	0	-15	-85	360	240	120	1.1	50	50	50	3	9	1	3
2014_3					4	-5	-20	-65	360	240	120	4.0	75	75	75	3	9	1	3
2015_3					5	15	-20	-75	360	240	120	1.1	50	50	50	3	9	1	3
2016_3					6	-15	-20	-65	360	240	120	4.0	75	75	75	3	9	1	3
2017_3					7	-30	-30	-50	360	240	120	1.1	50	50	50	3	9	1	3
2021_3			2	Ign	1	20	-20	-80	360	240	120	1.1	50	50	50	3	9	1	3
2022_3					2	45	-20	-90	360	240	120	1.1	50	50	50	3	9	1	3
2023_3					3	0	-15	-85	360	240	120	1.1	50	50	50	3	9	1	3
2024_3					4	-5	-20	-65	360	240	120	1.1	50	50	50	3	9	1	3
2025_3					5	15	-20	-75	360	240	120	1.1	50	50	50	3	9	1	3
2026_3					6	-15	-20	-65	360	240	120	1.1	50	50	50	3	9	1	3
2027_3					7	-30	-30	-50	360	240	120	1.1	50	50	50	3	9	1	3
2031_3			3	Qmd	1	20	-20	-80	360	240	120	1.1	50	50	50	3	9	1	3
2032_3					2	45	-20	-90	360	240	120	1.1	50	50	50	3	9	1	3
2033_3					3	0	-15	-85	360	240	120	0.8	50	50	50	3	9	1	3
2034_3					4	-5	-20	-65	360	240	120	0.8	50	50	50	3	9	1	3
2035_3					5	15	-20	-75	360	240	120	0.8	50	50	50	3	9	1	3
2036_3					6	-15	-20	-65	360	240	120	0.8	50	50	50	3	9	1	3
2037_3					7	-30	-30	-50	360	240	120	0.8	50	50	50	3	9	1	3
2041_3			4	HWS	1	20	-20	-80	360	240	120	0.8	50	50	50	3	9	1	3
2042_3					2	45	-20	-90	360	240	120	1.2	50	50	50	3	9	1	3
2043_3					3	0	-15	-85	360	240	120	1.2	50	50	50	3	9	1	3
2044_3					4	-5	-20	-65	360	240	120	0	50	50	50	3	9	1	3
2045_3					5	15	-20	-75	360	240	120	1.2	50	50	50	3	9	1	3
2046_3					6	-15	-20	-65	360	240	120	1.2	50	50	50	3	9	1	3
2047_3					7	-30	-30	-50	360	240	120	0	50	50	50	3	9	1	3

Table 14.27 Copper Search Parameters, Inside 1% Cu Grade Shell, Hugo North

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
3011_1	Pass 1	Inside 1% Cu Grade Shell	1	Va	1	20	-20	-80	120	80	40	0	50	50	50	9	15	1	3
3012_1					2	45	-20	-90	120	80	40	0	50	50	50	9	15	1	3
3013_1					3	0	-15	-85	120	80	40	1.2	50	50	50	9	15	1	3
3014_1					4	-5	-20	-65	120	80	40	0.5	50	50	50	9	15	1	3
3015_1					5	15	-20	-75	120	80	40	0.5	50	50	50	9	15	1	3
3016_1					6	-15	-20	-65	120	80	40	0.5	50	50	50	9	15	1	3
3017_1					7	-30	-30	-50	120	80	40	0.5	50	50	50	9	15	1	3
3021_1			2	Ign	1	20	-20	-80	120	80	40	0.5	50	50	50	9	15	1	3
3022_1					2	45	-20	-90	120	80	40	0.5	50	50	50	9	15	1	3
3023_1					3	0	-15	-85	120	80	40	0.5	50	50	50	9	15	1	3
3024_1					4	-5	-20	-65	120	80	40	0.5	50	50	50	9	15	1	3
3025_1					5	15	-20	-75	120	80	40	0.5	50	50	50	9	15	1	3
3026_1					6	-15	-20	-65	120	80	40	0.5	50	50	50	9	15	1	3
3027_1					7	-30	-30	-50	120	80	40	0.5	50	50	50	9	15	1	3
3031_1			3	Qmd	1	20	-20	-80	120	80	40	0	50	50	50	9	15	1	3
3032_1					2	45	-20	-90	120	80	40	0	50	50	50	9	15	1	3
3033_1					3	0	-15	-85	120	80	40	0	50	50	50	9	15	1	3
3034_1					4	-5	-20	-65	120	80	40	0	50	50	50	9	15	1	3
3035_1					5	15	-20	-75	120	80	40	0	50	50	50	9	15	1	3
3036_1					6	-15	-20	-65	120	80	40	0	50	50	50	9	15	1	3
3037_1					7	-30	-30	-50	120	80	40	0	50	50	50	9	15	1	3
3011_2	Pass 2	Inside 1% Cu Grade Shell	1	Va	1	20	-20	-80	180	120	60	0	50	50	50	6	12	1	3
3012_2					2	45	-20	-90	180	120	60	0	50	50	50	6	12	1	3
3013_2					3	0	-15	-85	180	120	60	1.2	50	50	50	6	12	1	3
3014_2					4	-5	-20	-65	180	120	60	0.5	50	50	50	6	12	1	3
3015_2					5	15	-20	-75	180	120	60	0.5	50	50	50	6	12	1	3
3016_2					6	-15	-20	-65	180	120	60	0.5	50	50	50	6	12	1	3
3017_2					7	-30	-30	-50	180	120	60	0.5	50	50	50	6	12	1	3
3021_2			2	Ign	1	20	-20	-80	180	120	60	0.5	50	50	50	6	12	1	3
3022_2					2	45	-20	-90	180	120	60	0.5	50	50	50	6	12	1	3
3023_2					3	0	-15	-85	180	120	60	0.5	50	50	50	6	12	1	3
3024_2					4	-5	-20	-65	180	120	60	0.5	50	50	50	6	12	1	3
3025_2					5	15	-20	-75	180	120	60	0.5	50	50	50	6	12	1	3
3026_2					6	-15	-20	-65	180	120	60	0.5	50	50	50	6	12	1	3
3027_2					7	-30	-30	-50	180	120	60	0.5	50	50	50	6	12	1	3

Estimation ID	Estimation Pass	Copper Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
3031_2			3	Qmd	1	20	-20	-80	180	120	60	0	50	50	50	6	12	1	3
3032_2					2	45	-20	-90	180	120	60	0	50	50	50	6	12	1	3
3033_2					3	0	-15	-85	180	120	60	0	50	50	50	6	12	1	3
3034_2					4	-5	-20	-65	180	120	60	0	50	50	50	6	12	1	3
3035_2					5	15	-20	-75	180	120	60	0	50	50	50	6	12	1	3
3036_2					6	-15	-20	-65	180	120	60	0	50	50	50	6	12	1	3
3037_2					7	-30	-30	-50	180	120	60	0	50	50	50	6	12	1	3
3011_3	Pass 3	Inside 1% Cu Grade Shell	1	Va	1	20	-20	-80	360	240	120	0	50	50	50	3	9	1	3
3012_3					2	45	-20	-90	360	240	120	0	50	50	50	3	9	1	3
3013_3					3	0	-15	-85	360	240	120	1.2	50	50	50	3	9	1	3
3014_3					4	-5	-20	-65	360	240	120	5.0	75	75	75	3	9	1	3
3015_3					5	15	-20	-75	360	240	120	0.5	50	50	50	3	9	1	3
3016_3					6	-15	-20	-65	360	240	120	0.5	50	50	50	3	9	1	3
3017_3					7	-30	-30	-50	360	240	120	5.0	75	75	75	3	9	1	3
3021_3			2	Ign	1	20	-20	-80	360	240	120	0.5	50	50	50	3	9	1	3
3022_3					2	45	-20	-90	360	240	120	0.5	50	50	50	3	9	1	3
3023_3					3	0	-15	-85	360	240	120	0.5	50	50	50	3	9	1	3
3024_3					4	-5	-20	-65	360	240	120	0.5	50	50	50	3	9	1	3
3025_3					5	15	-20	-75	360	240	120	0.5	50	50	50	3	9	1	3
3026_3					6	-15	-20	-65	360	240	120	0.5	50	50	50	3	9	1	3
3027_3					7	-30	-30	-50	360	240	120	0.5	50	50	50	3	9	1	3
3031_3			3	Qmd	1	20	-20	-80	360	240	120	0	50	50	50	3	9	1	3
3032_3					2	45	-20	-90	360	240	120	0	50	50	50	3	9	1	3
3033_3					3	0	-15	-85	360	240	120	0	50	50	50	3	9	1	3
3034_3					4	-5	-20	-65	360	240	120	0	50	50	50	3	9	1	3
3035_3					5	15	-20	-75	360	240	120	0	50	50	50	3	9	1	3
3036_3					6	-15	-20	-65	360	240	120	0	50	50	50	3	9	1	3
3037_3					7	-30	-30	-50	360	240	120	0	50	50	50	3	9	1	3

Table 14.28 Copper Search Parameters, Inside BiGd, Hugo North

Estimation ID	Estimation Pass	Domain	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
1051_1	Pass 1	BiGd High grade Domain	5	BiGd	1	20	−20	−80	120	80	40	3	50	50	15	9	15	1	3
1052_1					2	45	−20	−90	120	80	40	3	50	50	15	9	15	1	3
1053_1					3	0	−15	−85	120	80	40	3	50	50	15	9	15	1	3
1054_1					4	−5	−20	−65	120	80	40	3	50	50	15	9	15	1	3
1055_1					5	15	−20	−75	120	80	40	3	50	50	15	9	15	1	3
1056_1					6	−15	−20	−65	120	80	40	3	50	50	15	9	15	1	3
1057_1					7	−30	−30	−50	120	80	40	3	50	50	15	9	15	1	3
1051_h2	Pass 2				1	20	−20	−80	180	120	60	3	50	50	15	6	12	1	3
1052_h2					2	45	−20	−90	180	120	60	3	50	50	15	6	12	1	3
1053_h2					3	0	−15	−85	180	120	60	3	50	50	15	6	12	1	3
1054_h2					4	−5	−20	−65	180	120	60	3	50	50	15	6	12	1	3
1055_h2					5	15	−20	−75	180	120	60	3	50	50	15	6	12	1	3
1056_h2					6	−15	−20	−65	180	120	60	3	50	50	15	6	12	1	3
1057_h2					7	−30	−30	−50	180	120	60	3	50	50	15	6	12	1	3
1051_h3	Pass 3				1	20	−20	−80	360	240	120	3	50	50	15	3	9	1	3
1052_h3					2	45	−20	−90	360	240	120	3	50	50	15	3	9	1	3
1053_h3					3	0	−15	−85	360	240	120	3	50	50	15	3	9	1	3
1054_h3					4	−5	−20	−65	360	240	120	3	50	50	15	3	9	1	3
1055_h3					5	15	−20	−75	360	240	120	3	50	50	15	3	9	1	3
1056_h3					6	−15	−20	−65	360	240	120	3	50	50	15	3	9	1	3
1057_h3					7	−30	−30	−50	360	240	120	3	50	50	15	3	9	1	3
1051_I2	Pass 1	BiGd Outside of Grade Shell	5	BiGd	1	20	−20	−80	180	120	60	3	50	50	15	6	12	1	3
1052_I2					2	45	−20	−90	180	120	60	3	50	50	15	6	12	1	3
1053_I2					3	0	−15	−85	180	120	60	3	50	50	15	6	12	1	3
1054_I2					4	−5	−20	−65	180	120	60	3	50	50	15	6	12	1	3
1055_I2					5	15	−20	−75	180	120	60	3	50	50	15	6	12	1	3
1056_I2					6	−15	−20	−65	180	120	60	3	50	50	15	6	12	1	3
1057_I2					7	−30	−30	−50	180	120	60	3	50	50	15	6	12	1	3
1051_I3	Pass 2				1	20	−20	−80	360	240	120	3	50	50	15	3	9	1	3
1052_I3					2	45	−20	−90	360	240	120	3	50	50	15	3	9	1	3
1053_I3					3	0	−15	−85	360	240	120	3	50	50	15	3	9	1	3
1054_I3					4	−5	−20	−65	360	240	120	3	50	50	15	3	9	1	3
1055_I3					5	15	−20	−75	360	240	120	3	50	50	15	3	9	1	3
1056_I3					6	−15	−20	−65	360	240	120	3	50	50	15	3	9	1	3
1057_I3					7	−30	−30	−50	360	240	120	3	50	50	15	3	9	1	3

Table 14.29 Gold Search Parameters, Outside 0.3 g/t Au Grade Shell, Hugo North

Estimation ID	Estimation Pass	Au Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
1011_1	Pass 1	Outside 0.3 g/t Au Grade shell	1	Va	1	55	-70	20	120	40	80	0	50	50	15	9	15	1	3
1012_1					2	-15	15	20	120	40	80	0	50	50	50	9	15	1	3
1013_1					3	15	-15	90	120	80	40	0	50	50	50	9	15	1	3
1021_1			2	Ign	1	55	-70	20	120	40	80	1.5	50	50	50	9	15	1	3
1022_1					2	-15	15	20	120	40	80	1.5	50	50	50	9	15	1	3
1023_1					3	15	-15	90	120	80	40	1.5	50	50	50	9	15	1	3
1031_1			3	Qmd	1	55	-70	20	120	40	80	1.0	100	50	100	9	15	1	3
1032_1					2	-15	15	20	120	40	80	1.0	100	50	100	9	15	1	3
1033_1					3	15	-15	90	120	80	40	1.0	100	50	100	9	15	1	3
1011_2	Pass 2	Outside 0.3 g/t Au Grade shell	1	Va	1	55	-70	20	180	60	120	0	50	50	15	6	12	1	3
1012_2					2	-15	15	20	180	60	120	0	50	50	50	6	12	1	3
1013_2					3	15	-15	90	180	120	60	0	50	50	50	6	12	1	3
1021_2			2	Ign	1	55	-70	20	180	60	120	1.5	50	50	50	6	12	1	3
1022_2					2	-15	15	20	180	60	120	1.5	50	50	50	6	12	1	3
1023_2					3	15	-15	90	180	120	60	1.5	50	50	50	6	12	1	3
1031_2			3	Qmd	1	55	-70	20	180	60	120	1.0	100	50	100	6	12	1	3
1032_2					2	-15	15	20	180	60	120	1.0	100	50	100	6	12	1	3
1033_2					3	15	-15	90	180	120	60	1.0	100	50	100	6	12	1	3
1011_3	Pass 3	Outside 0.3 g/t Au Grade shell	1	Va	1	55	-70	20	360	120	240	0	50	50	15	3	9	1	3
1012_3					2	-15	15	20	360	120	240	0	50	50	50	3	9	1	3
1013_3					3	15	-15	90	360	240	120	0	50	50	50	3	9	1	3
1021_3			2	Ign	1	55	-70	20	360	120	240	1.5	50	50	50	3	9	1	3
1022_3					2	-15	15	20	360	120	240	1.5	50	50	50	3	9	1	3
1023_3					3	15	-15	90	360	240	120	1.5	50	50	50	3	9	1	3
1031_3			3	Qmd	1	55	-70	20	360	120	240	1.0	100	50	100	3	9	1	3
1032_3					2	-15	15	20	360	120	240	1.0	100	50	100	3	9	1	3
1033_3					3	15	-15	90	360	240	120	1.0	100	50	100	3	9	1	3

Table 14.30 Gold Search Parameters, Inside 0.3 g/t Au Grade Shell and Outside 1 g/t Grade Shell, Hugo North

Estimation ID	Estimation Pass	Au Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
2011_1	Pass 1	0.3 g/t Au Grade Shell	1	Va	1	55	-70	20	120	40	80	1.1	50	50	50	9	15	1	3
2012_1					2	-15	15	20	120	40	80	1.1	50	50	50	9	15	1	3
2013_1					3	15	-15	90	120	80	40	1.1	50	50	50	9	15	1	3
2021_1			2	Ign	1	55	-70	20	120	40	80	1.1	50	50	50	9	15	1	3
2022_1					2	-15	15	20	120	40	80	1.1	50	50	50	9	15	1	3
2023_1					3	15	-15	90	120	80	40	1.1	50	50	50	9	15	1	3
2031_1			3	Qmd	1	55	-70	20	120	40	80	1.1	50	50	50	9	15	1	3
2032_1					2	-15	15	20	120	40	80	1.1	50	50	50	9	15	1	3
2033_1					3	15	-15	90	120	80	40	0.8	50	50	50	9	15	1	3
2041_1			4	HWS	1	55	-70	20	120	40	80	0.8	50	50	50	9	15	1	3
2042_1					2	-15	15	20	120	40	80	1.2	50	50	50	9	15	1	3
2043_1					3	15	-15	90	120	80	40	1.2	50	50	50	9	15	1	3
2011_2	Pass 2	0.3 g/t Au Grade Shell	1	Va	1	55	-70	20	180	60	120	1.1	50	50	50	6	12	1	3
2012_2					2	-15	15	20	180	60	120	1.1	50	50	50	6	12	1	3
2013_2					3	15	-15	90	180	120	60	1.1	50	50	50	6	12	1	3
2021_2			2	Ign	1	55	-70	20	180	60	120	1.1	50	50	50	6	12	1	3
2022_2					2	-15	15	20	180	60	120	1.1	50	50	50	6	12	1	3
2023_2					3	15	-15	90	180	120	60	1.1	50	50	50	6	12	1	3
2031_2			3	Qmd	1	55	-70	20	180	60	120	1.1	50	50	50	6	12	1	3
2032_2					2	-15	15	20	180	60	120	1.1	50	50	50	6	12	1	3
2033_2					3	15	-15	90	180	120	60	0.8	50	50	50	6	12	1	3
2041_2			4	HWS	1	55	-70	20	180	60	120	0.8	50	50	50	6	12	1	3
2042_2					2	-15	15	20	180	60	120	1.2	50	50	50	6	12	1	3
2043_2					3	15	-15	90	180	120	60	1.2	50	50	50	6	12	1	3
2011_3	Pass 3	0.3 g/t Au Grade Shell	1	Va	1	55	-70	20	360	120	240	1.1	50	50	50	3	9	1	3
2012_3					2	-15	15	20	360	120	240	1.1	50	50	50	3	9	1	3
2013_3					3	15	-15	90	360	240	120	1.1	50	50	50	3	9	1	3
2021_3			2	Ign	1	55	-70	20	360	120	240	1.1	50	50	50	3	9	1	3
2022_3					2	-15	15	20	360	120	240	1.1	50	50	50	3	9	1	3
2023_3					3	15	-15	90	360	240	120	1.1	50	50	50	3	9	1	3
2031_3			3	Qmd	1	55	-70	20	360	120	240	1.1	50	50	50	3	9	1	3
2032_3					2	-15	15	20	360	120	240	1.1	50	50	50	3	9	1	3
2033_3					3	15	-15	90	360	240	120	0.8	50	50	50	3	9	1	3
2041_3			4	HWS	1	55	-70	20	360	120	240	0.8	50	50	50	3	9	1	3
2042_3					2	-15	15	20	360	120	240	1.2	50	50	50	3	9	1	3
2043_3					3	15	-15	90	360	240	120	1.2	50	50	50	3	9	1	3

Table 14.31 Gold Search Parameters, Inside 1 g/t Au Grade Shell, Hugo North

Estimation ID	Estimation Pass	Au Grade Shell	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
3011_1	Pass 1	1 g/t Au Grade Shell	1	Va	1	55	-70	20	120	40	80	0	50	50	50	9	15	1	3
3012_1					2	-15	15	20	120	40	80	0	50	50	50	9	15	1	3
3013_1					3	15	-15	90	120	80	40	1.2	50	50	50	9	15	1	3
3021_1			2	Ign	1	55	-70	20	120	40	80	0.5	50	50	50	9	15	1	3
3022_1					2	-15	15	20	120	40	80	0.5	50	50	50	9	15	1	3
3023_1					3	15	-15	90	120	80	40	0.5	50	50	50	9	15	1	3
3031_1			3	Qmd	1	55	-70	20	120	40	80	0	50	50	50	9	15	1	3
3032_1					2	-15	15	20	120	40	80	0	50	50	50	9	15	1	3
3033_1					3	15	-15	90	120	80	40	0	50	50	50	9	15	1	3
3011_2	Pass 2	1 g/t Au Grade Shell	1	Va	1	55	-70	20	180	60	120	0	50	50	50	6	12	1	3
3012_2					2	-15	15	20	180	60	120	0	50	50	50	6	12	1	3
3013_2					3	15	-15	90	180	120	60	1.2	50	50	50	6	12	1	3
3021_2			2	Ign	1	55	-70	20	180	60	120	0.5	50	50	50	6	12	1	3
3022_2					2	-15	15	20	180	60	120	0.5	50	50	50	6	12	1	3
3023_2					3	15	-15	90	180	120	60	0.5	50	50	50	6	12	1	3
3031_2			3	Qmd	1	55	-70	20	180	60	120	0	50	50	50	6	12	1	3
3032_2					2	-15	15	20	180	60	120	0	50	50	50	6	12	1	3
3033_2					3	15	-15	90	180	120	60	0	50	50	50	6	12	1	3
3011_3	Pass 3	1 g/t Au Grade Shell	1	Va	1	55	-70	20	360	120	240	0	50	50	50	3	9	1	3
3012_3					2	-15	15	20	360	120	240	0	50	50	50	3	9	1	3
3013_3					3	15	-15	90	360	240	120	1.2	50	50	50	3	9	1	3
3021_3			2	Ign	1	55	-70	20	360	120	240	0.5	50	50	50	3	9	1	3
3022_3					2	-15	15	20	360	120	240	0.5	50	50	50	3	9	1	3
3023_3					3	15	-15	90	360	240	120	0.5	50	50	50	3	9	1	3
3031_3			3	Qmd	1	55	-70	20	360	120	240	0	50	50	50	3	9	1	3
3032_3					2	-15	15	20	360	120	240	0	50	50	50	3	9	1	3
3033_3					3	15	-15	90	360	240	120	0	50	50	50	3	9	1	3

Table 14.32 Gold Search Parameters, Inside BiGd, Hugo North

Estimation ID	Estimation Pass	Domain	Nlith	Rock Type	Zone Code	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis	HY Threshold	HY Major Radius	HY Semi-Major Radius	HY Minor Radius	Minor Samples per Estimate	Maximum Samples per Estimate	Limit Samples per Hole	Maximum Samples per Hole
1051h1	Pass 1	BiGd High Grade Domain	5	BiGd	1	55	−70	20	120	40	80	0	50	50	50	9	15	1	3
1052h1					2	−15	15	20	120	40	80	0	50	50	50	9	15	1	3
1053h1					3	15	−15	90	120	40	80	0	50	50	50	9	15	1	3
1051h2	Pass 2				1	55	−70	20	180	60	120	0	50	50	50	6	12	1	3
1052h2					2	−15	15	20	180	60	120	0	50	50	50	6	12	1	3
1053h2					3	15	−15	90	180	60	120	0	50	50	50	6	12	1	3
1051h3	Pass 3				1	55	−70	20	360	120	240	0	50	50	50	3	9	1	3
1052h3					2	−15	15	20	360	120	240	0	50	50	50	3	9	1	3
1053h3					3	15	−15	90	360	120	240	0	50	50	50	3	9	1	3
1051_1	Pass 1	BiGd Outside of Au Grade shell	5	BiGd	1	55	−70	20	120	40	80	0	50	50	50	9	15	1	3
1052_1					2	−15	15	20	120	40	80	0	50	50	50	9	15	1	3
1053_1					3	15	−15	90	120	40	80	0	50	50	50	9	15	1	3
1051_2	Pass 2				3	55	−70	20	180	60	120	0	50	50	50	6	12	1	3
1052_2					3	−15	15	20	180	60	120	0	50	50	50	6	12	1	3
1053_2					3	15	−15	90	180	60	120	0	50	50	50	6	12	1	3
1051_3	Pass 3				1	55	−70	20	360	120	240	0	50	50	50	3	9	1	3
1052_3					2	−15	15	20	360	120	240	0	50	50	50	3	9	1	3
1053_3					3	15	−15	90	360	120	240	0	50	50	50	3	9	1	3

14.1.8.4 Hugo South

The cell size was 20 m east x 20 m north x 15 m high. Domain codes were assigned by a simple majority. In the case of assigning dyke codes to the block model, a slightly higher percentage of 60% was necessary. For the ore percent model, the default value was set to 100% but modified to reflect where blocks were intersected by dykes. Percent below topography was also calculated into the model blocks.

Modelling consisted of grade interpolation by OK, and also NN for validation purposes. The search ellipsoids were oriented preferentially to the orientation of the copper grade shells, with the long axis at 20° east of north, with no plunge, and dipping 55° to the south-east.

A two-pass approach was instituted for interpolation within the grade shells. The first pass allowed a single hole to place a grade estimate in a block, while the second allowed a minimum of two holes from the same estimation domain. A single-pass, two-hole minimum rule was used in the background domains. All blocks received a minimum of three and maximum of four composites from a single drillhole (for the two-hole minimum pass). Maximum composite limits varied by metal: nine for copper, 15 for gold, and 12 for molybdenum. Most interpolation runs implemented an outlier restriction to limit grade smearing in areas of limited data.

Bulk density data were assigned to an assay database file. These data were composited into 15 m fixed-length downhole values to reflect the block model bench height.

14.1.8.5 Heruga

The selected block size was 20 m east x 20 m north x 15 m high for consistency with previous modelling at the Oyu Tolgoi deposits. It was also considered to be a suitable block size for mining studies using the block cave approach, the assumed mining method for the Heruga deposit. The parent blocks were divided into sub-cells when flagging the model with dyke wireframes. The block model was coded according to zone, lithological domain, and grade shell. Post-mineralization dykes and the late quartz monzodiorite were assumed to represent zero-grade waste cutting the mineralized lithologies.

Only the mineralized lithologies were estimated, i.e. Qmd and Va. All other units in the model were set to zero grade. Modelling consisted of grade interpolation by OK. As part of the model validation, grades were also interpolated using NN, ID3, and OK of uncapped composites. Density was interpolated by ID3.

The search ellipsoids were oriented preferentially to the general trend of the grade shells. A staged search strategy was applied, 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 populated in a third pass that removed the two-hole constraint. 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 (50–100 m). Outside of this distance the lower capped value was used.

The sub-cells in the final model were regularized to parent cell size after estimation was complete.

14.1.9 Results of Estimation

The results of the Mineral Resource estimates are presented below as grade and tonnage tables at variable copper-equivalent cut-off grades for different resource classifications, and as copper and gold grade-tonnage curves. The information is arranged as follows:

- Oyut: Table 14.33 and Figure 14.2.
- Hugo North: Table 14.34 and Figure 14.3.
- Hugo North Extension: Table 14.35 and Figure 14.4.
- Hugo South: Table 14.36 and Figure 14.5.
- Heruga: Table 14.37 and Figure 14.6.

Table 14.33 Oyut Deposit – Grade and Tonnage Calculations at Variable Copper-Equivalent Cut-off Grades
Includes Open Pit Resources >0.22% CuEq and Underground Resources >0.37% CuEq
Excludes material mined up to 31 December 2015

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured	0.22	390	0.52	0.36	1.34	53.3	0.72	4,455	4,369	16,294	46
	0.30	354	0.55	0.39	1.38	55.6	0.77	4,289	4,270	15,137	43
	0.37	320	0.58	0.42	1.41	57.4	0.81	4,089	4,160	14,026	41
	0.50	251	0.64	0.49	1.48	60.3	0.92	3,554	3,860	11,555	33
	0.60	197	0.70	0.58	1.53	61.7	1.02	3,018	3,556	9,354	27
	0.70	152	0.75	0.68	1.59	63.6	1.13	2,513	3,227	7,498	21
	0.80	116	0.80	0.80	1.66	65.6	1.24	2,059	2,893	5,983	17
	0.90	90	0.85	0.93	1.72	68.0	1.36	1,693	2,602	4,839	14
	1.00	71	0.89	1.06	1.79	71.0	1.47	1,387	2,339	3,939	11
	1.25	41	0.97	1.39	1.93	79.6	1.73	879	1,776	2,460	7
	1.50	24	1.02	1.79	2.07	88.2	2.00	540	1,333	1,541	5
	2.00	8	1.12	2.67	2.31	93.1	2.57	201	678	585	2
	2.50	3	1.33	3.47	2.62	90.2	3.22	89	327	248	1
	3.00	2	1.47	4.11	2.80	91.9	3.70	51	201	137	0.32
	3.50	0.8	1.61	4.64	2.76	93.2	4.13	30	122	73	0.17
Indicated	0.22	808	0.38	0.27	1.12	53.9	0.53	6,747	6,784	28,099	96
	0.30	683	0.41	0.30	1.15	56.6	0.58	6,178	6,445	24,417	85
	0.37	566	0.44	0.34	1.19	58.1	0.63	5,507	5,995	20,874	72
	0.50	359	0.51	0.43	1.27	60.9	0.75	4,032	4,791	14,129	48
	0.60	237	0.56	0.52	1.33	61.7	0.85	2,951	3,826	9,813	32
	0.70	152	0.61	0.64	1.39	60.4	0.97	2,061	3,013	6,582	20
	0.80	97	0.66	0.78	1.46	58.5	1.09	1,415	2,360	4,418	13
	0.90	64	0.71	0.93	1.54	60.0	1.22	996	1,854	3,056	8
	1.00	45	0.74	1.09	1.60	61.9	1.34	734	1,515	2,238	6
	1.25	21	0.81	1.45	1.75	65.5	1.60	376	948	1,147	3
	1.50	11	0.87	1.76	1.86	66.6	1.84	204	580	611	2
	2.00	2	1.00	2.58	2.13	68.4	2.40	47	170	140	0.3
	2.50	1	1.16	3.41	2.46	74.9	3.01	14	59	43	0.1
	3.00	0.19	1.40	4.12	2.56	65.0	3.64	6	24	15	0.03
	3.50	0.10	1.53	4.46	2.52	56.8	3.96	3	14	8	0.01
Measured + Indicated	0.22	1,198	0.42	0.30	1.19	53.7	0.59	11,203	11,153	44,393	142
	0.30	1,036	0.46	0.33	1.23	56.3	0.65	10,467	10,715	39,554	128
	0.37	886	0.49	0.37	1.27	57.8	0.70	9,596	10,154	34,901	113
	0.50	611	0.56	0.46	1.35	60.6	0.82	7,586	8,651	25,684	82
	0.60	434	0.62	0.55	1.42	61.7	0.93	5,969	7,381	19,167	59
	0.70	304	0.68	0.66	1.49	62.0	1.05	4,574	6,241	14,081	42
	0.80	213	0.74	0.79	1.57	62.4	1.18	3,473	5,252	10,401	29
	0.90	154	0.79	0.93	1.65	64.7	1.30	2,688	4,455	7,895	22
	1.00	116	0.83	1.07	1.72	67.5	1.42	2,121	3,854	6,177	17
	1.25	62	0.92	1.41	1.87	74.8	1.69	1,256	2,724	3,607	10
	1.50	35	0.98	1.78	2.00	81.6	1.95	744	1,913	2,152	6
	2.00	10	1.10	2.66	2.27	88.0	2.54	248	848	725	2
	2.50	4	1.31	3.46	2.60	87.8	3.18	103	387	290	1
	3.00	2	1.46	4.11	2.77	89.0	3.70	57	225	151	0.34
	3.50	0.9	1.60	4.62	2.73	89.3	4.11	33	136	80	0.19
Inferred	0.22	548	0.32	0.20	0.85	38.7	0.43	3,808	3,466	14,524	46
	0.30	415	0.35	0.23	0.89	40.9	0.49	3,239	3,011	11,504	37
	0.37	315	0.39	0.27	0.91	40.4	0.54	2,680	2,607	8,970	28
	0.50	141	0.48	0.34	0.92	38.1	0.67	1,491	1,486	4,037	12
	0.60	75	0.56	0.40	0.94	34.9	0.78	925	936	2,195	6
	0.70	43	0.64	0.45	0.94	30.9	0.89	600	591	1,251	3
	0.80	26	0.72	0.47	0.92	25.2	0.98	412	381	747	1
	0.90	15	0.79	0.54	0.98	24.2	1.09	251	244	444	1
	1.00	8	0.86	0.63	1.08	24.9	1.21	149	154	264	0.4
	1.25	2	0.91	1.06	1.34	30.7	1.50	44	72	91	0.1
	1.50	1	1.00	1.44	1.70	35.2	1.79	16	33	39	0.1
	2.00	0.14	1.58	1.33	2.08	27.8	2.32	5	6	9	0.01
	2.50	0.04	1.91	0.86	1.45	24.3	2.93	2	1	2	0.002
	3.00	0.01	0.57	0.39	1.05	5.4	3.92	0.09	0.09	0.24	0.0001
	3.50	0.01	0.57	0.39	1.05	5.4	3.92	0.09	0.09	0.24	0.0001

Figure 14.2 Oyut Deposit – Grade-Tonnage Curves at Various Copper-Equivalent Cut-off Grades

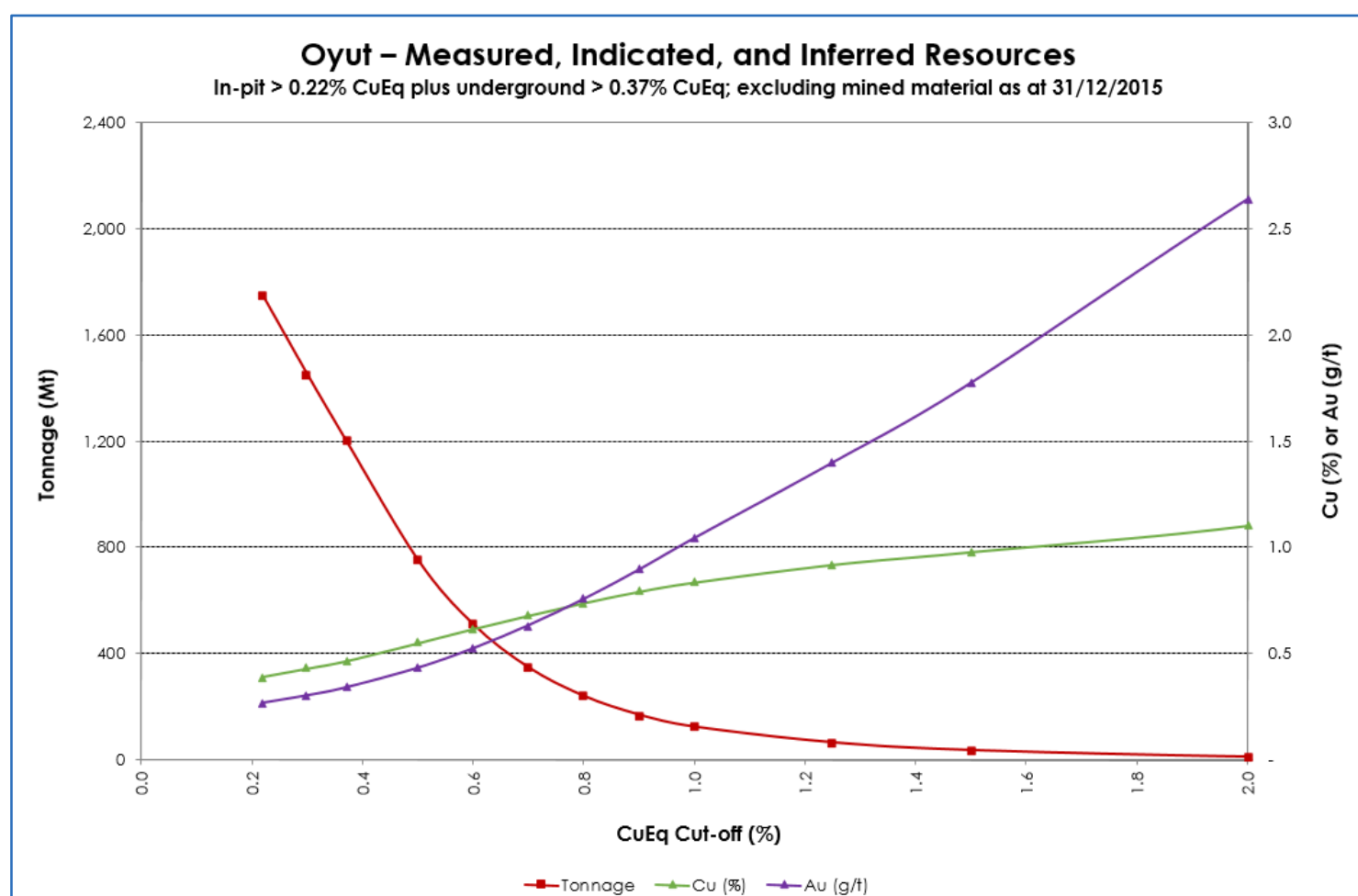


Table 14.34 Hugo North – Grade and Tonnage Calculations at Variable Copper-Equivalent Cut-off Grades (Unchanged since 2014 OTTR)

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured	0.10	108	1.79	0.42	4.11	29.2	2.06	4,277	1,461	14,327	7
	0.22	103	1.87	0.44	4.29	30.0	2.16	4,262	1,455	14,219	7
	0.30	100	1.93	0.45	4.40	30.1	2.22	4,246	1,450	14,128	7
	0.37	98	1.97	0.46	4.48	30.3	2.26	4,231	1,446	14,046	7
	0.50	94	2.03	0.48	4.61	30.8	2.34	4,200	1,435	13,890	6
	0.60	91	2.08	0.49	4.69	31.1	2.39	4,174	1,426	13,768	6
	0.70	89	2.12	0.50	4.78	31.4	2.44	4,142	1,417	13,636	6
	0.80	86	2.17	0.51	4.88	31.6	2.49	4,100	1,403	13,466	6
	0.90	83	2.21	0.52	4.98	31.9	2.54	4,056	1,390	13,306	6
	1.00	79	2.28	0.54	5.12	32.2	2.62	3,985	1,373	13,068	6
	1.25	69	2.47	0.59	5.56	32.7	2.85	3,753	1,320	12,338	5
	1.50	62	2.62	0.64	5.90	33.1	3.02	3,559	1,260	11,708	5
	2.00	50	2.87	0.69	6.40	33.2	3.30	3,184	1,117	10,374	4
	2.50	38	3.16	0.75	6.94	34.1	3.64	2,667	929	8,542	3
	3.00	28	3.45	0.81	7.35	35.2	3.96	2,138	732	6,650	2
	3.50	19	3.74	0.87	7.79	35.4	4.29	1,589	542	4,829	2
Indicated	0.10	923	1.30	0.29	2.83	30.4	1.49	26,484	8,503	84,061	62
	0.22	835	1.42	0.31	3.08	32.6	1.62	26,223	8,411	82,716	60
	0.30	787	1.50	0.33	3.23	33.6	1.71	25,983	8,338	81,705	58
	0.37	749	1.56	0.34	3.35	34.3	1.78	25,737	8,268	80,718	57
	0.50	687	1.67	0.37	3.56	35.1	1.90	25,202	8,128	78,667	53
	0.60	643	1.75	0.39	3.72	35.3	1.99	24,729	8,003	76,884	50
	0.70	607	1.81	0.40	3.85	35.4	2.07	24,276	7,879	75,186	47
	0.80	572	1.88	0.42	3.99	35.4	2.15	23,760	7,737	73,270	45
	0.90	534	1.96	0.44	4.14	35.4	2.24	23,131	7,577	71,062	42
	1.00	492	2.06	0.47	4.32	35.2	2.36	22,333	7,377	68,332	38
	1.25	396	2.32	0.54	4.82	34.4	2.65	20,225	6,824	61,338	30
	1.50	328	2.54	0.60	5.24	33.0	2.92	18,416	6,304	55,297	24
	2.00	238	2.94	0.69	5.92	31.9	3.37	15,403	5,310	45,300	17
	2.50	176	3.29	0.77	6.46	32.7	3.77	12,771	4,379	36,616	13
	3.00	132	3.59	0.85	6.90	32.4	4.12	10,407	3,601	29,194	9
	3.50	94	3.88	0.94	7.32	31.5	4.47	8,078	2,858	22,204	7

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured + Indicated	0.10	1,031	1.35	0.30	2.97	30.3	1.55	30,761	9,964	98,388	69
	0.22	938	1.47	0.33	3.21	32.3	1.68	30,485	9,866	96,935	67
	0.30	887	1.55	0.34	3.36	33.2	1.77	30,230	9,788	95,833	65
	0.37	847	1.61	0.36	3.48	33.9	1.83	29,968	9,714	94,764	63
	0.50	780	1.71	0.38	3.69	34.6	1.95	29,401	9,563	92,557	60
	0.60	734	1.79	0.40	3.84	34.8	2.04	28,903	9,429	90,652	56
	0.70	696	1.85	0.42	3.97	34.9	2.12	28,418	9,295	88,821	53
	0.80	658	1.92	0.43	4.10	34.9	2.20	27,860	9,141	86,736	51
	0.90	618	2.00	0.45	4.25	34.9	2.28	27,188	8,967	84,367	48
	1.00	571	2.09	0.48	4.43	34.8	2.39	26,318	8,750	81,400	44
	1.25	465	2.34	0.54	4.93	34.1	2.68	23,978	8,144	73,676	35
	1.50	390	2.56	0.60	5.34	33.0	2.94	21,975	7,565	67,004	28
	2.00	288	2.92	0.69	6.01	32.2	3.36	18,587	6,427	55,674	20
	2.50	215	3.26	0.77	6.54	32.9	3.75	15,438	5,308	45,158	16
	3.00	160	3.56	0.84	6.98	32.9	4.09	12,546	4,333	35,844	12
	3.50	114	3.86	0.93	7.40	32.2	4.44	9,667	3,400	27,034	8
Inferred	0.10	1,033	0.65	0.23	1.97	31.3	0.79	14,698	7,681	65,552	71
	0.22	935	0.70	0.25	2.14	33.3	0.86	14,455	7,437	64,375	69
	0.30	874	0.74	0.26	2.24	34.2	0.90	14,171	7,284	63,025	66
	0.37	811	0.77	0.27	2.34	34.8	0.94	13,807	7,058	60,964	62
	0.50	678	0.85	0.30	2.53	34.7	1.04	12,783	6,468	55,264	52
	0.60	571	0.93	0.32	2.72	33.5	1.13	11,730	5,897	49,926	42
	0.70	486	1.00	0.34	2.88	33.2	1.22	10,769	5,328	45,092	36
	0.80	415	1.08	0.36	3.03	32.7	1.30	9,828	4,756	40,434	30
	0.90	341	1.16	0.38	3.21	31.8	1.40	8,730	4,134	35,225	24
	1.00	281	1.25	0.39	3.41	31.1	1.49	7,725	3,529	30,866	19
	1.25	173	1.46	0.42	3.98	29.1	1.73	5,590	2,328	22,149	11
	1.50	103	1.68	0.47	4.65	28.4	1.97	3,812	1,550	15,435	6
	2.00	33	2.19	0.57	5.55	25.7	2.56	1,594	608	5,887	2
	2.50	13	2.70	0.64	5.98	31.3	3.10	764	263	2,467	1
	3.00	6	3.16	0.59	6.39	36.9	3.55	417	114	1,228	0.5
	3.50	3	3.63	0.51	7.07	41.9	3.98	210	43	594	0.2

Figure 14.3 Hugo North – Grade–Tonnage Curves at Various Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)

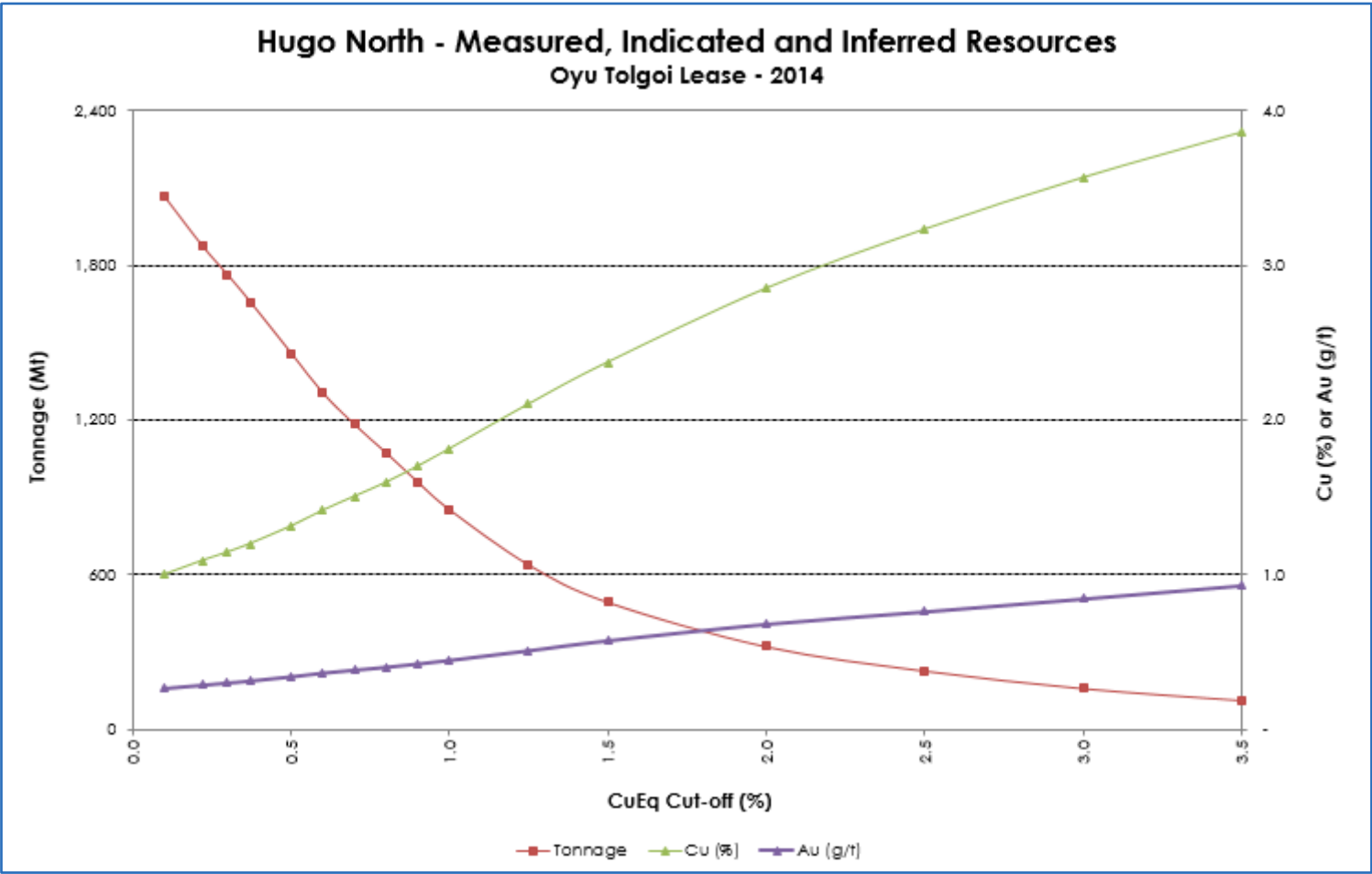


**Table 14.35 Hugo North Extension – Grade and Tonnage Calculations at Variable Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)**

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured	0.10	1.2	1.35	0.12	2.71	37.5	1.44	36	4.5	105	0.1
	0.22	1.2	1.36	0.12	2.72	37.5	1.45	36	4.5	105	0.1
	0.30	1.2	1.37	0.12	2.74	37.8	1.46	36	4.5	105	0.1
	0.37	1.2	1.38	0.12	2.77	38.4	1.47	36	4.4	104	0.1
	0.50	1.1	1.43	0.12	2.86	39.4	1.52	35	4.4	103	0.1
	0.60	1.1	1.47	0.12	2.93	40.6	1.56	35	4.2	101	0.1
	0.70	1.0	1.52	0.13	3.04	41.1	1.62	34	4.2	99	0.1
	0.80	0.9	1.61	0.14	3.20	41.6	1.72	32	4.1	94	0.1
	0.90	0.9	1.63	0.14	3.25	42.0	1.74	32	4.1	93	0.1
	1.00	0.9	1.66	0.14	3.29	42.2	1.77	31	4.0	90	0.1
	1.25	0.7	1.84	0.17	3.72	45.6	1.97	26	3.5	78	0.1
	1.50	0.5	2.04	0.20	4.27	52.1	2.19	21	3.1	66	0.1
	2.00	0.2	2.56	0.18	4.53	31.0	2.71	12	1.3	31	0.02
	2.50	0.1	2.98	0.23	5.22	28.3	3.16	7	0.8	18	0.01
	3.00	0.1	3.35	0.37	6.32	27.8	3.62	4	0.7	12	0.004
	3.50	0.03	3.55	0.67	6.46	30.3	3.98	2	0.6	6	0.002
Indicated	0.10	151	1.43	0.47	3.61	31.7	1.73	4,764	2,291	17,520	11
	0.22	139	1.54	0.51	3.86	33.0	1.86	4,728	2,282	17,296	10
	0.30	133	1.60	0.53	4.02	33.4	1.94	4,694	2,276	17,127	10
	0.37	128	1.65	0.55	4.12	33.6	1.99	4,663	2,271	16,988	10
	0.50	121	1.73	0.58	4.31	33.6	2.09	4,597	2,261	16,720	9
	0.60	112	1.82	0.62	4.53	33.0	2.20	4,504	2,247	16,372	8
	0.70	103	1.93	0.67	4.82	32.2	2.35	4,377	2,228	15,942	7
	0.80	92	2.07	0.74	5.18	31.5	2.53	4,219	2,200	15,416	6
	0.90	85	2.18	0.79	5.46	31.3	2.67	4,101	2,172	14,995	6
	1.00	80	2.26	0.83	5.67	31.2	2.77	4,008	2,142	14,642	6
	1.25	70	2.45	0.92	6.11	31.6	3.02	3,790	2,065	13,763	5
	1.50	63	2.61	0.98	6.44	32.1	3.21	3,605	1,981	12,976	4
	2.00	52	2.86	1.09	6.94	32.9	3.52	3,263	1,809	11,553	4
	2.50	44	3.04	1.16	7.27	33.4	3.75	2,944	1,640	10,274	3
	3.00	33	3.30	1.27	7.72	33.8	4.07	2,403	1,349	8,199	2
	3.50	22	3.65	1.41	8.43	34.3	4.51	1,739	982	5,862	2
Measured + Indicated	0.10	152	1.43	0.47	3.60	31.7	1.72	4,800	2,295	17,626	11
	0.22	140	1.54	0.51	3.85	33.0	1.85	4,764	2,286	17,401	10
	0.30	134	1.60	0.53	4.00	33.5	1.93	4,730	2,281	17,232	10
	0.37	129	1.65	0.55	4.11	33.7	1.99	4,699	2,276	17,093	10
	0.50	122	1.72	0.58	4.30	33.6	2.08	4,632	2,265	16,823	9
	0.60	114	1.81	0.62	4.51	33.1	2.20	4,539	2,251	16,473	8
	0.70	104	1.93	0.67	4.80	32.2	2.34	4,411	2,232	16,040	7
	0.80	93	2.06	0.73	5.17	31.6	2.52	4,251	2,204	15,510	7
	0.90	86	2.17	0.78	5.44	31.4	2.66	4,133	2,176	15,088	6
	1.00	81	2.26	0.82	5.64	31.4	2.76	4,039	2,146	14,733	6
	1.25	71	2.45	0.91	6.09	31.8	3.01	3,816	2,069	13,841	5
	1.50	63	2.60	0.98	6.42	32.2	3.20	3,626	1,984	13,041	4
	2.00	52	2.86	1.08	6.93	32.9	3.52	3,275	1,811	11,584	4
	2.50	44	3.04	1.16	7.27	33.4	3.75	2,951	1,640	10,291	3
	3.00	33	3.30	1.27	7.72	33.8	4.07	2,407	1,350	8,210	2
	3.50	22	3.65	1.41	8.43	34.3	4.51	1,741	983	5,868	2
Inferred	0.10	205	0.88	0.31	2.46	25.1	1.08	3,992	2,062	16,200	11
	0.22	196	0.92	0.32	2.53	25.4	1.12	3,972	2,034	15,958	11
	0.30	190	0.94	0.33	2.59	25.5	1.15	3,947	2,010	15,775	11
	0.37	179	0.99	0.34	2.68	25.4	1.20	3,887	1,963	15,418	10
	0.50	158	1.07	0.37	2.87	24.8	1.30	3,721	1,877	14,572	9
	0.60	141	1.14	0.40	3.06	24.6	1.39	3,549	1,803	13,866	8
	0.70	126	1.21	0.43	3.24	24.5	1.48	3,363	1,731	13,105	7
	0.80	110	1.30	0.47	3.50	24.7	1.59	3,129	1,656	12,340	6
	0.90	96	1.38	0.51	3.73	25.0	1.69	2,919	1,573	11,558	5
	1.00	88	1.43	0.53	3.87	25.1	1.76	2,772	1,495	10,909	5
	1.25	68	1.58	0.59	4.22	25.6	1.95	2,375	1,297	9,231	4
	1.50	50	1.75	0.68	4.56	27.1	2.16	1,908	1,080	7,258	3
	2.00	22	2.19	0.88	5.27	30.1	2.73	1,064	621	3,728	1
	2.50	12	2.56	0.99	5.84	34.5	3.16	678	382	2,258	0.9
	3.00	6	2.96	1.11	6.27	39.9	3.63	375	204	1,159	0.5
	3.50	3	3.24	1.28	6.70	39.0	4.01	208	119	627	0.3

Figure 14.4 Hugo North Extension – Grade-Tonnage Curves at Various Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)

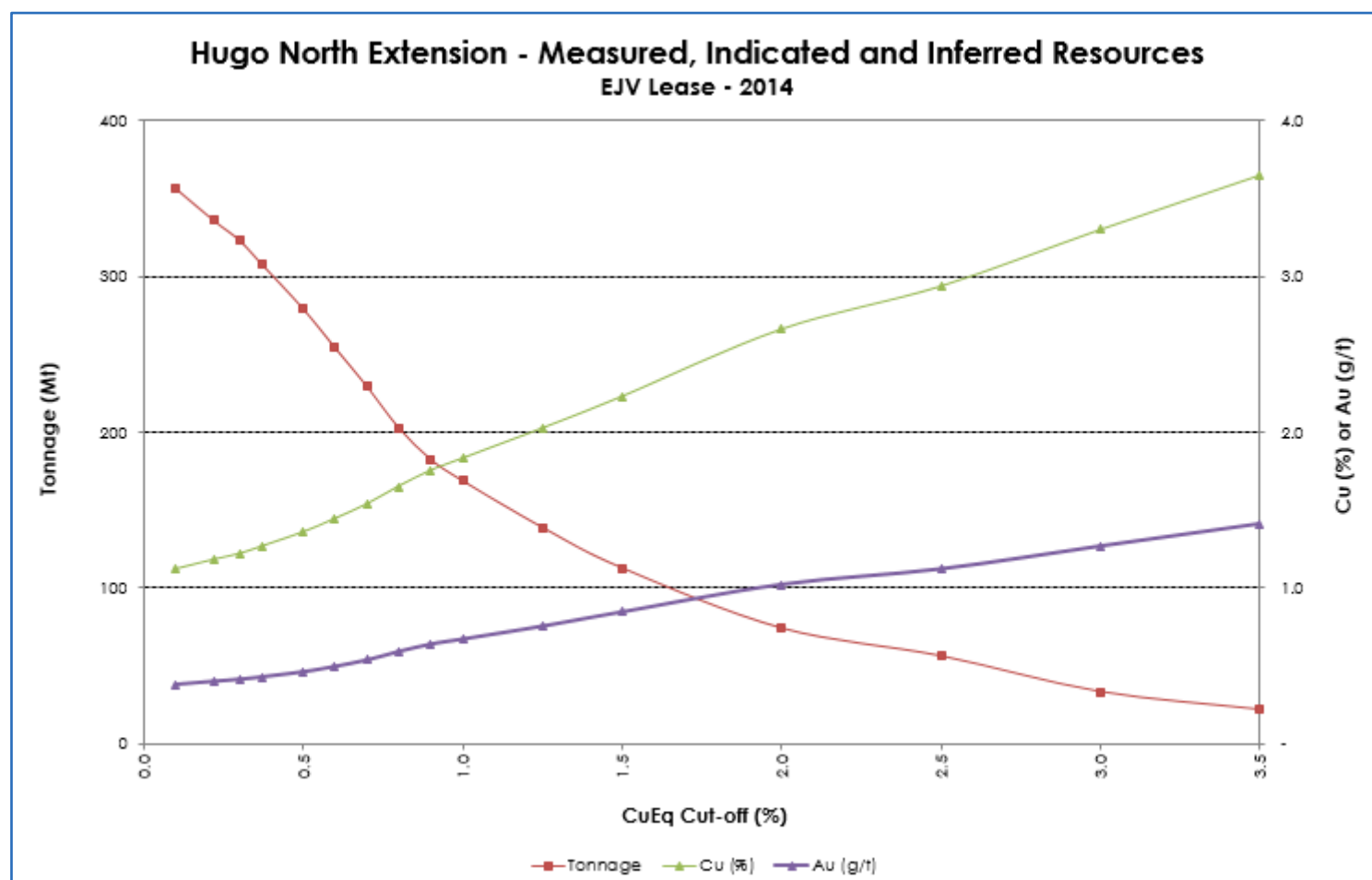


Table 14.36 Hugo South – Grade and Tonnage Calculations at Variable Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured	0.10	1,700	0.49	0.05	1.39	54.0	0.53	18,436	2,843	75,720	202
	0.22	1,372	0.58	0.06	1.51	59.1	0.62	17,409	2,561	66,743	179
	0.30	1,085	0.67	0.06	1.64	63.4	0.72	15,952	2,212	57,359	152
	0.37	845	0.77	0.07	1.78	66.4	0.83	14,372	1,861	48,406	124
	0.50	567	0.96	0.08	2.06	67.5	1.02	12,004	1,416	37,613	84
	0.60	466	1.06	0.08	2.20	66.6	1.13	10,896	1,220	33,060	68
	0.70	392	1.15	0.08	2.31	64.8	1.22	9,911	1,056	29,119	56
	0.80	303	1.28	0.09	2.47	60.3	1.35	8,551	838	24,071	40
	0.90	233	1.43	0.09	2.65	55.4	1.50	7,354	646	19,834	28
	1.00	198	1.53	0.09	2.77	53.2	1.60	6,670	549	17,575	23
	1.25	127	1.79	0.09	3.16	49.1	1.87	5,009	387	12,894	14
	1.50	70	2.18	0.12	3.81	46.3	2.28	3,380	274	8,628	7
	2.00	39	2.64	0.15	4.59	44.5	2.76	2,269	186	5,763	4
	2.50	23	2.99	0.17	5.08	43.8	3.13	1,493	126	3,706	2
	3.00	12	3.35	0.21	5.57	41.1	3.52	857	77	2,078	1
	3.50	5	3.68	0.25	6.17	40.7	3.87	428	42	1,046	0.5

Figure 14.5 Hugo South – Grade-Tonnage Curves at Various Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)

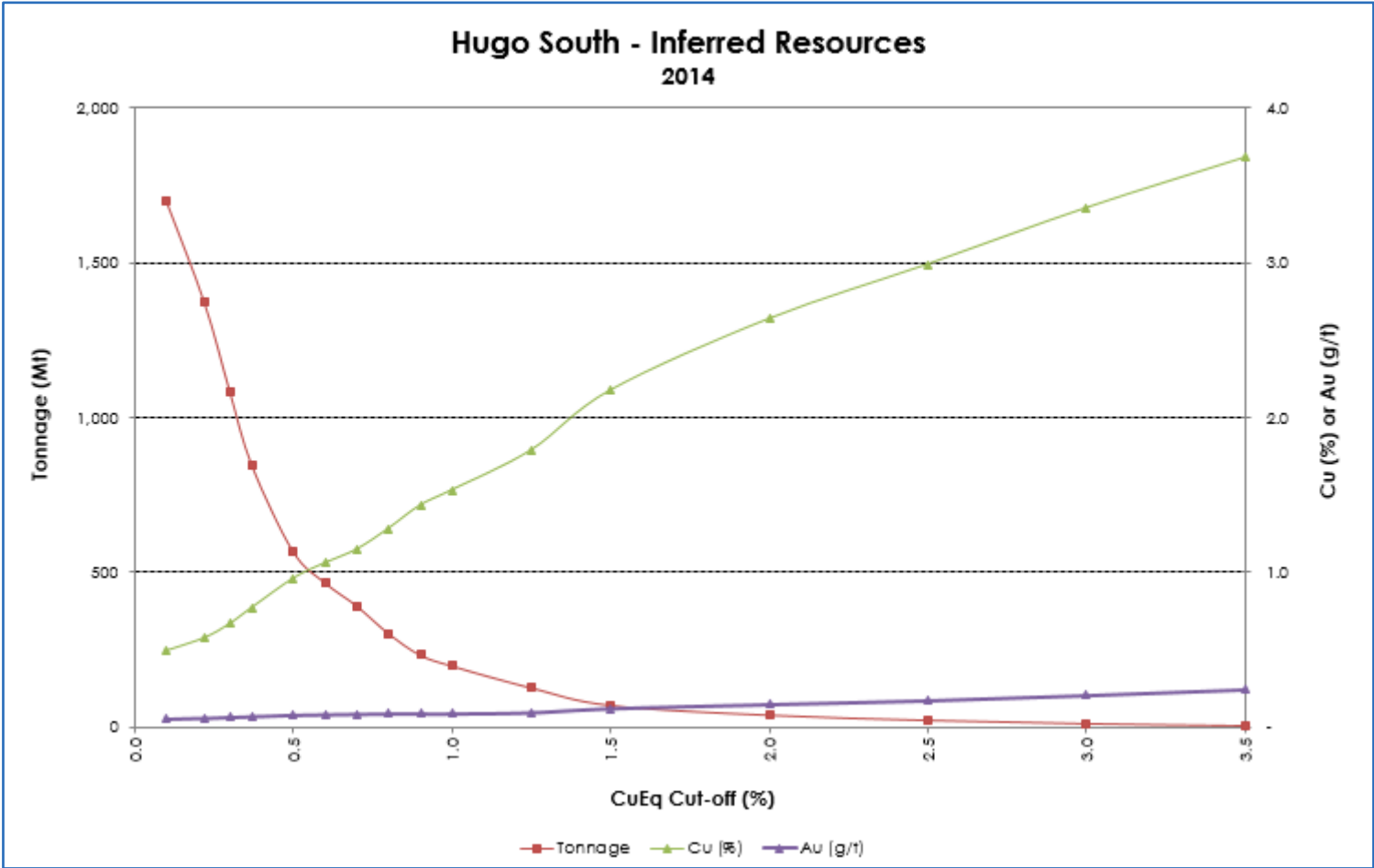
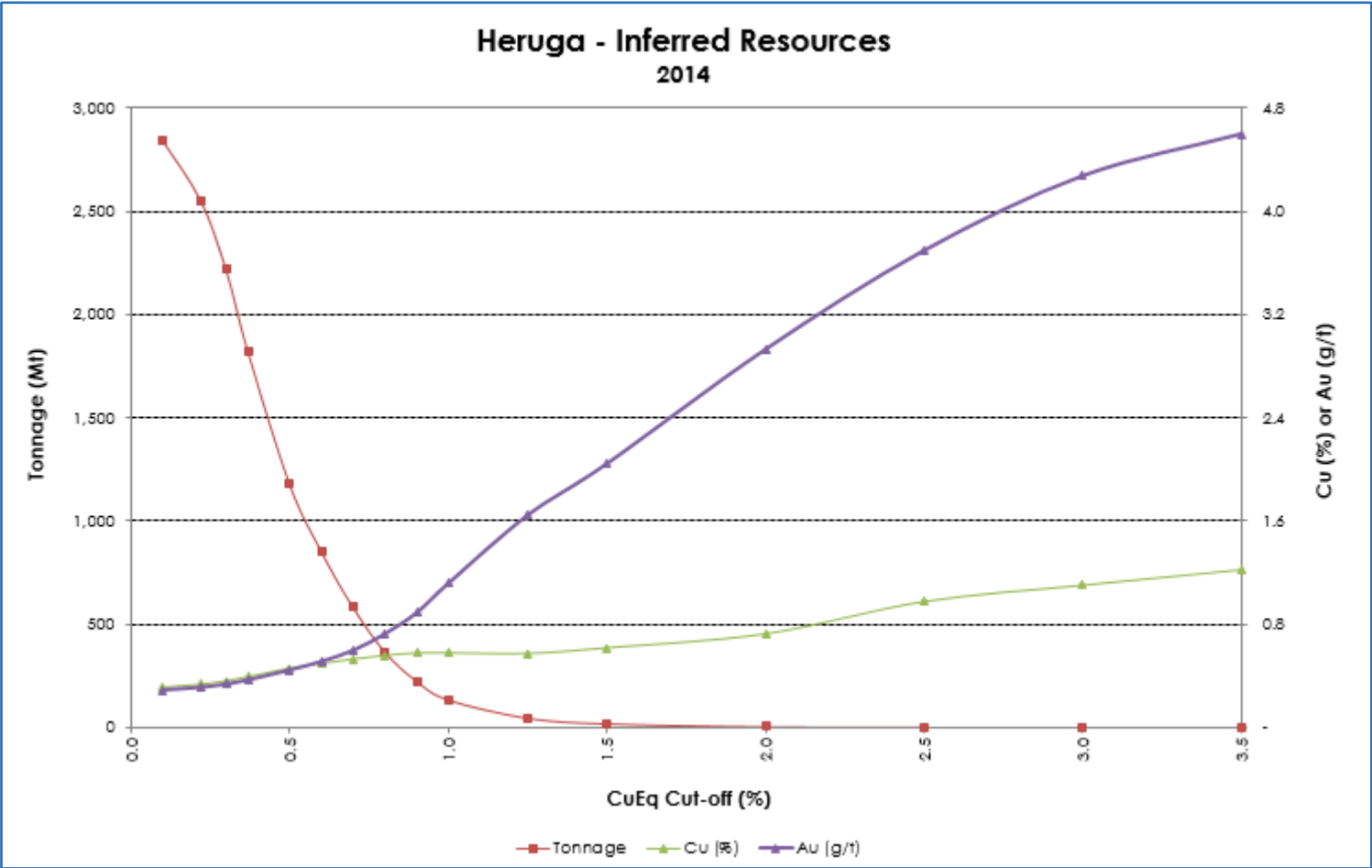


Table 14.37 Heruga – Grade and Tonnage Calculations at Variable Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)

CLASS	CuEq Cut-off (%)	Tonnage (Mt)	Grade					Contained Metal			
			Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)
Measured	0.10	2,845	0.31	0.29	1.17	89.5	0.50	19,347	26,298	106,930	561
	0.22	2,554	0.33	0.31	1.24	96.3	0.54	18,699	25,405	102,211	542
	0.30	2,219	0.36	0.33	1.31	103.4	0.58	17,505	23,862	93,758	506
	0.37	1,816	0.39	0.37	1.40	113.0	0.64	15,647	21,508	81,774	453
	0.50	1,180	0.46	0.44	1.57	132.5	0.75	11,850	16,841	59,492	345
	0.60	854	0.49	0.51	1.69	142.8	0.83	9,279	14,004	46,445	269
	0.70	583	0.53	0.60	1.81	152.1	0.92	6,768	11,173	33,861	195
	0.80	367	0.55	0.72	1.92	156.0	1.02	4,484	8,531	22,566	126
	0.90	224	0.57	0.89	2.00	152.6	1.13	2,834	6,404	14,419	75
	1.00	134	0.57	1.12	2.07	134.3	1.25	1,694	4,794	8,906	40
	1.25	45	0.57	1.64	2.39	112.6	1.53	568	2,386	3,464	11
	1.50	18	0.61	2.04	2.99	137.1	1.81	238	1,153	1,688	5
	2.00	3	0.72	2.93	3.89	99.7	2.40	49	292	389	1
	2.50	1	0.97	3.69	4.02	71.5	3.07	16	89	97	0.1
	3.00	0.3	1.10	4.27	4.03	73.0	3.52	8	47	44	0.1
	3.50	0.2	1.22	4.60	4.42	74.8	3.82	4	22	22	0.03

Figure 14.6 Heruga – Grade-Tonnage Curves at Various Copper-Equivalent Cut-off Grades
(Unchanged since 2014 OTTR)



14.1.10 Model Validation

14.1.10.1 Oyut

Validation was completed at the time of updating the model in 2011.

A visual validation of the variogram models and search ellipsoids was completed as a check for reasonableness against grade trends and grade shells. These checks were performed for copper and gold in the major domains at a minimum and for less-significant domains when deemed appropriate.

Checks were also performed on the ID2 bulk density model. A visual review of the density model was completed on 15 m level plans and 50 m east-west vertical sections. The model slices were displayed together with the 8 m bulk density composite database. The composites and estimated blocks were scrutinized to identify blocks with unreasonable density values compared to the composite values. The ID2 block model estimate, the NN interpolation, and bulk density composites were compared to assess global bias. Mean density value comparison charts and swath plots were also prepared to assist in validating the density estimation.

A NN interpolated model was created to approximate the declustered composite distribution. This NN model was used to check for global bias of the block grade estimates above zero cut-off and for local bias using swath plots. The NN model was also used as the first step in checking model smoothing, using volume-variance corrections from the composite scale to the SMU scale.

Visual checks were completed comparing composites to blocks as well as behaviour of grade estimates near firm boundaries. Histograms of blocks with and without firm boundaries were compared to assess the impact on removal of troughs that were not seen in the composite histograms.

A final set of checks was performed to compare the 2005 model to the 2011 model and to compare elements of the 2007 model to the 2011 model:

- Examination of dilution inside the 2005 and 2011 models, first, by assuming whole-block dilution, (i.e. selective mining cannot occur on a scale finer than 20 m x 20 m x 15 m) and, second, by assuming selective mining of dykes as was applied in 2005, although 100% of the dykes interpreted in 2011 were assumed to be 'selectively mineable' for the purposes of those dilution calculations.
- Comparison of the 2007 and 2011 model copper and gold grade maps on level plans. The most notable areas of grade change between the 2007 and 2011 models occurs at the grade shell margins; often for only a single block width.
- Comparison of the 2007 and 2011 model 'grade shell difference maps' for copper and gold on plans and sections. Very significant differences were noted where the grade shell strategy had changed, e.g., the creation of Cu grade shells in Southwest and Far South zones in 2011 where none were used in 2007, and of 0.3 g/t Au grade shells in Southwest and Far South zones where only 0.7 g/t Au shells had been used in 2007. Outside of these areas, the 2011 grade shells generally appear to be slightly more conservative than the 2007 grade shells.

- Block model estimates were checked for global bias by comparing the average metal grades (with no cut-off) from the model (OK) with means from NN estimates. Results showed a good relationship (Table 14.38).
- Evaluation of the 2007 versus 2011 volumes of each primary lithology in 30 m swathes for blocks classified as Measured or Indicated in 2007. Results indicated an acceptable comparison.
- Comparison of 2007 versus 2011 dyke volumes by calculating directly from percent variables. The 2011 dyke volume falling within the designated pit was 3%–4% higher than in 2007, with 63.8 Mm³ in 2007 and 65.8 Mm³ in 2011. A visual plan-view comparison of dyke difference between 2007 and 2011 was also completed.
- Examination of histograms of 2005 block grades, 2011 block grades, and 2011 composites for Cu and Au in select domains. The histograms display the reduction in 2011 of troughs seen in the 2005 block distribution that were not observed in the composite distributions.
- Review of plans and sections of classification differences where the block class was '1' or '2' in the 2005 model but not the same as assigned in the 2011 model, or vice versa, or where both models shared the same values. Overall volumes were similar, though the 2005 classification was considered to be more liberal near the grade shell boundaries, and it was determined that some of the documented classification criteria may not have been strictly followed in the 2005 classification.

A block-by-block comparison of the resource model to the current ore control (OC) model indicates that the two models are in good agreement. The resource model appears to be a global predictor of tonnes above the range of cut-offs. At all grade ranges, the resource model is within 1% agreement with the OC model. The comparison does however indicate a slight reduction (–1%) in tonnes above 0.6% copper. However, this grade category represents only 3% of the total tonnes.

Reconciliation of Resource Model to Grade Control Model

A block-by-block comparison of copper grades in the OC model to the grade of the resource model indicates that, at Cu >0% (in both models), the OC grade is approximately +6.6% (relative) higher, (i.e. 0.306% Cu in the OC model versus 0.287% Cu in the resource model). As the cut-off is raised, the observed variance (var) also changes from a high of 12% at a cut-off of 0.1% Cu (var=0.431 OC model versus var=0.381 resource model) to a low of –3% at a cut-off of 0.9% Cu (var=0.960 OC model versus var=0.989). Again, the negative variations of grade at the higher grade ranges should be balanced by the relative tonnes in this range.

Model reconciliation will continue to be used to validate and improve the resource estimations based on actual production data.

Ore control activities undertaken through the mining process in the Oyut open pit enabled a further validation of the Oyut resource model. Key findings include:

- Overall tonnage increased by +4% from the resource model to OC model, which includes:
 - Resource model grade has been changed by the grade control estimation (–7%) to reflect a change in ore tonnes, i.e. ore converting to waste. Note, that this change

- includes an upgrade of some waste to ore. The downgrade of tonnage also includes ore loss and dilution during the mining operations.
- Additional Inferred Mineral Resources have been mined (+1%). Those Inferred volumes were compiled by operational cut-off grades, which is the same as the declared Ore Reserve cut-off.
 - Unclassified oxide materials have been mined from the upper benches of the Phase 3 pit and subsequently milled as sulphide ore (+10%).
 - The milled ore tonnage matches with the OC model.
 - Contained copper metal is increased by +13% in the OC model compared to the resource model, which includes:
 - Approximately +2% contained copper metal increase observed in the OC model due to grade increase of resources.
 - Approximately +1% contained copper metal increase observed due to Inferred material having been mined using operational cut-off.
 - Approximately +10% contained copper metal increase observed as unclassified oxide materials have been mined from upper benches in Phase 3 of the pit.
 - The mill confirms the additional metal in the OC model, with an additional +5% copper metal.
 - There is a +6% increase in contained gold metal in the OC model relative to the resource model:
 - Approximately +4% contained gold metal increase observed in the OC model due to grade increase of Reserves.
 - Approximately +2% contained gold metal increase observed as Unclassified oxide materials have been mined.
 - The mill is indicating a close agreement on contained gold metal predicted by the OC model.

Highlights of the resource to mill reconciliation are shown in Figure 14.7 to Figure 14.9.

Figure 14.7 Resource to Mill Reconciliation Highlights (Tonnes)

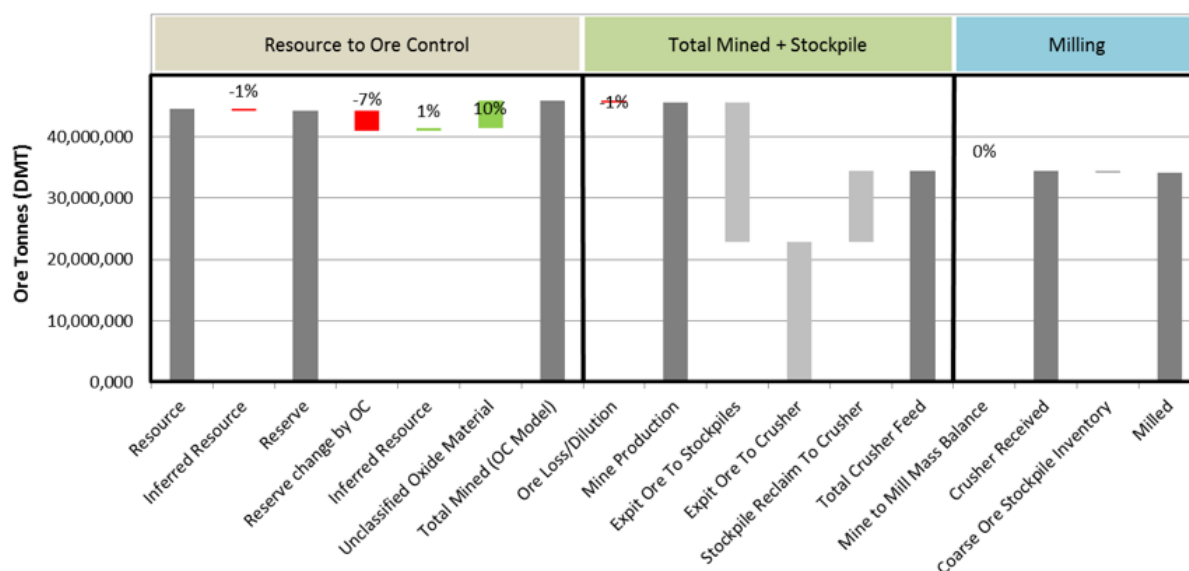


Figure 14.8 Resource to Mill Reconciliation Highlights (Copper Metal)

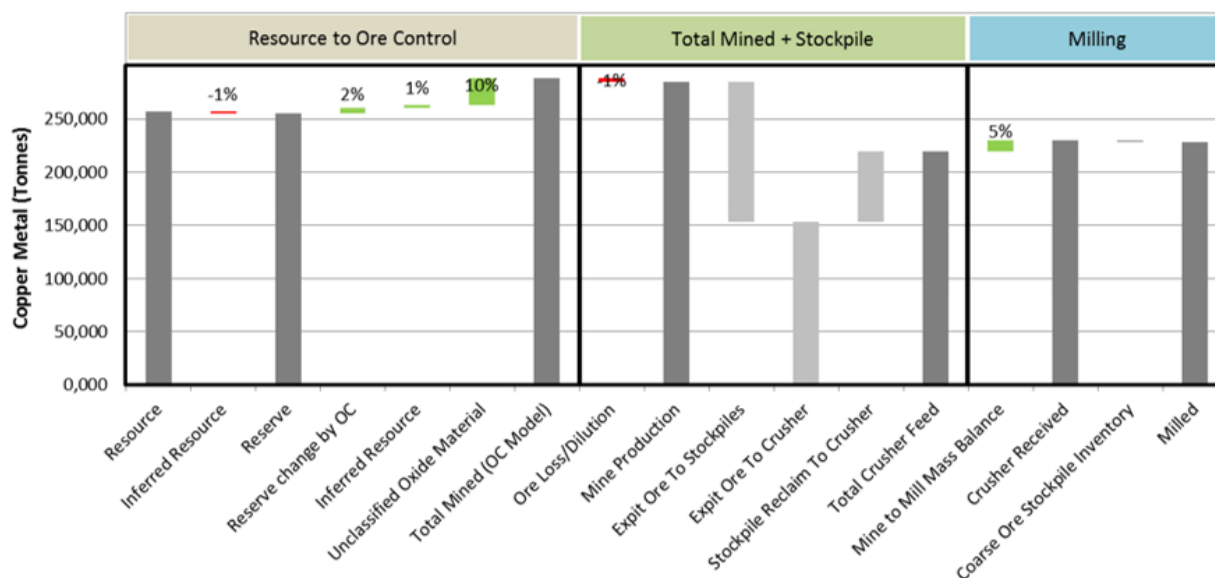
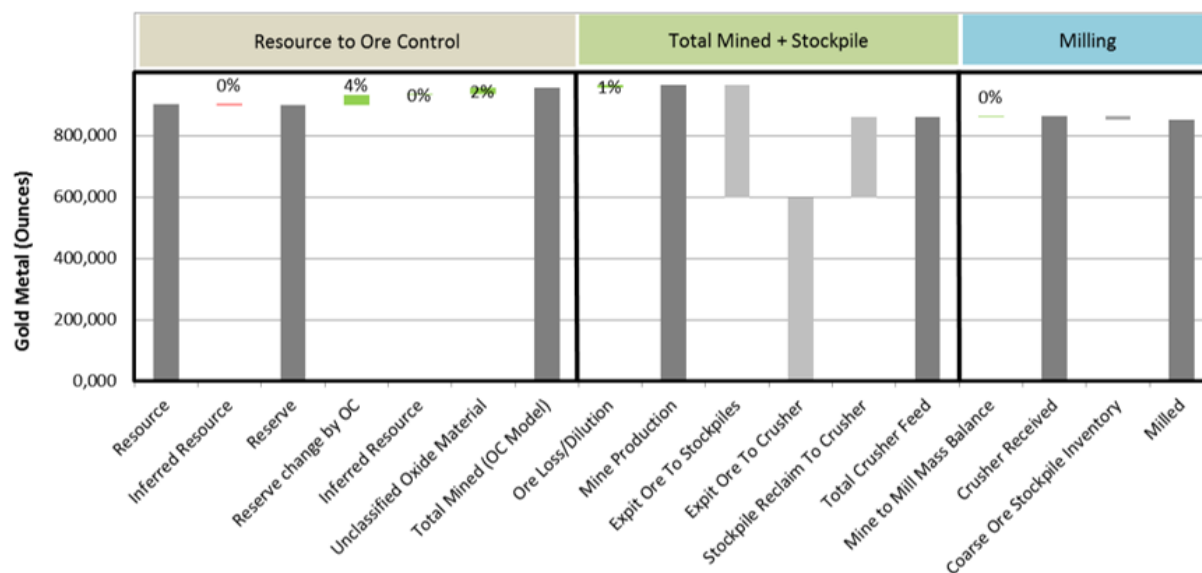


Figure 14.9 Resource to Mill Reconciliation Highlights (Gold Metal)



Detailed investigations are ongoing to compare resource to reserve, reserve to ore control, and ore control to mill. Model reconciliation will continue to validate and improve the resource estimations based on actual production data.

Table 14.38 Oyut Global Model Mean Grade Values by Domain in each Zone (NN vs. OK Estimates)

Domain/Zone	NN Estimate	OK Estimates	% Difference
Cu (%) –Oyut Zones			
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	0
Wedge Cu Shell	0.501	0.485	-3.2
Wedge background	0.113	0.116	+2.6
Au (g/t) – Oyut Zones			
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	0
Wedge	0.054	0.052	-3.8

14.1.10.2 Hugo North

Detailed visual validation of the Hugo North block model was performed 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 addition of the outlier restriction values succeeded in minimizing grade smearing.

Block model estimates were checked for global bias by comparing the average metal grades (with no cut-off) from the model (OK) with means from NN estimates. Results showed a good relationship (Table 14.39).

Models were also checked for local trends in the grade estimates (swath plot). This was undertaken by plotting the mean values from the NN estimate versus the OK results for benches in 30 m swathes and for northings and eastings in 40 m swathes.

The OK estimate would be expected to be smoother than the NN estimate, thus the NN estimate should fluctuate around the kriged estimate on the plots. The two trends behaved as predicted and showed no significant trends of copper or gold in the estimates.

Swath plots of uncapped copper estimations along easting, northing, and elevation are shown in Figure 14.10. The same information for gold estimation is shown in Figure 14.11.

Table 14.39 Hugo North Global Model Mean Grade Values by Domain in each Zone (NN vs. OK Estimates)

Domain/Zone	NN Estimate	OK Estimates	% Difference
Cu (%) – Hugo North			
All Zones	0.896	0.901	-1.0
Qtz-vein Domain	2.712	2.697	-0.5
0.6% Cu Domain	0.938	0.915	0.8
Cu background (outside 0.6%)	0.289	0.289	0.0
Au (g/t) – Hugo North			
All Zones	0.255	0.252	-2.6
1.0 g/t Au Zone	1.291	1.243	-2.1
0.3 g/t Au Zone	0.504	0.530	-3.7
Au background	0.127	0.117	-7.8

Figure 14.10 Swath Plot Comparison of Hugo North Kriged Nearest Neighbour Cu Estimates (uncapped) and 5 m Cu Composites with Depth

(Va+Ign+Qmd+BiGd)

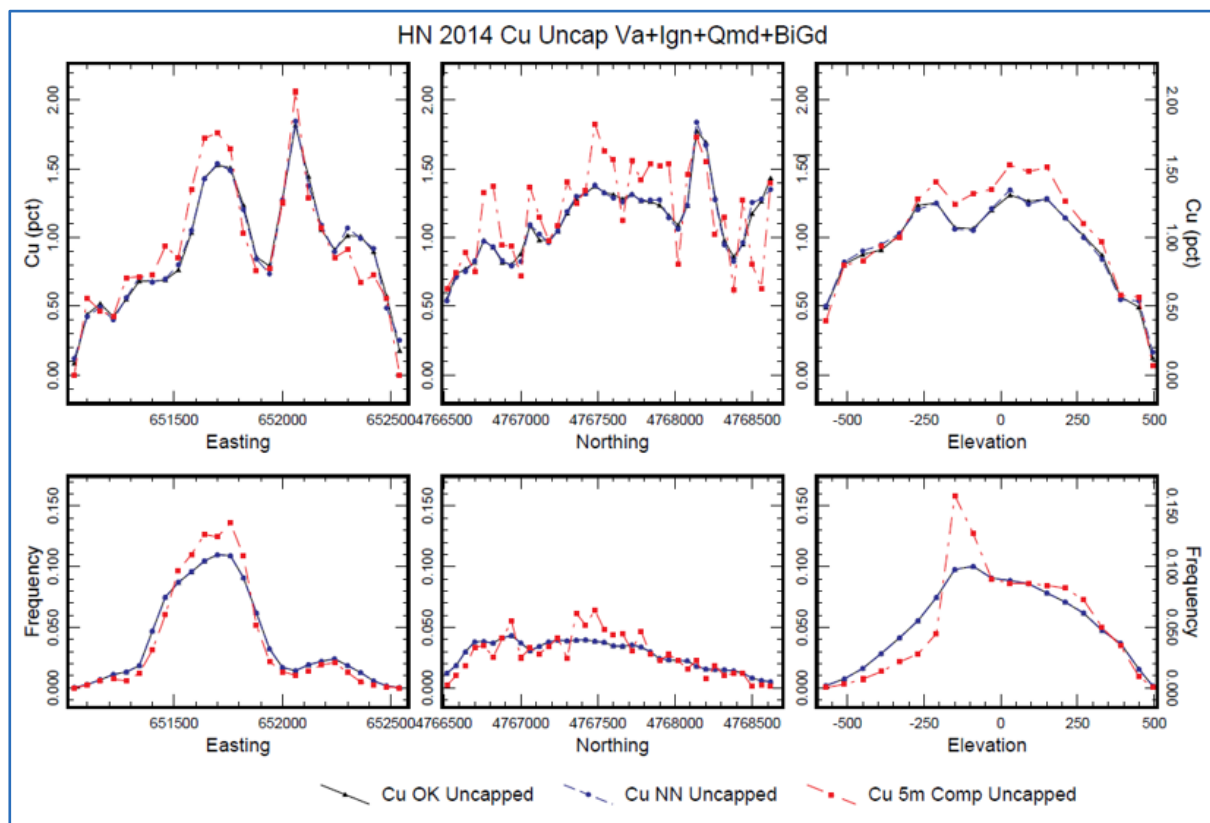
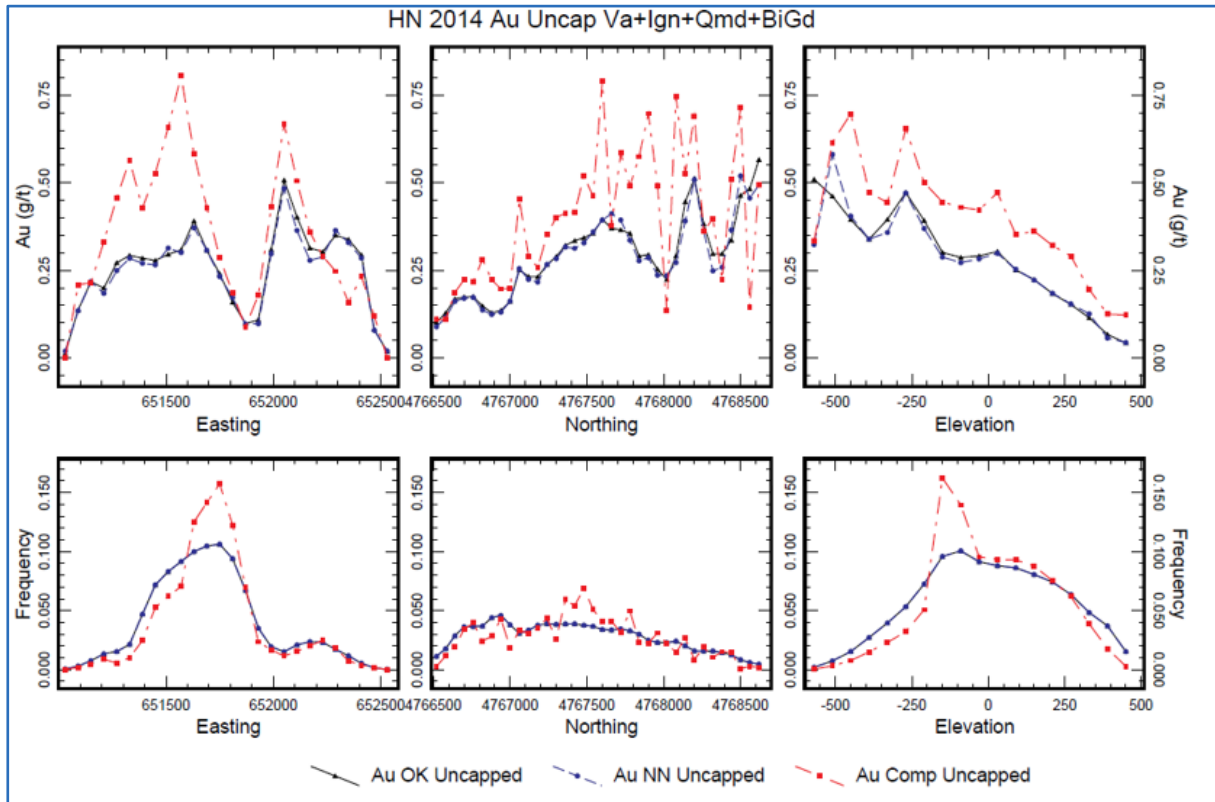


Figure 14.11 Swath Plot Comparison of Hugo North Kriged Nearest Neighbour Au Estimates (uncapped) and 5 m Au Composites with Depth
(Va+Ign+Qmd+BiGd)



14.1.10.3 Hugo South

Visual inspection of the estimates indicated no significant issues with the model. The smoothing in the estimates was checked independently. Grade-tonnage predictions produced for the model show that grade and tonnage estimates were validated by the change-of-support calculations over the likely range of mining grade cut-off values (0.8%–1.2% Cu).

The block model estimates were checked for global bias by comparing the OK grades (with no cut-off) from the model with the NN estimates. No bias was identified.

Local trends in the grade estimates were verified using swath plots. The trends behaved as predicted and showed no significant bias in the estimates.

14.1.10.4 Heruga

A detailed visual validation of the Heruga resource model found that flagging of the drill data file and the block model was performed correctly. The block model estimates were checked for global bias by comparing the average metal grades from the model with means from unrestricted NN estimates. No bias was identified.

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 NN model. In all cases, although smoothed due to the kriging interpolation method, the model was found to reflect the underlying data used to build it. The degree of smoothing occurring within the model was considered reasonable for the type of deposit and the likely block cave mining method.

The resource model was also checked for trends and local bias using 50 m swath plots that compared the restricted OK estimates to NN estimates. The trends behaved as predicted and showed no significant bias in the estimates.

14.1.11 Mineral Resource Confidence Classification

An initial classification was assigned using a set of rules based on estimation (OK) passes, distances to nearest composites, and numbers of holes used to inform the block estimates. A separate ID2 estimation was run for copper grade composites at Oyut, which was used to derive the initial classification in the model. At Oyut, the initial block classification was then assessed and modified with an algorithm designed to eliminate isolated blocks of a particular classification by 'switching' them to majority classification of the surrounding blocks.

At Hugo North, block classification confidence is based on the copper grade variable. A three pass ID2 estimation of Cu composites was used to capture the distance from a block centroid to the nearest composites; the closest anisotropic distance was captured from Pass 1 and Pass 2, and the closest Cartesian distance was captured from Pass 3. For the ID2 estimation the principal mineralized rock types Va, Ign, and Qmd were treated as one group, and unmineralized rock types BiGd and Rhy were treated as another group. Contacts between these two groups were treated as hard. The HWS rocks were not classified. The seven search anisotropy zones used for 2012 copper grade estimation were used for the ID2 estimation.

At Hugo North, block confidence classification is based on three processes: preliminary block classification using a script based on distance to a drillhole and number of drillholes used to estimate a block, generation of probability model for the three confidence categories, and manual 'cleaning' using polygons generated in sectional view.

A series of probability models were generated using the preliminary classification code of '1' for Measured, '2' for Indicated, and '3' for Inferred. Using a threshold value of 50%, the probability shells were compared to the preliminary classification block code. Boundary polygons reflecting the three categories were then manually digitized to eliminate the inclusion of isolated blocks and incorporate geologic and grade continuity. The probability shells were used as a guide for confidence.

The polygons were then connected to create a 3D solid. Blocks were then recoded as Measured, Indicated, or Inferred based on these solids.

Mineral Resources were classified based on the criteria outlined in Section 14.3.1.

14.2 Assessment of Reasonable Prospects for Economic Extraction

14.2.1 Copper Equivalence Formula

In order to assess the value of the total suite of minerals of economic interest in the mineral inventory, formulae have been developed to calculate copper equivalency (CuEq) based on given prices and recoveries.

14.2.1.1 2014 Formula Derivation

The base 2014 copper-equivalent formula incorporates copper, gold, silver, and molybdenum. The assumed metal prices are US\$3.01/lb for copper, US\$1,250/oz for gold, US\$20.37/oz for silver, and US\$11.90/lb for molybdenum. Copper is expressed in block grade in the form of percentages (%). Gold and silver are expressed in block grades in the form of grams per tonne (g/t). Molybdenum is expressed in block grades in the form of parts per million (ppm). Metallurgical recovery for gold, silver, and molybdenum are expressed as percentage relative to copper recovery.

The unit conversions used in the calculation are:

1 tonne = 1 million grams

grams per tonne (g/t) to ounces per tonne (oz/t) = 31.103477

pounds per kilogram (lb/kg) = 2.20462

tonne to pounds (lb) = 2,204.62

This leads to a base formula of:

$$\text{CuEq}_{14} = \text{Cu} + ((\text{Au} \times \text{AuRev}) + (\text{Ag} \times \text{AgRev}) + (\text{Mo} \times \text{MoRev})^{\dagger}) / \text{CuRev}$$

[†] Mo and MoRev are only incorporated into CuEq calculations for Heruga

Where:

$$\text{CuRev} = (3.01 \times 22.0462)$$

$$\text{AuRev} = (1,250/31.103477 \times \text{RecAu})$$

$$\text{AgRev} = (20.37/31.103477 \times \text{RecAg})$$

$$\text{MoRev} = (11.90 \times 0.00220462 \times \text{RecMo})$$

$$\text{RecAu} = \text{Au recovery} / \text{Cu Recovery}$$

$$\text{RecAg} = \text{Ag recovery} / \text{Cu Recovery}$$

$$\text{RecMo} = \text{Mo recovery} / \text{Cu Recovery}$$

Different metallurgical recovery assumptions lead to slightly different copper-equivalent formulas for each of the deposits; these are outlined in the following tables for Oyut, Hugo North, Hugo North Extension, Hugo South, and Heruga. In all cases, the metallurgical recovery assumptions are based on metallurgical testwork. For Oyut, actual mill performance has been used to further refine the recovery assumptions. Recoveries are relative to copper because copper contributes the most to the equivalent calculation.

All elements included in the copper-equivalent calculation have a reasonable potential to be recovered and sold, except for molybdenum. Molybdenum grades are only considered high enough to support construction of a molybdenum recovery circuit at Heruga, and hence the recoveries of molybdenum are zeroed out for the other deposits.

Table 14.40 Oyut – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.794	0.704	0.754	0		
	Recovery Relative to Cu	1	0.887	0.949	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.537	35.63
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	0.45				0.45	29.86
	Au Grade		0.31			0.166	11.05
	Ag Grade			1.23		0.012	0.76
	Mo Grade				0	0	–
	CuEq Grade & Revenue	0.45	0.31	1.23	0.	0.628	41.67

From Table 14.40 above, the base formula is adjusted as follows:

$CuEq_{14}(Oyut) =$

$Cu + ((Au \times 1,250 \times 0.0321507 \times 0.887) + (Ag \times 20.37 \times 0.0321507 \times 0.949)) / (3.01 \times 22.0462)$

Table 14.41 Hugo North – Copper Equivalence Assumptions and Calculation

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.92	0.83	0.86	0		
	Recovery Relative to Cu	1	0.906	0.941	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.549	36.43
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.66				1.66	110.16
	Au Grade		0.34			0.187	12.38
	Ag Grade			3.37		0.031	2.08
	Mo Grade				27.43	0	–
	CuEq Grade & Revenue	1.66	0.34	3.37	27.43	1.878	124.62

From Table 14.41 above, the base formula is adjusted as follows:

$$\text{CuEq}_{14(\text{HN})} =$$

$$\text{Cu} + ((\text{Au} \times 1,250 \times 0.0321507 \times 0.906) + (\text{Ag} \times 20.37 \times 0.0321507 \times 0.941)) / (3.01 \times 22.0462)$$

Table 14.42 Hugo North Extension – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.92	0.84	0.86	0.00		
	Recovery Relative to Cu	1.00	0.913	0.942	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.553	36.69
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.59				1.59	105.51
	Au Grade		0.55			0.304	20.18
	Ag Grade			3.72		0.035	2.29
	Mo Grade				25.65	0	–
	CuEq Grade & Revenue	1.59	0.55	3.72	25.65	1.929	127.98

From Table 14.42 above, the base formula is adjusted as follows:

$$\text{CuEq}_{14(\text{HNE})} =$$

$$\text{Cu} + ((\text{Au} \times 1,250 \times 0.0321507 \times 0.913) + (\text{Ag} \times 20.37 \times 0.0321507 \times 0.942)) / (3.01 \times 22.0462)$$

Table 14.43 Hugo South – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.89	0.81	0.85	0		
	Recovery Relative to Cu	1	0.909	0.945	0		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.551	36.54
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	1.07				1.07	71.00
	Au Grade		0.06			0.033	2.19
	Ag Grade			2.07		0.019	1.28
	Mo Grade					0	–
	CuEq Grade & Revenue	1.07	0.06	2.07		1.122	74.48

From Table 14.43 above, the base formula is adjusted as follows:

$$\text{CuEq}_{14(\text{HS})} =$$

$$\text{Cu} + ((\text{Au} \times 1,250 \times 0.0321507 \times 0.909) + (\text{Ag} \times 20.37 \times 0.0321507 \times 0.945)) / (3.01 \times 22.0462)$$

Table 14.44 Heruga – Copper Equivalence Assumptions and Calculation based on Average Grades

		Cu	Au	Ag	Mo		
	Metal Price (US\$)	3.01/lb	1,250/oz	20.37/oz	11.90/lb		
	Recovery (%)	0.86	0.79	0.82	0.635		
	Recovery Relative to Cu	1	0.911	0.949	0.736		
	Conversion Factor	22.0462	0.0321507	0.0321507	0.0022046		

		Cu%	Au g/t	Ag g/t	Mo ppm	CuEq%	US\$/t
Assumed Grade	Cu Credit	1				1	66.36
	Au Credit		1			0.552	36.61
	Ag Credit			1		0.009	0.62
	Mo Credit				1	0	0.03
Average Grade of Deposit	Cu Grade	0.42				0.42	27.87
	Au Grade		0.41			0.226	15.01
	Ag Grade			1.47		0.014	0.91
	Mo Grade				138.47	0.055	2.67
	CuEq Grade & Revenue	0.42	0.41	1.47	138.47	0.70	46.47

From Table 14.44 above, the base formula is adjusted as follows:

$$\text{CuEq}_{14}(\text{HERUGA}) = \text{Cu} + ((\text{Au} \times 1,250 \times 0.0321507 \times 0.911) + (\text{Ag} \times 20.37 \times 0.0321507 \times 0.949) + (\text{Mo} \times 11.9 \times 0.0022046 \times 0.736)) / (3.01 \times 22.0462)$$

14.2.2 Derivation of Cut-off Grades

Cut-off grades were determined using Base Data Template 31 (BDT31) assumptions. The Net Smelter Return (NSR) per tonne of ore needs to equal or exceed the production cost of a tonne of ore for the mine to break even or make money.

For the underground mine, the break-even cut-off grade needs to cover the costs of mining, processing, and general and administrative (G&A). A NSR of US\$15.34/t would be required to cover costs of US\$8.00/t for mining, US\$5.53/t for processing, and US\$1.81/t for G&A. This translates to a CuEq break-even underground cut-off grade of approximately 0.37% CuEq for Hugo North ore. This cut-off grade has been used for tabulating underground Mineral Resources in this report.

In the open pit, the mining cost for a tonne of waste is the same as for a tonne of ore. Therefore, the marginal open pit cut-off grade is determined where a tonne of material covers the processing and G&A. A NSR of US\$8.36/t would be required to cover process costs of US\$6.20 and G&A of US\$2.16/t. This NSR translates to a copper-equivalent marginal open pit cut-off grade of 0.22% for Southwest zone ore. This cut-off grade has been used for tabulating open pit Mineral Resources in this report.

14.2.3 Reasonable Prospects for Eventual Economic Extraction

CIM Definition Standards require reported Mineral Resources to have reasonable prospects for eventual economic extraction. The following subsections address the reasonable prospects for eventual economic extraction together with commentary on conceptual mining considerations and other constraints used in tabulating the Mineral Resources.

Constraining 3D shapes were generated by ascribing positive values that defray mining, processing, and G&A costs to blocks that have been assigned to Measured, Indicated, and Inferred (MII) Mineral Resource categories.

14.2.3.1 Open Pit Mineral Resources Constraints

In 2014, OT LLC undertook pit optimization that was used to confirm the current reserve pit designs. The parameters used for the pit optimization work are described in BDT31. The metal prices used were: copper price of US\$3.01/lb, silver price of US\$20.37/oz, and gold price of US\$1,250/oz. A pit shell was generated from the pit optimization work using all the estimated blocks classified as Measured, Indicated, or Inferred. The result was a pit shell (MII Pit Shell) that was larger than the reserve pit design to be used to constrain the estimates that would go into the Oyut Mineral Resource reporting.

The reserve pit design contains Inferred Mineral Resources that are ignored in the declaration of Mineral Reserves. Given that this Inferred material is located within the reserve pit, it is considered to be amenable to mining and, provided the Inferred blocks are above the economic cut-off, they are considered to have reasonable prospects of future economic extraction and thus can be stated as Mineral Resources.

Using the BDT31 prices and other parameters, a marginal copper-equivalent cut-off of 0.22% CuEq was calculated for Oyut. This cut-off grade represents the point at which the copper revenue is equal to the processing and G&A costs. The use of a marginal cut-off is based on industry standard practice. It is calculated assuming that there is a decision point at the pit rim, if the revenue from processing is less than the cost then the block is assumed to be waste. As a result, the Oyut open pit Mineral Resources are tabulated at a marginal cut-off grade of 0.22% CuEq.

Blocks coded as being within the oxide horizon are excluded from the Mineral Resource on the basis of being considered to be mineralogically unfavourable within the existing processing options.

Mining depletion has also been accounted for by using the 2015 year-end mining surface to constrain Inferred Mineral Resource material reported within the reserve pit design.

14.2.3.2 Oyut Underground Mineral Resource Constraints

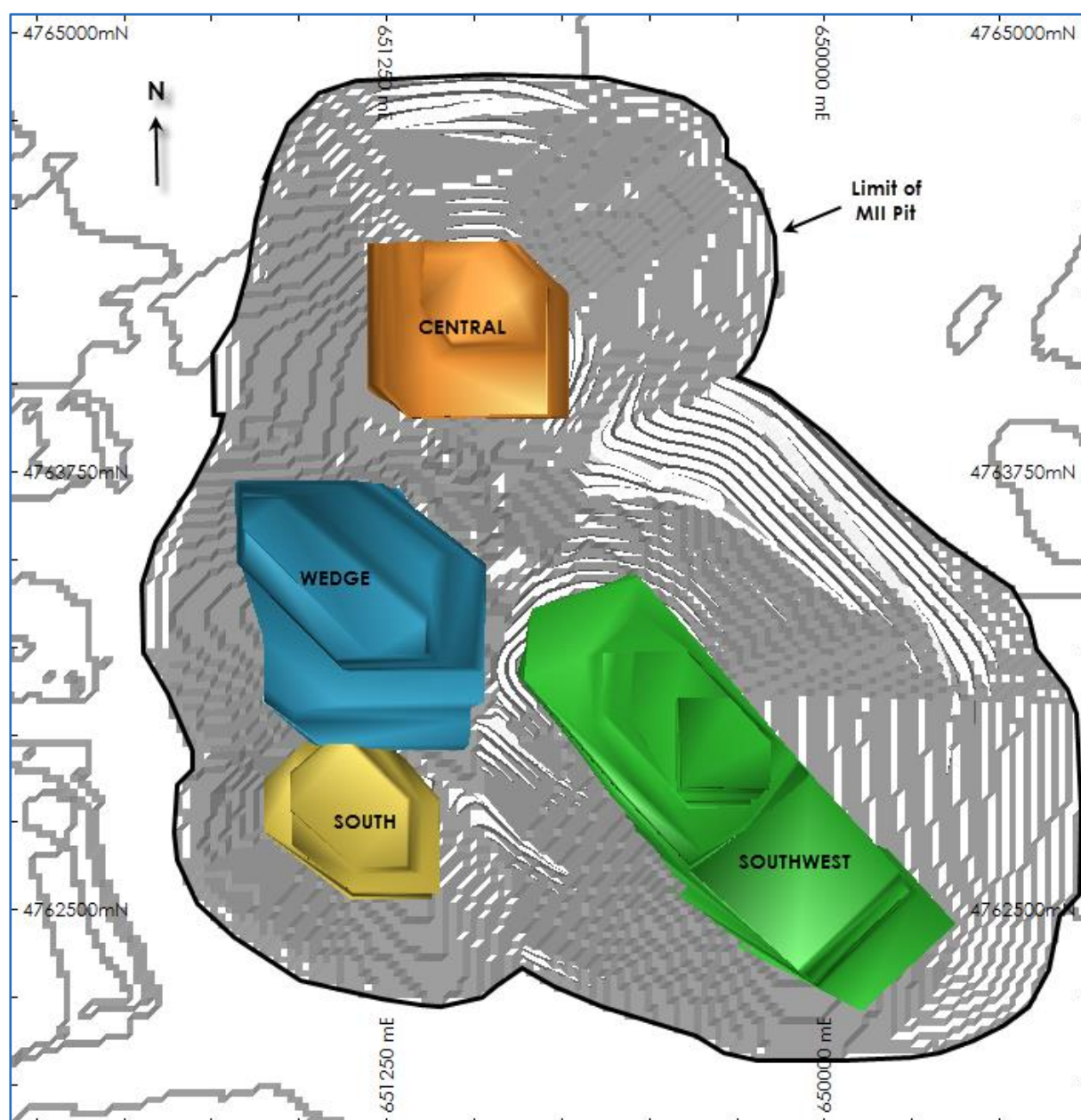
The Oyut deposit has several mineralized areas beneath the MII Pit Shell that are considered to be targets for future underground mining. Located within 3 km of the planned Hugo North underground mine and associated infrastructure, these target areas could also be used for access to and hoisting or conveying from any underground operation at Oyut.

Underground constraining stope-block shapes (based on assumed mechanized block

caving) were defined with at least a 100 m x 200 m footprint for the four areas (Figure 14.12). Constraining 3D stope-block shapes were outlined based on estimation of economic criteria that would pay for primary and secondary development, mining, ventilation, tramming, hoisting, processing, and G&A costs, i.e. the underground stope shape cut-off. In delineating the constraining stopes-block shapes, a mining costs of US\$12/t was used, which included primary and secondary development, mining, ventilation, tramming, and hoisting costs. These stope-block shapes and the cost assumptions used in generating them have not been updated in this study because the infrastructure built for Hugo North Lift 1, and perhaps in future for Hugo South, could provide synergies for lower capital intensity underground development at Oyut.

Using the MII Pit Shell and the underground constraining shapes, Mineral Resources have been stated for those blocks that are located within the constraining underground cave shapes and that met a marginal cut-off grade of 0.37% CuEq (Section 14.2.2).

Figure 14.12 Projected Plan View (looking upwards from below) of Oyut MII Pit showing Shapes used to Constrain Oyut Underground Resources



14.2.3.3 Hugo North and Hugo South Mineral Resource Constraints

To assess reasonable prospects for eventual economic extraction (RPEE) and to declare underground resources at Hugo North, an underground resource-constraining shape (the RPEE, shell) was prepared on vertical sections using economic criteria that would pay for development, block-cave mining, ventilation, haulage, hoisting, processing, and G&A costs.

A primary and secondary development cost of US\$8.00/t and a mining, process, and G&A cost of US\$12.45/t were used to delineate the RPEE shape cut-off. Using OT LLC's Base Data Template 29 (BDT29) gold price of US\$970/oz and a revised copper price of US\$3.00/lb, it was estimated that a 0.50% copper cut-off would return US\$21.74/t, which would cover the RPEE shape cut-off costs stated above. The OT LLC RPEE shell was developed in 2012 and was not updated in 2014. OreWin has undertaken a separate RPEE analysis to confirm that the Mineral Resource RPEE is still current and valid. The infrastructure built for Hugo North Lift 1 would provide synergies for lower capital intensity underground development at subsequent Hugo North panels. Mineral resources within the RPEE shell at Hugo North are reported at a break-even copper-equivalent cut-off grade of 0.37% CuEq.

Estimates classified as Inferred Mineral Resource within the Lift 1 block cave shape are assigned a zero grade and treated as dilution in the reserve. Thus, they are not treated in the same way as Inferred Mineral Resources within the open pit final reserve shell.

Inferred Mineral Resources at Heruga and Hugo South have been constrained only by using a cut-off of 0.37% CuEq.

14.3 Tabulating Mineral Resources

Once the open pit and underground constraining shapes were generated, resources were stated for those model cells within the constraining underground stope-block shapes that met a given CuEq cut-off grade.

14.3.1 Mineral Resource Confidence Classification

Classification was undertaken using a set of rules based on estimation (OK) passes, distances to nearest composites, and numbers of holes used to inform the cell estimates to establish an initial classification. A separate ID2 estimation was run for copper grade composites at Oyut, which was used to interpolate the initial classification model. The initial classification was then assessed and modified with an algorithm designed to eliminate isolated cells of a particular classification by switching them to majority classification of the surrounding cells.

Confidence categories, contained in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves, were applied to the resource block models.

Mineral Resources were classified based on the criteria outlined in the following sections.

14.3.1.1 Oyut

Measured Mineral Resources

A three-hole rule was applied where cells contained an estimated grade from three or more composites from different holes, all within 50 m and at least one composite within 30 m of the cell centroid. These cells were classified as Measured Mineral Resource.

Indicated Mineral Resources

The drillhole spacing over much of the Oyut area is approximately 70 m. The drillhole spacing and geological and grade continuity over this area was considered to support an Indicated Mineral Resource classification in this area. A two-hole rule was used where cells containing an estimated grade were required to have been informed by two or more composites from different holes. Furthermore, for the Southwest zone, the two holes were required to be within a distance of 75 m, with at least one hole within 55 m of the cell centroid. For the remaining Oyut zones, the two holes were required to be within 65 m with at least one hole within 45 m of the block centroid.

Inferred Mineral Resources

Estimates in the Oyut area with that did not meet the classification criteria for Measured Mineral Resource or Indicated Mineral Resources were assigned to the Inferred Mineral Resources category if the cell centroid was within 150 m of a copper composite.

14.3.1.2 Hugo North

Measured Mineral Resources

Underground drilling has resulted in sufficient confidence in geological and grade continuity to support Measured Mineral Resources in proximity to underground drillholes. Blocks classified as Measured Mineral Resources have satisfied the following criteria:

- A three-hole rule was used for OK-estimated Cu blocks with three or more composites from three different holes, from three different search octants, all within 50 m and at least one composite within 35 m of the block centroid, all distances from ID2 Pass 1. The distance used is the closest anisotropic distance.
- Blocks were constrained by the Measured classification solid generated using sectional interpretation and block probabilities.

Indicated Mineral Resources

The drillhole spacing over much of the Hugo North area is approximately 125 m x 75 m. The minimum nominal drillhole spacing of 75 m (horizontal) between drillholes and 150 m between drill lines for Indicated Mineral Resources was determined in the course of a study on drillhole spacing conducted in 2004. The following conditions need to be met to classify blocks as Indicated Mineral Resources:

- A three-hole rule was used for OK-estimated Cu blocks not classified as Measured and with three or more composites from three different holes, all within 50 m distance from ID2 Pass 1. The distance used is the closest anisotropic distance.
- A three-hole rule was used for OK-estimated Cu blocks with three or more composites from three different holes, all within 150 m and at least one composite within 105 m of the block centroid, all distances from ID2 Pass 2. The distance used is the closest anisotropic distance.

- A two-hole rule was used for OK-estimated Cu blocks with two or more composites from two different holes, all within 150 m with at least one hole within 75 m of the block centroid, all distances from ID2 Pass 2. The distance used is the closest anisotropic distance.
- Blocks were constrained by the Indicated classification solid generated using sectional interpretation and block probabilities.

Inferred Mineral Resources

All blocks in the Hugo North model with an OK-estimated Cu grade that did not meet the classification criteria for Measured or Indicated Mineral Resources were assigned to Inferred Mineral Resources if the block centroid was within 150 m of a composite. The distance used is the closest Cartesian distance captured from Pass 3 of the ID2 estimation described above.

- Blocks were constrained by the inferred classification solid generated using sectional interpretation and block probabilities.

14.3.1.3 Hugo South

There are no Measured or Indicated Mineral Resources at Hugo South.

Inferred Mineral Resources

Interpolated cells were classified as Inferred Mineral Resources if they fell within 150 m of a drillhole composite. All mineralization at Hugo South is currently classified as Inferred Mineral Resources.

14.3.1.4 Heruga

There are no Measured or Indicated Mineral Resources at Heruga.

Inferred Mineral Resources

Interpolated cells were classified as Inferred Mineral Resources if they fell within 150 m of a drillhole composite. All mineralization at Heruga is currently classified as Inferred Mineral Resources.

14.4 Mineral Resource Statement

The summary of the Oyu Tolgoi Mineral Resources is presented in Table 14.45. Mineral Resources are classified in accordance with the 2010 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves.

The Mineral Resources have various effective dates:

- Oyut 19 March 2012
- Hugo North 28 March 2014
- Hugo South 1 November 2003
- Heruga 30 March 2010

The contained copper, gold, silver, and molybdenum estimates in the Mineral Resource tables have not been adjusted for metallurgical recoveries. However, the differential recoveries were taken into account when calculating the copper equivalency formula explained in 14.2.1. The various recovery relationships at Oyu Tolgoi are complex and relate both to grade and Cu : S ratios.

Mineral Resources were also estimated for trace and impurity elements, including arsenic, fluorine, and sulphur, as well as the copper, gold, silver, and molybdenum estimates reported in these tables.

A sulphide mineral abundance model was created for Oyut and Hugo North that will allow improved estimates of geometallurgical modelling and assist with characterization of tailings acid rock drainage (ARD) capacity.

Table 14.45 Oyu Tolgoi Mineral Resource Summary – 31 December 2015

Classification	Deposit	Tonnage (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Contained Metal				
								Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)	CuEq (Mlb)
Oyut Deposit – Open Pit (0.22% CuEq Cut-off) (Excludes material mined up to 31 December 2015)												
Measured		377	0.52	0.35	1.35	53.9	0.72	4,335	4,038	15,804	45	5,947
Indicated		715	0.38	0.23	1.11	56.4	0.51	6,039	5,082	24,705	89	8,110
Measured + Indicated		1,092	0.43	0.27	1.19	55.5	0.58	10,374	9,120	40,509	134	14,057
Inferred		389	0.29	0.16	0.86	44.2	0.38	2,461	1,888	10,381	37	3,247
Oyut Deposit – Underground (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Measured		14	0.40	0.78	1.15	38.8	0.83	121	342	509	1.2	250
Indicated		93	0.35	0.59	1.19	34.3	0.67	713	1,766	3,562	7.1	1,386
Measured + Indicated		107	0.35	0.61	1.18	34.8	0.69	833	2,108	4,072	8.2	1,636
Inferred		159	0.39	0.32	0.85	25.4	0.56	1,354	1,638	4,382	8.9	1,985
Hugo Dummett Deposits (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Measured	OT LLC	98	1.97	0.46	4.48	30.3	2.26	4,231	1,446	14,046	6.5	4,865
	EJV	1	1.43	0.12	2.86	39.4	1.52	35	4	103	0.1	38
	All Hugo North	99	1.96	0.46	4.46	30.4	2.25	4,267	1,450	14,149	6.6	4,902
Indicated	OT LLC	749	1.56	0.34	3.35	34.3	1.78	25,737	8,268	80,718	57	29,362
	EJV	128	1.65	0.55	4.12	33.6	1.99	4,663	2,271	16,988	10	5,633
	All Hugo North	877	1.57	0.37	3.46	34.2	1.81	30,400	10,539	97,707	66	34,994
Measured + Indicated	OT LLC	847	1.61	0.36	3.48	33.85	1.83	29,968	9,714	94,764	63	34,226
	EJV	129	1.65	0.55	4.11	33.70	1.99	4,698	2,276	17,091	10	5,670
	All Hugo North	976	1.61	0.38	3.56	33.83	1.85	34,667	11,989	111,856	73	39,897
Inferred	OT LLC	811	0.77	0.27	2.34	34.8	0.94	13,807	7,058	60,964	62	16,851
	EJV	179	0.99	0.34	2.68	25.4	1.20	3,887	1,963	15,418	10	4,730
	All Hugo North	990	0.81	0.28	2.40	33.1	0.99	17,695	9,021	76,382	72	21,581
Inferred	Hugo South	845	0.77	0.07	1.78	66.4	0.83	14,372	1,861	48,406	124	15,384
Heruga Deposit (0.37% CuEq Cut-off) (Unchanged since 2014 OTTR)												
Inferred Heruga EJV		1,700	0.39	0.37	1.39	113.2	0.64	14,610	20,428	75,955	424	24,061
Inferred Heruga TRQ		116	0.41	0.29	1.56	109.8	0.61	1,037	1,080	5,819	28	1,565
Inferred (All Heruga)		1,816	0.39	0.37	1.40	113.0	0.64	15,647	21,508	81,774	453	25,626
Oyu Tolgoi All Deposits Grand Total (Excludes material mined up to 31 December 2015))												
Measured		489	0.81	0.38	1.97	48.7	1.03	8,722	5,971	30,996	53	11,098
Indicated		1,686	1.00	0.32	2.34	43.6	1.20	37,152	17,572	126,797	162	44,486
Measured + Indicated		2,175	0.96	0.34	2.26	44.8	1.16	45,875	23,543	157,792	215	55,584
Inferred		4,200	0.56	0.27	1.64	75.1	0.73	51,531	35,980	221,670	695	67,821

Notes:

- The Mineral Resources include Mineral Reserves.
- The contained gold and copper estimates in the tables have not been adjusted for metallurgical recoveries.
- The 0.22% CuEq cut-off is equivalent to the open pit Mineral Reserve cut-off determined by OT LLC.
- The 0.37% CuEq cut-off is equivalent to the underground Mineral Reserve cut-off determined by OT LLC.
- Oyut open pit Mineral Resources exclude material mined in the open pit as at 31 December 2015.
- CuEq has been calculated using assumed metal prices (US\$3.01/lb for copper, US\$1,250/oz for gold, US\$20.37/oz for silver, and US\$11.90/lb for molybdenum). Mo grades outside of Heruga are assumed to be zero for CuEq calculations.
 - Oyut CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.887) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.949)) / (3.01 \times 22.0462)$
 - HN (OT LLC) CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.906) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.941)) / (3.01 \times 22.0462)$
 - HN (EJV) CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.913) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.942)) / (3.01 \times 22.0462)$
 - HS CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.909) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.945)) / (3.01 \times 22.0462)$
 - Heruga CuEq% = $\text{Cu\%} + ((\text{Au (g/t)} \times 1,250 \times 0.0321507 \times 0.911) + (\text{Ag (g/t)} \times 20.37 \times 0.0321507 \times 0.949) + (\text{Mo (ppm)} \times 11.9 \times 0.0022046 \times 0.736)) / (3.01 \times 22.0462)$
- Totals may not match due to rounding.
- EJV is the Entrée-OT LLC Joint Venture. The Shivee Tolgoi and Javkhant licenses are held by Entrée. The Shivee Tolgoi and Javkhant licenses 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. See Section 4.2.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

14.5 Factors that Could Affect the Mineral Resource Estimates

Areas of uncertainty that could materially affect the Mineral Resource estimates include the following:

- Commodity pricing.
- Interpretations of fault geometries.
- Effect of alteration as a control on mineralization.
- Lithological interpretations on a local scale, including dyke modelling and discrimination of different Qmd phases.
- Pit slope angles.
- Geotechnical assumptions related to the proposed block cave design and material behaviour.
- Metal recovery assumptions.
- Dilution considerations.
- Contaminants such as arsenic and fluorine.
- Estimates of operating costs used to support reasonable prospects assessment.
- Changes to drill spacing's and number of drillhole composites used to support classification categories.

14.6 Reconciliation with 2014 Mineral Resources

Mineral Resources were previously updated in the 2014 Technical Report (2014 OTTR). The 2016 OTTR reports Mineral Resources from the same resource models as those reported in the 2014 OTTR. The Oyut open pit Mineral Resource has been depleted since 2014 as a consequence of mining, but apart from this, there have been no other changes made to the Mineral Resources since 2014. A comparison of the 2014 OTTR and 2016 Mineral Resources is shown in Table 14.46.

The Measured and Indicated Mineral Resources have decreased by approximately 80 Mt since the 2014 OTTR due to the extraction (mining) of open pit resources at Oyut. This represents a depletion of –3.5% of the overall (Measured plus Indicated) tonnage, –2% of copper metal, –7% of gold metal, and –2% of silver metal.

Relative to the grades in the 2014 OTTR Mineral Resources, the remaining in situ Measured plus Indicated Mineral Resources show an overall 2% increase in Cu grade, a 3% decrease in gold grade, and a 1% increase in silver grade.

Table 14.46 Mineral Resource Reconciliation 2016 OTTR versus 2014 OTTR

Classification	Tonnage (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	CuEq (%)	Contained Metal				
							Cu (Mlb)	Au (koz)	Ag (koz)	Mo (Mlb)	CuEq (Mlb)
2016 OTTR											
Measured	489	0.81	0.38	1.97	48.7	1.03	8,722	5,971	30,996	53	11,098
Indicated	1,686	1.00	0.32	2.34	43.6	1.20	37,152	17,572	126,797	162	44,486
Measured + Indicated	2,175	0.96	0.34	2.26	44.8	1.16	45,875	23,543	157,792	215	55,584
Inferred	4,200	0.56	0.27	1.64	75.1	0.73	51,531	35,980	221,670	695	67,821
2014 OTTR											
Measured	544	0.78	0.43	1.93	47.4	1.03	9,372	7,486	33,713	57	12,356
Indicated	1,711	0.99	0.32	2.33	43.4	1.19	37,394	17,782	127,995	164	44,851
Measured + Indicated	2,255	0.94	0.35	2.23	44.3	1.15	46,766	25,268	161,708	220	57,207
Inferred	4,201	0.56	0.27	1.64	75.0	0.73	51,533	35,979	221,805	695	67,830
Absolute Difference (2016 - 2014)											
Measured	-55	0.028	-0.048	0.044	1.311	-0.001	-649	-1,515	-2,717	-4.3	-1,258
Indicated	-25	0.008	0.001	0.013	0.277	0.008	-242	-210	-1,199	-1.4	-365
Measured + Indicated	-80	0.016	-0.012	0.026	0.443	0.008	-891	-1,725	-3,916	-5.7	-1,623
Inferred	-0.32	0.00003	0.00003	-0.00087	0.054	-0.00004	-1.47	1.03	-135.1	0.45	-9.06
Relative Percentage Difference ((2016 - 2014) / 2014)											
Measured	-10.1%	3.5%	-11.3%	2.3%	2.8%	-0.1%	-6.9%	-20.2%	-8.1%	-7.6%	-10.2%
Indicated	-1.5%	0.8%	0.3%	0.5%	0.6%	0.7%	-0.6%	-1.2%	-0.9%	-0.8%	-0.8%
Measured + Indicated	-3.5%	1.7%	-3.4%	1.2%	1.0%	0.7%	-1.9%	-6.8%	-2.4%	-2.6%	-2.8%
Inferred	-0.008%	0.005%	0.011%	-0.053%	0.072%	-0.006%	-0.003%	0.003%	-0.061%	0.064%	-0.013%

Notes: Mineral Resources are reported using CuEq cut-offs and Mineral Reserves are reported using NSR cut-offs resulting in differences in depletion shown.

Total may vary due to rounding.

15 MINERAL RESERVE ESTIMATES

15.1 Mineral Reserves

The Mineral Reserves for the project have been estimated using the Oyut and Hugo North Mineral Resources. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT). 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 the 2016 OTTR are based on mine planning work prepared by OT LLC in OTFS16.

The Mineral Reserves for the Oyut open pit are based on the same modifying parameters and Mineral Resources, the change since the 2014 OTTR has been the mining depletion. The Hugo North Mineral Reserves in the 2016 OTTR are the same as the Mineral Reserves in the 2014 OTTR.

The Hugo North Mineral Reserve contains ore that is on the OT License and on the EJV Shivee Tolgoi license. The EJV Shivee Tolgoi license is subject to a joint venture agreement (EJV) between OT LLC and Entrée.

The 2016 OTTR Mineral Reserves are reported as at 31 December 2015. This date was selected for reporting of the Mineral Reserves to remain consistent with OTFS16.

The metal prices and assumptions used for the cut-off grades were denominated in NSR US\$/t and are the same as those used for cut-off grade determination in the 2014 OTTR. The economic analysis has been updated with current long-term metal prices and assumptions.

OT LLC undertook pit surveys and reported the depletion from the Oyut Mineral Reserves. The Oyut Mineral Reserves shown in Table 15.1 are the Proven and Probable remaining in the pit. Stockpiles have not been included in the 2016 OTTR Oyut Mineral Reserves reporting as they will include some inferred and unclassified materials as well as low grade Measured and Indicated Mineral Resources.

The 2016 OTTR only considers Mineral Resources in the Measured and Indicated categories, and engineering that has been carried out to a feasibility level or better to estimate the open pit and underground Mineral Reserves. Mine designs were prepared using industry-standard mining software, assumed metal prices as described in the notes to the Mineral Reserves, and smelter terms as set forth in Section 22.

Table 15.1 Total Oyu Tolgoi 2016 Mineral Reserves – 31 December 2015

Deposit by Classification	Ore (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
					Cu (Mlb)	Au (koz)	Ag (koz)
Oyut Mineral Reserves							
Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
Probable	598	0.39	0.23	1.11	4,058	3,103	15,977
Oyut Total (Proven and Probable)	951	0.45	0.28	1.22	7,325	5,878	27,814
Hugo North Mineral Reserves							
Probable (OT LLC)	464	1.66	0.34	3.37	15,592	4,199	43,479
Probable (EJV)	35	1.59	0.55	3.72	1,121	519	3,591
Hugo North Total (Probable)	499	1.66	0.35	3.40	16,713	4,717	47,070
Total Mineral Reserves							
Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
Probable	1,097	0.97	0.29	2.15	20,771	7,820	63,047
Total (Proven and Probable)	1,450	0.86	0.30	1.97	24,037	10,595	74,884

Notes:

1. Metal prices used for calculating the financial analysis are as follows: long-term copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
2. For mine planning the metal prices used to calculate block model NSR were copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz.
3. The Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs at Oyu Tolgoi, therefore cut-off is denominated in US\$/t. By definition the cut-off is the point at which the costs are equal to the NSR. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t and the underground (including some mining costs) costs were based on US\$15.34/t.
4. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes Indicated Mineral Resources below the resource cut-off grade. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
5. The Oyut open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
6. For Oyut, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
7. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves.
8. EJV is the Entrée-OT LLC Joint Venture. The Shivee Tolgoi and Javkhant licenses are held by Entrée. The Shivee Tolgoi and Javkhant licenses 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. See Section 4.2.
9. The Mineral Reserves reported above were not additive to the Mineral Resources.
10. Totals may not match due to rounding.
11. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

15.2 Reconciliation with 2014 OTTR Reserves

Mineral Reserves were previously updated in the 2014 OTTR. The resource model used for and cut-off grades in the Mineral Reserves have not been changed from the 2014 OTTR. The only change to the Mineral Reserves has been from mining depletion in the Oyut open pit. The mining depletion is approximately 81 Mt and is the depletion from the Oyut resource block model. A summary of the Mineral Reserves depletion between the 2014 OTTR and 2016 OTTR Mineral Reserves is shown in Table 15.2 and a comparison of the 2016 OTTR and 2014 OTTR Mineral Reserves is shown in Table 15.3.

Table 15.2 Mineral Reserves Depletion between 2014 OTTR and 2016 OTTR

Deposit by Classification	Ore (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
					Cu (Mlb)	Cu (Mlb)	Cu (Mlb)
Oyut Mineral Reserves							
Proven	57	0.52	0.82	1.24	563	1,177	1,931
Probable	23	0.57	0.30	1.61	304	130	1,145
Oyut Total (Proven and Probable)	81	0.53	0.67	1.34	867	1,308	3,076
Hugo North Mineral Reserves							
Probable (OT LLC)	–	–	–	–	–	–	–
Probable (EJV)	–	–	–	–	–	–	–
Hugo North Total (Probable)	–	–	–	–	–	–	–
Total Mineral Reserves							
Proven	57	0.52	0.82	1.24	563	1,177	1,931
Probable	23	0.57	0.30	1.61	304	130	1,145
Total (Proven and Probable)	81	0.53	0.67	1.34	867	1,308	3,076

Notes:

1. See notes to Table 15.1.
2. Depletion is a result of production from 1 January 2014 through 31 December 2015.

Table 15.3 Mineral Reserves Reconciliation 2016 OTTR versus 2014 OTTR

Case	Mineral Reserves	Ore (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
						Cu (Mlb)	Au (koz)	Ag (koz)
2016 OTTR	Proven	353	0.54	0.35	1.40	3,266	2,775	11,837
	Probable	1,097	0.97	0.29	2.15	20,771	7,820	63,047
	Total	1,450	0.86	0.30	1.97	24,037	10,595	74,884
2014 OTTR	Proven	410	0.54	0.42	1.38	3,829	3,952	13,768
	Probable	1,120	0.96	0.29	2.14	21,075	7,951	64,192
	Total	1,530	0.85	0.32	1.94	24,905	11,903	77,960
Absolute Difference	Proven	-57	0	-0.06	0.02	-563	-1,177	-1,931
	Probable	-23	0.01	0	0.01	-304	-130	-1,145
	Total	-81	0.02	-0.02	0.03	-867	-1,308	-3,076
Relative Difference	Proven	-16%	0%	-18%	2%	-17%	-42%	-16%
	Probable	-2%	1%	0%	1%	-1%	-2%	-2%
	Total	-6%	2%	-6%	2%	-4%	-12%	-4%

Notes:

1. 2014 OTTR Mineral Reserves have the effective date 20 September 2014.
2. 2016 Oyu Tolgoi Technical Report Mineral Reserves have the effective date of 31 December 2015.
3. Metal prices used in 2014 for calculating the Oyu open pit Net Smelter Return (NSR) and the Hugo North underground NSR are as follows: copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz, all based on long-term metal price forecasts at the beginning of the Mineral Reserves work. The analysis indicates that the Mineral Reserves are still valid at these metal prices.
4. Metal prices used in 2016 for calculating the financial analysis are as follows: long-term copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties. Prices are assumed to increase from current prices to the long-term prices which apply from 2021.
5. The NSR has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
6. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t.
7. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes Indicated Mineral Resources below the resource cut-off grade. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
8. For Oyu, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
9. The Oyu open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
10. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves (no Measured Mineral Resource for Hugo North in 2014 OTTR).
11. The Mineral Reserves reported above are not additive to the Mineral Resources.
12. Totals may not match due to rounding.

15.3 Key Assumptions

Key assumptions are summarized below; other assumptions are documented in the 2016 OTTR:

- Metal prices used for calculating the Oyut open pit NSR and the Hugo North Underground NSR are copper US\$3.01/lb, gold US\$1,250/oz, and silver US\$20.37/oz based on long-term metal price forecasts at the beginning of the Mineral Reserves work. Analysis indicates that the Mineral Reserves are 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: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t.
- For the underground, a footprint cut-off of US\$15.34/t NSR and column height shut-off of US\$15.34/t NSR were used to maintain grade and productive capacity. The shut-off is the grade used to determine the point at which each drawpoint is closed.
- For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes low-grade Indicated Mineral Resource and Inferred Mineral Resource assigned zero grade treated as dilution.
- For the Oyut open pit only Measured Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
- For the Hugo North Lift 1 underground both Measured and Indicated Mineral Resources were converted to Probable Mineral Reserves.
- EJV is the Entrée–OT LLC Joint Venture. The Shivee Tolgoi and Javkhlant licenses are held by Entrée. The EJV Shivee Tolgoi and Javkhlant licenses 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. See Section 4.2
- The Mineral Reserves are not additive to the Mineral Resources.
- The underground Mineral Resource cell models used for reporting the Mineral Reserves are the models reported in the Mineral Resource section of 2016 OTTR. The 2012 Oyut resource model prepared by OT LLC has been used for the open pit Mineral Reserves.
- 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 Reserves size are overestimated.

15.4 US SEC Industry Guide 7

The Mineral Reserves reported for NI 43-101 are also applicable for reporting the Ore Reserve under the US SEC Industry Guide 7. OreWin estimated the Oyu Tolgoi Mineral Reserves for the NI 43-101 2016 OTTR, which are based on 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 (US SEC) reporting, OreWin considers that the 2016 OTTR 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 three-year average price for the commodity that is expected to be mined in determining economic viability.
- Primary environmental analysis has been submitted to government authorities.

15.4.1 Bankable Study

OTFS16 is a Bankable Feasibility Study that has been used as the basis of a US\$4.4b project finance facility. The facility has been provided by a syndicate of international financial institutions and export credit agencies representing the governments of Canada, the United States and Australia, along with 15 commercial banks. Drawdown of the loan has been completed. OreWin therefore considers 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 economic analysis has been prepared using current long-term metal price estimates of:

- Copper US\$3.00/lb
- Gold US\$1,300/oz
- Silver US\$19.00/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 three years, their average and the metal prices used for the Mineral Reserve estimates are shown in Table 15.4. The sensitivity analysis using the three-year averages shows the after tax NPV8% is US\$4.17b. This demonstrates the forward-looking discounted cash flow is still positive.

Table 15.4 Metal Price Summary

Year Ended	Copper (US\$/lb)	Gold (US\$/oz)	Silver (US\$/oz)
2014	3.10	1,265	19.07
2015	2.49	1,160	15.71
2016	2.13	1,220	15.83
Average	2.57	1,215	16.87
Reserve NSR	3.01	1,250	20.37
Base Case Financial Analysis	3.00	1,300	19.00

15.4.3 Primary Environmental Analysis Submission

The 2007 SME Guide Section 56 describes how the permitting and legal requirements of US SEC 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.

OT LLC has completed a comprehensive Environmental and Social Impact Assessment (ESIA) for Oyu Tolgoi. The culmination of nearly 10 years of independent work and research carried out by both international and Mongolian experts, the ESIA identifies and assesses the potential environmental and social impacts of the project, including cumulative impacts, focusing on key areas such as biodiversity, water resources, cultural heritage, and resettlement.

The ESIA also sets out measures through all project phases to avoid, minimize, mitigate, and manage potential adverse impacts to acceptable levels established by Mongolian regulatory requirements and good international industry practice, as defined by the requirements of the Equator Principles, and the standards and policies of the International Finance Corporation (IFC), European Bank for Reconstruction and Development (EBRD), and other financing institutions.

Corporate commitment to sound environmental and social planning for the project is based on two important policies: TRQ's Statement of Values and Responsibilities (March 2010), which declares its support for human rights, social justice, and sound environmental management, including the United Nations Universal Declaration of Human Rights (1948); and The Way We Work 2009, Rio Tinto's Global Code of Business Conduct that defines the way Rio Tinto manages the economic, social, and environmental challenges of its global operations.

OT LLC has commenced the development and implementation of an environmental management system (EMS) that conforms to the requirements of ISO 14001:2004. Implementation of the EMS during the construction phases will focus on the environmental policy; significant environmental aspects and impacts and their risk prioritization; legal and other requirements; environmental performance objectives and targets; environmental management programmes; and environmental incident reporting. The EMS for operations will consist of detailed plans to control the environmental and social management aspects of all project activities following the commencement of commercial production in 2013. The Oyu Tolgoi ESIA builds upon an extensive body of studies and reports, and DEIA's that have been prepared for project design and development purposes and for Mongolian approvals under the following laws:

- The Environmental Protection Law (1995).
- The Law on Environmental Impact Assessment (1998, amended in 2001).
- The Minerals Law (2006).

These initial studies, reports and DEIA's were prepared over a six-year period between 2002 and 2008, primarily by the Mongolian firm Eco-Trade LLC, with input from RPS Aquaterra on water issues.

The original DEIA's provided baseline information for both social and environmental issues. These DEIA's covered impact assessments for different project areas, and were prepared as separate components to facilitate technical review as requested by the GOM.

The original DEIA's were in accordance with Mongolian standards and while they incorporated World Bank and IFC guidelines, they were not intended to comprehensively address overarching IFC policies such as the IFC Policy on Social and Environmental Sustainability, or the EBRD Environmental and Social Policy.

Following submission and approval of the initial DEIA's, the GOM requested that OT LLC prepare an updated, comprehensive ESIA whereby the discussion of impacts and mitigation measures was project-wide and based on the latest project design. The ESIA was also to address social issues, meet GOM (legal) requirements, and comply with current IFC good practice.

For the ESIA the baseline information from the original DEIA's was updated with recent monitoring and survey data. In addition, a social analysis was completed through the commissioning of a Socio-Economic Baseline Study and the preparation of a Social Impact Assessment (SIA) for the project.

The requested ESIA, completed in 2012, combines the DEIA's, the project SIA, and other studies and activities that have been prepared and undertaken by and for OT LLC.

15.5 Mongolian Commercial Minerals

Mongolia has its own system for reporting Mineral Reserves and Mineral Resources. OT LLC registered a Mineral Reserve with the GOM in 2009. A key difference between the two standards is the classification of material contained in Hugo North Lift 2, Hugo South, and Heruga under Mongolian standards as reserves. This contrasts to the Canadian National Instrument (NI) 43-101 definitions, which include only Oyut and Hugo North Lift 1 in the Mineral Reserve category.

The system is based on a review by Mongolian experts in a number of disciplines. A significant difference between the Mongolian system and NI 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. OT LLC has submitted an update of the Mongolian Mineral Resource and Reserve to the GOM, which is based on the production and costing in a production case assuming that the plant is not expanded and that additional mines are developed (OTFS16 Resources Case).

16 MINING METHODS

16.1 Open Pit Mining

Open pit mine plans for this technical report have not materially changed from those reported in the 2014 OTTR since they have been developed with the same or similar mine design and scheduling criteria. While open pit plans have not materially changed, a parallel review performed as part of the planning process identified potential upside in the current valuations of the open pit and its ability to support development of the underground mine. These potential upsides are in the areas of pit slopes, pit phase design and sequence, mining intensity, cut-off grade strategy and location of waste dump and tailings facilities. These opportunities will be investigated in future mine plans.

16.1.1 Open Pit Geotechnical

In the 2013 Pit Slope Management Programme (PSMP), nine boreholes were drilled totaling 4,120 m, using conventional geotechnical mud rotary and coring methods. The boreholes were drilled from surface to a maximum depth of 600 m at inclinations between 55° and 65° from the horizontal. Some of the boreholes were equipped with Time Domain Reflectometry (TDR) and hydrogeology monitoring instrumentation (Vibrating Wire Piezometer, or VWP). Some of the holes were also used for packer testing and geophysical logged using an Acoustic Televiewer (ATV).

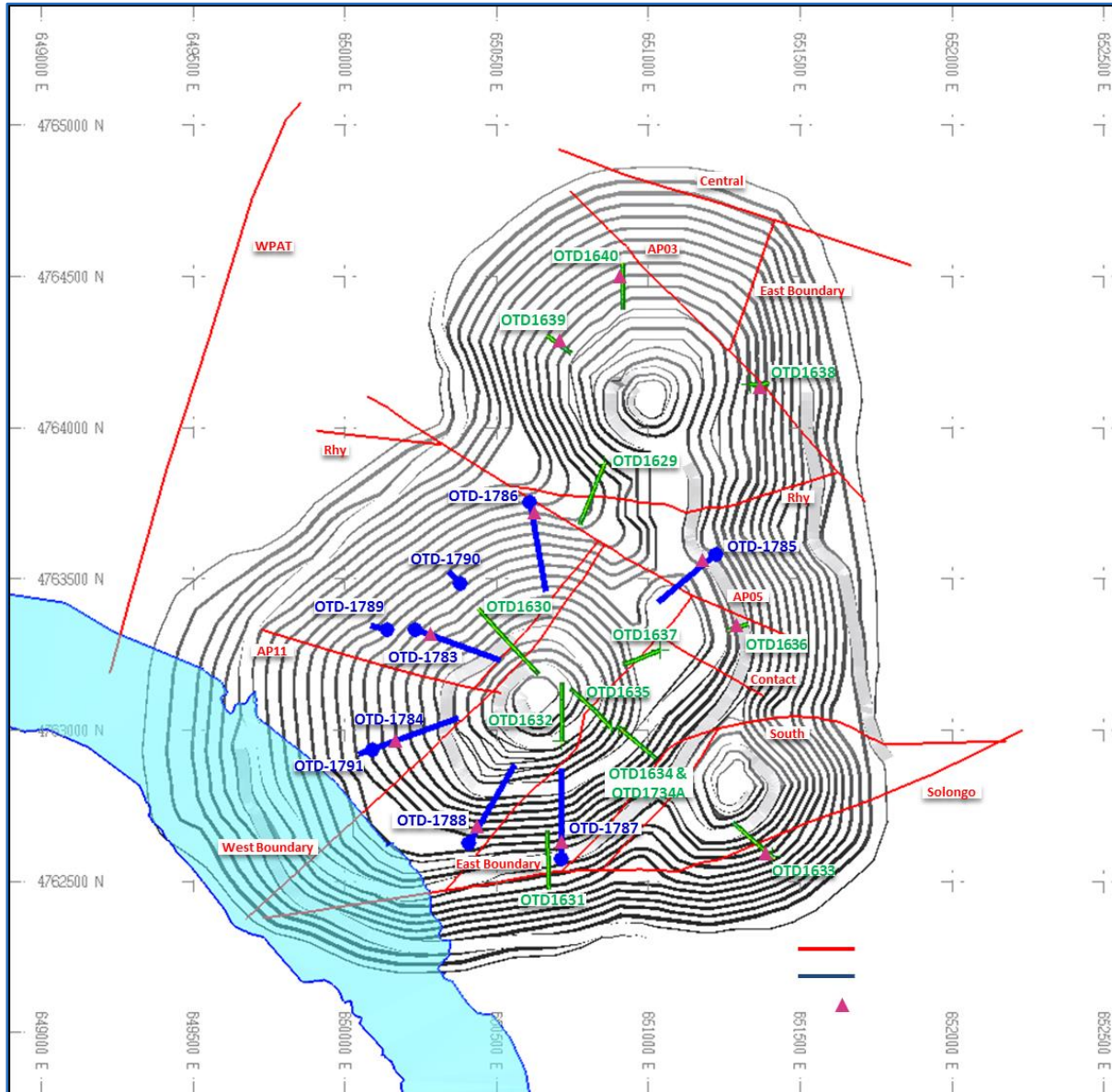
The next geotechnical drilling programme, consisting of 380 m of drilling over three holes, was completed in June 2015. VWPs and TDRs have been installed in these holes for use in geotechnical pit slope monitoring, hydro-geotechnical modelling, and stability evaluations, focusing on determining the distribution of pore pressures within the assumed overbreak zone (which current hydro-geotechnical models assume is 40 m).

Four oriented core boreholes were completed in July 2015 in the Phase 6 pit. A total of 1,390 m were drilled using conventional geotechnical mud rotary and coring methods (triple-tube PQ3 size). The holes were drilled from surface to a maximum of 400 m at inclinations of between 65° and 70° from the horizontal. All boreholes were geophysically logged by ATV and are equipped with piezometers.

An Ace core orientation device was run down the boreholes as part of the PQ core barrel assembly. Triple-tube core barrels were used for the core drilling, which allow the core to be hydraulically pumped out from the splits to preserve the quality of the core prior to logging.

The borehole locations for the drilling programmes completed in 2013 and to mid-2015 are shown in Figure 16.1 to Figure 16.3. Collection and analysis of drilling, hydrogeology, and actual slope performance data is ongoing. Slope performance prism monitoring is also ongoing from two Robotic Total Stations (RTS) installed on opposite sides of the pit. The pit, dumps, and stockpile slopes are also monitored regularly.

Figure 16.1 2013 Geotechnical Drilling Programme



Pit slope stability investigations and design for the Oyu Tolgoi open pit mine were originally completed by SRK (Australia) in 2005. In May 2009, OT LLC retained Golder Associates Limited (Golder) to carry out a review of the proposed pit slope designs. In mid-2010, OT LLC commissioned Rio Tinto Technology and Innovation (RTTI) to lead a study to examine the hydrogeological and geotechnical conditions of the open pit area to determine the anticipated pit slope performance and design criteria, the expected groundwater conditions, and dewatering and slope depressurization requirements for the planned pit sequence. RTTI prepared scopes of work and engaged three consulting firms, Golder, Schlumberger Water Services (Schlumberger), and RPS Aquaterra, to complete the study. The work was intended to provide feasibility-level geotechnical assessment for the first seven

years of planned mining and prefeasibility-level assessment for Year-8 through the planned life-of-mine (LOM).

Golder conducted the following field investigations from 4 December 2010 through 21 February 2011:

- Drilled 12 new geotechnical core holes (denoted OTD1629/GT01 to OTD1640/GT12) with core orientation.
- Collected rock samples for laboratory testing.

OT LLC performed ATV surveys in the 12 new geotechnical holes and in 14 existing open pit exploration holes. Golder interpreted the survey results to determine structural data. RPS Aquaterra completed the hydrogeological field testing (packer testing) on the geotechnical drillholes, and Schlumberger provided pit wall pore pressure predictions.

Figure 16.2 Phase 2 Boreholes Completed in June 2015

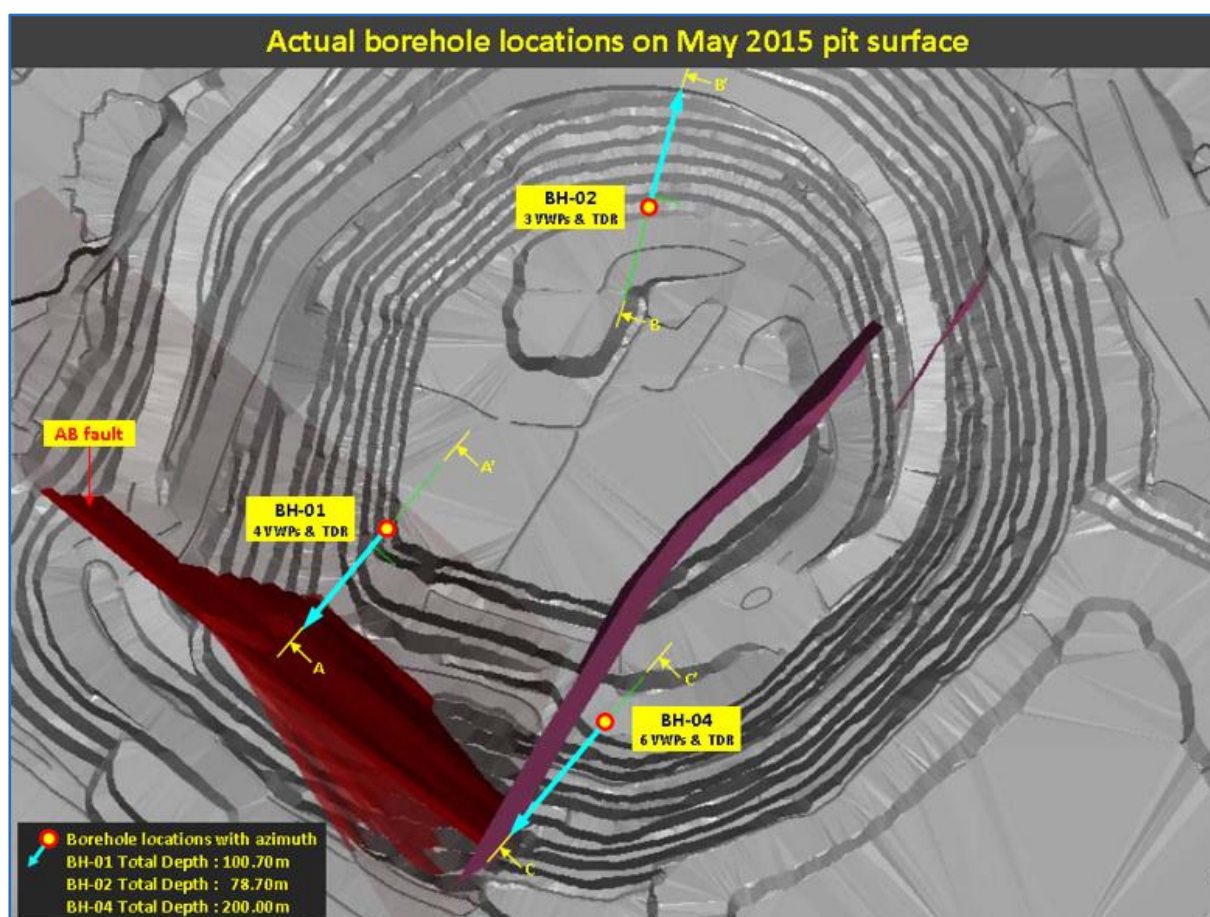
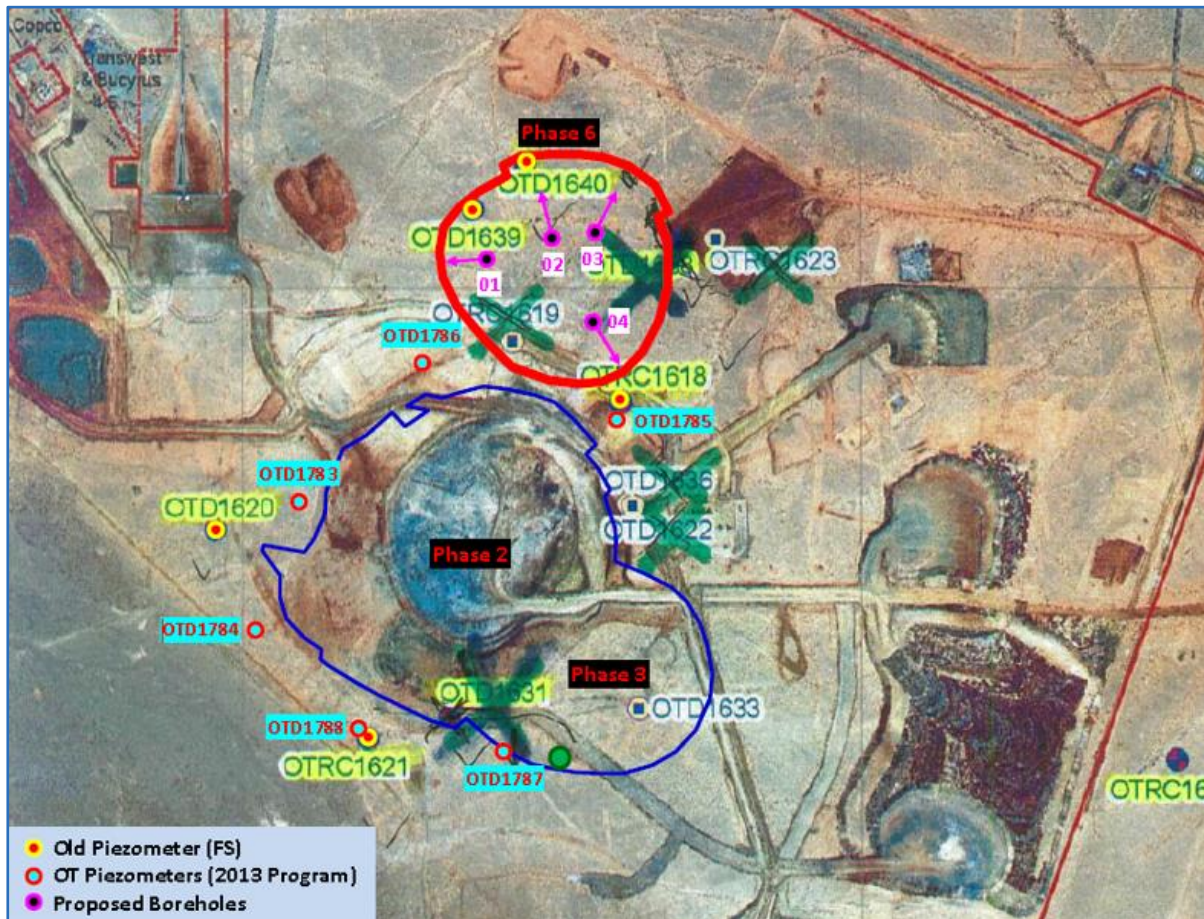


Figure 16.3 Phase 6 Boreholes Completed in July 2015



16.1.1.1 Geotechnical Assessment

Two main rock zones are present in the open pit area: the weathered zone and the bedrock zone. Field assessment of rock hardness, point load tests, and laboratory rock test results indicated that the rocks found within the weathered zone are weak and those within the bedrock zone are moderately strong to very strong; uniaxial compressive strength (UCS) values are shown in Table 16.1. The basalt, granodiorite, andesite, and rhyolite dykes are the strongest rock units (UCS values varying from 62–133 MPa), followed by the porphyritic augite basalt, quartz monzodiorite, dacite, and conglomerate rocks (R4, 50 MPa < UCS < 100 MPa). The siltstone-sandstone, ignimbrite, and fault zone rocks have the lowest strengths. The average fresh and altered ignimbrite UCS tests indicated moderately strong rock (R3, UCS in the range of 25–50 MPa).

Based on the Rock Mass Rating (RMR) system of classification, rock mass quality is expected to be fair within the weathered zone (40 < RMR < 60). Within the Bedrock zone, the main rock masses to be exposed on the pit walls are estimated to be generally of good rock mass quality (RMR > 60). The ignimbrite may show fair rock mass quality where the argillic alteration is more intense. Fair to poor rock mass quality is expected within some fault zones. The

intrusion of dykes on the 'old' faults, like the Solongo Fault, improved the rock mass conditions, creating fair to good rock mass quality.

Table 16.1 Rock Hardness in Open Pit Area

Zone	Hardness	Uniaxial Compressive Strength (UCS)
Weathered zone	Weak	R2, 5 MPa < UCS < 25 MPa
Bedrock zone	Moderately strong	R3, 25 MPa < UCS < 50 MPa
Bedrock zone	Very strong	R5, UCS > 100 MPa

Limit equilibrium stability assessments were carried out for rock mass failure, considering the potential for circular and bi-linear failure surfaces formed by the combination of the major geological structures and through the rock mass. Based on the recommended inter-ramp slope angles, the stability analyses confirmed that the overall pit wall would exhibit a Factor of Safety (FoS) higher than the accepted criterion of FoS = 1.3, even when considering a lower overall rock mass strength with a disturbance factor of 0.7. A more conservative inter-ramp angle (IRA) was recommended within the ignimbrite.

Given the strong, fair to good rock quality of the rock masses and the results of the limit equilibrium analyses, the main consideration for rock slope failure will be structurally controlled mechanisms (kinematics) at either a small scale (i.e. wedge fallout from benches) or a larger scale (i.e. slope failure along persistent joints or faults).

The structural data collected from the geotechnical and reconciled exploration drillholes were used to develop and assess seven structural design domains, labelled West, East, South, Solongo, Southwest, Northwest, and Middle. The locations of these structural domains were predominantly based on the orientations of major faults, lithological boundaries, and rock mass quality. Where required, these domains were further subdivided into sub-domains based on the rock type. For example, the East domain, corresponding to the east wall of the Central and South pits, was sub-divided into the sediments and intrusive structural sub-domains.

Based on stereonet generated for each structural domain, kinematic analyses were conducted to investigate structurally controlled failure in rock slopes – planar, wedge, and toppling failures. Each domain was analyzed kinematically every 30° to cover all wall orientations expected to daylight through the different phases of mining.

16.1.1.2 Slope Design Criteria

Feasibility level pit slope design recommendations were made for each design domain, considering the proposed bench face angles (BFA, or batter angle) and berm widths based on the rock mass quality and the structural controls. For simplicity, three BFA's were used in the design: 60°, 65°, and 70°.

- BFA = 60° corresponds to areas with high structural controls,
- BFA = 65° to areas with fair rock mass quality and some structural controls, and
- BFA = 70° to areas with minimum structural controls and good rock mass quality.

In addition, the pit design recommendations considered vertical separation of single (15 m) or double (30 m) benches, and varying berm widths and IRAs.

The convention used to define the orientation of each design sector was the wall dip direction, which corresponds to the direction perpendicular to the wall when looking into the pit. However, these orientations were redefined because the Oyu Tolgoi team uses a programme that requires the strike of the wall, which would be equivalent to the wall dip direction of minus 90°, to be considered as well.

The resulting design criteria recommendations are summarized as follows:

- Single, 15 m bench height (or batter) configuration in all rock slopes.
- Maximum BFA of 65° in Weathered zone.
- Average BFA of 70° in Bedrock zone.
- Minimum berm width of 7.5 m.
- Where structural controls are present, other berm widths of 9 m, 12 m, and 15 m to match the recommended IRA's.

A summary of the geotechnical parameters for domains in weathered rock is shown in Table 16.2 followed by the pit slope design recommendations for the geotechnical domains in Table 16.3.

Table 16.2 Summary of Geotechnical Recommendations for Domains in the Weathered Zone

Domain	'ndeposit' Flag	Bench Face Angle (BFA)	Bench Height (m)	Berm Width (m)	Inter-Ramp Angle (IRA)	GT_DOMCODE
West	1	65°	15	12	38°	14
East	2	65°	15	9	43°	15
South	3	65°	15	9	43°	16
Solongo	4	65°	15	9	43°	17
Southwest	5	65°	15	13.5	36°	18
Northwest	6	65°	15	13.5	36°	19
Middle	7	65°	15	9	43°	20
Solongo Fault		65°	15	8	45°	13

Note: Current design inter-ramp angle varies by domain

In addition to the berm and batter configurations, Golder recommended that the final and interim walls be subdivided into a series of bench stacks, either by providing haul road traverses or by including geotechnical berms at least 15 m wide or more at about 90 m vertical intervals (six benches x 15 m high). These berms would also protect personnel from potential major rock falls, allow for horizontal drain hole water controls, provide some flexibility in wall development, provide access for instrumentation, and allow for periodic bench clean-up.

Table 16.3 Recommended Inter-Ramp Angles for Geotechnical Domains

Wall Dip Direction	From	000°	030°	060°	090°	120°	150°	180°	210°	240°	270°	300°	330°
	To	030°	060°	090°	120°	150°	180°	210°	240°	270°	300°	330°	360°
Wall Strike Direction	From	270°	300°	330°	000°	030°	060°	090°	120°	150°	180°	210°	240°
	To	300°	330°	360°	030°	060°	090°	120°	150°	180°	210°	240°	270°
1_5 West (Combined) GT_DOMCODE = 1													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		8.5	8.5	8.5	13.5	17.5	13.5	13.5	16	15	12.4	12.4	8.5
Inter-Ramp Angle (IRA)		47	47	47	38.3	33	38.3	38.3	35	36	40.3	40.3	47
2_2 East (Volcanic) GT_DOMCODE = 2													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		9	9	9	11	11	12	12	9	9	9	9	9
Inter-Ramp Angle (IRA)		46	46	46	42	42	41	41	46	46	46	46	46
2_3 East (Intrusive) GT_DOMCODE = 3													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		8	8	8	11	11	11	14	8	8	9	8	8
Inter-Ramp Angle (IRA)		48	48	48	42	42	42	38	48	48	46	48	48
2_4 East (Sediment) GT_DOMCODE=4													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		8	8	9.5	11	15	9.5	7.5	7.5	9.5	9.5	7.5	7.5
Inter-Ramp Angle (IRA)		48	48	45	42	36	45	49	49	45	45	49	49
3_2 South (Volcanic) GT_DOMCODE = 5													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		7.5	10.6	11	13.5	13.5	13.5	7.5	7.5	7.5	7.5	7.5	7.5
Inter-Ramp Angle (IRA)		49	43	41	38	38	38	49	49	49	49	49	49
3_3 South (Intrusive) DOMS = 6													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		7.5	13.5	13.5	13.5	10	10	7.5	7.5	7.5	7.5	7.5	7.5
Inter-Ramp Angle (IRA)		49	38	38	38	44	44	49	49	49	49	49	49
3_4 South (Sediment) – 2_4 GT_DOMCODE = 7													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		9	9	10	11	16	9.5	7.5	7.5	9.5	9.5	7.5	7.5
Inter-Ramp Angle (IRA)		46	46	44	42	35	45	49	49	45	45	49	49
4_5 Solongo (Combined) GT_DOMCODE = 8													
Bench Face Angle (BFA)		70°	70°	–	–	–	–	–	–	–	–	70°	70°
Bench Height (m)		15	15	–	–	–	–	–	–	–	–	15	15
Berm Width (m)		7.5	11	–	–	–	–	–	–	–	–	7.5	7.5
Inter-Ramp Angle (IRA)		49	42	–	–	–	–	–	–	–	–	49	49
5_5 Southwest (Combined) GT_DOMCODE = 9													
Bench Face Angle (BFA)		70°	70°	70°	70°	–	–	–	–	–	–	–	70°
Bench Height (m)		15	15	15	15	–	–	–	–	–	–	–	15
Berm Width (m)		7.5	11	17.5	17.5	–	–	–	–	–	–	–	7.5
Inter-Ramp Angle (IRA)		49	42	33	33	–	–	–	–	–	–	–	49
6_5 Northwest (Combined) GT_DOMCODE = 10													
Bench Face Angle (BFA)		–	70°	70°	70°	70°	70°	70°	–	–	–	–	–
Bench Height (m)		–	15	15	15	15	15	15	–	–	–	–	–
Berm Width (m)		–	14.4	14.4	14.4	12.5	12.5	8	–	–	–	–	–
Inter-Ramp Angle (IRA)		–	37	37	37	40	40	48	–	–	–	–	–
7_5 Middle (Combined) GT_DOMCODE = 11													
Bench Face Angle (BFA)		70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°	70°
Bench Height (m)		15	15	15	15	15	15	15	15	15	15	15	15
Berm Width (m)		7.5	7.5	7.5	7.5	7.5	12	12	11	11	8	7.5	7.5
Inter-Ramp Angle (IRA)		49	49	49	49	49	41	41	42	42	48	49	49

16.1.1.3 Hydrogeological Assessment

The hydrogeological study was completed by Schlumberger and RPS Aquaterra. The hydrogeological field programme (packer tests) was carried out in six boreholes and involved 3,500 m of diamond drilling with in situ downhole testing in each hole to quantify the hydrogeological characteristics of targeted fault / dyke structures and the rock mass fabric. Each hole was also completed with either standpipe monitoring wells or cemented VWP transducers.

The rock mass is expected to exhibit low to very low hydraulic conductivity; however, Schlumberger recommended that long horizontal drains be installed to ensure adequate depressurization behind the pit walls. Pore-pressure monitoring will be required to investigate whether natural (or gravitational) drainage is adequate or if sub-horizontal drains are required. Installation of additional piezometers may be required for this monitoring.

Surface drainage ditches should be maintained along the outside perimeter of the pit to collect and convey surface water away from the pit slopes in areas where water run-off could affect the stability of the pit face.

For the overburden, slope stability will also vary with slope height and overburden thickness. It was recommended that the slopes be fully dewatered and then excavated with different slopes as a function of height, varying from 2.4H : 1V (horizontal : vertical) for slopes of 30 m in height, to 0.8H : 1V for slopes of <5 m in height.

16.1.1.4 Geotechnical Hazards

Golder reviewed the updated phased pit shells from the geotechnical viewpoint and developed a preliminary hazard model. This was subsequently refined by the Oyu Tolgoi planning team by applying the following individual hazard ratings, varying from 0–2, for the OyuT pit slope design:

- Rock Mass Rating (RMR) values:
 - RMR 0–20 : hazard rating = 1.0
 - RMR 21–40 : hazard rating = 0.75
 - RMR 41–60 : hazard rating = 0.5
 - RMR 61–80 : hazard rating = 0.25
 - RMR 81–100 : hazard rating = 0
- Rock type:
 - Ignimbrite : hazard rating = 1.0
 - Sediments : hazard rating = 0.75
 - Other rocks : hazard rating = 0
- Proximity to fault type of alteration:
 - Clay : hazard rating = 1.0
 - Chlorite : hazard rating = 0.6

- Sericite : hazard rating = 0.4
- Variation of inter-ramp angle:
 - 36°–38° : hazard rating = 2.0
 - 38°–42° : hazard rating = 1.6
 - and so on, until >50° : hazard rating = 0

An overall hazard index was estimated by summing all the individual hazard ratings; the index varies from 0–6. As an example, a pit wall located close to faults, with poorer rock mass quality and argillic alteration, would receive a high hazard index rating.

The overall hazard index mapping identified the following areas as having higher geotechnical hazards:

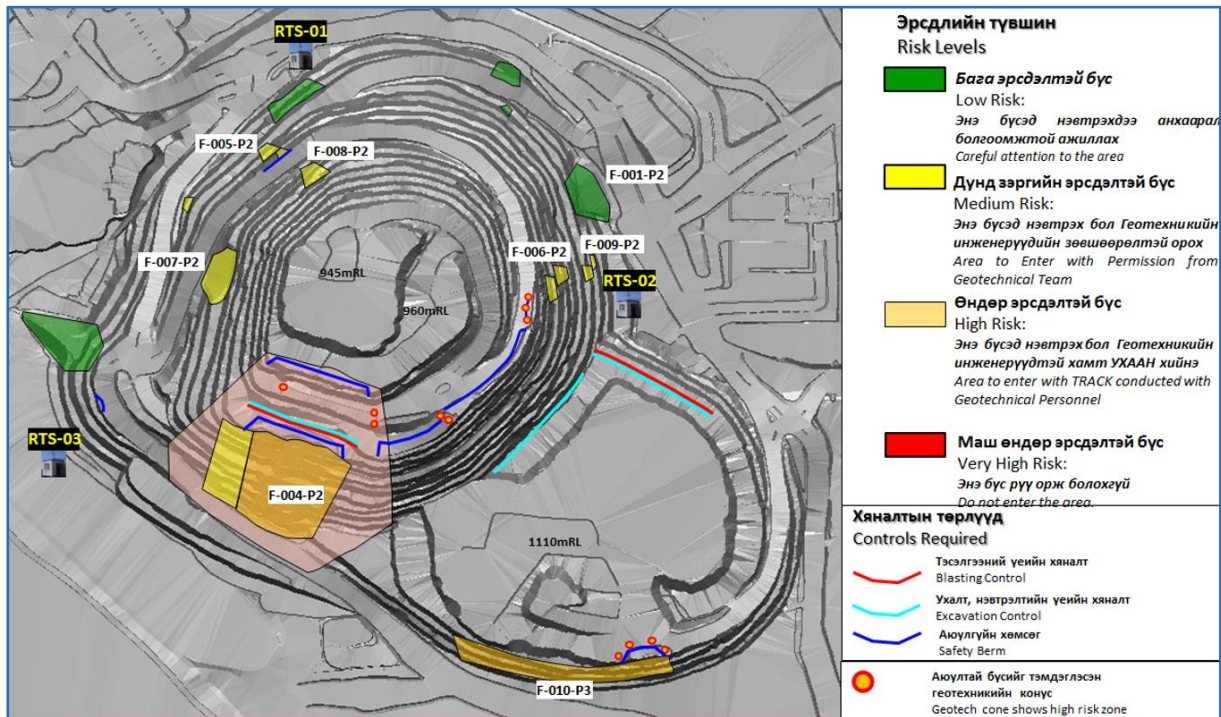
- Rhyolite Fault separating the Central and Southwest pit areas.
- Inclined AP01 Fault daylighting on the southern wall of the Central pit.
- Occurrence and proximity to the other major faults, including the Solongo Fault.
- Occurrence of the potentially weaker ignimbrite rock mass on the East domain.
- Unfavourable dipping (and continuous) major set on the West wall of the final pit.

The hazard index mapping was also used to assist in determining the best locations for the switchbacks on the ramps.

Slightly conservative pit slope configurations in areas with high hazards are recommended. As a result, the geotechnical hazards (and risks) are considered to be low for the recommended slope design. Small bench-scale failures can be expected locally where structural conditions are unfavourable or blasting practices are poor. Additionally, seasonal variations in temperature such as winter freeze and spring thaw may adversely affect bench stability and increase rock fall hazards.

The current geotechnical hazards for the open pit in Phase 2 are divided into four risk categories, Low, Medium, High, and Very High. Figure 16.4 shows the current risk areas and categorization in the open pit. The risk level determines the access control and attention paid to each area.

Figure 16.4 Geotechnical Hazard Map 22 – June 2015



16.1.1.5 3D Numerical Modelling

The open pit mine area has been divided into 10 phases. Three-dimensional (3D) stability numerical analyses were carried out for two critical periods over the LOM: 2017 (end of Phase 3), and 2032 (end of Phase 8).

The 3D results indicate that the proposed pit walls are expected to be stable based on the pit shell configurations, material properties, and groundwater conditions considered in these analyses. These results confirmed the results of the two-dimensional (2D) limit equilibrium stability assessments that were carried out on 10 representative cross-sections.

The 3D numerical analyses also indicated that while faulting clearly influences the patterns of deformation in the pit slopes, complex 3D failure mechanisms associated with faulting were not predicted at the end of 2017 (Phase 3) or 2032 (Phase 8).

Areas where yielding and ground movements were predicted, were relatively localized within the pit walls and did not extend to pit infrastructure on the ground surface.

16.1.1.6 Pit Slope Operational Considerations

Good quality operational practices are essential for the safe development of stable pit slopes, particularly effective controlled blasting and excavation procedures. Optimized controlled blasting designs and operational outcomes will be incorporated into slope design criteria.

Recommendations related to operational considerations include:

- Double-benching to maximize slope angles.
- Controlled blasting with pre-split blasting to maximize slope angles.
- Thorough bench clean-up and scaling using equipment that can safely reach the crest of the bench.

With the exception of the eastern walls within the ignimbrite, double-benching may be possible if the bench faces are clean and stable to enable safe drilling and excavation.

For practical reasons, the Oyu Tolgoi team decided to use four berm widths in design: 7.5 m, 9 m, 12 m, and 15 m. However, berm widths of 10.5 m and 13.5 m, for example, could be considered for the final wall to increase the IRA by 1°–2° in some design sectors.

To better define pit slope angles, the continuing geotechnical drilling through the Pit Slope Management Programme (PSMP), begun in 2013, has targeted the areas scheduled for mining within the Southwest mineralization boundary in the next three to five years. A better understanding of the geological conditions in the field is particularly important for setting the pit slope parameters for Phases 4, 5, and 9.

16.1.1.7 Current Geotechnical Programmes

Pit Slope and Inter-Ramp Angle

A primary purpose of the PSMP geotechnical drilling programme is to increase the level of confidence in slope design criteria and to support the evaluation of options to steepen the pit slopes (inter-ramp slope angle).

Data gathered from the geotechnical drilling programmes included structural geological information from oriented core, ATV images, and samples for laboratory testing of intact rock and discontinuity strength. The finished geotechnical holes were subsequently used for the installation of VWPs; time domain reflectometry monitoring; and packer testing in selected boreholes for hydrogeological design purposes.

Packer testing and installation of piezometers was carried out to determine the natural hydraulic conductivity of bedrock essential for the design of in-pit pumping, wall depressurization planning, and slope design. In addition, it facilitated:

- monitoring for possible transient increases in bedrock groundwater pressures and flux rates due to occasional Undai River recharge events
- monitoring the performance of the Undai River diversion (piezometers between the Undai River and the west pit wall)
- monitoring potential recharge and increased pore pressure as the TSF expands (piezometers between the planned TSF and the east wall).

The hydrogeological outcomes are fully integrated into the geotechnical analysis, using sophisticated 2D and 3D numerical techniques with the aim of providing pit wall design parameters that are compatible with achievable depressurization targets relative to the mine excavation schedule.

Actual batter and IRA slope performance is reconciled against pit slope designs after the excavation of every trim blast. The results are subsequently fed back into the ongoing analysis of pit slope design criteria for future phases. The results of this reconciliation for Phase 2 are summarized in Figure 16.5 to Figure 16.8.

Figure 16.5 Phase 2 Excavation Compliance Index – Crest

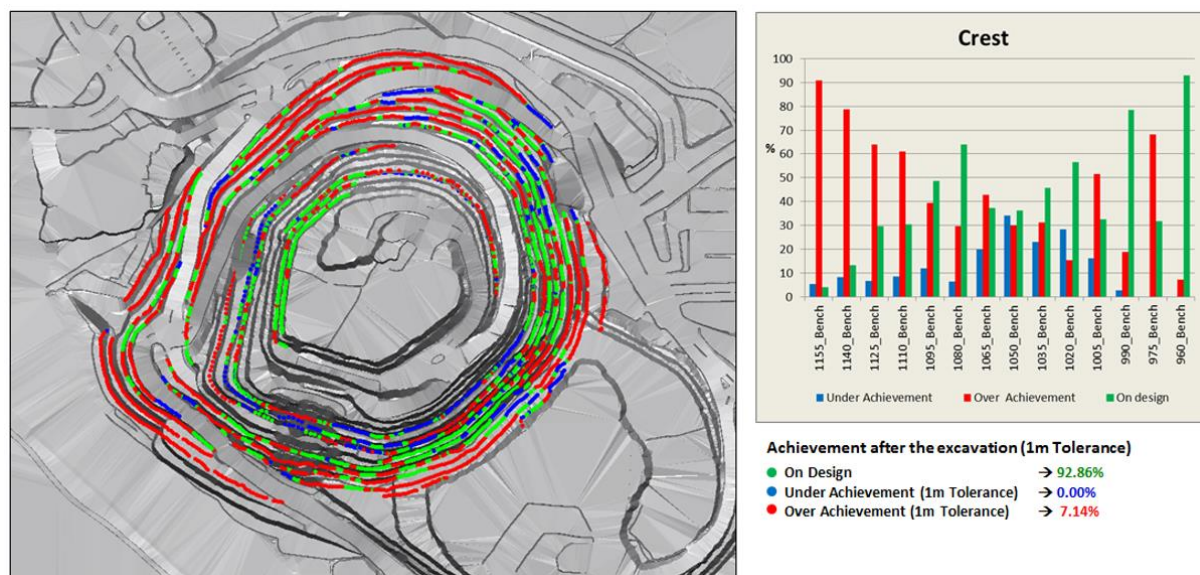


Figure 16.6 Phase 2 Excavation Compliance Index – Toe

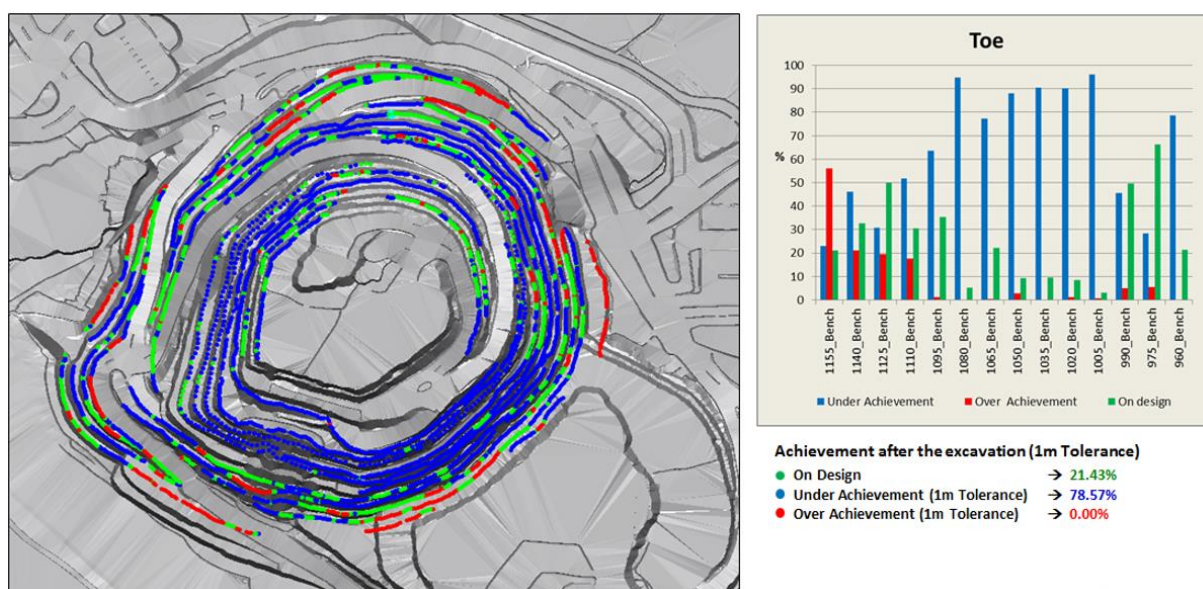


Figure 16.7 Excavation Compliance Index – Bench Face Angle

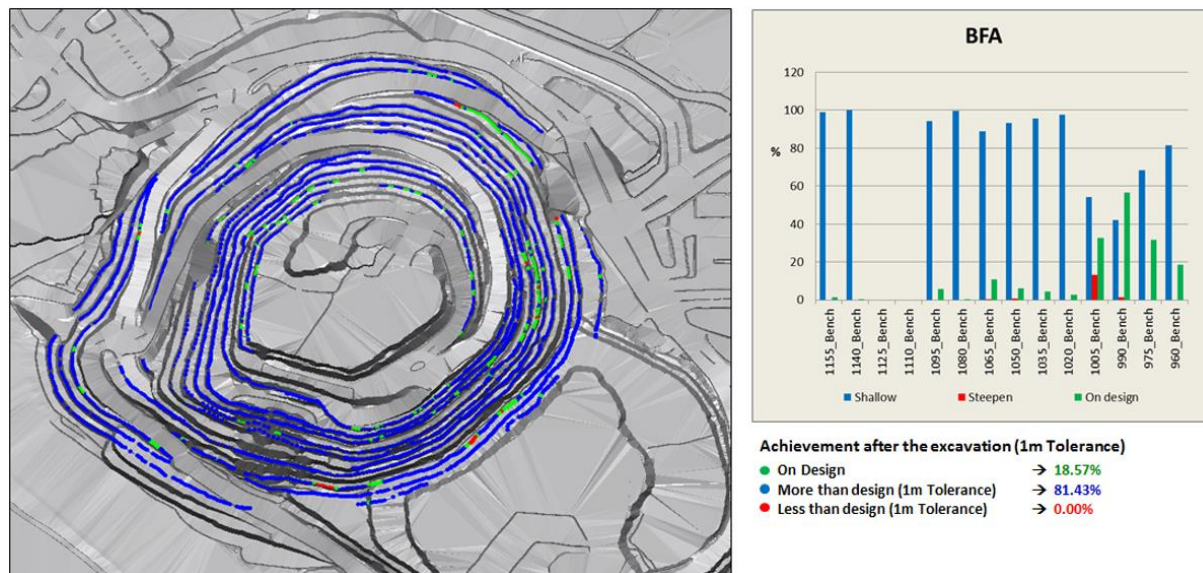
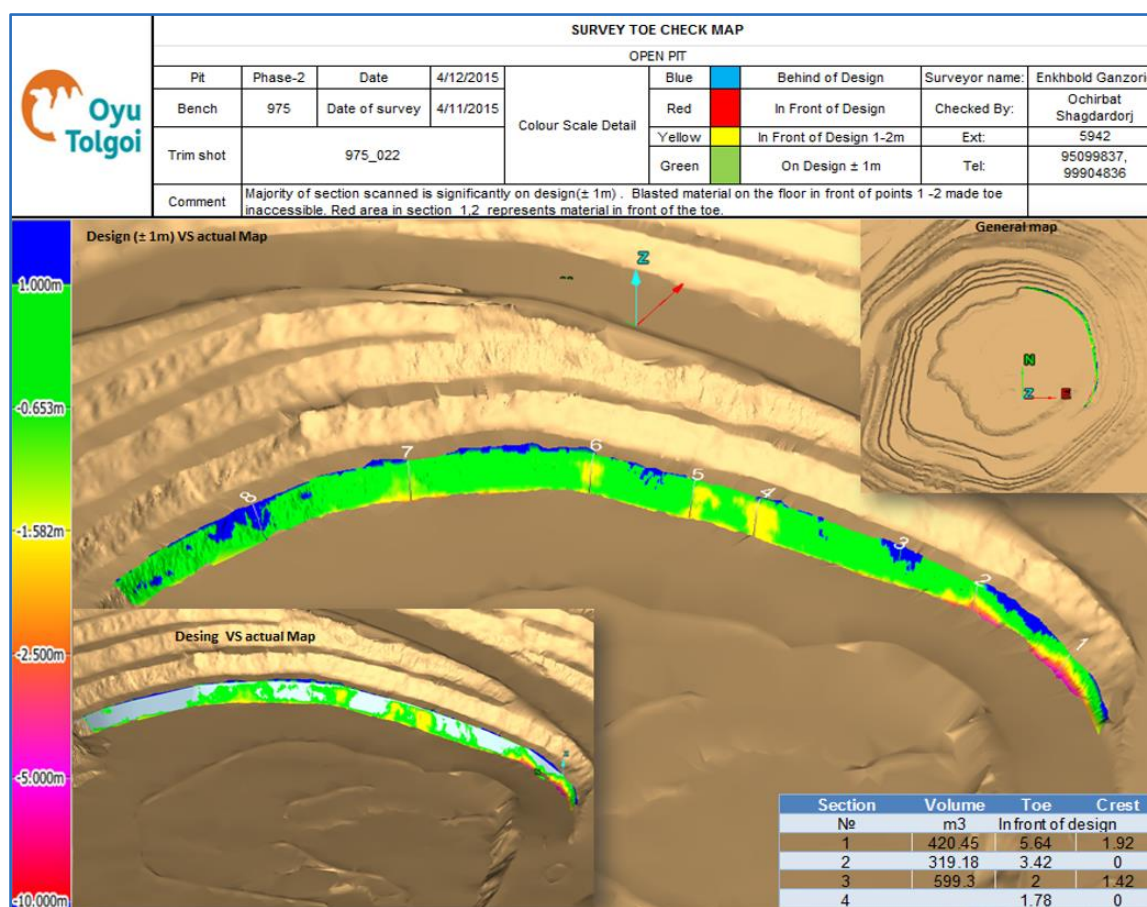


Figure 16.8 Batter Check Report – 975 bench



16.1.2 Open Pit Mine Plan

OT LLC carried out the mine planning and scheduling work for the Oyu Tolgoi open pits, including the integration of the underground plans. The mine plan has a start date of January 2016 and schedules 1,069 Mt of ore, along with 1,970 Mt of waste, in eight pit phases. The open pit reserves are mined over approximately 40 years, continuing while underground mining commences with the Hugo North Lift 1 block cave. The OTFS16 work is based on the latest resource model ('mpst15.v9') and Base Data Template 31 (BDT31) metallurgical response parameters, unchanged from the 2014 Technical Report.

The optimization, designs, and production scheduling were completed using Measured and Indicated resources only, with Inferred Resources treated as waste.

16.1.2.1 Mining Model

The recovered copper, gold and silver grades, and the NSR were calculated for each block. The calculated 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, and excludes costs for mining, processing, and G&A. NSR is the in situ value after allowances have been made for:

- Recovery to concentrate
- Smelter deductions
- Concentrate transport
- Smelter treatment and refining charges
- Royalties

The resulting NSR values were used to classify the ore blocks in the pit optimization process. The NSR values for the 2016 OTTR were calculated using the parameters described in Table 16.4. The NSR calculation does not include penalties for impurities such as arsenic and fluorine. The penalties have been minimized through blending in the production schedule and are accounted for in the financial analysis.

Table 16.4 Base Data Template 31 (BDT31)

Parameter	Copper	Gold	Silver
Payment and Deductions			
Metal Prices	US\$3.01/lb	US\$1,250/oz	US\$20.37/oz
Payment Levels	96% 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, 97.5% of Au content	–
Deductions	Minimum one unit/dmt	n/a	All <30 g/dmt
Treatment and Refining			
Treatment (Concentrate)	–	US\$80/dmt	–
Refining and PP	US\$0.08/lb	US\$8/oz	US\$0.45/oz
Price Participation	US\$0.00/lb	–	–
Penalties	Arsenic: No penalty to 3,000 ppm. Thereafter US\$2/dmt penalty per 1,000 ppm. Rejection level 5,000 ppm.		
	Fluorine: No penalty to 1,000 ppm. Rejection level 1,000 ppm.		
Royalties			
GOM	5% of payable metal value + IVN Royalty 2% (non-tax deductible)		–
Transport			
Moisture Content	8%	–	–
Transportation	US\$25/wmt	–	–

16.1.2.2 Pit Optimization

The pit optimization was completed by OT LLC using the current resource model. Only resources classified as Measured or Indicated were used in the optimization. Inferred resources were treated as waste. The pit phases have been selected to provide increasing value between the phases. The pit shells for Phases 4–5 were determined by expanding the pit out of Phase 2 at the minimum 80 m bench width. The goal is to reach the bottom of the gold core as soon as possible. Phase 10 is the ultimate pit. Phases 8 and 9 are mainly a pushback designed to trim Phase 10. Pit Phase Selection. Phases 8, 9 and 10 would be split into smaller phases that would be designed in the years before they are mined.

16.1.2.3 Mine Design

The open pit design work focused on the Southwest and Central zones of the Oyut deposit. The zones fall within the following pit phases:

- Phase 2 Southwest
- Phase 3 Southwest
- Phase 4 Southwest
- Phase 5 Southwest
- Phase 6 Central
- Phase 7 Central (predominately chalcocite) and Southwest
- Phase 8 Southwest limit with Central (stripping phase)
- Phase 9 Southwest
- Phase 10 All Oyut zones

The optimized pit shells were developed into mining phases to accomplish the following objectives:

- Minimize mining costs and maximize economic return by exposing the highest value ore with minimum waste mining
- Address operational requirements for loading, hauling, slope stability, and rockfall, as follows:
 - Loading – the phases were designed with a minimum operational width of 80 m between phases to allow efficient mining for the equipment scale
 - Hauling – generally, two exit haul roads per phase were included: the west-bound exit to the crusher, low-grade stockpile, and west dump; and the east-bound exit to the NAF and PAF dumps. Haul roads were generally 40 m wide at a 10% gradient.
 - Slope Stability and Rockfall – the phase designs are presented in toe-and-crest format with 15 m bench height, recommended pit slope angles, and 6 m to 12 m wide berms.

Consideration was given to narrowing the width of the haul road in the last few benches of all phases. This involved having a 25 m wide haul road in the bottom of the phase to provide enough room to add a few benches in these phases. Although this could be achievable, the delays associated with this concept need to be further understood before this could be implemented. In practice when the phases are mined each phase will be split into smaller stages to optimize the operational efficiency. For example, in 2016 Phase 4 has been split into two stages, referred to as Phase 4a and Phase 4b.

The open pit slope angles implemented in the pit optimization were used as a guide for the phase designs. This method allows the bench face angle (BFA), catch bench width (CBW), and ramps to be incorporated into the phase designs.

A plan view of the 10 pit phases is shown in Figure 16.9 and Figure 16.10. Phases 1 is complete and there is a residual amount in Phase 2 that will be mined with Phase 4.

Figure 16.9 Open Pit – Individual Phase Designs

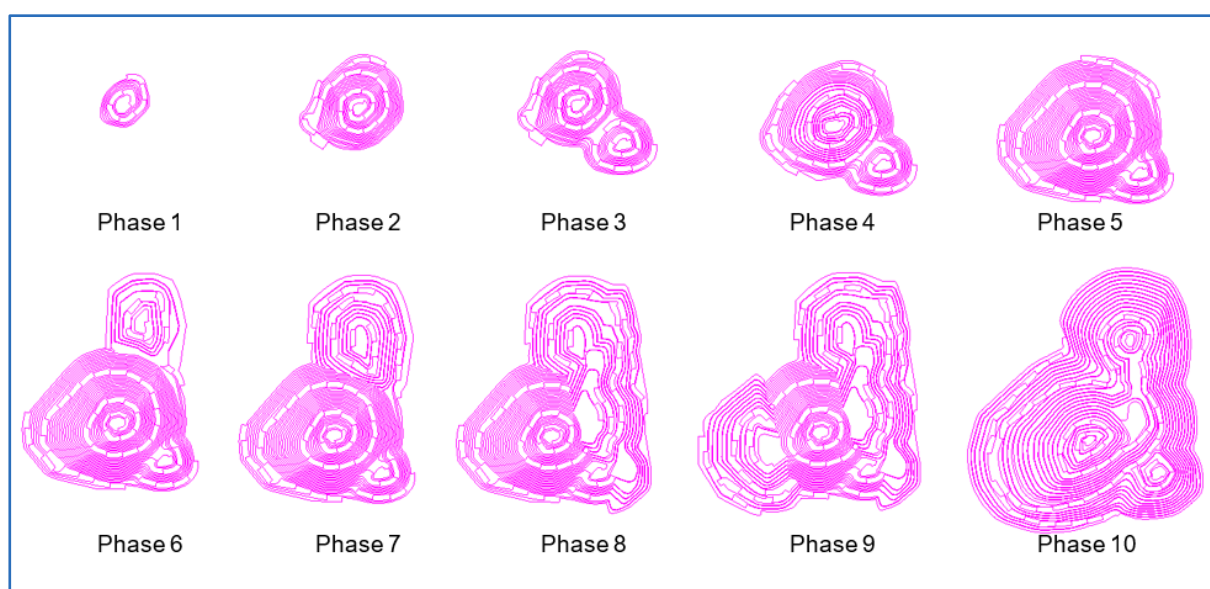
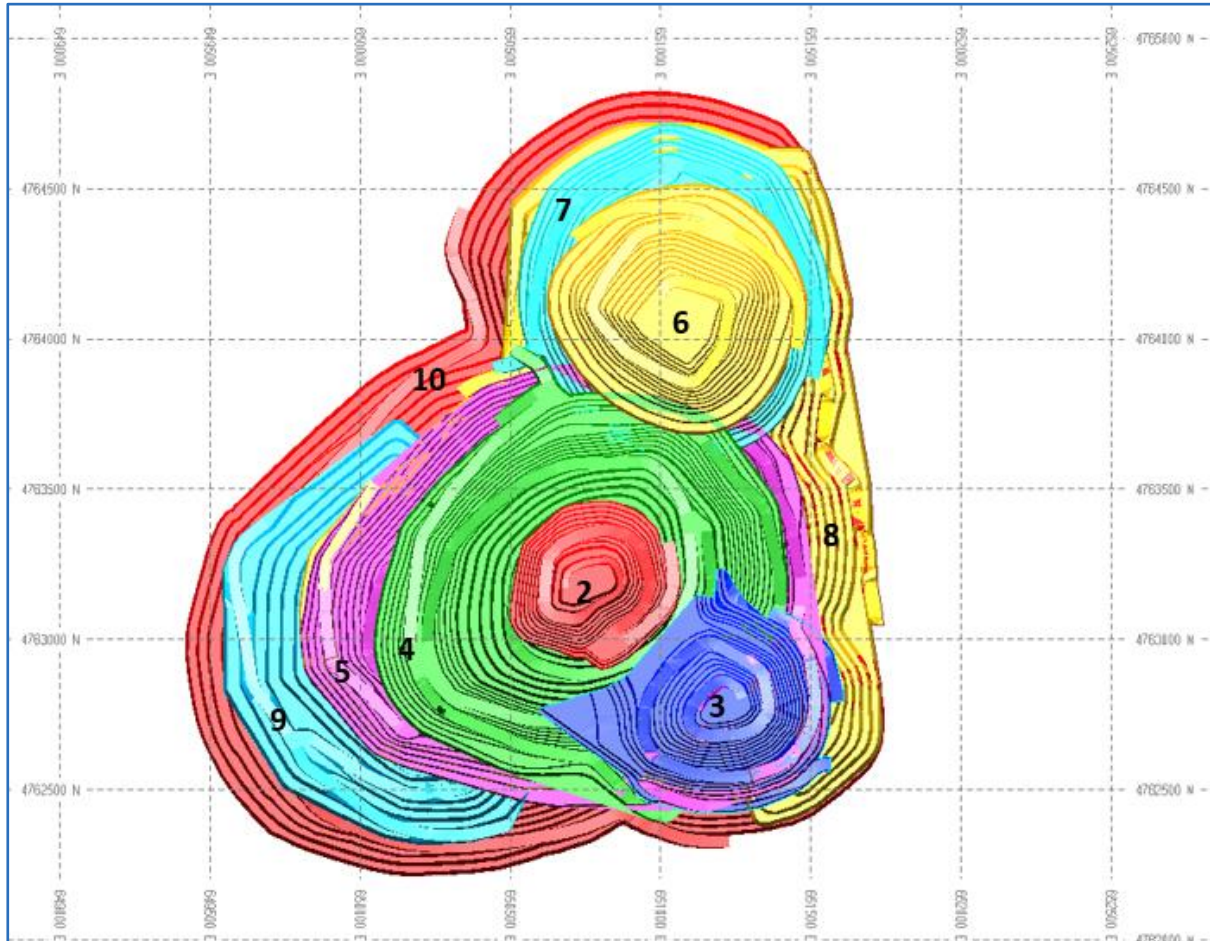


Figure 16.10 Open Pit – Nested Phase Designs



16.1.2.4 Open Pit Mining Inventory Summary

The open pit mining inventory is summarized by pit phase in Table 16.5.

Table 16.5 Summary of Total Material by Pit Phases

Phase	Ore (Mt)	Waste (Mt)	Strip Ratio	NSR (US\$/t)	Cu (%)	Au (g/t)	Ag (g/t)	F (ppm)	As (ppm)	Total Material (Mt)
Phase 2	5	–	–	82.31	0.83	1.48	1.94	2,244	10	5
Phase 3	25	4	0.15	29.73	0.53	0.17	1.73	1,916	76	28
Phase 4	140	180	1.28	28.87	0.41	0.38	1.23	1,716	22	320
Phase 5	120	221	1.84	24.34	0.38	0.27	1.14	1,554	45	341
Phase 6	51	34	0.66	24.13	0.63	0.08	1.43	2,240	257	85
Phase 7	81	75	0.92	21.89	0.55	0.10	1.16	1,920	196	157
Phase 8	60	236	3.95	20.38	0.43	0.05	0.88	1,868	123	296
Phase 9	33	199	5.97	21.89	0.33	0.30	1.00	1,360	19	232
Phase 10	435	833	1.92	26.57	0.44	0.32	1.23	1,715	73	1,267
Total	950	1,782	1.88	25.90	0.45	0.28	1.21	1,745	83	2,732

Note: As at 31 December 2015.

16.1.2.5 Open Pit Operating Schedule

Open pit mining operations are scheduled on a nominal 365 d/a calendar, 24 h/d, resulting in a total of 8,760 calendar hours per year.

The effective utilization and productivities for the primary earth-moving fleet are based on consideration of the 2015 performance. Table 16.6 and Table 16.7 compare the 2015 demonstrated and OTFS16 parameters for equipment performance.

Table 16.6 Demonstrated Equipment Performance 2015

Equipment	Annual Production (Mt)	Rate (t/h)	Availability (%)	Utilisation (%)	Operator Efficiency (%)	Effective Utilisation (PA x UA x OE) (%)
Electric Shovel	29.8	6,860	88	86.9	64.9	49.6
Hydraulic Shovel	17.4	4,436	88	76.2	66.6	44.7
Loader	9.0	2,663	85	64.8	69.9	38.5
Haul Truck	-	280 (payload)	92	85.6	87.1	68.6

Table 16.7 OTFS16 Equipment Operating Assumptions

Equipment	Annual Production (Mt)	Rate (t/h)	Availability (%)	Utilisation (%)	Operator Efficiency (%)	Effective Utilisation (PA x UA x OE) (%)
Electric Shovel	31.8	6,860	85	80.5	77.3	52.9
Hydraulic Shovel	19.2	4,436	85	76.2	76.3	49.4
Loader	10.6	2,663	85	67.3	79.5	45.4
Haul Truck	-	285 (payload)	87	85.6	87.1	64.9

16.1.2.6 Labor

A labor model for Oyu Tolgoi open pit operations has been prepared by taking the planned 2016 organizational structure and projecting forward the anticipated requirements based on mining fleet numbers and scale of operations. This includes a substantial reduction in expat roles. Although total material movement almost halves over the next 10 years with Hugo North Lift 1 coming on line, the open pit labor levels are not affected proportionally. This is because almost the full range of support services needs to be kept in place to maintain ore production along with waste stripping for tailings dam wall construction. In addition, the haulage fleet reduces by less than 30% to meet increasing haulage burdens. In the longer term, even though total material movement is not as great, the haulage fleet numbers increase, pushing labor levels up.

16.1.2.7 Mining Equipment

The open pit mine at Oyu Tolgoi is a conventional shovel-truck operation. OT LLC's workforce carries out drilling, loading, hauling, and associated production support roles. Equipment maintenance is conducted under Service Agreements with the original equipment manufacturers in-country dealers. A blasting contractor provides blasting products and down the-hole services.

This operation makes use of a mixed fleet of 34 m³ diesel hydraulic shovels and 56 m³ electric rope shovels working in tandem with 290 t haul trucks.

The equipment fleet requirements 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.

Table 16.8 Open Pit Mining Equipment in Service

Unit	2016	2020	2025	2030	2035	2040	2045
Shovel – 34 m ³ Hydraulic Diesel, RH340	2	2	1	1	1	2	2
Shovel – Rope Electric, 495HR	2	2	2	2	2	2	2
Front End Loader – WA1200 18 m ³	2	2	0	0	0	0	0
Komatsu PC9400	1	1	1	1	1	1	1
Truck 290 t – 930E-4SE	28	33	33	28	31	34	39
Production Drill – Diesel, PV-351D	2	2	1	1	2	2	2
Production Drill – Electric, PV-351E	2	2	2	2	2	2	2
Small Drill	2	2	1	1	2	2	2
Caterpillar 16M	3	4	4	4	4	4	4
Komatsu D475	2	2	2	2	2	2	2
Komatsu D375	4	4	4	4	4	4	4
Komatsu WD600	2	2	2	2	2	2	2
Komatsu HD785WT	2	3	3	3	3	3	3
Komatsu PC600	1	1	1	1	1	1	1
Komatsu WA500	1	1	1	1	1	1	1
Komatsu WA250	2	2	2	2	2	2	2
HM400 Service Trucks	2	2	3	2	2	2	3
CAT735 Service Truck	1	1	1	1	1	1	2

The existing mine equipment fleet is monitored through Dispatch and Mine Care, a computerized system available in the industry. The information collected is used for fleet performance management and continuous improvement.

High-precision GPS systems are currently installed in the heavy mobile equipment (HME), including the electric shovels, drills, hydraulic shovels, and loaders. This will allow live monitoring and decision-making with regard to mining direction and performance once a steady supply of ore is available to the concentrator.

After three years of operation, the equipment productivities being achieved in the open pit are aligned with prior assumptions, as well as those in this study.

16.1.2.8 Drilling and Blasting

Production blast holes are drilled 16.5 m deep x 311 mm diameter. The OT LLC team designs, fires, and monitors the blasts, while a contractor (Maxam) provides down-the-hole services, including the supply and storage of explosives. It is assumed that ammonium nitrate fuel oil (ANFO) is used in dry holes and high-density explosives (RIOFLEX) in wet holes. Given the prevailing conditions, up to 90% of holes so far have been found to be wet, requiring the use of 90% high-density explosives for blasting. The proportion of wet blasting may reduce as the

height of the cutbacks of the phases increases with depth of the pit and the pit walls are drained.

Almost all materials within the open pit are assumed to be blasted; free digging is only possible in the first or second benches covered by weathered or clay material.

Two of the production drills are diesel units and two are electric. As replacements are required, one diesel drill will be maintained in service and the other will be replaced with an electric power drill. Table 16.9 shows the drilling and blasting design parameters.

Table 16.9 Drill and Blast Design

Drilling Parameters	Unit	Production Shot		Trim Shot		
		Ore	Waste	Front Row	Buffer Row	Trim Row
Drillhole diameter	mm	311	311	311	172	172
Penetration rate	m/h	30	32	30	32	32
Bench height	m	15	15	15	15	15
Burden	m	7.5	8.4	7.5	4.6	4.2
Spacing	m	8.6	9.7	8.6	5.3	3.7
Subdrill	m	1.5	1.5	1.5	0.0	0.0
Stemming	m	7.0	7.0	8.0	8.0	1.5
Charge length	m	8.5	8.5	8.5	7.0	2.5
Total hole depth	m	16.5	16.5	16.5	15.0	15.0
Productivity						
Effective minutes per hour	min	60	60	60	60	60
Effective hole volume	bcm	968	1,222	968	366	233
Drilling time per hole	min	33	31	33	28	28
Steel change time per hole	min	0.5	0.5	0.5	0.5	0.5
Setup time per hole	min	1.5	1.5	1.5	1.5	1.5
Total time per hole	min	35	33	35	30	30
Holes per hour	holes/h	2	2	2	2	2
Re-drills	%	5	5	5	5	5
Effective production	bcm/h	1,576	2,115	1,576	692	441
Density	t/m ³	2.80	2.70	2.76	2.76	2.76
Effective drill rate per hour	m/h	27	29	27	28	28
Tonnes per Hour	t/h	4,412	5,922	4,412	1,937	1,235
Production Rate	Mt/a	38.7	51.9	387	17.0	10.8

16.1.2.9 Loading

The primary loading fleet consists of two 34 m³ diesel hydraulic shovels and two 56 m³ electric rope shovels. One of the shovels mines ore from the lower pit benches, and the other(s) mines waste for the development of the next phase. Another shovel moves between the ore and waste phases to assist in mining ore, low-grade stockpiling, and waste stripping as required.

In addition to the four production shovels, two 18 m³ front end loaders (FEL) are in operation, providing additional flexibility in ore/waste loading capacity and stockpile management.

The smaller loading units are considered support units for clean-up, stockpile retrieval, and mining support projects.

Table 16.10 shows loading fleet data used for production scheduling.

Table 16.10 Loading Unit Production

	56 m³ Bucket Shovel	34 m³ Bucket FEL	18 m³ Bucket FEL
Annual production (Mt/a)	31.8	19.2	10.6

16.1.2.10 Hauling

The mine plan takes into account the increasing haul road distances and travel times as the pit deepens over time. This is calculated and repeated for each phase and destination combination. Additional fixed times are added to every cycle to account for loading, spotting, and dumping. This information is used to calculate the required 290 t truck hours for the duration of the production plan. A truck payload of 285 t is used based upon the results of an ongoing truck payload study and reconciliation.

16.1.2.11 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 10% compaction of the dumps. It is also assumed that the blasted waste rock will settle at the natural angle of repose of 37°. The plan is to build the dumps in 15 m high lifts, forming 30 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, based on the Environmental Impact Assessment (EIA) submitted to the GOM in 2007. However, an additional 15 m lift would be required on the West and South dump for additional capacity to provide flexibility for non-acid-forming (NAF) placement and the tailings stockpile.

The design criteria consider selective placement of rock on the dumps to ensure 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, in an area where the surface does not contain clay.
- Placement of a 1–3 m thick NAF cap on the reclaimed dump surface to mitigate the effects of erosion and to inhibit water penetration into the dump.

The following dumps have been designed:

- Tailings stockpile (TSF dump) – This dump stores NAF oxide and sediment material to be used in the construction of the tailings storage facility (TSF). This stockpile is dynamic, with material dumped from the open pit being removed for TSF construction as required.

- West waste dump – This dump overlies some clay material and the adjacent Undai River; primarily used for storing PAF material.
- South waste dump – Approximately half of this dump area overlies clay and so does not require the NAF bottom lift. This dump can be managed to store different types of material depending on cycle times and waste rock characteristics.
- SOM dump – This dump stores any oxide material that contains copper grades greater than 0.25%.

The mine schedule is NPV-optimized based upon declining cut-off grade theory for mill feed and uses stockpiles to manage ore that is above the marginal cut-off but below the grade of ore more readily available from the pit. The stockpiles have been divided into three categories:

- High-grade/tactical (HG), which is direct-feed ore and variable between US\$20/t and US\$30/t NSR based upon available stockpile areas
- Medium-grade (MG), from US\$15/t NSR
- Low-grade (LG), above the marginal process cost (BDT31) at US\$8.36/t SW ore, US\$8.49/t Central Chalcocite, US\$5.68/t Central Covellite, and US\$6.34/t Central Chalcopyrite.

The waste dump and stockpile designs consider an overall slope angle of 18° after reclamation. The designs and a cross-section of the dump design geometry are shown in Figure 16.11 and Figure 16.12, and the ultimate capacities of the dumps and stockpiles are listed in Table 16.11.

Figure 16.11 Waste Dump Designs

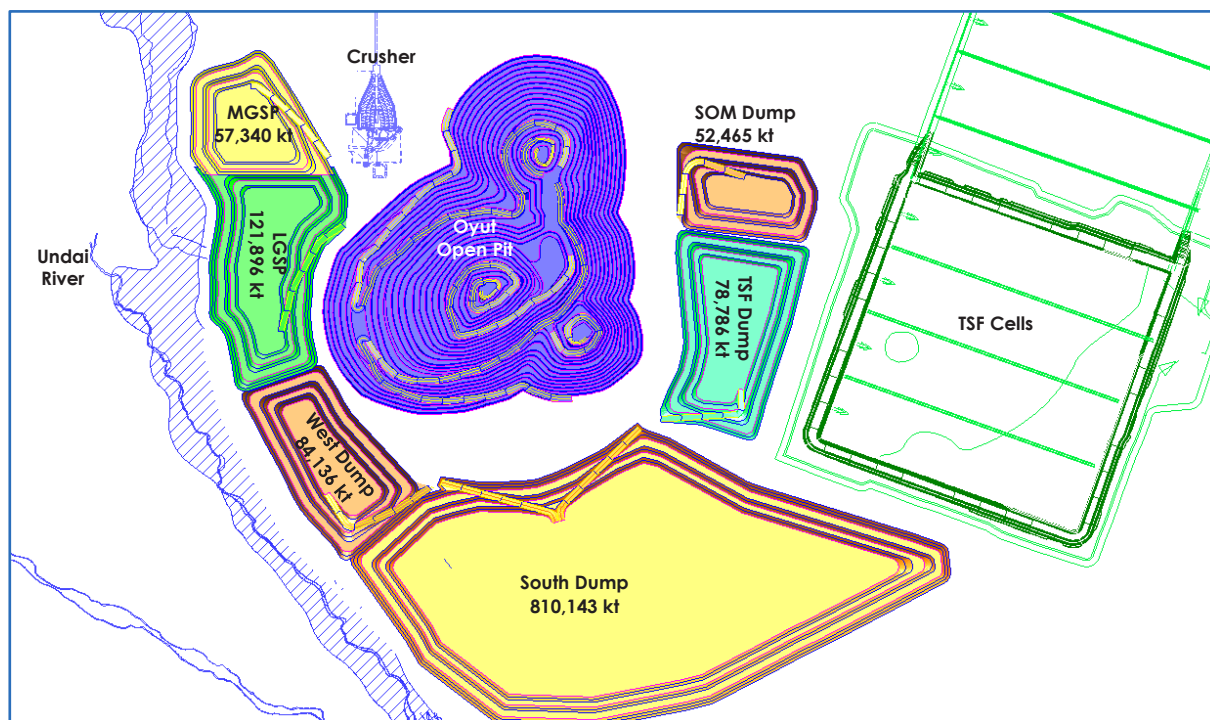


Figure 16.12 Cross-Section Showing Dump Design Geometry

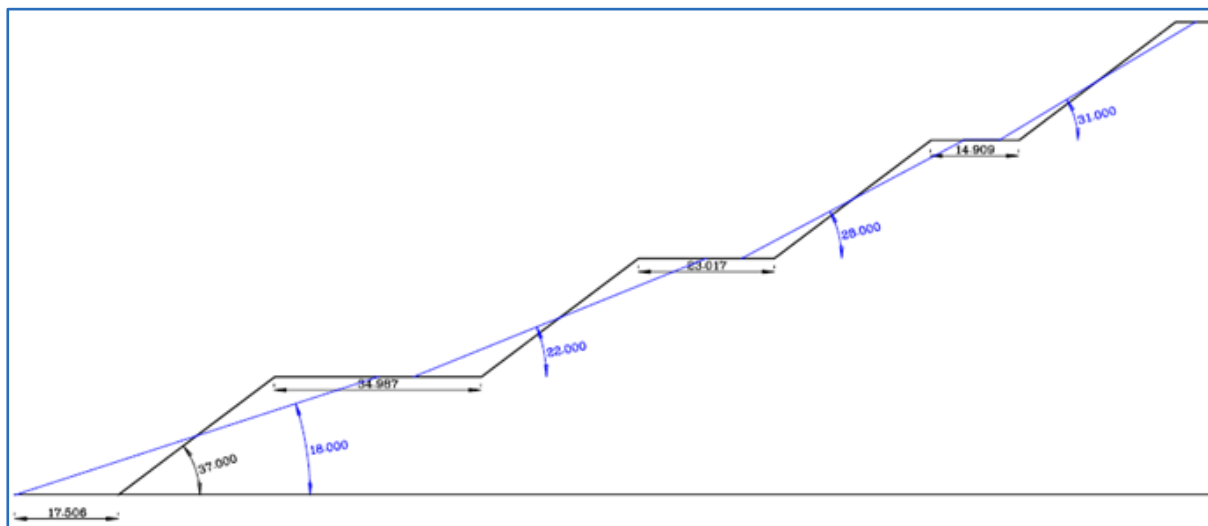


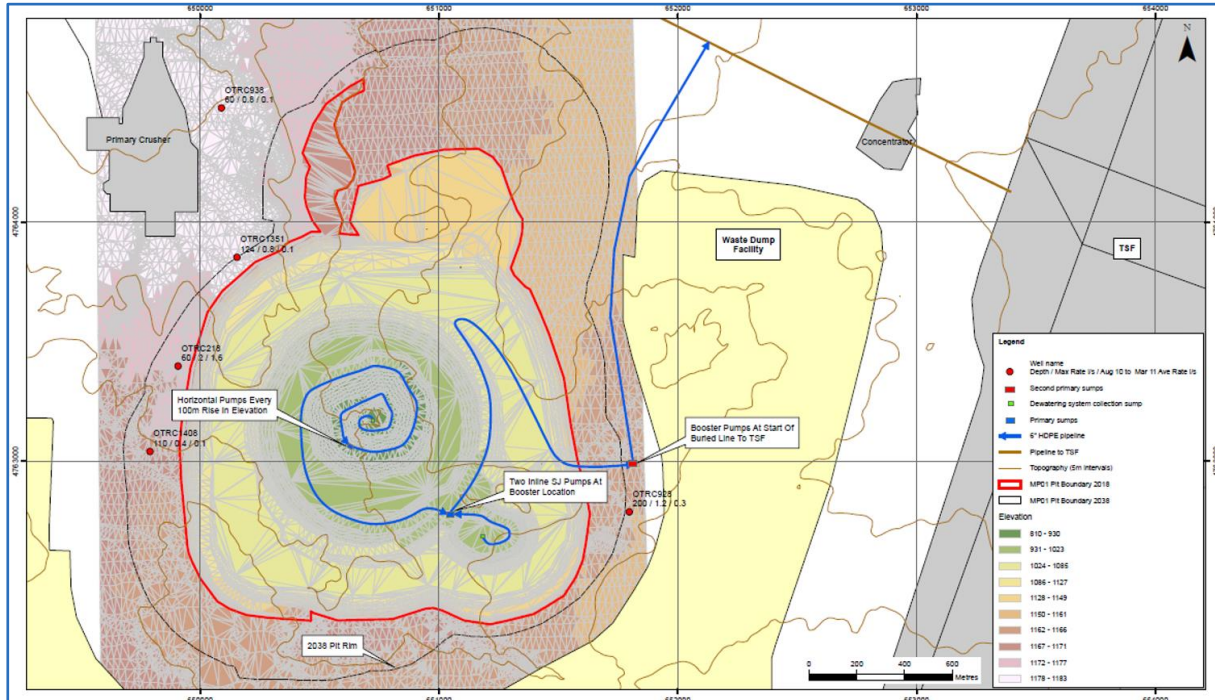
Table 16.11 Ultimate Open Pit Waste Dump and Stockpile Capacities

Waste Dump	Capacity (Mt)
TSF Dump (East)	79
West Dump	84
South Dump	810
SOM Dump	52
Total Waste	1,026
Low Grade Stockpile	122
Medium Grade Stockpile	57
Total Stockpile	179

16.1.2.12 Open Pit Mine Dewatering

As the open pit deepens, it is planned to install diesel-powered pumps for pit dewatering. The water will be pumped to a dam at the process plant. Pumps have been sized depending on pit depths. Figure 16.13 shows the dewatering system for the open pit. Pumping requirements are dependent on the mining schedule. Pump replacement and repair allowances are included in the operating costs. The pump requirement estimates are based on the bench positions of each pit phase in each year.

Figure 16.13 Open Pit Dewatering System



16.1.2.13 Open Pit Ore Definition

The open pit mine geology team updates the ore control (OC) block model with mapping data and daily blasthole assay data (copper, gold, and silver) and deleterious elements (arsenic, talc, sulphur). These data are interpolated onto a 10 x 10 x 15 m block model. This updated block model is transferred to the ore control team to continue the process of defining ore / stockpile / waste boundaries.

NSR values based on short-term metal prices are calculated for each block by the Ore Control team who then define material types and digging boundaries for the loader operators.

Reconciliation work to compare the Oyut geological block model to the OC model (blast hole data based) and mill performance shows that models are performing largely to expectations, and there are no significant concerns about grade performance from mine to mill.

16.2 Underground Mining

16.2.1 Introduction

16.2.1.1 Overview

The Hugo North, Hugo South, and Heruga orebodies are planned to be mined by underground panel caving methods. The first underground orebody to be mined is Hugo North, where two mining lifts are planned. The first three panels of Hugo North Lift 1, the basis for OTFS16, contains the highest grades of copper and gold and has the highest value.

The Hugo North Lift 1 underground construction formally re-commenced in July 2016. Development and construction activities will ramp up and continue through to the start of production in late 2019, defined as the point of commissioning the initial 30 kt/d production ore handling system. Production will ramp up to deliver an average of 95 kt/d of ore to the process plant during its peak production period from 2027 to 2035, ramping down to completion in 2039. The Hugo North Lift 1 reserves total 499 Mt at a grade of 1.66% Cu and 0.35 g/t Au.

To support mining of Hugo North Lift 1, five shafts, 203 km of lateral development, 6.8 km of vertical raise-boring, and 115,000 m³ of mass excavations will be undertaken. The Lift 1 mining levels are approximately 1,300 m below surface. The orebody has average dimensions of 2,000 m long x 280 m wide. A total of 2,231 drawpoints are planned to be developed within the mining footprint, accessed from 52 extraction drives.

During the ramp-up to 30 kt/d production, from 2020 through 2022, crushed ore will be conveyed to a storage bin at Shaft 2, where the rock will be loaded into skips and hoisted to the surface. Post-2022, following commissioning of the 6,500 t/h conveyor-to-surface system, crushed material will be transferred by incline conveyors from crushers to the surface run-of-mine (ROM) stockpile; the Shaft 2 ore hoist will provide backup capacity during maintenance of the conveyor-to-surface system.

Figure 16.14 illustrates the planned mine development superimposed with site layout, Figure 16.15 illustrates an isometric of the mine design, Figure 16.16 illustrates the annual tonnage and grade profile.

Figure 16.14 Hugo North Lift 1 Mine Design Projection

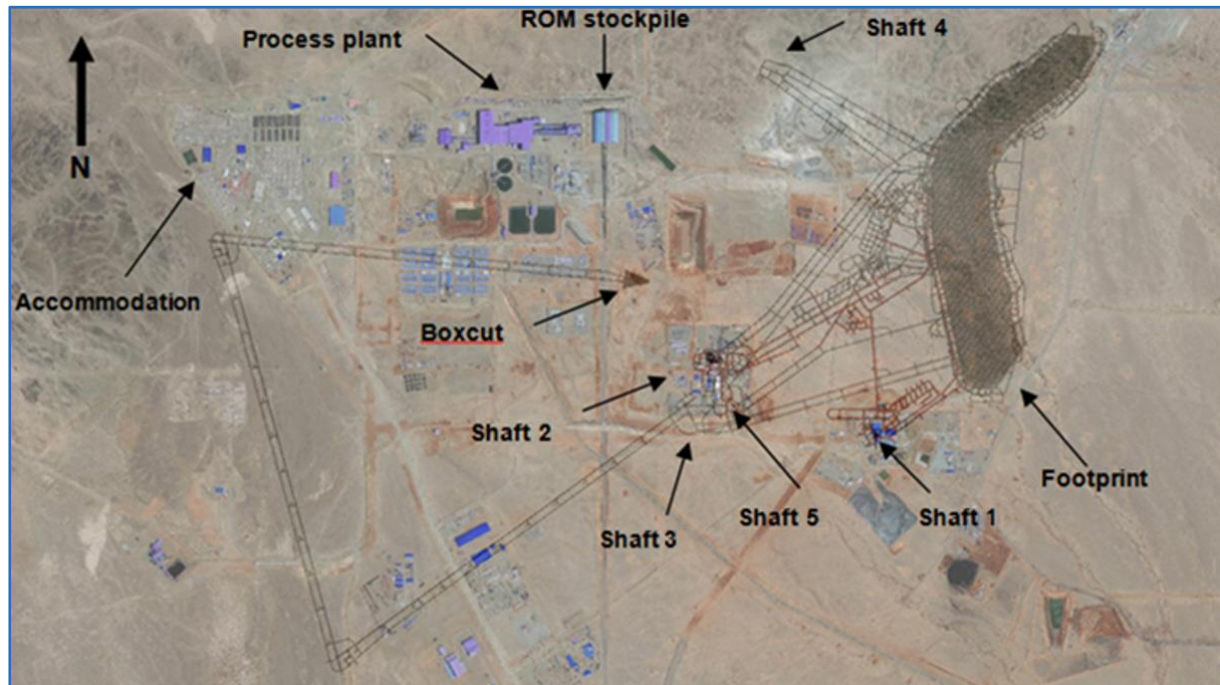


Figure 16.15 Isometric of Mine Design

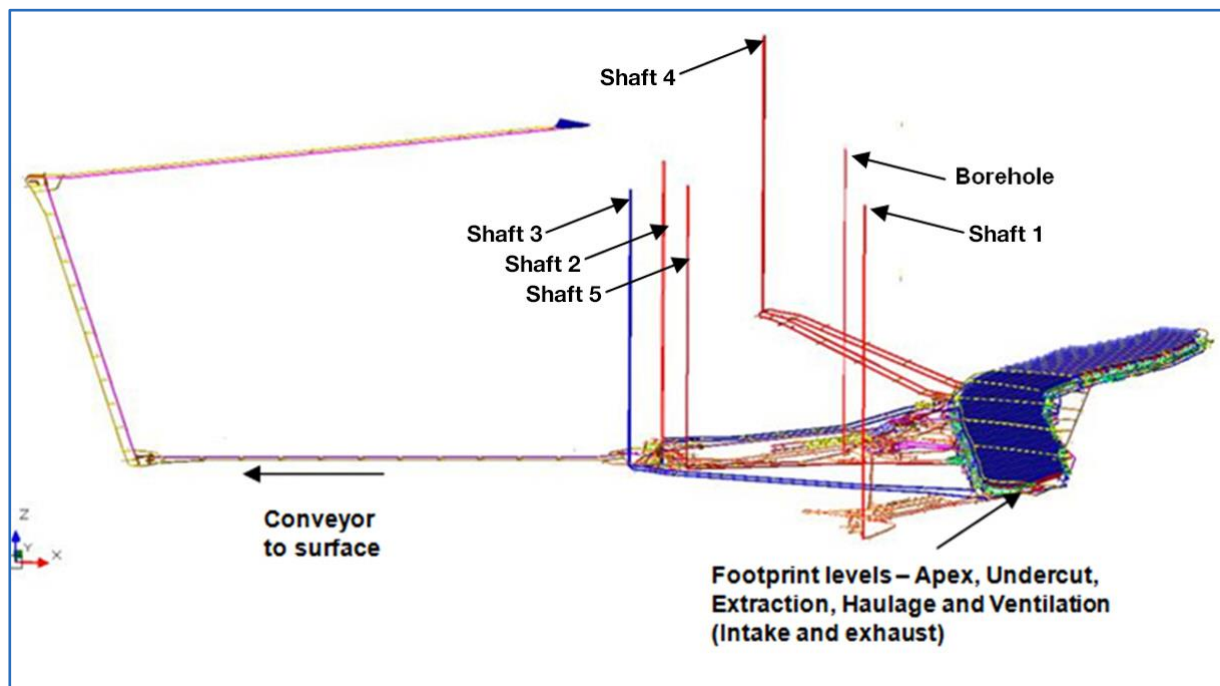
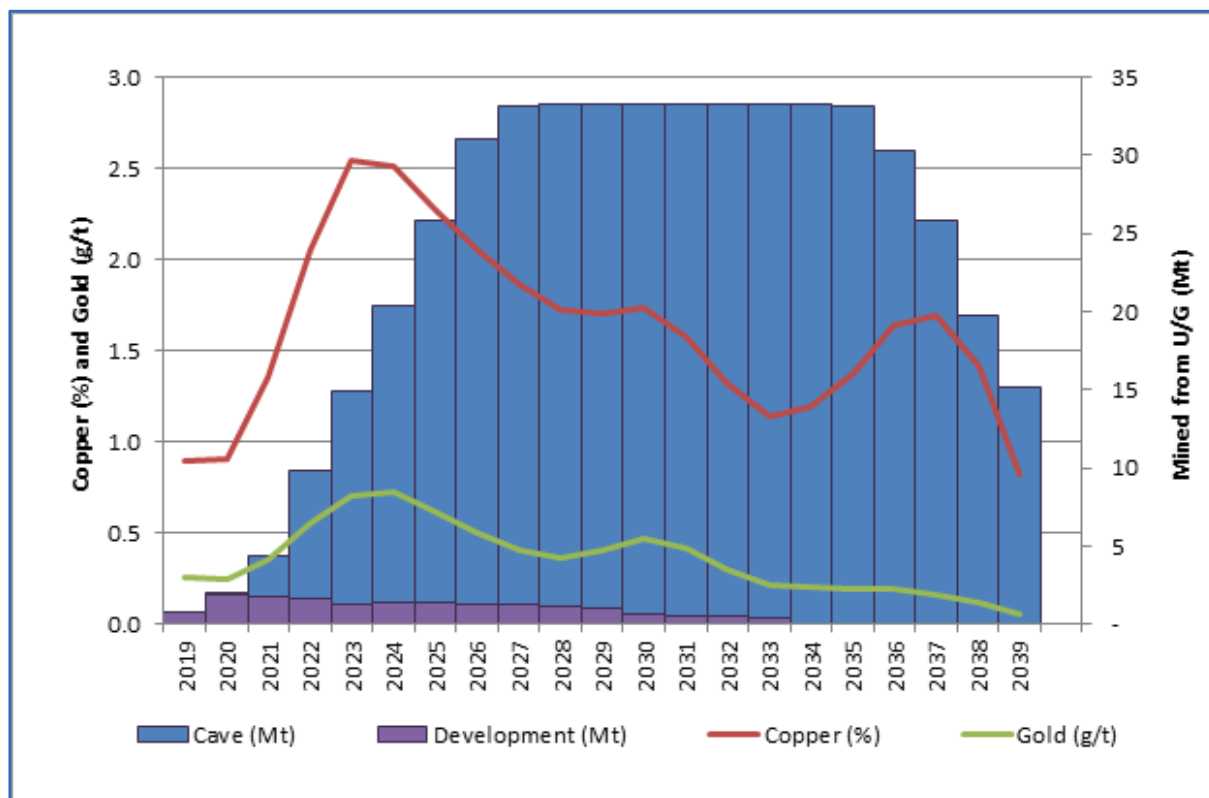


Figure 16.16 Annual Tonnage and Grade Profile for Underground Mine



16.2.1.2 Site Actuals

The first underground development at Oyu Tolgoi started with sinking the bulk sample Camel Well shaft (3.6 m diameter x 74 m deep) in the open pit area between October 2004 and January 2005. Surface works for the first shaft to access the Hugo North deposit, Shaft 1, began in February 2004, and sinking started in February 2005. The 1,300 m level station was reached in October 2007, with the station developed and sinking continuing to a final depth of 1,385 m by January 2008. A temporary bucket loadout arrangement was fitted by March 2008 to support off-shaft lateral development.

Two kilometres of horizontal development from Shaft 1 were completed initially, primarily to provide access to and geotechnical characterization of the orebody and the surrounding conditions to support the Prefeasibility Study. This development was subsequently expanded in parallel with the beginning of the Feasibility Study in 2010, focusing on additional data gathering to support studies as well as advancing pre-production access. In 2012, after the completion of 11 km of horizontal development, the Shaft 1 bucket hoisting arrangement was converted to a skip-and-cage arrangement to support a more intensive development and construction programme. A total of 16 km of lateral development was undertaken from Shaft 1 before the underground project was placed into care and maintenance in August 2013.

Lateral development performance has been very good, with advance rates matching and exceeding feasibility study rates and excavation and ground support being of high quality. Ground conditions and response to mining have been as expected. Negligible water has been intersected.

Surface works for Shaft 2, a 10 m finished diameter shaft, began in July 2006 and was placed on hold in December 2007. Shaft 2 works recommenced in April 2010 and by August 2013, Shaft 2 sinking reached a depth of 1,167 m. Shaft 2 has now established an independent second means of egress and additional ventilation.

A surface-to-underground raisebore was commenced in mid-2010 adjacent to Shaft 1, with a planned 500 m upper leg and 800 m lower leg. Significant challenges were encountered while both piloting and reaming, and the raise excavation was abandoned. The ventilation strategy for the underground project was reviewed and changed, resulting in the ventilation raises being replaced by Shaft 5. Shaft 5 will have a 6.7 m finished diameter and commenced surface works in August 2012 and sinking in April 2013. As at August 2013, sinking had reached a depth of 208 m, progressing on schedule with ground conditions as expected.

The Hugo North Lift 1 underground development and construction formally re-commenced in July 2016.

16.2.2 Geotechnical Conditions and Design

16.2.2.1 Overview

Hugo North is considered to be highly suitable for the caving method of mining. Three caveability assessments were undertaken for OTFS16 using the Laubscher Modified Rock Mass Rating classification system (MRMR), the Mathews extended stability chart, and Flac3D numerical modelling. The risks associated with caveability and propagation are considered to be low. High stress conditions, a highly fractured rock mass, and a large caving footprint are key factors. Surface subsidence analysis does not raise any concern for surface infrastructure in place or planned.

Fragmentation analysis illustrates fine fragmentation for all geotechnical domains. Secondary breaking requirements are not expected to pose a risk to production schedule ramp-up or full production rates.

Predicted abutment stresses are considered to be high, and focus has been placed on optimizing mine design and ground support to manage excavation stability.

16.2.2.2 Subsidence

The predicted fracture limits (determined as the point of having a notable impact on key infrastructure such as hoisting shafts) by the end of mining Hugo North Lift 1 are shown by the red outline in Figure 16.17. A fence will be constructed 100 m outside this red outline to restrict access. The subsidence angles are predicted to be nearly vertical at the northern and southern limits of the cave, where confinement is highest, and are approximately 55° in the east and west, where confinement is lowest. All shafts and permanent infrastructure are planned to be situated outside the predicted fracture limits.

Figure 16.17 Subsidence Predictions from Modelling



Note: fracture limits = thick red line.

Shaft 1 is closest to the fracture limits. Shaft 1 will be used as a hoisting shaft until the Shaft 2 loadout and primary crusher are commissioned. Thereafter the primary function of Shaft 1 is for intake ventilation although modelling predicts that the shaft could be used as a hoisting shaft up to 2035. This provides additional contingency against an unexpectedly larger cave subsidence damage area, as a bald concrete lined shaft can withstand higher ground movement than a shaft reliant on the close tolerances of operating hoisting infrastructure.

16.2.2.3 Rock Mechanics

The drilling programme prior to underground suspension, which concentrated on the cave initiation area and the first four years of production ramp-up, provided critical characterization data and confirmed that the Hugo North orebody is highly faulted and sheared. In situ stress (Sigma (σ)) measurements estimated at the extraction horizon are high: $\sigma^1 = 58$ MPa (sub-horizontal with a dip direction of 055°), $\sigma^2 = 33$ MPa (sub-horizontal with a dip direction of

145°), and $\sigma^3 = 27$ MPa (sub-vertical). An analysis of geotechnical domain data confirmed that a lithology basis for domain assignment remains valid. MRMR values for the different lithologies vary between 43–53.

Stress orientations have been calculated, with the sub-horizontal major principal stress bearing towards 055°. The in situ stress regime, summarized in Table 16.12, represents the latest version from the geotechnical assessment in the study.

The rock mass strengths of the orebody units were divided by a range of mining stress levels as predicted from the cave-scale modelling: isolated drifts under in situ stresses (60 MPa), average abutment stresses (80 MPa), and high abutment stresses (100 MPa). Results indicated that closure strains up to 5% were possible from high abutment stress loading on the extraction and undercut levels.

Table 16.12 In Situ Stress Regime

	Depth Range (m)	σ_1 (MPa)	σ_2 (MPa)	σ_3 (MPa)
Linear	0–1,330	0.049 z	0.028 z	0.022 z
Domain 1	0–600	0.047 z	0.031 z	0.024 z
Domain 2	600–800	0.071 z – 13.95	0.012 z + 11.08	0.027 z – 1.59
Domain 3	>800	0.031 z + 17.50	0.026 z – 0.33	0.015 z + 7.66

Note: 'z' = depth below surface.

16.2.2.4 Caveability and Fragmentation

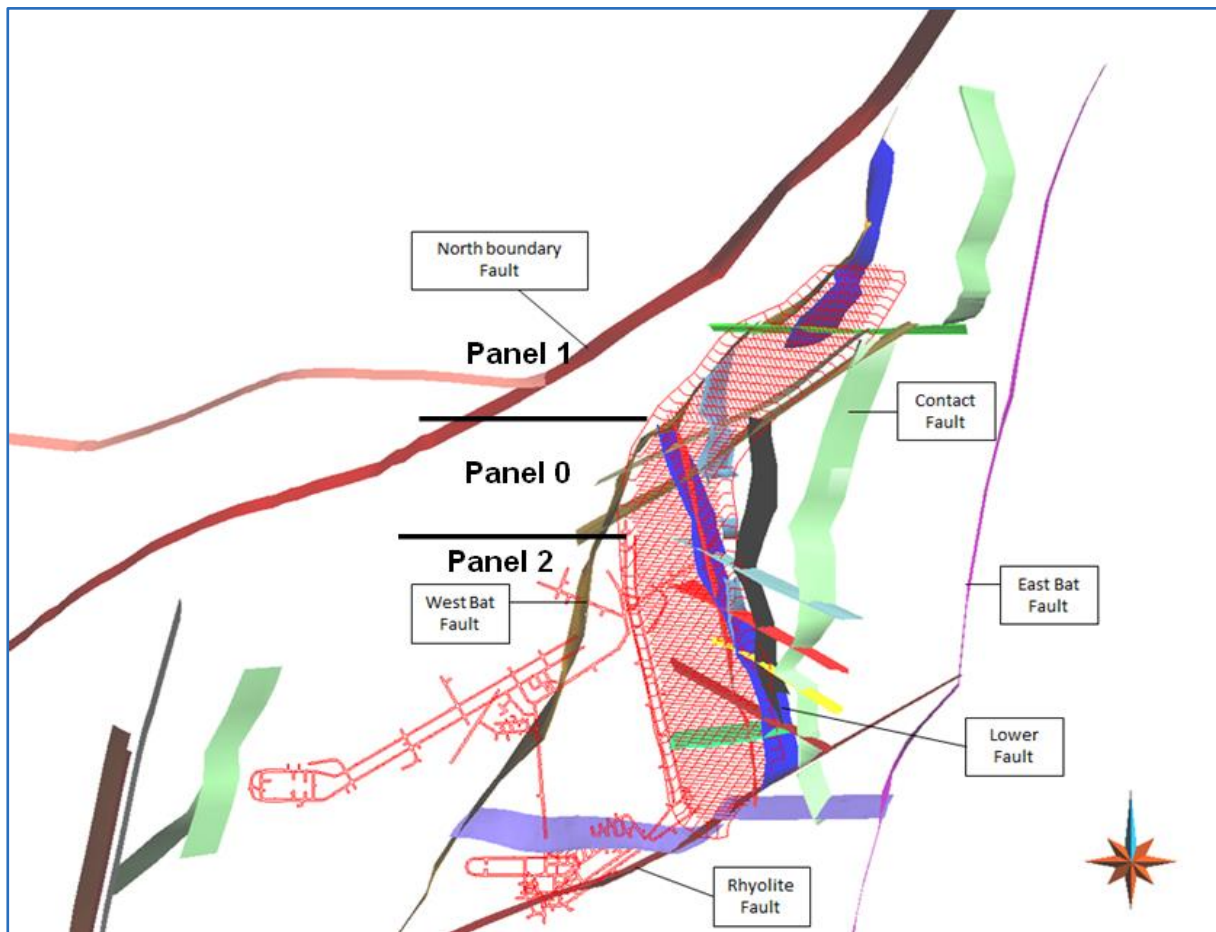
The Hugo North orebody is a highly jointed rock mass classed as fair to poor. Based on current data, the critical hydraulic radius (HR) to initiate and sustain caving of the rock mass is approximately 20–23 m. The MRMR of 40–45 is highly caveable at a critical hydraulic radius >20–23 m, for the median values for Qmd, Va, and Ign units. Values for the Qmd and Va represent most of the orebody. This HR range represents approximate dimensions of 80 m x 80 m to 100 m x 100 m. Other key points from the analysis include:

- Major faulting will significantly influence caving and should promote cave propagation.
- Stress caving is likely to dominate the cave propagation.
- 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 EJV part of the orebody, is sufficient to initiate and sustain caving.

16.2.2.5 Mining Layout

To best manage the risk of drive and pillar damage from high abutment stresses and the highly fractured orebody, the footprint design has been divided into three production panels. The footprint layout is shown in Figure 16.18.

Figure 16.18 Footprint Layout and Current Development in Relation to Major Faults



The design was influenced by the following key geotechnical criteria:

Principal Stress:

- Extraction drives parallel or sub-parallel ($\pm 20^\circ$) to principal stress direction of 055° strike: This provides best use of the clamping forces from the high horizontal stress on the major apex to improve pillar stability. It also improves the development quality and advance rate of extraction panel drives, undercut drill drives, and apex inspection drives.
- Undercut face perpendicular or sub-perpendicular to principal stress direction of 055° strike: This provides lower stress conditions immediately in front of the undercut face, and, coupled with drive pillars parallel to principal stress as mentioned above, provides highest stability.

Structure:

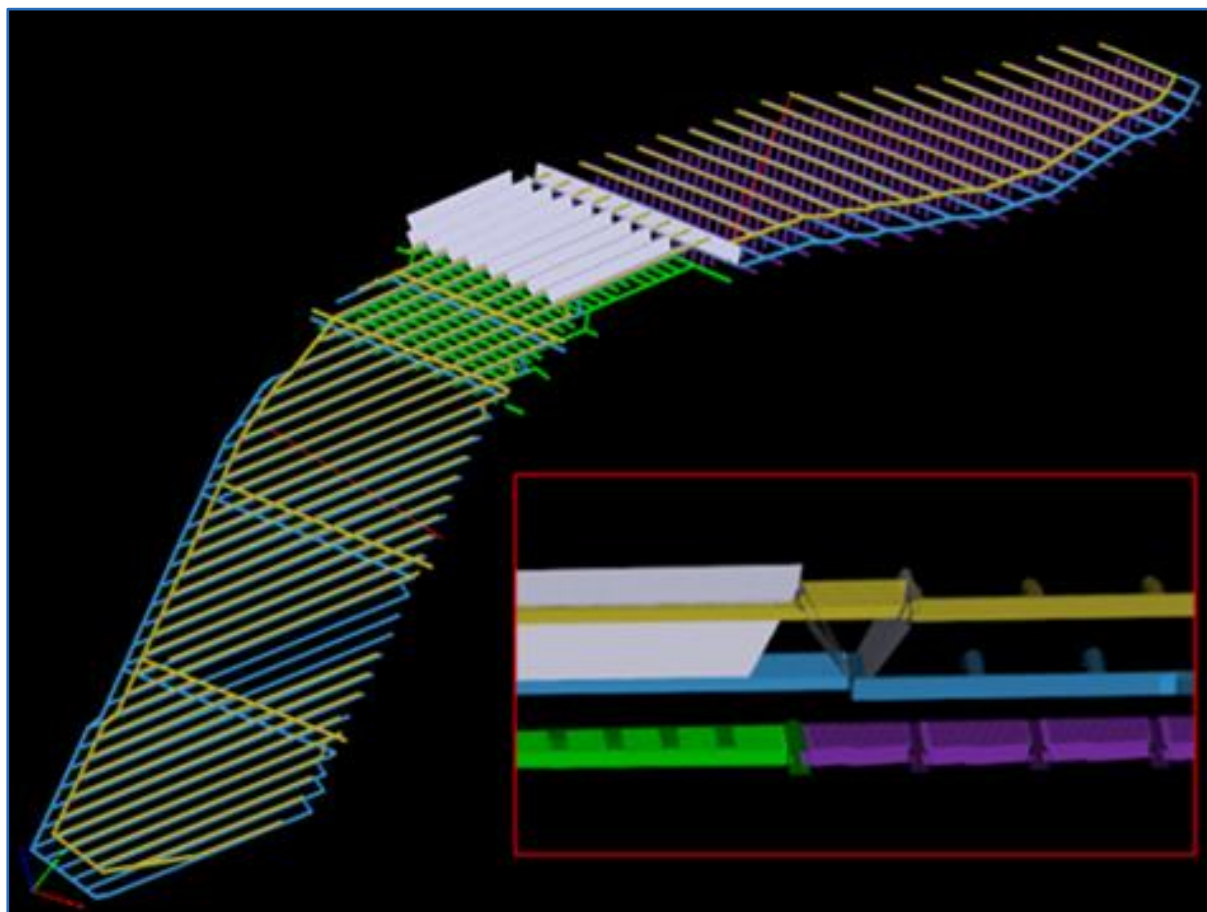
- Undercut face intersecting major structure at an angle, not parallel, to structure: A key structure in the central and southern areas of the footprint is the north-south trending Lower Fault. Key structures in the northern area of the footprint are the H Fault, trending north-east / south-west, and other similar faults parallel to the West Bat and Contact faults, which are major orebody bounding faults.
- Extraction drives intersecting major structure at an angle, not parallel, to major structure. This provides best conditions for development quality and advance rate, along with long-term stability of key development openings.

Undercut Stability:

- Implementation of the 'Wide W' undercut design with apex level: the 'Wide W' design increases the volume of the major apex pillar and doubles the pillar width between undercut drives, significantly improving pillar and drive stability both for the undercut and for the extraction level. The apex level, elevated above the undercut level with drives located along the peak of the major apex pillar, provides several benefits: improved ability to inspect undercut blasts and confirm breakage particularly, at the peak of the major apex pillar; ability to confirm swell void for the toe area of undercut blast rings and monitor the amount of undercut swell mucking; and provides an additional platform for remedial drilling of un-blasted stubs. Ventilation management is simplified as the number of active drill and blast headings on the undercut horizon is reduced. These operational benefits decrease the risk of undercut delay causing stress loading and potential damage and collapse.
- Undercut length under 350 m: Longer undercut faces increase the number of active undercut drives that are required to operate in series, increasing the risk of one undercut drive delaying the retreat of the undercut face. Additional active undercut drives add complexity and congestion. Long undercut faces are considered to aid in concentration of undercut stresses near the centre of the undercut face.
- Undercut rate of retreat greater than 80 m per annum, or 7 m per month, as measured along the undercut drill drive: Slower undercut retreat rates increase the time dependent stress deterioration along the zone in front of the undercut face, in particular the half blasted pillar area that establishes the undercut lead-lag.
- Undercut lead-lag of 10 m (± 1 m) undercut blast (2.5 m): Larger lead-lags increase the half-blasted pillar area in front of the undercut face, increasing the risk of ground damage and collapse.

To implement the design criteria over the Hugo North Lift 1 footprint, a three-panel approach was adopted. This approach includes two panel boundaries. The northern panel boundary between Panel 0 and Panel 1 changes in undercut orientation and development drive orientation, as illustrated in Figure 16.19. The panel interfaces are considered manageable and have been used at several operations in the past. These boundaries are in isolated areas and, with increased ground support and detailed drill and blast management, present lower risk than ongoing issues with long undercut faces and slow-moving caves.

Figure 16.19 Illustration of Panel 0 – Panel 1 Boundary



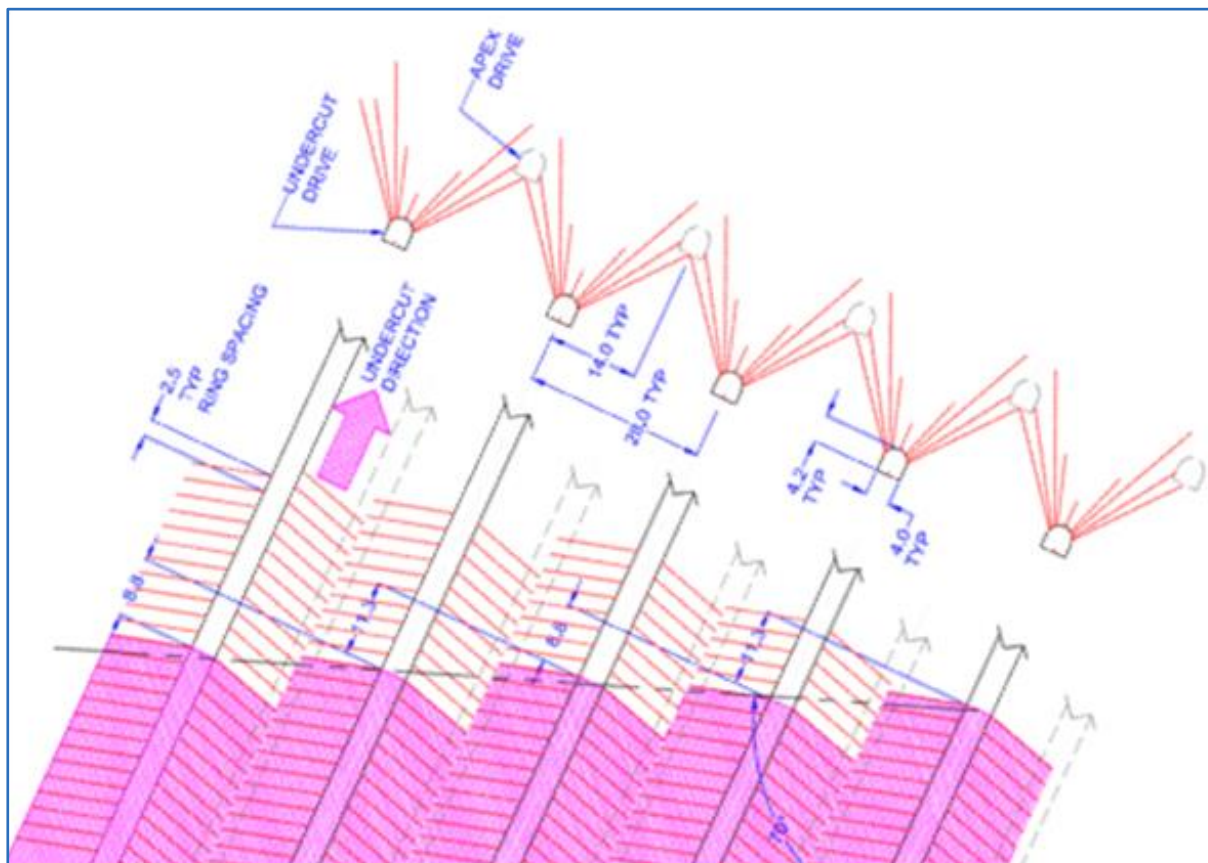
16.2.2.6 Undercut Level

Undercut pillars next to the advancing cave front are likely to experience vertical stresses on the order of 45 MPa. Sigma 1 (σ_1) comes into the undercut face in a horizontal direction from a bearing of 055° (normal to the cave front) and angles down through the undercut pillars at up to 110 MPa, then reduced to in situ levels two or three pillars (30–45 m) behind the undercut front. These stresses result in high modelled closure strains (up to 5%) immediately at the undercut front and lower strains (2%–5%) behind the undercut face.

Figure 16.20 illustrates the undercut blasting area and cave front. The 10 m lead-lag shown between adjacent undercut drives to manage stress build-up near the undercut face results in an undercut face orientated at 70° to the undercut and extraction drive. The minimum undercut retreat rate (along an undercut drive) will be 7 m per month to prevent stress build-up and management of time-dependent ground deterioration.

The 4.0 m wide x 4.2 m high undercut drives spaced every 28 m are considered to be supportable and adequate. Apex inspection drives, located 17 m above the undercut level and spaced every 28 m, are situated along the peak of the major apex pillars and are considered important in managing successful undercutting activities.

Figure 16.20 Undercut and Cave Front



16.2.2.7 Extraction Level

Based on geotechnical modelling and cave flow models, 28 m x 15 m drawbell spacing with an El Teniente layout was selected. Pillar stability and recovery were major factors in selecting the drawpoint spacing. To promote interactive draw, drawcone centroids within a drawbell are spaced 10 m apart. Layout parameters are illustrated in Figure 16.21.

The advanced undercut sequence allows the extraction level panel drives to be mined ahead of the undercut face. A safety zone running the length of the undercut face will be established on the extraction level underneath the advancing undercut face. This zone will be 34 m wide, starting 17 m, or 45°, in front of the undercut, and ending 17 m, or 45°, behind the undercut face. The development of the drawpoint drives will begin 17 m behind the undercut face, and full drawbell excavation will begin at least 60° behind the undercut face. This is shown in Figure 16.22.

The 4.5 m wide x 4.5 m high extraction panel drives spaced every 28 m, and the 4.5 m wide x 4.2 m high drawpoint drives, are considered to be supportable and adequate. The potential of using smaller extraction panel and drawpoint drives has been identified as a further risk mitigation measure for drive and pillar instability.

Figure 16.21 Extraction Level Layout Parameters

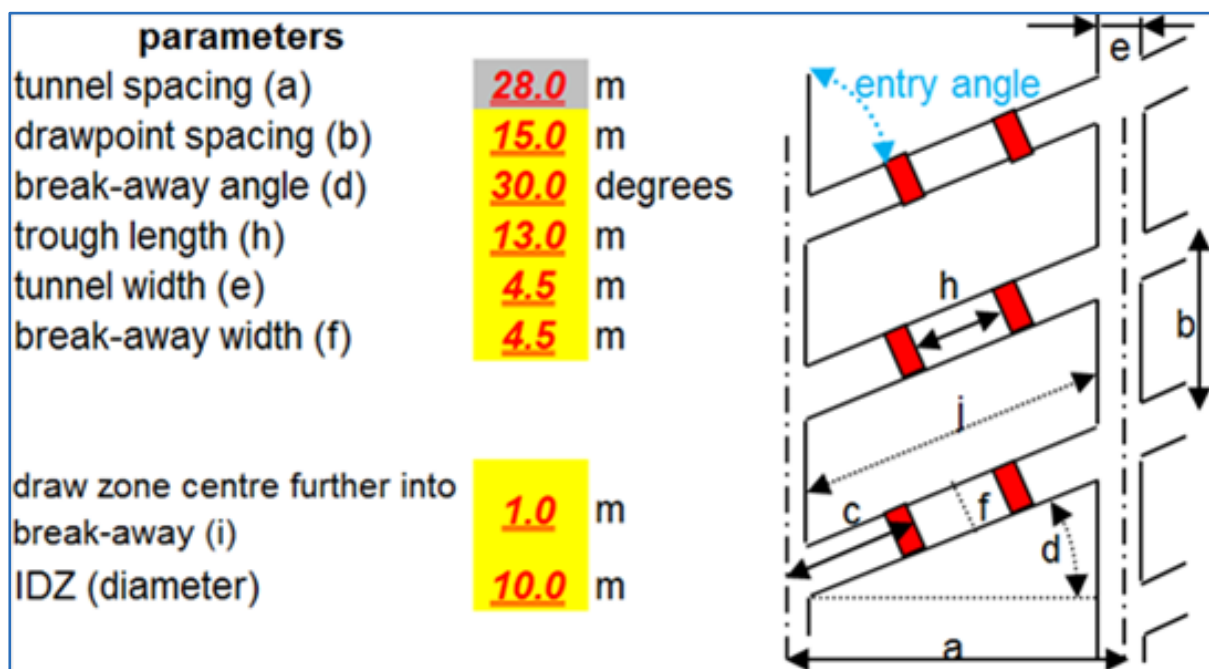
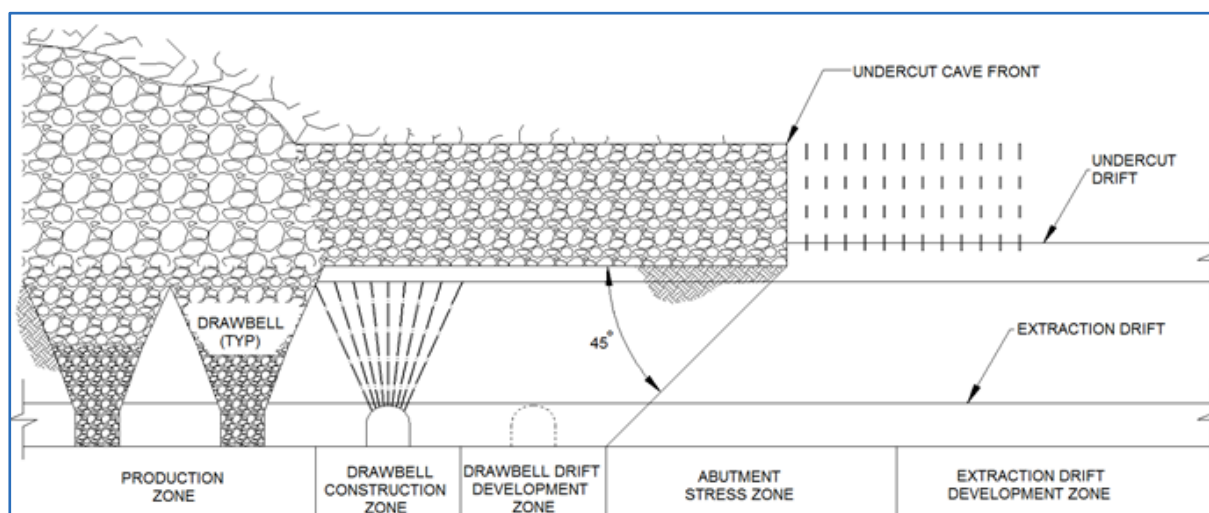


Figure 16.22 Cave Section along Extraction Drift



16.2.2.8 Haulage Level

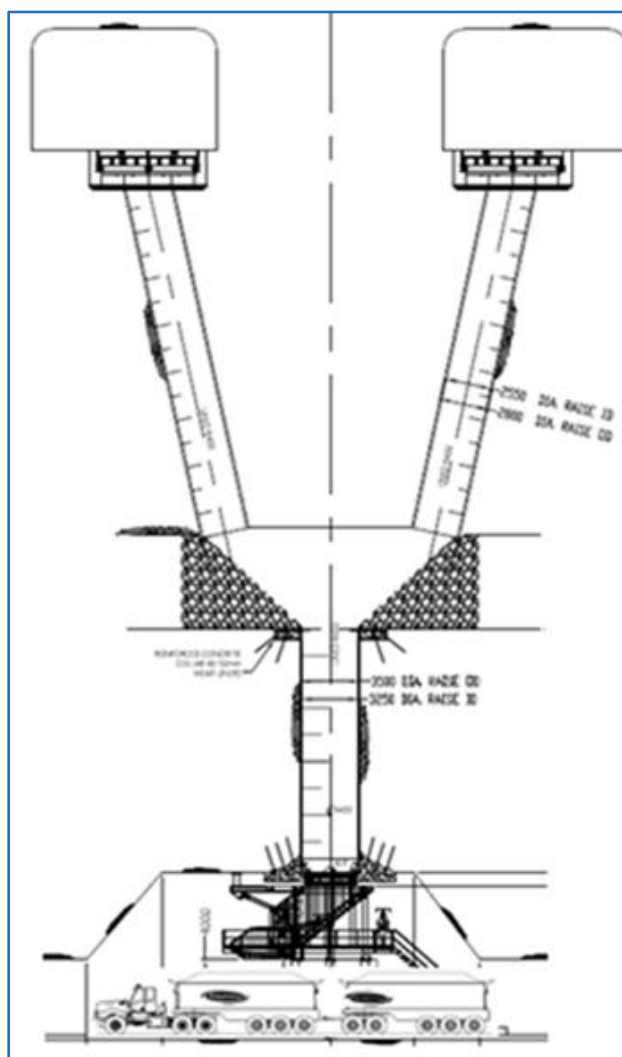
Modelling suggests haulage level excavations could be subject to short-term abutment stresses of up to 100 MPa associated with the passing of the leading edge of the cave. Stress levels along the haulage drives are reduced to 60–70 MPa underneath the active cave, then increase to 80–90 MPa once draw ceases. Because of its orientation relative to the major

stresses at the north limb haulage level are approximately 10 MPa lower than at the south limb for all stages.

Over time, immediately below the haulage level, stresses are modelled to increase to 100 MPa or higher and remain at elevated levels. This suggests that the haulage drives are near the bottom of the stress shadow beneath the cave.

The main production orepasses are located along the central axis of the footprint strike. As illustrated in Figure 16.23, each orepass system is configured in a 'Y' shape where two 2.8 m diameter x 15 m long upper orepass legs (each supporting tonnage from one extraction drive) feed ore through a small storage bin into one 3.5 m diameter x 13 m long lower orepass leg. A truck chute is used to load material from the base of the 3.5 m diameter raise into haulage trucks.

Figure 16.23 Illustration of 'Y' Orepass Arrangement



Geotechnical modelling of this arrangement, including exposure to elevated passing abutment stresses, illustrated highest stresses building in the area where the two upper orepass legs converge into the storage bin 'Y' junction. To manage this, the distance (or the pillar) between the converging upper orepass legs has been designed at a minimum of 5 m and will be cable-bolted. The orepass design is considered supportable and stable. From a geotechnical perspective, the short length of orepass legs minimizes the time duration between excavating and installing steel lining, which aids in managing the risk of time dependent orepass failure. The mid-level access also provides an extra working platform if pre or post-support is required. To minimize stress exposure and fluctuation the upper orepass legs are scheduled for construction after the cave stress shadow has passed.

16.2.2.9 Ground Control Methods/Support Regimes

Different support regimes are proposed as a function of the anticipated ground conditions and induced stress regimes that may be encountered during the development and operation of the Lift 1 caves, and the life of the excavation.

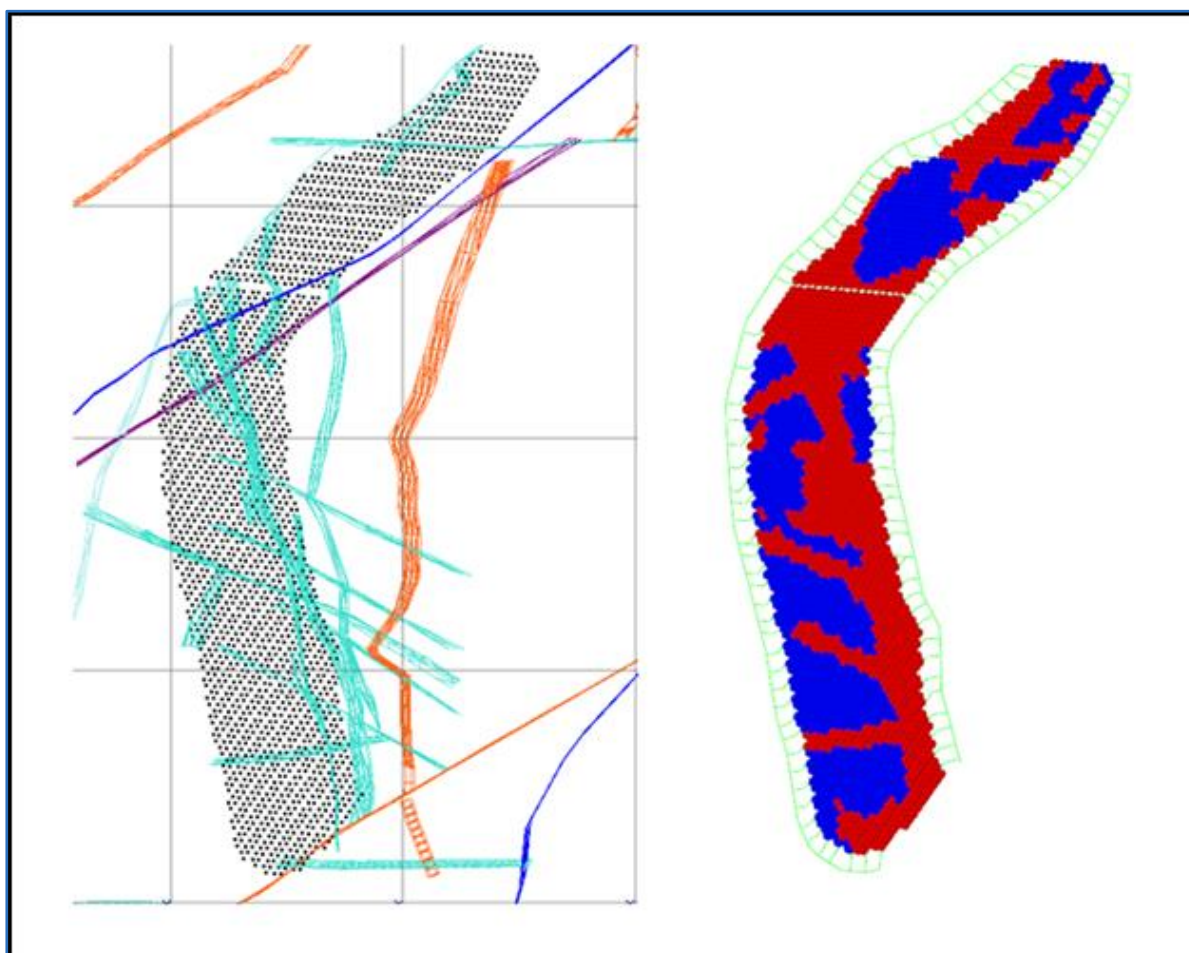
To manage variation in cave stresses, support categories are divided into 'on-footprint' and 'off-footprint'. To manage variation in ground conditions, support categories are further divided into 'good' ground ($MRMR > 30$) and 'poor' ground ($MRMR < 30$).

For on-footprint development, the OTFS16 basis averages 55% of the ground as good and 45% as poor. For off-footprint development, 90% of the ground is classified as good and 10% as poor (Figure 16.24). Geotechnical recommendations from Hazmap modelling have been used for the basis of classification.

The following ground support elements are included in all ground support designs:

- Fibre-reinforced shotcrete between 50–100 mm thickness, having a minimum UCS of 20 MPa in 72 hours; 30 MPa in seven days; and 40 MPa in 28 days.
- Rock bolts 25 mm diameter, threaded, fully-encapsulated resin-grouted thread bars with a minimum yield strength of 200 kN. The minimum bolt length is 2.4 m.
- Cable bolts installed at all intersections and any drive profile greater than 6 m wide will be 18 mm and 22 mm single-strand cables with a minimum yield strength of 331 kN and 510 kN, respectively, and a minimum length of 6 m. For zones of high deformations, such as strainburst-prone rock masses, cables are to be installed with a 2 m debonded section at the collar and pre-tensioned to 10 tonnes.

Figure 16.24 Major Faults (left) and Good (blue) and Poor (red) Ground Distribution (right) on the Extraction Horizon



The ground support recommendations proposed are based on the anticipated average ground conditions and stress regime; hence, these are minimum support requirements and additional ground support may be required where the conditions demand. The base Oyu Tolgoi cave support design uses the same methodology as implemented at benchmark mines to successfully recover collapsing drives. Additional ground support elements include:

- 10 mm diameter heavy-gauge woven steel mesh installed floor to floor (over previously applied first pass shotcrete and rockbolting).
- Cable strapping applied longitudinally along drives, on walls, and on backs. On the extraction level, cable strapping wraps around bullnoses and camel-backs into the drawpoints.
- An additional 200 mm layer of shotcrete from shoulder to floor, applied to each wall over the mesh.
- Spacing of cable bolting in ground classified as Poor ($MRMR < 30$) reduced from 2 m x 2 m to 1 m x 1 m pattern, with the cable extended to 8 m in length.

- Debonded cable bolts to be installed in high-stress areas such as extraction drives and haulage drives. A yielding rock bolt to be used for the strain-prone zones.

Steel sets will be installed on the extraction level in each drawpoint to support the drawpoint brow area. To support cave initiation and early ramp-up, all drawpoints in Panel 0 will be fitted with heavy-gauge 4-arch, 9-tonne steel sets. For the remaining drawpoints in Panel 1 and Panel 2, those classified in Poor ground will be fitted with the heavy-gauge 4 arch sets, and those classified in Good ground will be fitted with heavy gauge 2 arch sets.

The proposed cave monitoring system includes a micro-seismic system, Time Domain Reflectometers (TDR), extensometers, and open drillholes. Cave flow monitoring systems comprise Smart Markers, Network Markers, and Cave trackers (Elexon Pty Ltd) installed primarily down surface drillholes. These systems are safeguards against potential hazards and increase the understanding of cave flow for adjusting draw strategy to optimize recovery.

16.2.3 Mine Design

16.2.3.1 Mining Method

The focus of the Feasibility Study mine design is to demonstrate the technical and financial feasibility of Hugo North Lift 1, to optimize ore production potential, and to manage capital cost.

Panel caving has been the basis for underground mine planning at Hugo North since order-of-magnitude studies were carried out in 2005. The weak, massive nature of the orebody and its location between 700 m and 1,400 m below surface make it well suited both geotechnically and economically to the large-scale caving method of underground mining. Caving requires a large early capital investment but is highly productive and has low operating costs. The long operating life of the mine supports this initial capital investment.

16.2.3.2 Design Summary

The Lift 1 mining levels are approximately 1,300 m below surface. The orebody has average dimensions of 2,000 m long x 280 m wide. A total of 2,231 drawpoints are planned to be developed within the mining footprint, accessed from 52 extraction drives. The mine design consists of 203 km of lateral development, five shafts, and two decline tunnels from surface. Five shafts are required to provide access for mining personnel and equipment, for production, and for intake and exhaust ventilation-ways. The primary ore handling system will transport ore to surface by a series of conveyors. An overview of Lift 1 is shown in Figure 16.25. Table 16.13 outlines the total development quantities for the Hugo North Lift 1 mine design.

Figure 16.25 Lift 1 Mine Design

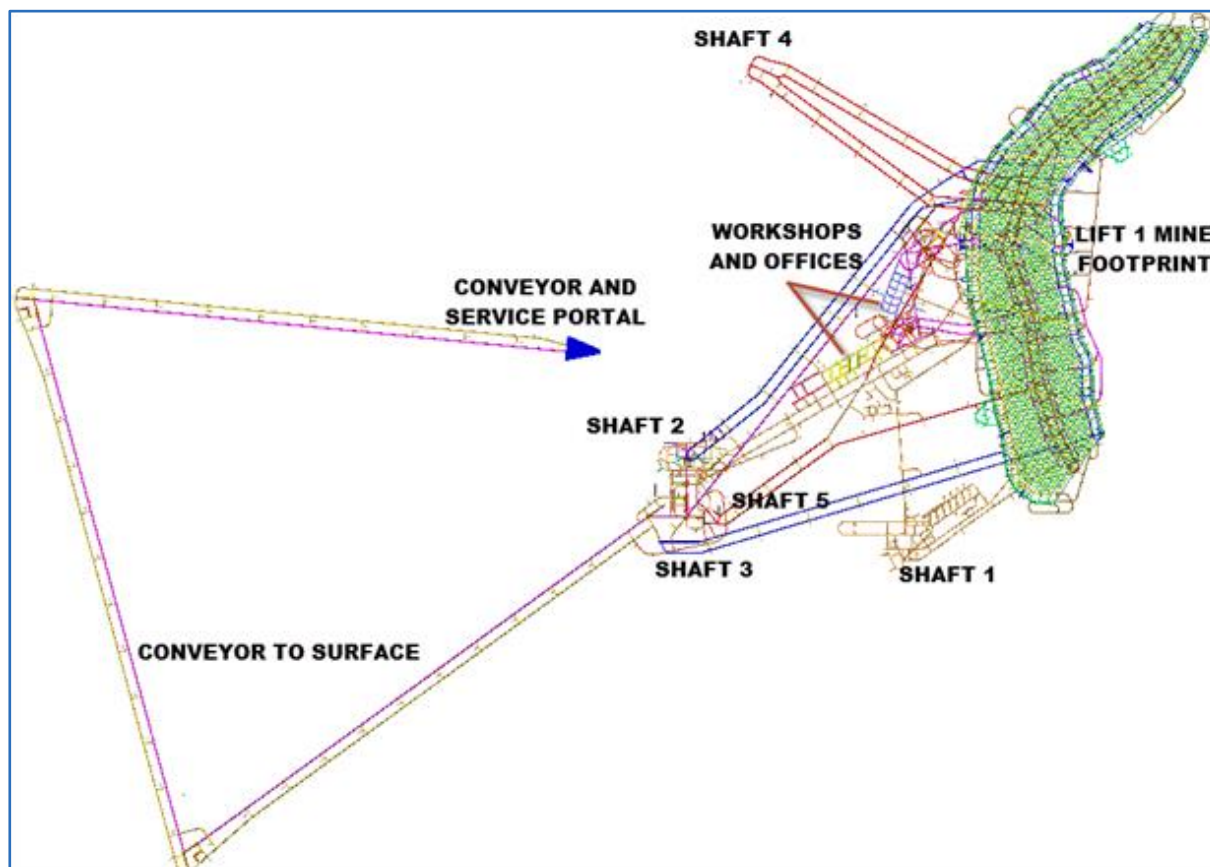


Table 16.13 Development Quantities

Lateral Development		Vertical Development and Mass Excavation		
Area	Metres	Area	Unit	Value
Development to July 2016	15,747	Shaft Development	m	6,159
General Access & Facilities	9,218			
Apex and Undercut Level	51,931	Raises 2–6 m diameter	m	6,826
Extraction Level	59,542	Ore Bins 10.8 m diameter	m	51
Haulage Level	14,656	Mass Excavation for Facilities	m ³	60,077
Intake Drives	17,018	Ore Handling Excavations	m ³	55,764
Exhaust Drives	16,168			
Conveyor	18,863			
Total Lateral Development	203,143			

16.2.3.3 Mine Access

Shaft 1 is the existing pre-production access and services shaft. It is 6.7 m in diameter, concrete-lined, equipped with fixed-guides, and, sunk to a depth of 1,385 m. The steel headframe supports two winders. One winder operates a double-deck, 6 t capacity cage (1.5 m x 3 m) with a personnel capacity of 32 people per deck. The other winder operates two 9.5 t skips with 3.5 kt/d muck hoisting capacity. Mine air heaters connected to a sub-collar plenum provide heated intake air. Underground fans will be connected to ducts in the shaft to provide exhaust ventilation until the commissioning of Shaft 5.

Shaft 2 will be a dual-purpose service and production shaft and a primary intake ventilation shaft. It is 10 m diameter, concrete-lined, equipped with fixed guides, and sunk to a depth of 1,284 m. The shaft will be equipped with a service cage and have a capacity of 39 t and be able to accommodate a peak of 150 persons on a single deck.

Mine personnel and material access will commence through Shaft 1. Personnel utilize the Shaft 1 1300-Level station for mine access with materials delivered to the 1344-Level station until commissioning of the Shaft 2 service cage. After this Shaft 2 will be the primary access for personnel, equipment, and materials.

Primary access from Shaft 2 is along two access drives that connect the Shaft 2 Service Level station on the 1,256 mRL to the main workshops, offices, and extraction level, and by ramps to other mining levels. A one-way traffic loop arrangement will minimize traffic interaction and congestion.

16.2.3.4 Lateral Development

Six distinct levels will be developed to support Lift 1 of the Hugo North mine. They are apex and undercut, extraction, haulage, intake and exhaust ventilation, and crushing and conveying levels. These are shown in Figure 16.26, followed by a cross-section through the production footprint in Figure 16.27. The mine footprint is divided into three mining panels: Panel 0, Panel 1, and Panel 2. The panel division allows the apex, undercut and extraction drives to be oriented to optimize the undercut face length. It also allows alignment to major fault structures and principal stress as the orebody dimension changes along strike.

Figure 16.26 Summary of Feasibility Study Mine Design

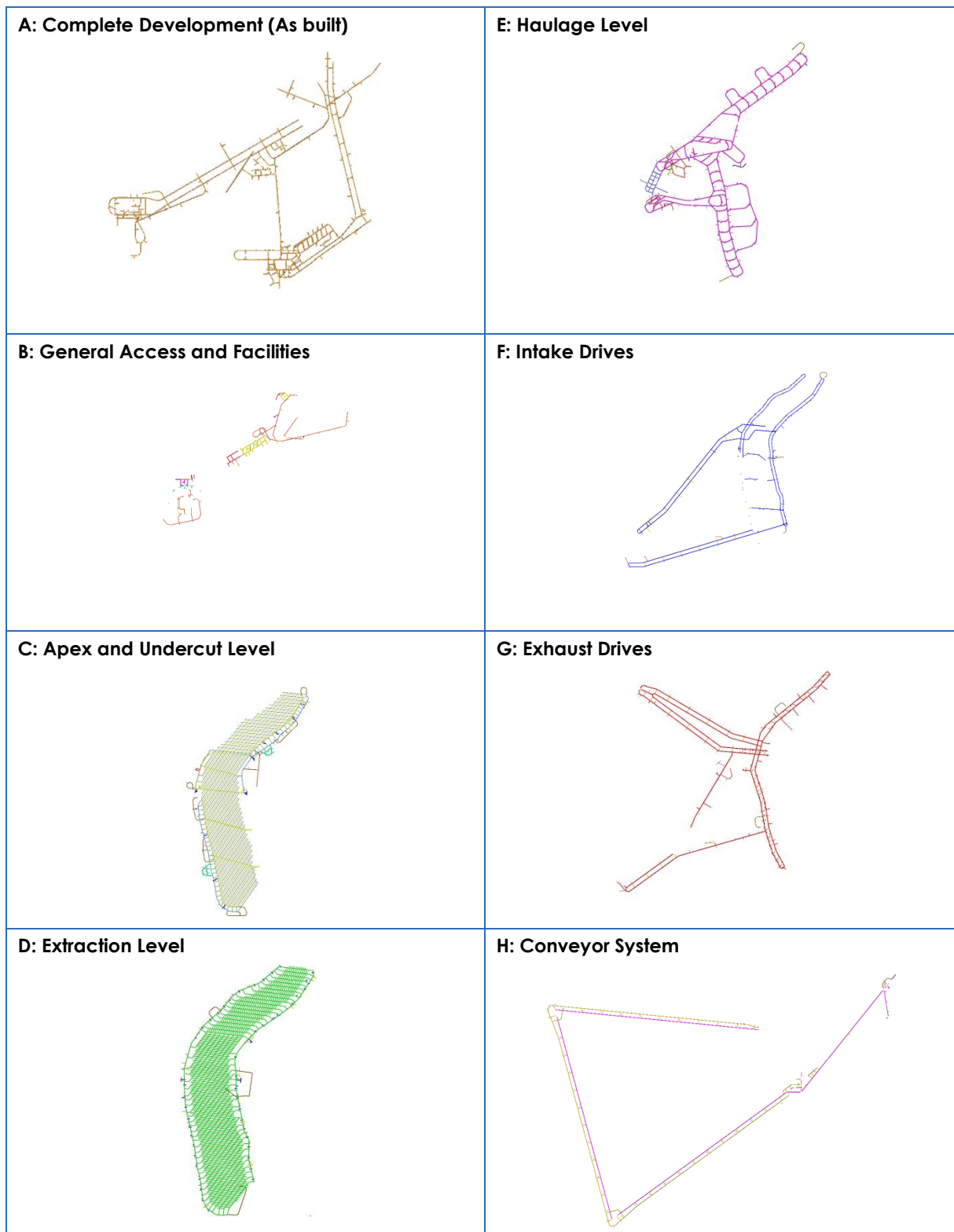
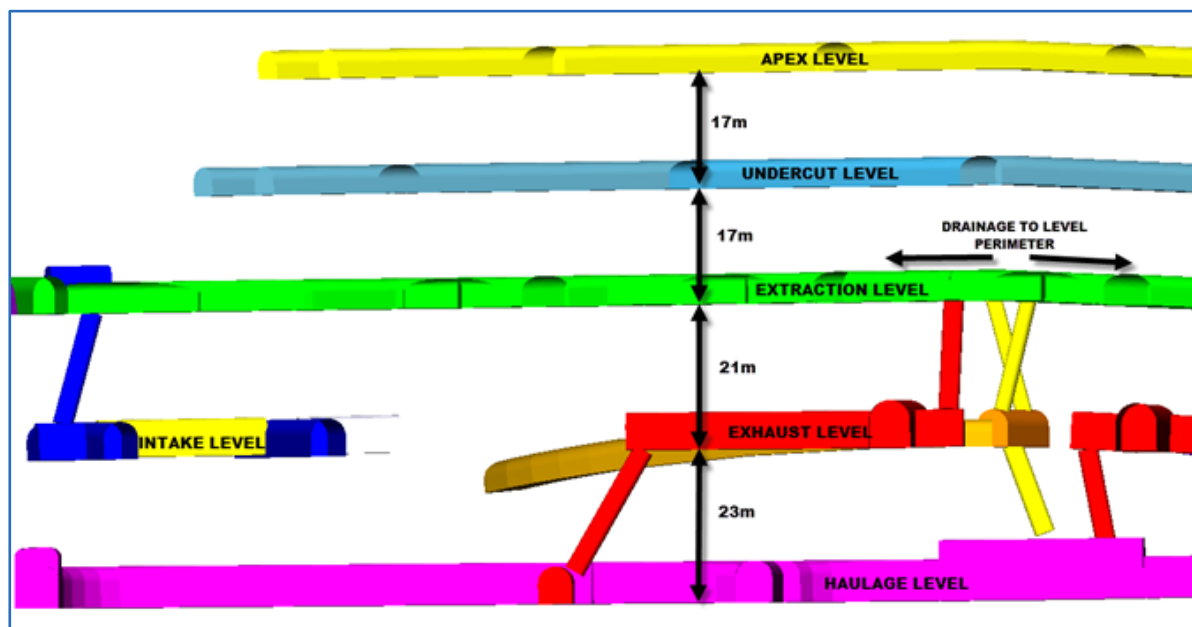


Figure 16.27 Cross-Section through Production Levels



Apex and Undercut Level

The apex and undercut levels provide access tunnels for production drilling and blasting with the purpose of undercutting the orebody. Production holes are drilled from undercut level tunnels up and into parallel tunnels on the apex level. Development of the apex level allows inspection for undercut drillhole deviation prior to each blast.

Extraction Level

The extraction level is designed for the efficient development of drawbells and load-haul-dump (LHD) operation to extract ore from drawpoints. The undercut is situated 17 m above the extraction level to provide an adequate pillar between the levels. The undercut and apex drives are parallel to the extraction level production drives.

The extraction level drives will be spaced 28 m apart, using an El Teniente-style drawbell layout with 15 m spacing. The drawpoints are oriented at a 60° angle to the extraction drives to optimize the pillar size between drawbells and accommodate loader access. The extraction drives drain from the centre to the perimeter drives to stop water from flowing into the exhaust raises and orepasses.

Haulage Level

The purpose of the haulage level is to collect material from the extraction level and undercut level and transport it to crushers for size reduction. The haulage level will be 44 m below the extraction level. It is designed to support one-way traffic from the crusher to the truck loading chutes and returning to the crusher. In general, it is located under the centre of the footprint,

serving orepasses from each extraction drift. Haulage drives will be driven 5.4 m wide x 6.1 m high with fully arched back profile.

Intake Ventilation Level

The intake ventilation system is designed to provide fresh air to the mining footprint level, main travel ways, working areas within the mine, and the fixed facilities.

Fresh air for the mining footprint level will be supplied through two sets of two intake tunnels parallel to the extraction perimeter drives, 5.0 m wide x 5.5 m high, running the length of the footprint. A series of 3 m diameter raises will connect the intake drives to the perimeter drives on the extraction level.

Exhaust Ventilation Level

The exhaust ventilation level allows passage of used air out of the mine. Fresh air enters the east and west side of the mining footprint, removing dust from Load Haul Dump (LHD) mucking, and exhausts down a two metre diameter central ventilation raise adjacent to the orepasses. Two parallel drives on the exhaust level, 5.8 m wide x 6.5 m high, will run the length of the orebody along the centre of orebody axis.

Exhaust drives also connect to exhaust raises from the haulage level in each truck loading chamber to collect dust from the truck-loading chutes. Return air will exit the mine through 6 m high x 7 m wide return air drives.

The conveyor to surface and parallel service drive are pressurized with fresh air from Shaft 3, allowing dust generated from the conveyors to be exhausted to surface.

Crusher and Conveying Levels

Trucks haul ore from chutes to crushers on the west side of the mining footprint. Ore is crushed and transferred by a series of conveyors directly to surface or to the Shaft 2 hoisting system. Truck shop facilities will be constructed west of Crusher 1 location to provide optimal access and to minimize truck downtime.

The conveyor-to-surface system consists of one 130 m conveyor that transport ores from Transfer 5 to three 2,200 m conveyors up to surface. This will be the primary ore handling route. Shaft 2 will serve as the initial ore handling route to surface until the conveyor-to-surface system is commissioned. At this time the Shaft 2 system will serve as a backup ore handling system.

Vertical Development

Vertical development will include shaft development, orepasses, and ventilation raises. With the exception of the shafts, all vertical development is planned to be done with raisebore and boxhole machines.

Shafts

Five shafts will be required to support Lift 1 of the Hugo North mine. A list of the shafts and their depths as at July 2016 and planned final depths is provided in Table 16.14.

Table 16.14 Shaft Station Elevations and Depths

Shafts	Diameter (m)	Depth as at July 2016 (m)	Planned Final Depth (m)	Function
Shaft 1	6.7	1,385	1,385	Early Development and Intake
Shaft 2	10.0	1,167	1,284	Skipping, Primary Cage Access, Intake
Shaft 3	10.0	–	1,148	Intake
Shaft 4	11.0	–	1,149	Exhaust
Shaft 5	6.7	208	1,178	Exhaust

Orepasses

Two types of orepasses will be constructed to handle the production and development muck from the extraction and undercut levels:

- Central orepasses
- Perimeter orepasses

The ore bin will be raisebored from the exhaust level at a 3.5 m excavated diameter, 14.5 m long with a dip angle of 70° to the haulage truck chute chamber. The orepasses will be raisebored at a 2.8 m excavated diameter, 18 m long with a dip angle of 65° from the exhaust level to the extraction level. After being excavated, the orepasses will be lined with 20–50 mm rolled-steel plate (thickness dependent on throughput) capable of handling rock flow wear up 24 Mt.

Ventilation Raises

Most of the ventilation raises will be 3 m diameter and range from 20–100 m long. An exception is the central exhaust raises, which are relatively short (16 m) and will be excavated at a 2 m diameter. All ventilation raises will be supported with remotely applied fibre shotcrete.

16.2.3.5 Mass Excavation

Several mass excavations will be required to support Lift 1 of the Hugo North mine. Each will have unique support requirements and excavation methodology, depending on ground conditions, geometry, access, and overall functionality. Feasibility level designs and excavation plans have been done for the largest and most complex of these excavations, as listed in Table 16.15.

Table 16.15 Planned Mass Excavations for Hugo North Lift 1

Excavation	Number	Size (W x L x H)	Volume (m ³)
Truck Chute Chamber	36	6.4 m x 30 m x 8.9 m	1,679
Shaft Ore Bins	1	10.8 m (diam. excavated) x 51 m	4,670
Crusher Chamber	2	23 m x 27 m x 45 m	13,000
Tail End Chamber	3	various	4,000
Conveyor Transfer Chamber	6	various	3,000–15,000
Drive Chamber	1	14.2 m x 23.3 m x 12.5 m	2,887
Loadout Feeders	1	–	15,400

Table excludes mass excavation on the conveyor to surface.

16.2.3.6 Surface Facilities

The underground mine will require a number of surface facilities to support the overall Oyu Tolgoi operation.

Shaft 1 Area

Current facilities include offices, dryhouse, warehouse, lamp room, shop, generators, boiler plant, and miscellaneous ancillary facilities. Most of these facilities will stay in service until the completion of the mine construction in 2022.

Production Shaft Farm

The production shaft area will include the collars of Shafts 2, 3, and 5. Also in this area are the 220 kV substation, shaft take-away conveyors, and overland conveyor to the concentrator coarse ore stockpile. The permanent mine office and dryhouse will be located near the collar of Shaft 2.

Shaft 4 Area

The Shaft 4 area will be equipped with the main exhaust fans and an electrical substation.

Conveyor to Surface Portal Area

The underground conveyor to surface system will connect to a surface take-away conveyor and onto the overland conveyor.

16.2.4 Ore Handling Design

LHD muckers will deliver run of mine ore from draw points to the grizzly stations on the extraction level.

Orepasses will connect the grizzly stations to the truck loading stations at the haulage level. Each in line truck loading station will be equipped with a hydraulically operated loading chute, complete with variable throat openings and active lip for total flow control, to load the haul trucks. The truck loading stations will be located at the perimeter and central orepasses to load the 160 t capacity (2 x 80 t trailers) side dump Powertrans road trains. The trucks will deliver ore to two crushers.

The design capacity of each of the crushers is 4 kt/h, which will satisfy the 95 kt/d production target. The crusher sizing has been refined to reflect the latest vendor information for two 1,600 mm x 2,400 mm top service ultra-duty (TSU) gyratory crushers. Crushed ore discharges into a 640 t surge bin. Each crusher station will be equipped with a rock breaker and an overhead bridge crane for service. The station will be operated remotely from a central control facility on the surface.

Primary ore flow will be diverted toward the conveyor to surface system and will feed a short transfer conveyor and then onto the first of a series of three incline conveyors to the surface. The conveyor to surface system will be the primary ore handling system.

Ore diverted towards Shaft 2 via the short horizontal conveyor feeds into a two-way diverter chute installed above a 5 kt ore storage bin. Ore is either fed directly into this bin or diverted onto the conveyor-to-surface system. Ore will be reclaimed from the ore bin onto the skip loadout conveyor via apron feeders and be discharged to skips for hoisting material to surface.

The conveyor to surface incline conveyor system will deliver ore to the new coarse ore stockpile feed conveyor and discharge material onto the stockpile. The stockpile feed conveyor will be parallel to the existing stockpile feed conveyor. The new stockpile feed conveyor will be similar in construction to the existing system for commonality of parts.

The total conveying and hoisting capacity from the underground is planned to be approximately 140 kt/d (50 Mt/a). This conveyor to surface system is planned to move an average daily throughput of 95 kt/d. Simulation work by OT LLC suggested trucking to the conveyor to surface system could have a capacity of 106 kt/d. The Shaft 2 hoisting capacity is designed to be 30 kt/d and Shaft 1 hoisting capacity is designed to be 3.5 kt/d. The ore handling system will include five apron feeders and 16 belt conveyors. A two-way diverter chute will be arranged to optimize operability and maintainability and to reduce geotechnical risk associated with large excavations. The underground conveyor system is shown in Figure 16.28 and Figure 16.29.

Figure 16.28 Underground Conveying System Layout – Conveyor to Surface

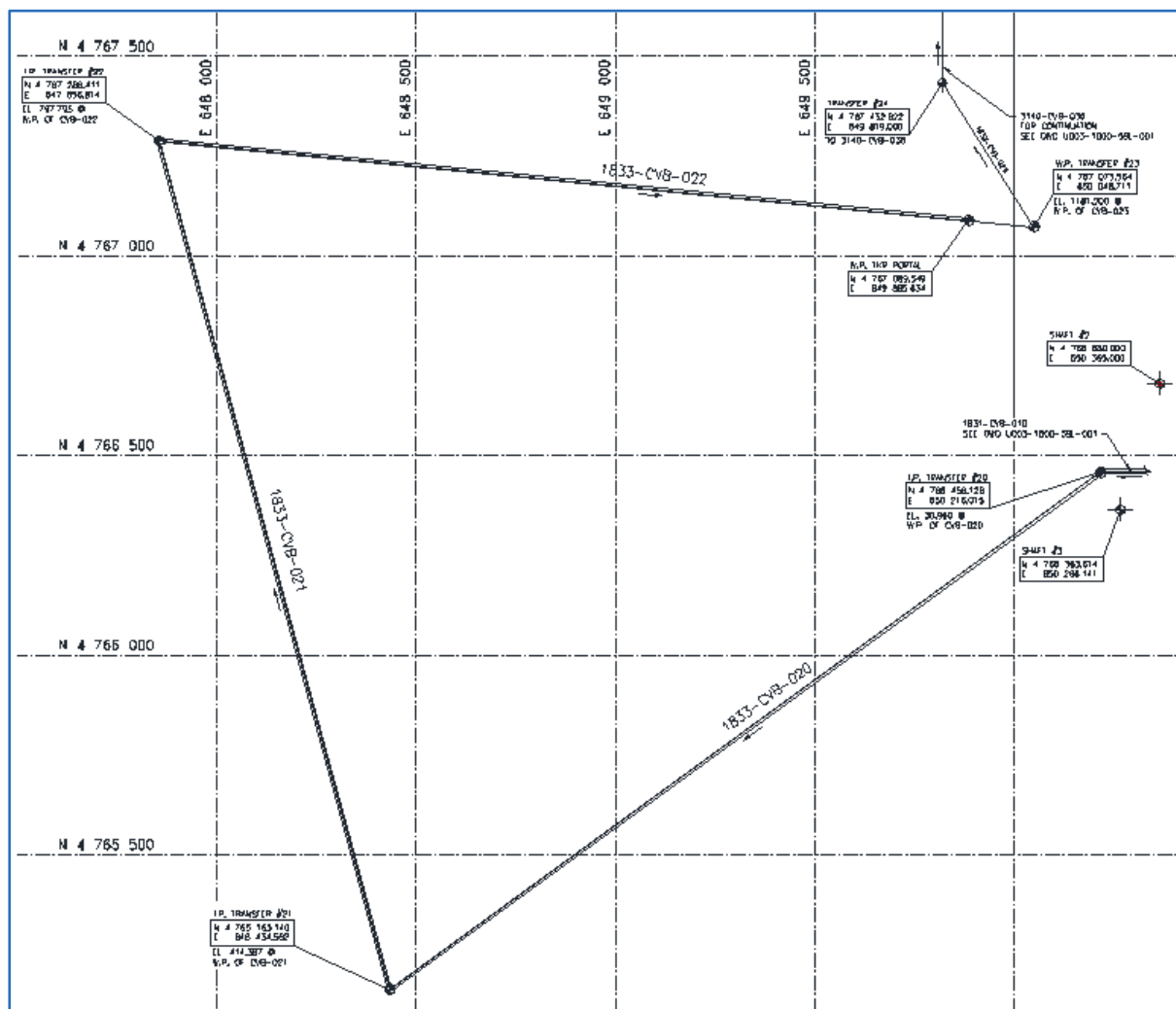
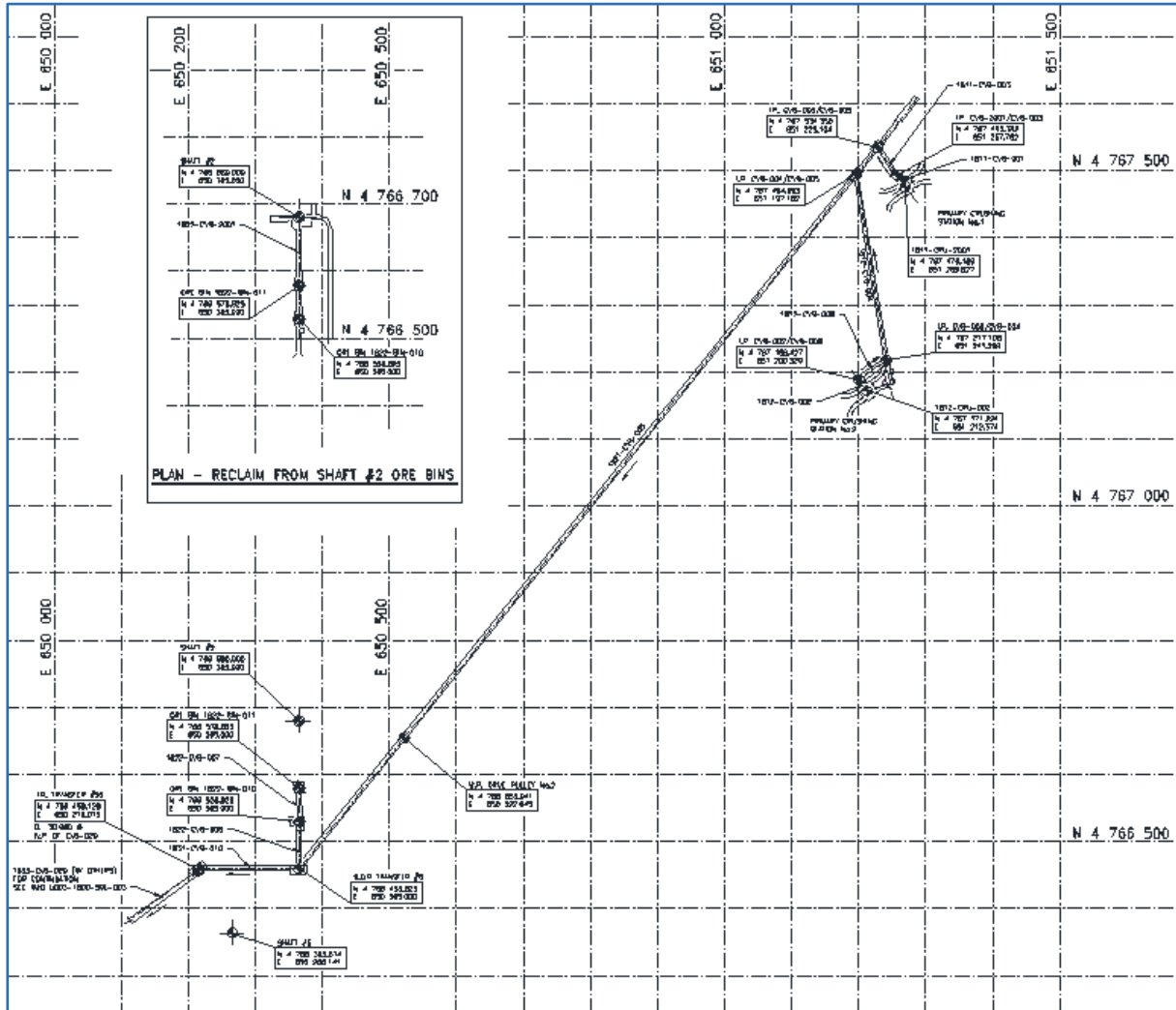


Figure 16.29 Underground Conveying System Layout – Crusher Stations and Shaft 2



Conveying and Hoisting System

Shaft 1 is equipped with skips and a hoist with a capacity of 3.5 kt/d. This is intended to be used for development hoisting only, and not for production hoisting.

Production hoisting will be through Shaft 2. Shaft 2 will utilize two 60 t capacity bottom discharge skips to hoist the ore from underground to the surface. The design capacity of the hoist is 30 kt/d over a planned operating time of 19.2 hours.

When completed, Shaft 2 will be 10 m in diameter and equipped with rigid guides. Skips will be loaded by a conveyor loading arrangement located on the -28.2 mRL, 1,202.2 m below collar. The surface discharge of ore from the Shaft 2 skips will take place below collar level into an ore bin. The skip loading system is designed as an automated and continuous operation for loading the skips without stopping the loadout conveyors.

The Shaft 2 skips will discharge ore into a 200 t capacity surface bin within the headframe of Shaft 2. Ore will be reclaimed from the bin on an apron feeder. Reclaimed ore will be discharged onto the 1,400 mm wide Shaft 2 discharge conveyor running at a belt speed of 2.5 m/s. The discharge conveyor will transfer ore onto the existing open pit overland conveyor and stockpile feed Conveyor 1 to deliver ore to the coarse ore stockpile.

The conveyor to surface incline conveyor will deliver ore to the new coarse ore stockpile feed Conveyor 2, and discharge material onto the stockpile. The stockpile feed Conveyor 2 will be parallel to the existing stockpile feed Conveyor 1. The new stockpile feed conveyor will be similar in construction to the existing system for commonality of parts.

16.2.5 Development Rock Handling

Before the Shaft 2 loadout and skip-hoisting system is commissioned, all rock will be hauled by 50 t trucks to the Shaft 1 hoisting system consisting of a jaw crusher on top of a storage bin, with rock conveyed to a flask-loaded skip system.

At the time of commissioning the Shaft 2 loadout and skip-hoisting system, a 6 kt/d jaw crusher will be installed on top of Shaft 2 ore storage bin 011, and the trucks will haul rock to this crusher as well as the Shaft 1 jaw crusher for a combined hoisting capacity of 9,500 t/d. The crusher discharge will be fed into the ore bin for loading into the skips in Shaft 2. This material will be delivered to the concentrator stockpile.

Once the production gyratory crusher and conveying system from the orebody is commissioned all production rock, and most development rock, will be handled through the production crusher and conveying system for delivery to the concentrator.

16.2.6 Mine Services and Support Infrastructure Design

16.2.6.1 Mine Ventilation

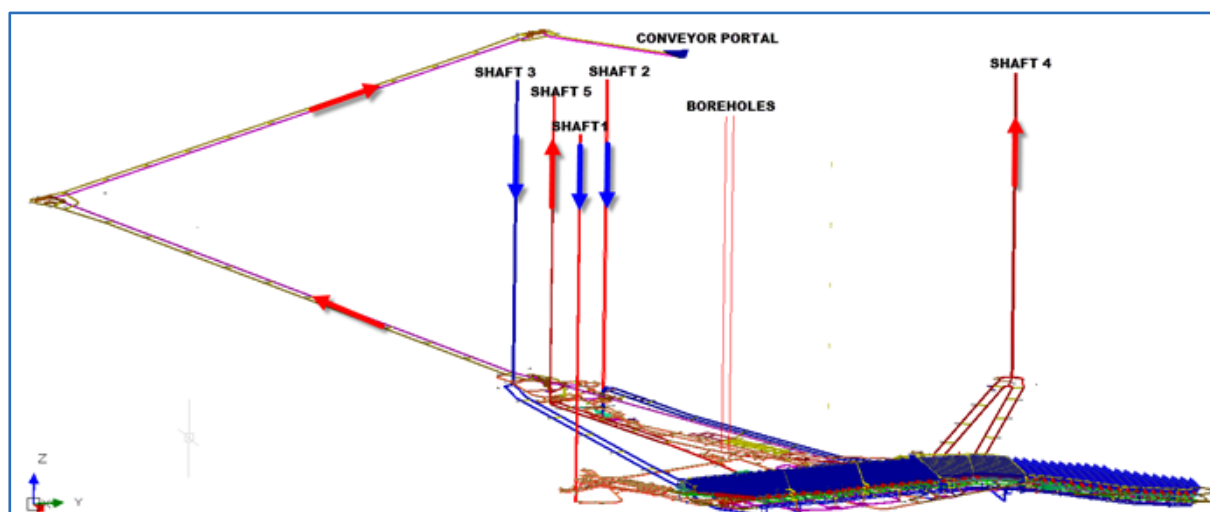
At full production, fresh air will enter the mine through one of three shafts and exit through two dedicated exhaust shafts as well as the conveyor to surface portal. The ventilation system is primarily a pull design with the main exhaust fans on exhaust shafts. The system components are outlined in Table 16.16 and Figure 16.30.

The ventilation system is designed to handle peak demand. This was determined by modelling the system for the year 2027 to simulate operating conditions at full production, when the maximum number of extraction drifts will have been opened for production. In addition, all off-footprint drifts will have been developed and all mine facilities incorporated into the ventilation plan.

Table 16.16 Ventilation Flow Summary

Shaft	Diameter (m)	Flow Direction	Start of Production (m ³ /s)	Full Production (m ³ /s)
Shaft 1	6.7	Intake	310	431
Shaft 2	10.0	Intake	550	908
Shaft 3	10.0	Intake	n/a	1,217
Shaft 4	11.0	Exhaust	n/a	1,765
Shaft 5	6.7	Exhaust	867	662
Conveyor to Surface	n/a	Exhaust	n/a	160
Total Intake at Mine Level Density			774	2,301

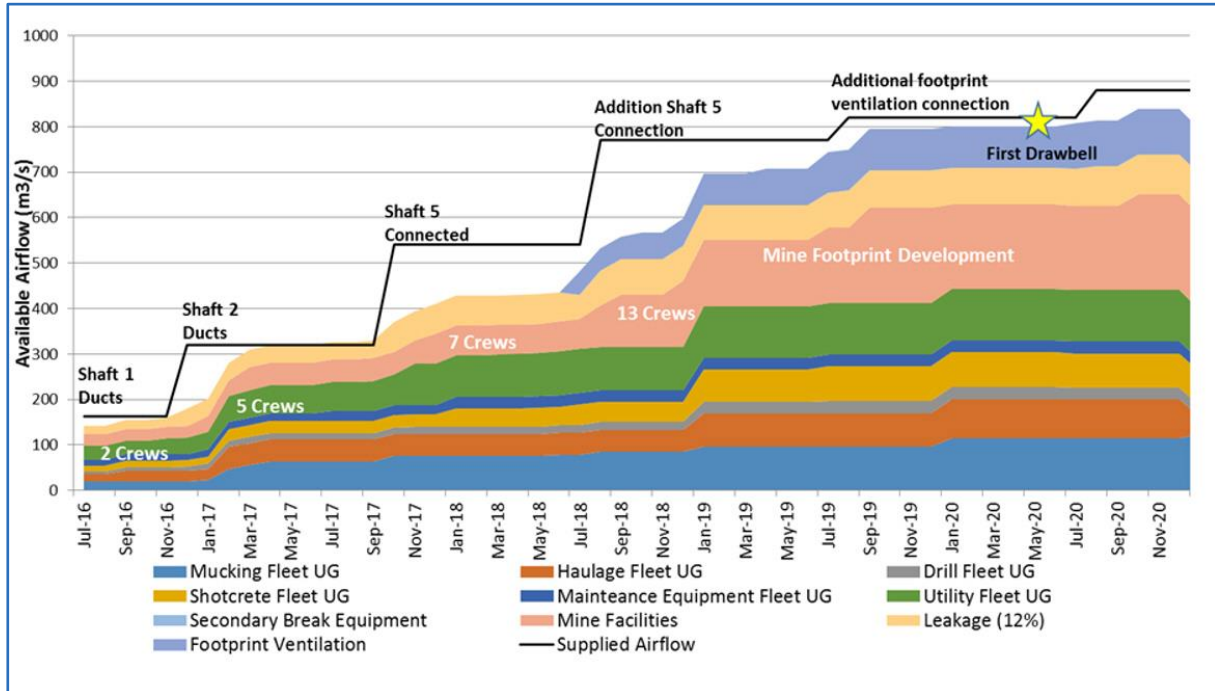
Figure 16.30 Shafts and Ventilation Raises



Future design may consider the potential to add a ventilation management system that could be used to reduce airflow to non-operating headings and drifts in non-peak times, thus optimizing ventilation performance and power usage.

The capacity of the ventilation system will increase incrementally as various shafts and fans are brought on line. Figure 16.31 shows the ventilation build-up and loads during the construction period to 2020.

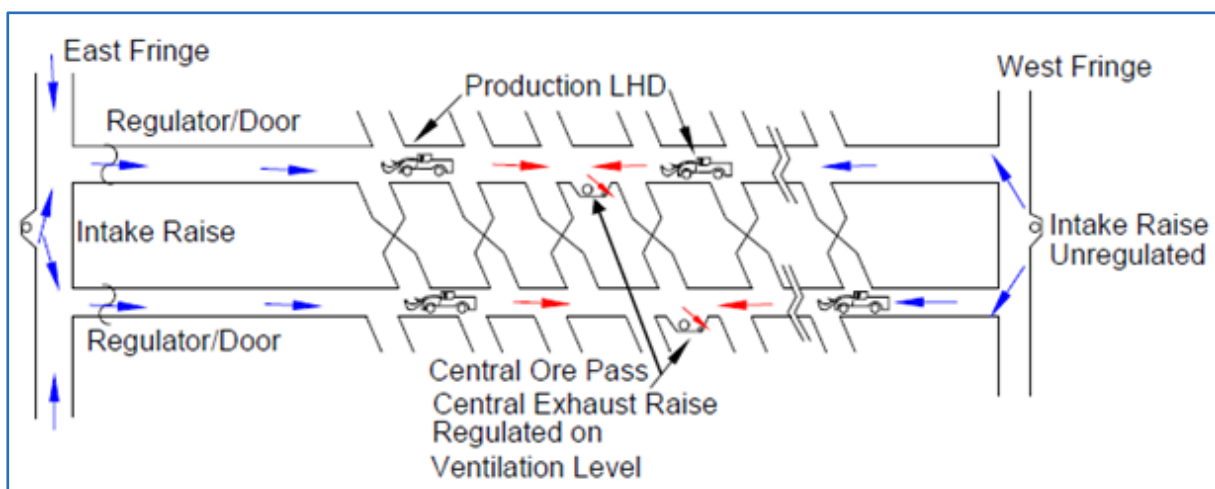
Figure 16.31 Ventilation Build-up and Consumption by Activity



16.2.6.2 Production Area Ventilation

The extraction level will be ventilated from both east and west fringes, as shown in Figure 16.32. Fresh air will enter the level through raises from the intake airways below the level. The air will travel through the extraction drifts to the centrally located exhaust raise and then down to the exhaust airway. This arrangement allows one mucker to work in fresh air at each end of the extraction drift.

Figure 16.32 Extraction Level Ventilation



16.2.6.3 Dust Management

Dust management will be required in various locations throughout the underground mine:

- Extraction level dust control – The two main dust creation areas are the drawpoint loading locations and the LHD dump locations. Each drawpoint will be fitted with water sprays to suppress dust. The extraction drift ventilation system will include intake raises along the rim drift and central exhaust raises.
- Haulage level truck chute – Each central truck-loading chute will be fitted with an adjacent exhaust raise to remove dust directly to the exhaust circuit. At each of the two crusher truck dumps, air will be drawn across the dumping area, capturing the dust created and depositing it directly to an exhaust raise that feeds to the return air ventilation system. The exhaust connection is shown in Figure 16.33. The crusher dump locations will also be fitted with water sprays to suppress dust.
- Ore handling – Dust collectors will be located on the apron feeders under each crusher. Transfer chutes will be enclosed and fitted with water sprays. Conveyors will be fitted with dust covers, with conveyor airflow along the direction of ore travel. Dust created from the Shaft 2 loadout will be removed directly to Shaft 5 as shown in Figure 16.34.

16.2.6.4 Heating

Mine air heaters will be installed on all intake shafts: Shaft 1, Shaft 2, and Shaft 3. Heaters will need to be running any time there is a possibility of the intake air being at freezing temperatures. The design temperature for the heated air entering the mine is +2°C. The system will use hot water from a central heating plant delivered to glycol heat exchangers to transfer heat to each mine air heater glycol loop, which in turn heats intake air from ambient to design discharge temperature.

Figure 16.33 Haulage Level Exhaust Connection

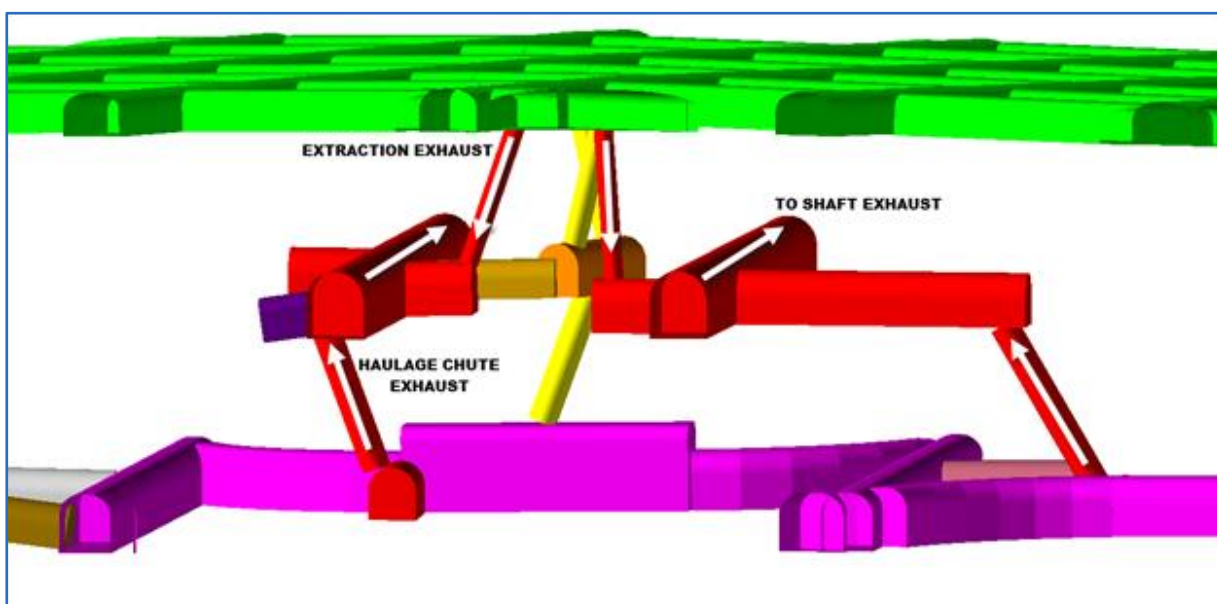


Figure 16.34 Shaft 2 Loadout Exhaust Connection



16.2.6.5 Permanent Infrastructure Design

Underground Warehouse

The main warehousing area for the mine is the central warehouse on the surface. Underground warehousing facilities will also be available, divided among the five underground shop areas. Mine development materials such as pipe, rock bolts, wire mesh, etc., will be stored in cut outs, muck bays, and other temporary facilities near the working places.

Maintenance Shops

The underground maintenance shops will consist of service bays / garages, auxiliary storage, and warehouse facilities for the maintenance of the underground mobile equipment fleet and fixed plant equipment. The maintenance facilities for the underground mine are summarized in Table 16.17.

Major equipment overhaul and rebuilds will be done in central service facilities on the surface. Tyre repairs and recapping will be done in a central tyre shop on the surface.

Table 16.17 Underground Mine Maintenance Shops

Shop Title	Location	Main Service
Shaft 1 Shop	Near Shaft 1 – 1300 Level	Development, construction, and service equipment
Main Shop	Extraction level	Production equipment and general fleet
Haulage Truck Shop	Haulage level	Production haul trucks
Drill Shops (two)	Undercut level (north-east and south-west)	Undercut drills and development equipment
Electrical Shop	Part of main shop	Electrical equipment, including temporary power reticulation systems
Fixed Plant Shop	Part of haulage shop	Crushers, conveyors, and fixed plant
Pipe Shop(s)	Part of main shop	Water and air piping

Fuel and Lube Station

Underground equipment will be fuelled at one of three locations: the Shaft 1 maintenance shops, the extraction-level fuel station near the main shop, and the fuelling station on the haulage level. In addition, fuel/lube service trucks will deliver fuel to slow moving equipment such as drills that are not working in the vicinity of the permanent fuel stations. The fuel will be received and stored in surface storage tanks dedicated to the underground mine. Storage capacity will equal six weeks of fuel use at 44 kL/d, the maximum rate of consumption.

Primary fuel will be delivered from surface to underground, to support extraction level and haulage level fuel stations, by means of fuel lines installed in boreholes adjacent to Shaft 5. From the base of the boreholes, fuel will flow by pipeline along the Shaft 5 exhaust drive to the fuel stations. A pipeline in a borehole adjacent to Shaft 1 will deliver fuel to the Shaft 1 fuel station. All fuel delivery is batched, and pipelines will be empty except during the transfer.

Lubricants will be supplied to the underground in totes, which will be stored in specifically designated drifts in the fuelling areas, sealed off with spillage bunds and fire doors. Waste oil will be collected in empty totes for delivery to the surface.

Explosives

A single international supplier of explosive products for Oyu Tolgoi will supply blasting agents selected for underground mine development and production during project execution. As per Mongolian regulations, a maximum of three days' production worth of explosives will be stored underground.

The main underground magazine will include storage areas for bulk explosive product reagents, packaged explosives, and detonators. A total of four storage bays will be provided within the magazine gates. Additional storage areas to house waste and equipment will be outside but in close proximity to the main magazine.

Underground Office and Lunchroom Complex

The main underground personnel facilities will be adjacent to the main maintenance shop on the extraction level, approximately 450 m from Shaft 2.

One section of the lunchroom is designed to function as the main refuge area. The capacity of this refuge area is 200 persons.

Refuge Stations

Portable refuge stations will be strategically located in areas of the mine for ease of access in the case of an emergency. These will include both permanent and a network of portable MineARC or similar 20-person refuge stations.

16.2.6.6 Mine Services

The underground mine will require the following services:

- Compressed air.
- Service water.
- Mine dewatering system.
- Fire protection.
- Shotcrete/concrete delivery.
- Electric power distribution.
- Communications, instrumentation, and data system.

16.2.7 Development and Construction Schedule

16.2.7.1 Milestones

The underground project schedule includes three major components: Development, Construction, and Production. Each schedule was developed separately but was linked to the others to develop an overall project schedule. Refining the schedules required multiple iterations to ensure all components matched and provided the optimal solution. Based on these schedules, the major milestones to first ore are as shown in Table 16.18.

Table 16.18 Underground Major Milestones

Milestone	Date
Commission Shaft 2 Exhaust Fans	Late-2016
Commission Explosives Magazine	Late-2016
Commission Shaft 5	Late-2017
Commission Shaft 2 – Cage	Mid-2018
Commission Shaft 2 – Hoist, Loadout, Jaw Crusher	Mid-2018
Start Undercut Ring Drill and Blast*	Late-2019
Commission 30 kt/d Ore Handling System	Late-2019
First Drawbell Blasted*	Mid-2020
Production Ramp-up Commences	Early-2021
Conveyor to Surface Commissioned	Early-2022
Crusher 2 Commissioned	Mid-2022
Concentrator Upgrade Complete	Late-2022
Expansion / Development Capital Complete	Late-2022
Full Production Achieved (95 kt/d)	Early-2027

* Includes five months' schedule range contingency

16.2.7.2 Development Schedule

The development heading advance rates are based on simulations developed to model various advance scenarios, heading configurations, and crew situations. These rates are calibrated against actual job site performance and used in the scheduling package. Table 16.19 outlines the development rates of the conveyor decline. The development rates used for the schedule are summarized in Table 16.20.

The conveyor to surface (C2S) is developed from both surface and underground with planned breakthrough along CVB-020. The breakthrough location is driven by the availability of underground resources to commence incline development. Transfers are developed off critical path behind the advancing face by a dedicated mass excavation crew.

Development requirements will increase to a maximum of 13 crews in the main mine ventilations district. Additional development crews will carry out the development of the C2S from the surface until breakthrough. Figure 16.35 shows the total metres per period, the general location of these metres, and the cumulative number of metres developed and the total design development quantities are shown in Table 16.13.

Figure 16.35 Development Metre Build-up

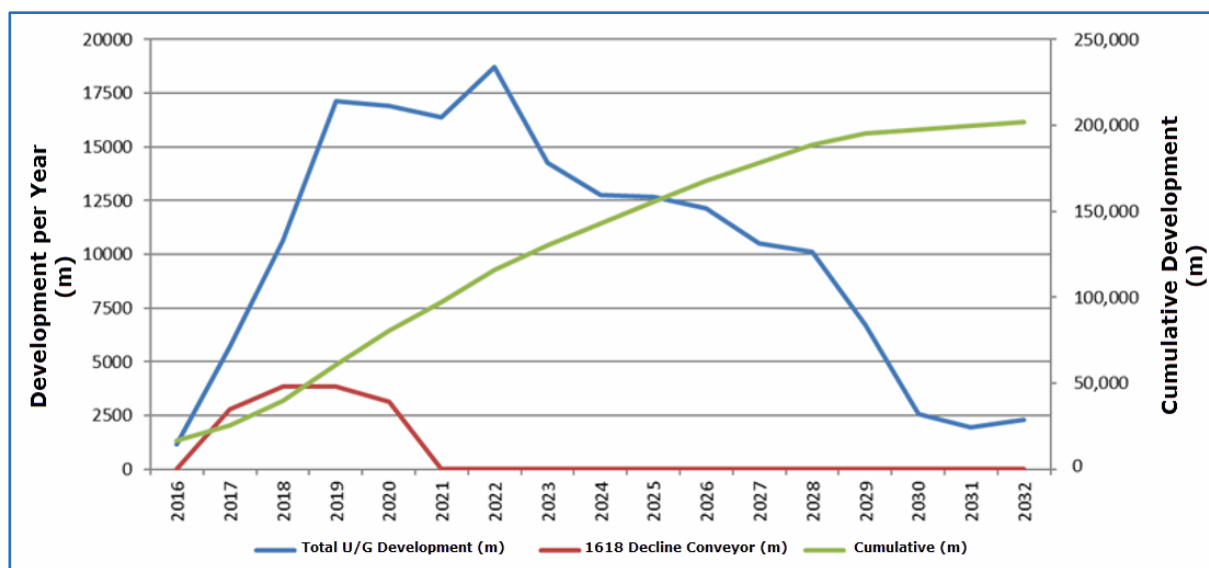


Table 16.19 Main C2S Decline Development Rates

Decline Development	Primary Heading (m/d)	Muckbays / Cross-cuts (m/d)
Conveyor Drive	4.6	1.6
Service Drive	5.0	1.2

C2S = conveyor to surface

Table 16.20 Development Rate Summary

Development	Type	Dimension (m) (width x height)	Off-Footprint			On-Footprint		
			m/d	Three Headings m/mo/crew	Each Heading m/mo/crew	m/d	Three Headings m/mo/crew	Each Heading m/mo/crew
All development except Panel 0, Subpanel B								
Undercut Drifts	B	4.0 x 4.2	-	-	-	5.3	154	51
Apex Rim	B3	4.5 x 5.5	-	-	-	5.3	154	51
Undercut Perimeter	B2	5.0 x 5.5	-	-	-	3.6	104	35
Undercut Cross-cut	B3	4.5 x 5.5	-	-	-	3.6	104	35
Extraction Drifts	E1	4.5 x 4.5	-	-	-	3.6	104	35
Drawpoint Drifts	E2	4.5 x 4.2	-	-	-	3.6	104	35
Drawbell Drifts	E3	4.5 x 4.2	-	-	-	3.6	104	35
Extraction Perimeter	E4	5.0 x 5.5	-	-	-	3.6	104	35
Vent Drifts (on FTP)	X1	5.0 x 5.5	-	-	-	3.6	104	35
Vent Drifts (on FTP)	X2	5.8 x 6.5	-	-	-	3.2	93	31
Vent Shaft 5 Exhaust	X3	6.0 x 6.0	4.8	139	46	-	-	-
Vent Drift	Y	6.0 x 7.0	4.8	139	46	3.3	96	32
Raisebore Cut-out	Y2	6.0 x 7.0	-	-	-	3.3	96	32
Haulage Drift Straight	Q	5.4 x 6.1	3.2	93	31	3.2	93	31
Haulage Drift Corner	QA	6.0 x 6.1	3.2	93	31	3.2	93	31
Conveyor	L	6.0 x 5.4	5.1	148	49	-	-	-
Ramp Access	H	5.0 x 5.5	6.2	180	60	4.3	125	42
Panel 0, Subpanel B								
Undercut Drifts	B	4.0 x 4.2	-	-	-	4.2	122	41
Apex Rim	B3	4.5 x 5.5	-	-	-	4.2	122	41
Undercut Perimeter	B2	5.0 x 5.5	-	-	-	2.9	84	28
Undercut Cross-cut	B3	4.5 x 5.5	-	-	-	2.9	84	28
Extraction Drifts	E1	4.5 x 4.5	-	-	-	2.9	84	28
Drawpoint Drifts	E2	4.5 x 4.2	-	-	-	2.9	84	28
Drawbell Drifts	E3	4.5 x 4.2	-	-	-	2.9	84	28
Extraction Perimeter	E4	5.0 x 5.5	-	-	-	2.9	84	28
Vent Drifts (on FTP)	X1	5.0 x 5.5	-	-	-	2.9	84	28
Vent Drifts (on FTP)	X2	5.8 x 6.5	-	-	-	2.8	82	28

16.2.8 Equipment Fleet

16.2.8.1 Mobile Equipment

The underground mobile equipment fleet is classified into seven broad categories:

- Mucking
- Haulage
- Drilling
- Raisebore / boxhole
- Utility and underground support
- Surface support
- Light vehicles

The underground equipment fleet will increase in size from currently more than 90 units to more than 290 units in 2020. Fleet size and composition will fluctuate with demand and changes in the work requirements. Figure 16.36 shows the underground equipment fleet. Each unit of equipment is scheduled for a major rebuild at 60% of its forecast life and will be replaced at 100% of scheduled life. Table 16.21 shows the planned life of select major classes of equipment.

Figure 16.36 Underground Mobile Equipment Fleet by Main Category

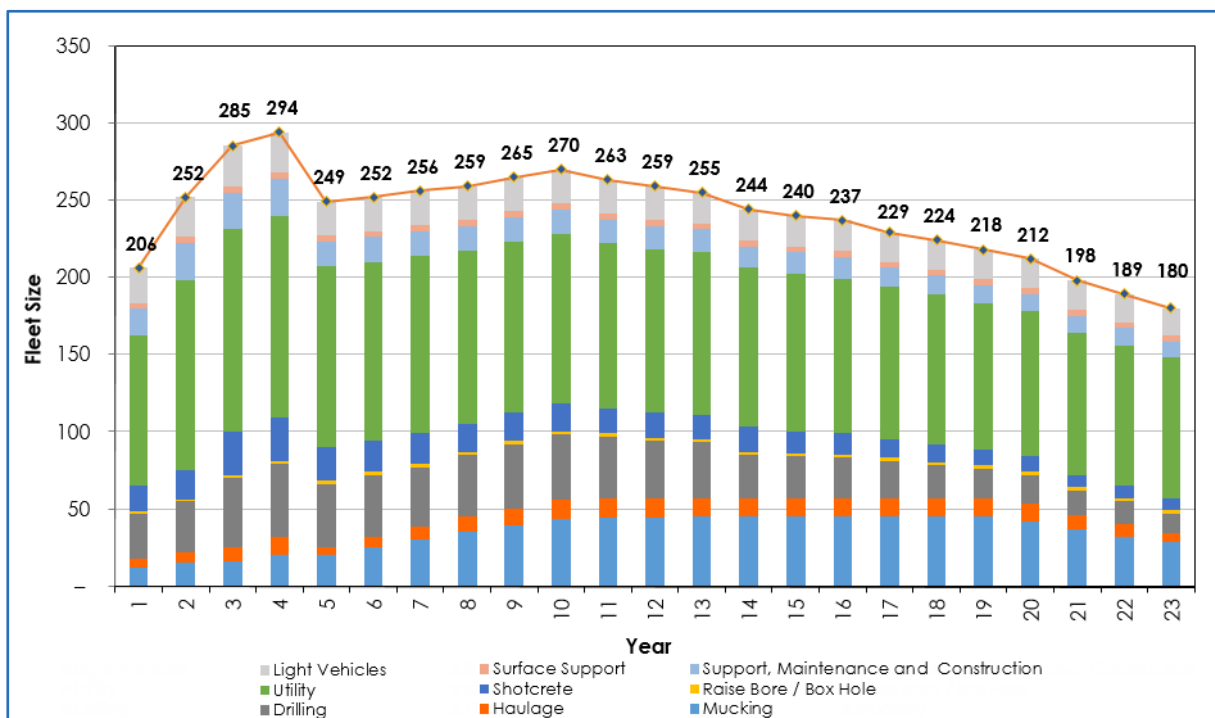


Table 16.21 Mobile Equipment Replacement Life

Equipment Type	Replacement Life (kh)
Mucking	21
Haulage	30
Drilling (Face)	25
Drilling (Ground Support)	25
Shotcrete Sprayer	20
Utility	30

16.2.8.2 Fixed Equipment

The major fixed equipment will include:

- Material handling (crushing and conveying).
- Fans and ventilation equipment.
- Pumping and water handling equipment.
- Power distribution equipment.
- Data and communications equipment.
- Maintenance equipment (shop furnishings).

16.2.8.3 Personnel

The IA with the GOM requires that the workforce consist of predominantly Mongolian nationals, but recognizes the need for foreign technical and managerial expertise during the early years of the operation. The IA sets specific ceilings for the ratio of Mongolian citizens to expatriates at different stages of mine construction and operations. The ratios used for the underground mine are outlined in Table 16.22.

Table 16.22 Workforce Make-up

Project Stage	Mongolian (%)	Expatriate (%)
Mining Contractor	75	25
Construction contractor	60	40
OT LLC Operations	90	10

During the life of the mine, the workforce for the underground mine will approach 100% Mongolian.

16.2.8.4 Training

OT LLC will develop and operate a training facility and training programme for miners, mechanics, process plant operators, and technicians. Development of the training programme is continuing with input from OT LLC and major contractors and in coordination with the GOM.

The area-specific training programmes are grouped into categories as defined in the Training Management Plan – Underground Operation. These categories are:

- Mobile equipment operations.
- Technical and trade.
- Fixed plant operations.
- Lifting equipment.
- Blasting.
- Safety.
- People management.

The Hugo North mine will be highly mechanized. To satisfy its need for skilled equipment operators, the Oyu Tolgoi underground mine department will use computerized equipment simulators for training operators. This programme has been successful to date and will continue to be expanded to match the ramp-up in the number of employees.

16.3 Mining Production Schedules

The 2016 OTTR has examined production from open pit mining of the Oyut zones and underground mining from Hugo North. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT). OT LLC has prepared both the open pit and underground mining work and schedules. The case adopted for the 2016 OTTR assumes no expansion of plant capacity, is based on Mineral Reserves only, does not include Inferred Mineral Resources, and does not include underground mining areas other than Hugo North Lift 1.

16.3.1 Scheduling Assumptions

The following scheduling methodology was used to balance mine, mill, and stockpile quantities:

- Underground ore is designated as 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.
- The production schedule is based on Proven and Probable Mineral Reserves only. No Inferred Mineral Resources were used.
- The open pit schedules were based on mining inventories by bench reported within the pit stages.
- Low-grade stockpiling was used to balance the mining rate where necessary.

The parameters in Table 16.23 are carried in the detailed schedule.

Table 16.23 Mining Schedule Parameters

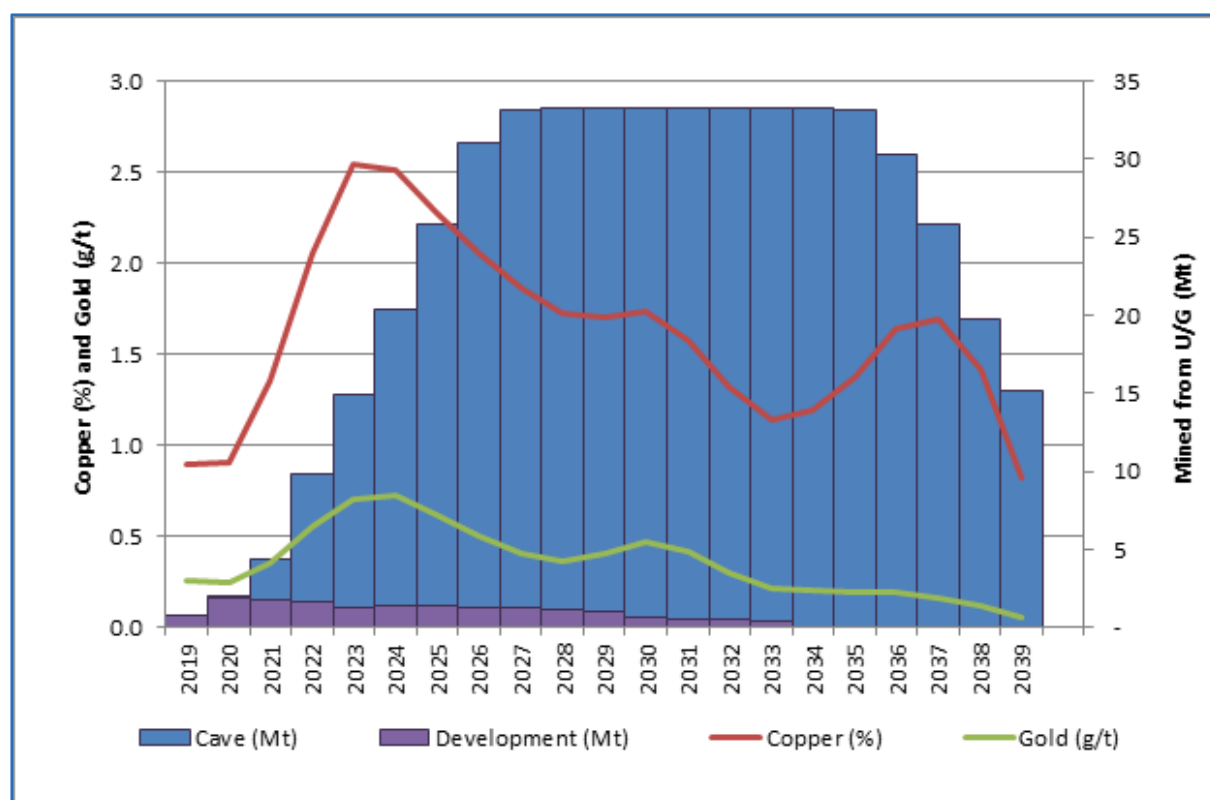
Variable	Unit	Variable	Unit
Copper grade	%	Mill hours for each kt of ore	HPKT
Gold grade	g/t	Copper recovery	%
Silver grade	g/t	Gold recovery	%
Arsenic grade	g/t	Silver recovery	%
Fluorine grade	ppm	Net smelter return	US\$/t
Sulphur grade	%	SAG power index	SPI
Iron grade	%	Modified bond index	MB
Molybdenum grade	ppm	Crusher index	Ci

Note: Annual throughput (Mt/a) = $(365 \times 24) / (1,000 \times \text{HPKT})$

16.3.2 Underground Production Schedule

The mine production schedule was developed using PCBC, a cave modelling software package. The start of production is scheduled for mid-2020. Critical hydraulic radius is expected to be reached in late-2020. The production rate will ramp up from 2021 through 2027, when the mine begins producing at a steady-state rate of 95 kt/d. Figure 16.37 shows annual underground production.

Figure 16.37 Underground Production Schedule



16.3.3 Open Pit Production Schedule

The Oyut open pit design consists of ten mining phases. Phases 1 and 2 are complete. The open pit reserves are mined over approximately 40 years, continuing while underground mining commences with the Hugo North Lift block cave. Various alternative operating scenarios for the open pit are currently being evaluated around pit phase design and sequencing. This evaluation shows strong potential for improving the value of the project, but the associated work streams for these improvements exceeded the time-frames available to finalise OTFS16.

The input assumptions for OTFS16 were generally kept the same as those used for OTFS14, with some minor adjustments due to differences between demonstrated and planned performance. While some productivity inputs for the mine and mill were adjusted based on

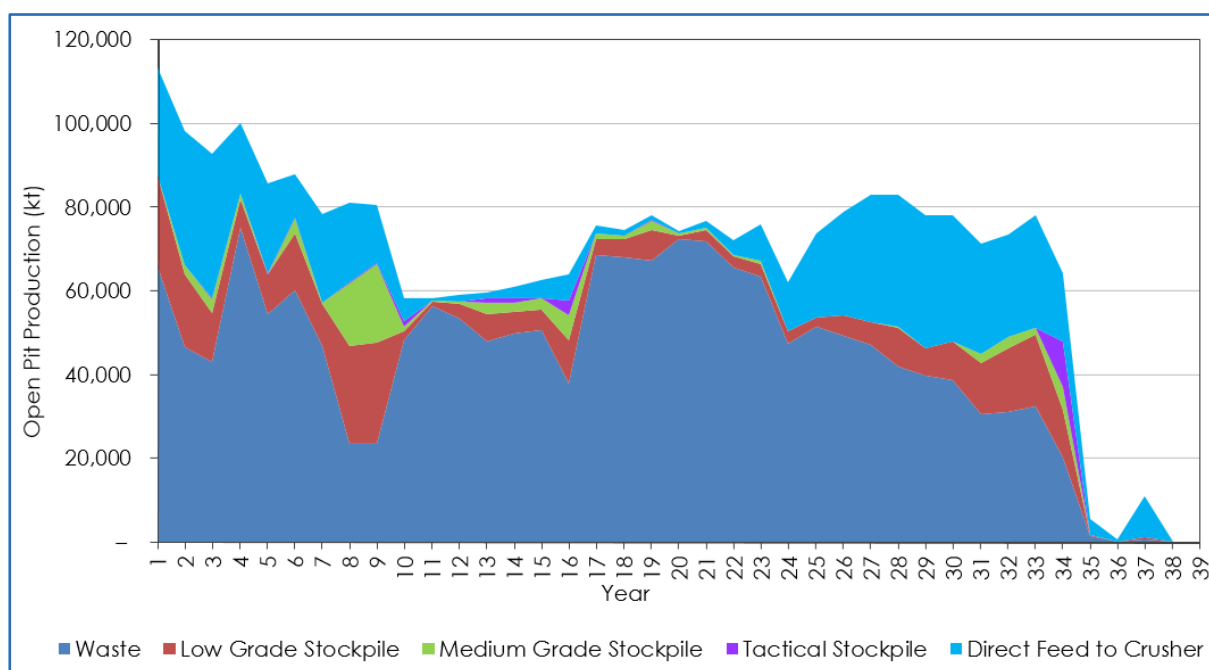
demonstrated performance, block model and metal recoveries were kept the same as those used for OTFS14.

The objective of the production schedule is 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 life-of-mine (LOM) scheduling include:

- Ensuring continuous ore supply to the concentrator by delivering the highest NSR value ore first and meeting physical mining and milling hours capacity constraints.
- Achieving shovel productivities and sinking rates to deliver ore at maximum utilization of milling hours available at the concentrator.
- Maximizing annual utilization hours for the mine loading equipment.
- Maintaining a balance of ore throughput hardness and mill cut-off grades that allows milling hours to be maximized.

The mine schedule incorporates strategic stockpiling considerations by optimizing the number of shovels on the benches of the early phases, increasing the opportunity to raise mill cut-off grades. This leads to stockpiling medium- and low-grade material and sending higher grade ore to the mill sooner. The open pit total movement (Year 1 is 2017) is shown in Figure 16.38.

Figure 16.38 Open Pit Production



16.3.4 Processing Schedule

The processing schedule was balanced to meet the available mill hours after the underground material was processed. The processing schedule by metallurgical ore type with the copper, gold, and silver feed grades are shown in Figure 16.39. Year 1 is 2017. Total concentrate production by ore type is shown in Figure 16.40. The recovered copper, gold, and silver production is in Figure 16.41 to Figure 16.43. The production schedule is in Table 16.24.

Figure 16.39 Ore Processing and Grade by Ore Type

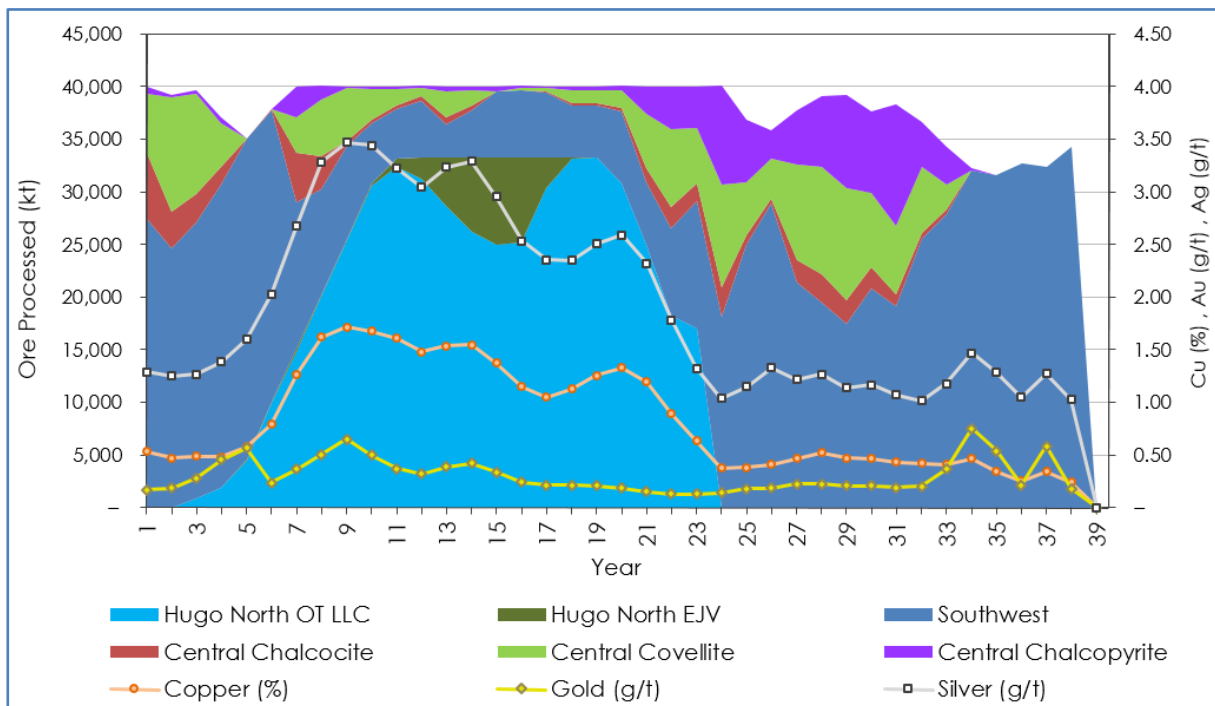


Figure 16.40 Concentrate Production by Ore Type

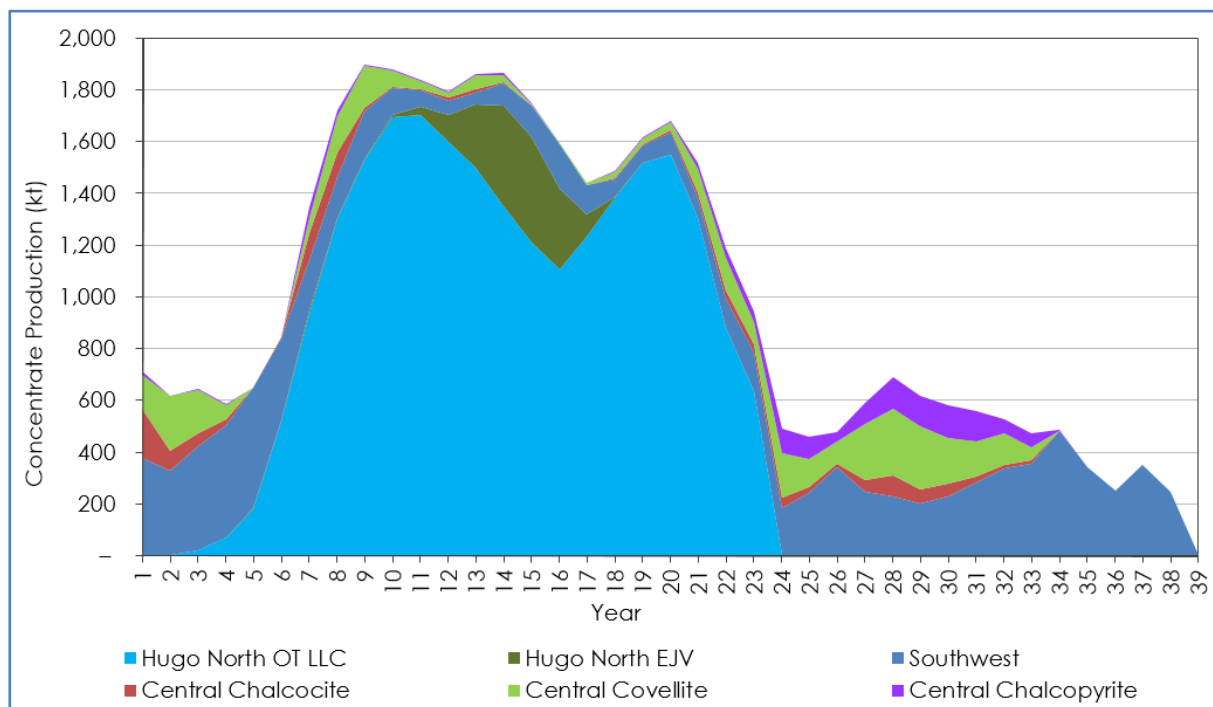


Figure 16.41 Recovered Copper Production

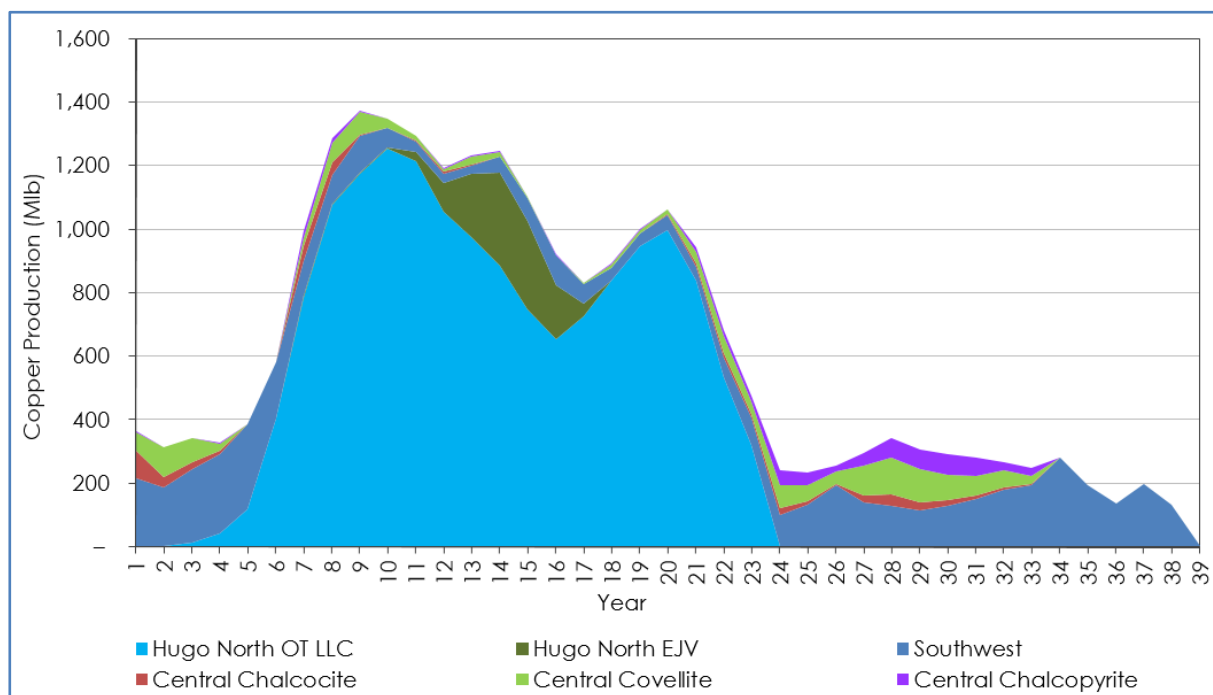


Figure 16.42 Recovered Gold Production

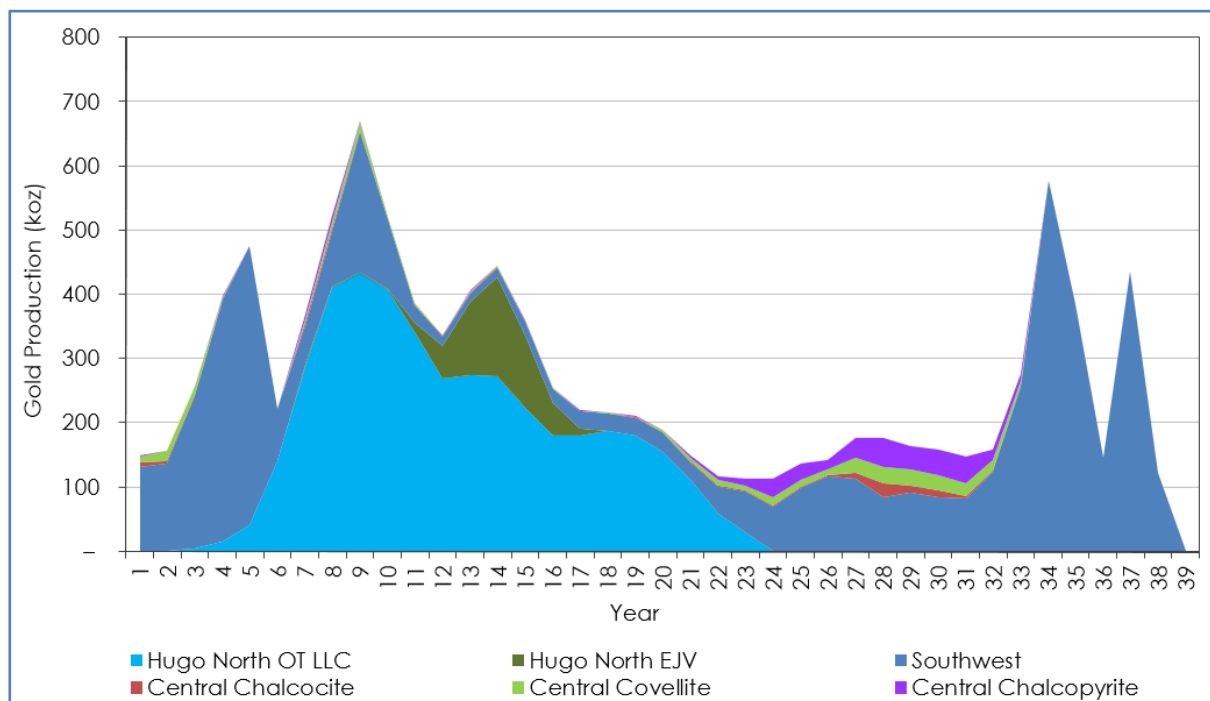


Figure 16.43 Recovered Silver Production

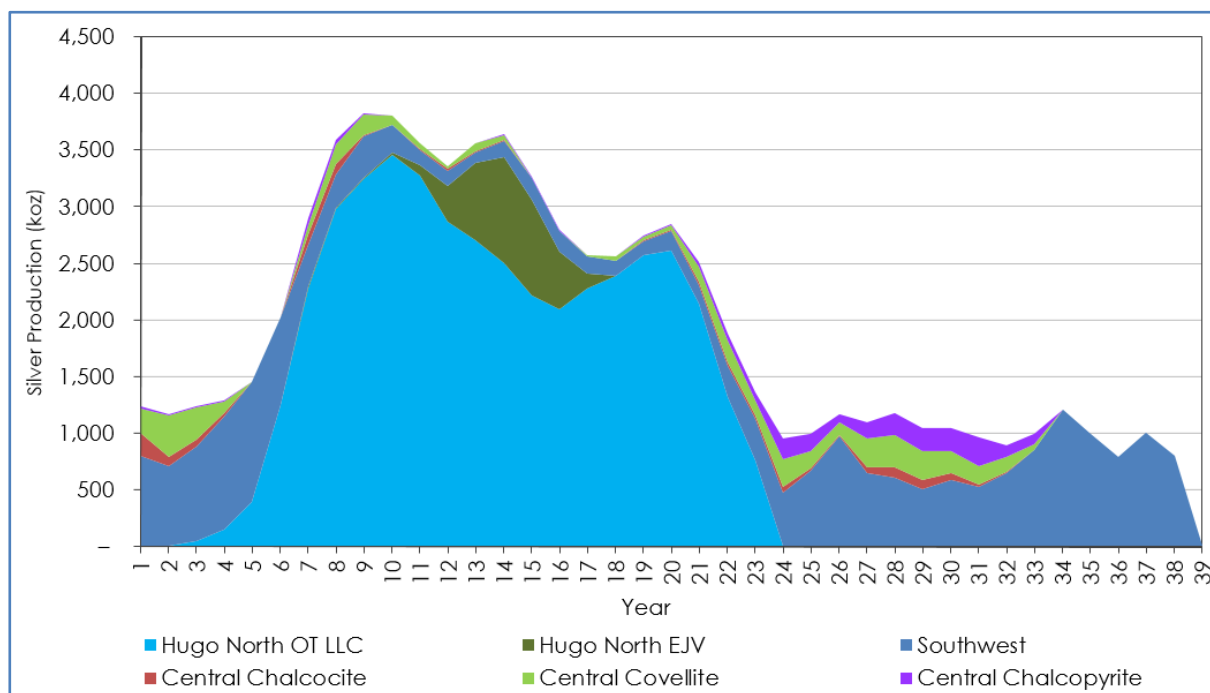


Table 16.24 Production Schedule

		Year												Total		
Year Number		1	2	3	4	5	6	7	8	9	10	11	21		31	
Year To												20	30		40	
Open Pit Ore Waste	kt	39,980	39,169	38,818	35,161	30,518	27,842	24,901	19,936	14,456	9,180	70,468	326,251	272,289	948,969	
	kt	65,788	46,565	43,109	75,153	54,397	60,194	46,786	23,343	23,362	48,171	571,594	516,711	116,278		
Total Movement	kt	105,768	85,734	81,927	110,314	84,915	88,035	71,687	43,279	37,818	57,351	642,061	842,962	388,567	2,640,419	
Underground	kt	–	–	881	1,905	4,527	10,046	15,101	20,177	25,547	30,823	329,890	60,269	–	499,164	
Processed	kt	39,980	39,169	39,699	37,066	35,045	37,887	40,003	40,112	40,003	40,003	400,358	386,520	272,289	1,448,133	
	NSR	25.33	23.03	28.51	34.14	42.99	45.96	73.96	97.01	107.11	101.77	77.61	30.33	25.03		49.98
	Cu%	0.53	0.47	0.49	0.48	0.57	0.79	1.27	1.62	1.71	1.67	1.34	0.58	0.36		
	Au g/t	0.17	0.18	0.28	0.45	0.56	0.24	0.36	0.51	0.65	0.50	0.29	0.18	0.37		
	Ag g/t	1.29	1.25	1.26	1.39	1.60	2.02	2.68	3.28	3.47	3.44	2.81	1.38	1.16		
	As ppm	103.60	139.21	82.65	47.97	25.33	52.14	75.41	81.87	52.97	56.87	72.77	90.43	54.21		
	Mo ppm	55.85	57.83	45.72	50.18	55.04	38.62	53.11	51.82	36.15	29.97	38.96	44.77	57.03		
Bulk Concentrate	Conc. kt	710	619	644	585	650	842	1,346	1,721	1,897	1,877	16,923	7,553	3,229	38,597	
	Conc. Cu %	23.24	23.05	24.13	25.42	26.86	31.33	33.40	33.87	32.81	32.61	28.87	24.41	24.30		28.06
	Conc. Au g/t	6.53	7.85	12.37	21.13	22.73	8.16	8.52	9.44	10.97	8.64	5.55	5.95	21.63		
	Conc. Ag g/t	54.23	58.59	59.75	68.66	69.62	74.70	66.91	64.91	62.67	63.10	56.80	54.37	73.55		
	Conc. As ppm	2,597	4,114	2,458	1,207	320	601	994	1,095	694	755	1,018	2,140	924		
	Conc. Mo ppm	–	–	–	–	–	–	–	–	–	–	–	–	–		
	Conc. F ppm	613	612	588	550	494	499	509	552	559	568	617	580	528		
Recovered Metal	Copper Mlb	364	314	343	328	385	581	992	1,285	1,372	1,349	10,770	4,064	1,730	23,878	
	Gold koz	149	156	256	397	475	221	369	522	669	521	3,021	1,445	2,246		10,449
	Silver koz	1,239	1,166	1,238	1,291	1,454	2,022	2,896	3,592	3,822	3,808	30,904	13,203	7,636		

17 RECOVERY METHODS

17.1 Summary

The Phase 1 concentrator commenced ore commissioning in January 2013 with production of first copper–gold concentrate on 31 January 2013 and commercial production was achieved in September 2013. Mill throughput reached the nominal 100 kt/d design capacity, with milled throughput exceeding design in April, 2014. Concentrator performance, as measured by recovery, has improved with operating experience and improving head grade.

Concentrate production has progressively increased, as a combination of increasing mill throughput and grade and improvements in concentrator performance with recovery. Concentrates grades have been at or near design, particularly for copper, despite lower feed grades, as a result metal production has improved as mill throughput and feed grade have improved.

The expansion scope (Phase 2) is all the additional work required to process Hugo North Lift 1 production plus open pit ore to match Phase 1 SAG mill capacity, including:

- the addition of a fifth ball mill to achieve a finer primary grind P80 of 140–160 µm for a blend of Hugo North and open pit ores
- additional roughing and column flotation capacity to process the higher level of concentrate production when processing the higher grade Hugo North ore
- additional concentrate dewatering and bagging capacity.

The above scope may vary slightly depending on the throughput outcome of future concentrator improvement work for Phase 1.

The parameters described below are considered the major capacity determinants for the Phase 2 concentrator conversion design.

Hugo North is planned to reach a peak production plateau of 95 kt/d (33.25 Mt/a) of the maximum concentrator capacity of 110 kt/d. To keep the concentrator at its maximum capacity, additional tonnage will be supplied from open pit orebodies to progressively higher limits set by:

- SAG milling capacity (annual average of daily tonnage), which varies primarily with SPI from 96–110 kt/d
- ball milling capacity
- tailings pumping volumetric capacity
- flotation and concentrate handling equipment
- current water permit limit of 918 L/s.

The concentrator volumetric capacity may be dependent on either the concentrate or tailings production rates. The basis of OTFS16 production is to operate at the lesser of TPUT or the tailings handling capacity (110 kt/d, or 40 Mt/a).

17.2 Concentrator Capacity Constraints

In the 2016 Reserves Case the underground mine production is 95 kt/d (33.25 Mt/a), with additional tonnage supplied from the Oyut open pit. The key concentrator capacity constraints from OTFS16 are:

- SAG milling capacity (annual average tonnage varies primarily with SPI from 96–110 kt/d).
- Ball milling capacity (soft constraint, with pumping and cyclone changes, but ultimately limited by flotation losses at coarser grind).
- Tailings pumping volumetric capacity (approximately 110 kt/d through one tailings line after applying 93.5% effective utilization).
- Flotation and concentrate handling equipment (approximately 125 kt/d at peak underground heads, whenever open pit ore is unavailable).
- Current water permit limit of 918 L/s with seasonal water balance variation at the tailings dam from summer evaporation, winter ice formation, and spring thaw. The OTFS16 average annual unit raw water projection assumed is 0.40 m³/t.

The capacity of the grinding circuit to receive ore is measured in terms of available mill hours at a specific rate. This is expressed as TPUT, the annual tonnage achievable in 8,191 running hours (93.5% effective utilization) through Lines 1–2. The Central zone ores are particularly soft and have correspondingly high TPUT values. The capacity to treat them is limited by other plant constraints such as the hydraulic limitations of the tailings system.

The concentrator volumetric capacity may be dependent on either the concentrate or tailings production rates. The OTFS16 production basis is to operate at the lesser of TPUT or the tailings handling capacity. This maximum TPUT from this assumption is 110 kt/d (40 Mt/a).

17.2.1 Phase 2 Concentrator Conversion

Completion of the concentrator conversion is driven by the underground cave production ramp-up schedule. Commissioning and ramp-up of the conversion are scheduled to be complete by the time underground ore is projected to comprise 26.5% of concentrator feed. Mining of Central zone ores with a high As : Cu ratio will elevate arsenic in the final concentrate, and so requires co-processing of low As : Cu ratio ores, such as Hugo North. Production sequencing to avoid penalty or rejection levels of arsenic is planned.

17.2.2 Blended Processing of Underground and Open Pit Ores

Separate processing of Hugo North ore was recommended in the 2009 Technical Evaluation Group (TEG) review and adopted in IDOP and DIDOP. However, following the independent decision to constrain Phase 2 to the original two grinding lines in Phase 1, there was found to be little potential to segregate processing conditions because of the Phase 1 plant design, where the products from the separate grinding lines are combined before flotation. Even if it proved possible to provide separate and equal flotation and pumping capacities for each line, and to replicate the sampling arrangements for separate processing and accounting for tailings streams, there remained little ability to separate the process water systems to allow independent pH control of both flotation circuits.

It was subsequently decided to select a compromise of grinding and flotation conditions for the different ores, and to process underground and open pit ores by blending through Lines 1 and 2. Hugo North would provide up to 95 kt/d of the maximum concentrator capacity of 110 kt/d. Open pit feed would vary as required to keep the concentrator at its maximum capacity. This also led to a change in philosophy, where grinding synergy would be maximized by combining hard and soft ores for maximum capacity and minimum unit cost, while trying to minimize the consequences of negative flotation synergy in terms of sub-optimal concentrate grade and recovery at compromise conditions. This has had a negative impact on Central zone recoverable metal, but a positive impact on project economics.

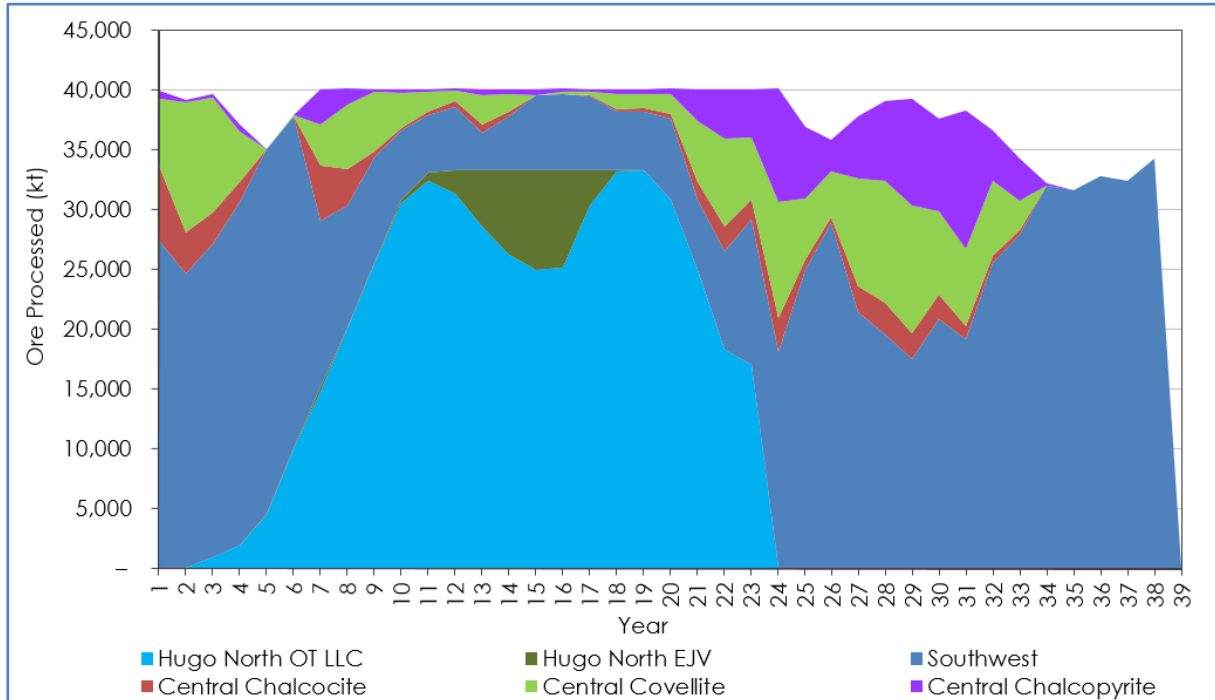
The optimum conditions for treatment of Southwest zone, Hugo North, and Central zone ores vary significantly in terms of primary grind and regrind sizes and also in operating pH in flotation roughing and cleaning. Hugo North ore carries the highest value and so compromise processing conditions are set much closer to those that are optimal for its copper and gold recovery, than for those that are optimal for Central zone ore metallurgy. As a consequence, recovery and grade from the Central zone ores have been reduced, as discussed in Section 13.3.

17.2.3 Reserve Production Schedule

The production schedule has evolved from Integrated Development Plan 2007 (IDP07) to the current OTFS16. The plan that has persisted is to process all of the high-value underground ore while displacing lower value open pit ore. As the cave matures, softer Central zone ore will be delivered up to the concentrator grinding or tailings handling capacity. When underground production eventually decreases, the total plant capacity will again decline with a return to harder Southwest zone ore, where the lower copper head grades and concentrate grades allow less blending of soft, high-arsenic Central zone ore, without exceeding maximum arsenic in concentrate limits or falling below minimum copper grade targets.

Figure 17.1 shows the Reserves Case production schedule by ore type. Soft constraints on maximum concentrate production and arsenic content and minimum concentrate grade are respected in the mine scheduling process, with a hard limit on volumetric capacity when processing soft ore types.

Figure 17.1 Production Schedule – 2016 Reserves Case



Conversion Schedule

Keeping the conversion equipment ramp-up and the transition of Phase 1 from Southwest zone to underground and Central zone ores in step presents several challenges and opportunities. The open pit and underground development schedules must be reviewed continuously in future work to maintain a feasible production schedule.

To ensure concentrate marketability with less than 5,000 ppm of arsenic (rejection limit) in every concentrate shipment, large volumes of higher arsenic Central zone ore can most safely be processed when accompanied by a suitable volume of low-arsenic, high-copper Hugo North ore. Such volumes are available only after Hugo North approaches full capacity through the concentrator.

Based on the cash flow potential of soft Central zone ore, Oyu Tolgoi's plan in 2016 is to process large volumes of Central zone ore from June 2016 and onwards. Contract limits for arsenic and copper in concentrate have been re-negotiated upwards and downwards accordingly to cater for an increased proportion of Central zone ore (up to 43%) from 2016 through 2020. This period of Central zone ore processing represents the first sustained opportunity to test Phase 2 volumetric constraints, albeit with tailings rheology that will not reflect Phase 2 operation.

After commissioning the conversion equipment during 2022, the Phase 2 circuit is scheduled to operate at 73% open pit ore and 27% Hugo North ore. It would have been appropriate to construct and commission the conversion equipment in reverse order from the traditional "front to back" process flow order, since the initial exposure is to lack of concentrate

handling capacity, rather than lack of grinding capacity. However, the Project Execution Plan (PEP) calls for a conventional approach to maximize effort earlier to align better with the scope of other surface facilities, which are assumed to be managed under the same contract.

A review and update of the conversion schedule should be carried out when the design study commences. This may indicate value in earlier installation of flotation capacity, especially if debottlenecking efforts upstream are successful.

Ramp-up Profile

The ramp-up profile for the conversion equipment is faster than for Phase 1, which was originally benchmarked at the 65th percentile for large copper concentrator start-ups, but was subsequently corrected for staggering the twin lines (Table 17.1).

- The dual line ramp-up profile is two months longer than the individual profile for either line to avoid overloading the construction, commissioning, and operating resources. Similarly, ramp-up for conversion equipment will be shorter than even for an individual line start-up, since each component can be brought on in parallel and is not required to run the other components.
- Concentrator utility tie-ins will be made in advance of Phase 2 commissioning, leaving fewer systems to troubleshoot.
- Hugo North and open pit ore will be delivered evenly onto the stockpiles for both Lines 1 and 2. However, for full-scale advance testing of operating conditions, Hugo North ore could be campaigned through one line during a scheduled shutdown of the other grinding circuit in Phase 1 in order to gain operating experience without overloading the flotation and dewatering areas.

Table 17.1 Ramp-up Profiles for Phase 1 and Phase 2

Phase	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7
Phase 1, Lines 1–2 ramp-up (%)	4	15	30	52	69	86	100
Conversion ramp-up (%)	15	50	85	100	–	–	–

Nominal Daily Capacity Definitions

For the conversion, with the feed change to softer underground and Central zone ores, the milling capacity could exceed the volumetric capacity of the Phase 1 tailings system. To take account of this limitation, the Phase 2 concentrator capacity in OTFS16 was redefined to 5 kt/h, (40 Mt/a, 110 kt/d), at 93.5% concentrator availability. Further elevation and revision of the limit is quite likely as de-bottlenecking and optimization of the plant continues. The OTFS16 limit has already been reached and may be exceeded as the Central zone ore is treated.

The final concentrate production schedule is shown in Figure 17.2. As compared to the ore processing schedule in Figure 17.1, the tonnage and concentrate production peaks occur in different years because throughput is limited either by the milling power or the volumetric constraint, while concentrate production is dependent on ore feed grade.

Table 17.2 compares the concentrator conversion limiting design criteria with the OTFS16 production schedule annual peak values.

Figure 17.2 Final Concentrate Production Schedule – 2016 Reserves Case

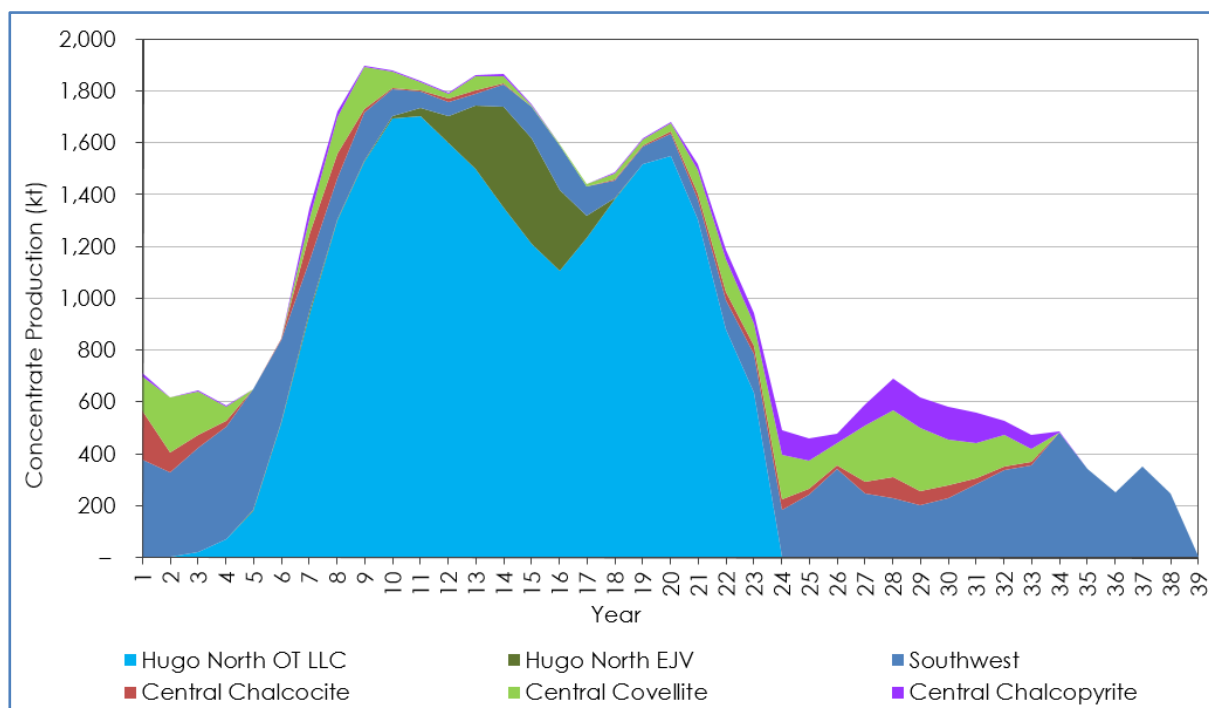


Table 17.2 Comparison of Concentrator Conversion Design Criteria and 2016 Reserve Peak Values

Description	Unit	Conversion Design Criteria	Reserve Mine Plan Peak	Year
Copper Feed Grade	%	3.0	1.71	2025
Annual Tonnage	Mt/a	44.3	40.1	2023–2040
Regrind Duty	kW	6,714	6,596	2043
Concentrate Production	kt/a	3,361	1,897	2027

Conventional Comminution / Flotation Flowsheet Options

In 2013, a concentrator conversion was investigated, incorporating incremental changes to the Phase 1 plant to treat a blend containing majority higher grade Hugo North underground ore instead of the Line 3 concentrator expansion to 160 kt/d ore throughput. The Hugo North orebody has characteristics compatible with the conventional SAG milling, ball milling, and flotation technology used in the Phase 1 plant. The conversion includes additional ball mill power to allow a finer flotation feed size compared with the Phase 1 grind size; this, in turn, will increase the flotation recovery of copper and gold. Use of the conventional flowsheet

maximizes use of the existing Phase 1 concentrator assets for treatment of Hugo North ore and has low process risk.

Several comminution variants on the basic process route were investigated to achieve the OTFS16 Phase 2 outcomes. These included: buffer stockpiles, gravity gold separation, mine-to-mill optimization, secondary crushing or HPGR, two-stage ball milling, rougher flotation, regrind milling, additional regrind ball storage, cleaner flotation, Jameson cell retrofit.

17.3 Mass Balance

Simulation of the process mass balance was undertaken for the Phase 2 expansion using Metsim.

For OTFS16, a concentrator design throughput of 110 kt/d was selected based on Oyu Tolgoi Operations' view on the near-term volumetric constraint.

The potential to overload the concentrate handling system due to high copper grades in the individual orebodies was examined and considered highly unlikely, given the capacity headroom available. The copper grades that would be required to overload the system exceed all values in the mine plan and any reasonable likelihood of short-term variation above those plan values, mainly because of the large number of underground drawpoints that are open at any one time.

The process inputs modelled in Metsim to match MP08v2 have been confirmed to meet the final production schedule in OTFS16.

At the feasibility level, there was no significant need to track particles by size, and so few unit operation types were required to simulate the concentrator. The most common were flotation cells, mix tanks, distributors, splitters, phase separators, filters, thickeners, and pumps.

The recovery and grades calculated in the model were based on correlations developed in the laboratory flotation programme and integrated with metallurgical predictions in the mine production schedule. Copper recovery has been correlated against copper head grade. Copper grade in concentrate has been correlated to the Cu : S ratio in ore.

In the model, no recovery distinction was made between individual copper species, e.g., for Hugo North the recoveries of chalcopyrite, bornite, and chalcocite were set as equal and were dictated by the calculated overall copper recovery. In the flotation model, recoveries were stated for gold, silver, arsenic, fluorine, and pyrite. The desired grade was achieved by dilution with gangue minerals in proportion to their presence in the feed stream to the flotation units. The correlations used in the model are shown in Table 17.3.

Table 17.3 Correlations Used in Mass Balance Model

Metallurgical Value	Equations and Parameters
Final Concentrates	$Cu_f = \% \text{ Cu in Ore}$, $S_f = \% \text{ S in Ore}$ $Cu_r = \% \text{ Cu in Ro feed}$, $Cu_c = \% \text{ Cu in final Conc.}$
Copper Recovery (%)	$a + (b \times Cu_f) / (1 + b \times Cu_f) \times (1 - e^{-bx\%Cu_f})$
Hugo North	$a = 95$; $b = 15$
Southwest Zone	$a = 98$; $b = 14.5$
Central Zone Covellite	$a = 80$; $b = 15$
Central Zone Chalcocite	$a = 72$; $b = 15$
Central Zone Chalcopyrite	$a = 88$; $b = 12.2$
Copper Grade (%)	
Hugo North	$(2.9 \times Cu_{\text{Recovery}}) + 11.4 \times (Cu_f : S_f) + 15.3$
Southwest Zone	$(12.8 \times Cu_f : S_f) - 3.6 \times (Cu_f : S_f)^2 + 22.5$
Central Zone Covellite	20
Central Zone Chalcocite	20
Central Zone Chalcopyrite	$(12.8 \times Cu_f : S_f) - 3.6 \times (Cu_f : S_f)^2 + 21$
Gold Recovery (%)	$c + (d \times Cu_f)$
Hugo North	$c = 9.8$; $d = 0.80$
Southwest Zone	$c = 4.8$; $d = 0.80$
Central Zone Covellite	$c = 4.8$; $d = 0.65$
Central Zone Chalcocite	$c = 4.8$; $d = 0.70$
Central Zone Chalcopyrite	$c = 4.8$; $d = 0.80$
Silver Recovery (%) All Ores	$0.8 \times Cu \text{ Recovery} + 13$
Rougher Concentrate Grade (%)	$Cu_r \times (Cu_c / Cu_r)^K$
Hugo North	$K = 0.6$
Southwest Zone + Central Zone Ores	$K = 0.5$

Recirculating loads have been altered between Phase 1 and Phase 2. In Phase 1, cleaner scavenger concentrate was recycled to the regrind circuit, increasing the flow through the regrind cyclones. In the current model, the scavenger concentrate is returned directly to the cleaner flotation cells. The assumption is that material once reground to a certain size is unlikely to benefit by return to the same circuit with pre-classification and the same product size criterion.

A 200% circulating load is assumed for the regrind circuit processing only rougher concentrate, compared to 100% in the Phase 1 design. This reflects the impact of material that has already been reground no longer being present in the new feed to the regrind circuit. These modifications allow Lines 1 and 2 to continue to operate without modification of the regrind classification and pumping circuits.

The mass balance model allowed the assessment of all unit operations from the coarse ore stockpiles to concentrate dewatering, tailings disposal, water reclaim, and tailings storage facility (TSF) evaporation. Thus it was possible to effectively evaluate the overall process and raw water demands and the impacts on shared facilities such as pebble crushing, concentrate handling, tailings thickening, and the TSF.

17.4 Process Design Criteria

The process design criteria are the key elements used for plant design. This section discusses the overall design assumptions and constraints for selection of equipment for the major process plant duties. The DIDOP design criteria have been modified to accommodate the concentrator conversion for OTFS16. They include, for example, criteria for sizing the SAG and ball mills, pebble crushers, flotation cells, regrind Vertimills, filtration and thickening units, and concentrate storage and bagging facilities.

The mass balance flows and ore analyses for plant design were taken from provisional mine plan MP08v2, based on year-2021 in that schedule. All concentrator conversion upgrades were based on MP08v2. Design factors have been re-calculated relative to the OTFS16 ore supply and concentrate production schedules and are shown to be conservative.

The development of the design criteria is an iterative process in which process assumptions must match and keep pace with test results, mine plans, economic constraints, vendor data, etc., for example:

- Grinding testwork and preliminary mill selection provide the key capacity input to the mine, resulting in a production plan. In many cases, increments are determined by the largest available equipment or the size of the equipment already installed to minimize holding costs for insurance spares.
- Flotation recoveries and concentrate analyses provide the head grade-related capacity and product quality constraints used to tune the mine plan to maximize NSR while still producing a readily marketable product.
- The production plan is incorporated into the design criteria and ultimately drives the next mass balance.
- The mass balance usually identifies shortfalls or inconsistencies that demonstrate the need for additional testwork before ultimately refining the production plan.

The capacity of the concentrator conversion plant is constrained by the tailings volumetric capacity when softer Hugo North and Central zone ores are processed, and then reverts to the grinding TPUT (throughput) when Oyut ore processing ramps up again in 2036. The Oyut deposit was formerly known as Southern Oyu Tolgoi (SOT).

The tailings volumetric capacity is based on the maximum motor power installed for the tailings thickener underflow pumps. This limit is similar to that derived from recent CFD analysis of the tailings thickener feed distributor. The TPUT constraint is based on parameters developed in the grinding testwork, notably SPI (SAG mill Performance Index), MBI (Modified Bond Index) and Ci (Minnovex Crushing Index). These parameters are inputs to the Minnovex formulae that generate the grinding model outputs, specifically TPUT (throughput, or instantaneous grinding rate) and the P₈₀ product sizing to flotation feed.

Metallurgical grades and recovery models developed from testwork used for flotation predictions are based on meeting the optimum primary and regrind ranges shown in Table 17.4, which has been the design objective. An additional ball mill (Ball Mill 5) was required to compensate for the higher SAG mill capacity with underground and Central zone ore feed. The softer ore results in a higher SAG throughput, but requires a higher ball mill to SAG mill power ratio to maintain flotation feed P_{80} . The Ball Mill 5 circuit is identical to the existing four ball mill lines and will be operated in parallel with them.

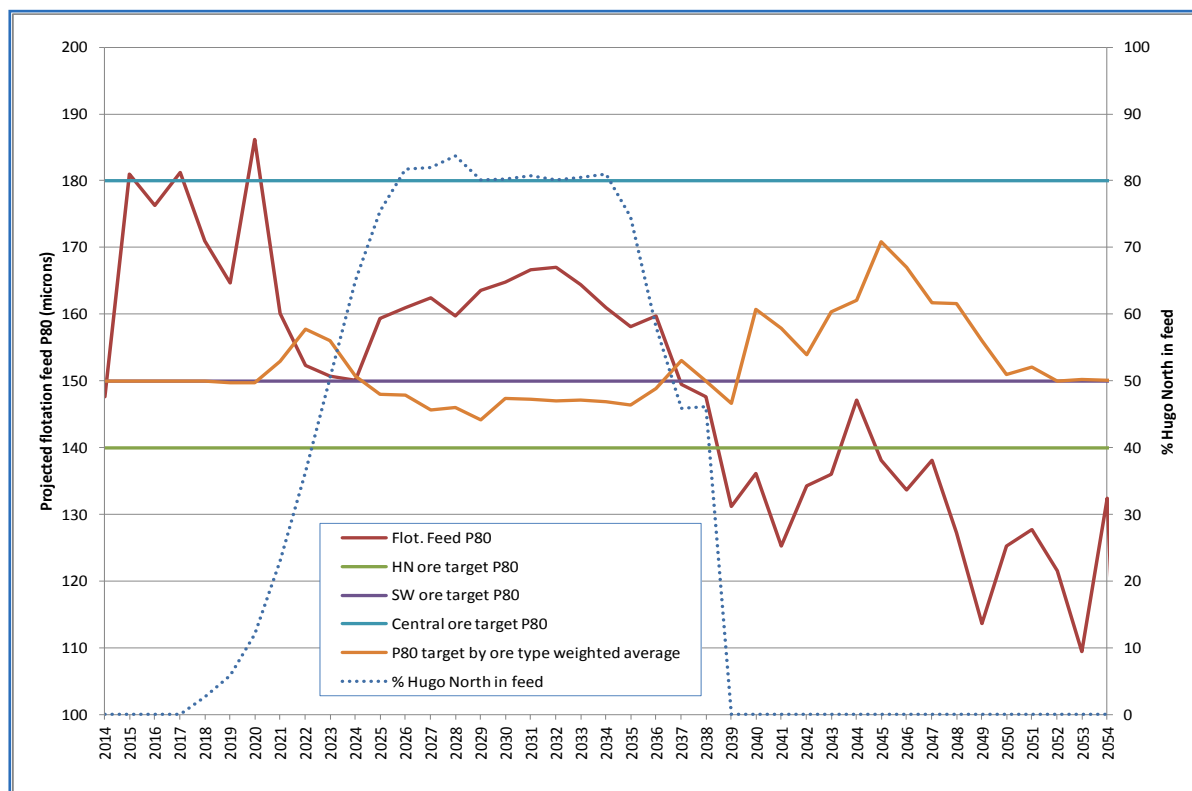
Table 17.4 Primary Grind and Regrind Target Size Ranges

Orebody	Primary Grind P_{80} , μm	Regrind P_{80} , μm
Hugo North	125–160	40–45
Southwest Zone	150–180	30–40
Central Zone Chalcopyrite	180–200	30–40
Central Zone Chalcocite	180–200	30–40
Central Zone Covellite	180–200	30–40

The prior 2015 mine plan (MP2015) is compared with the MP08v2 peak design mine plan and concentrator design envelopes in Figure 17.3. Whenever the plant is volumetrically limited, there is excess power for grinding finer than shown: the P_{80} values have not been recalculated for the reduction of the volumetric limit in OTFS16.

Concentrator design envelopes for OTFS16 are compared with the MP08v2 design mine plan for mill feed tonnage and concentrate production in Figure 17.4 and Figure 17.5, respectively.

Figure 17.3 MP2015 Flotation Feed P₈₀ with Ball Mill 5



Note: Uncorrected for reduction in volumetric limit applied in OTFS16

Figure 17.4 Comparison of MP08v2 and OTFS16 – Ore Delivery

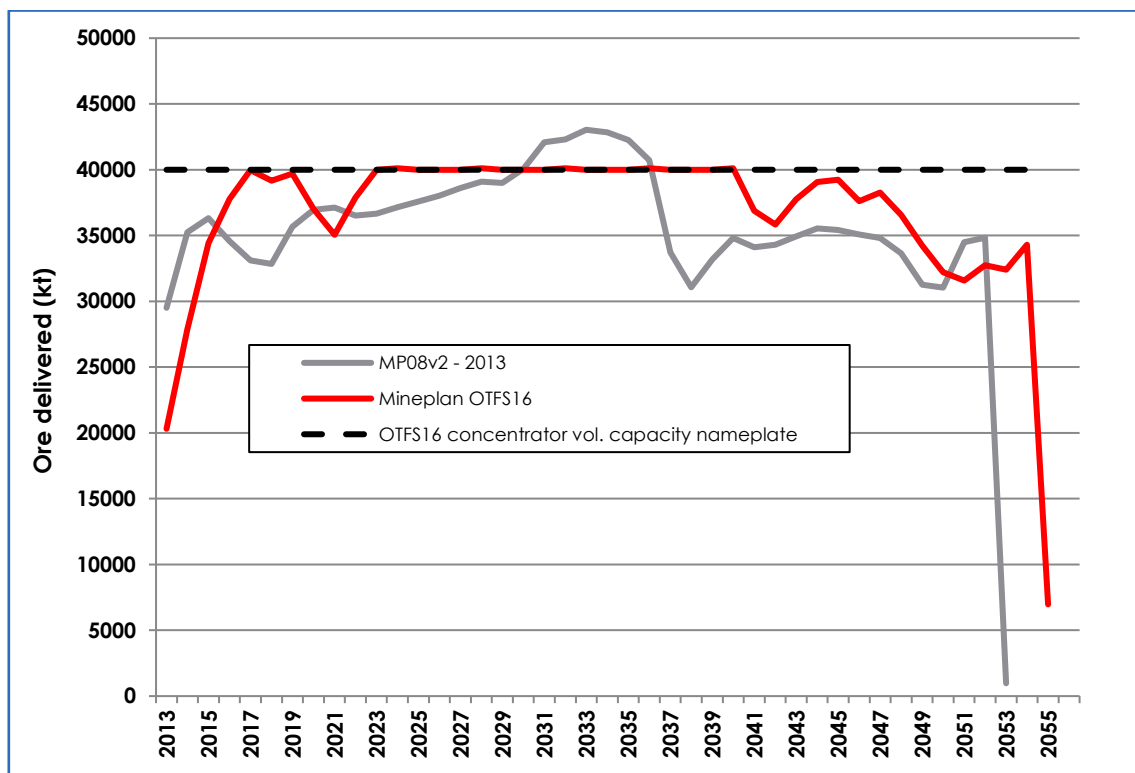
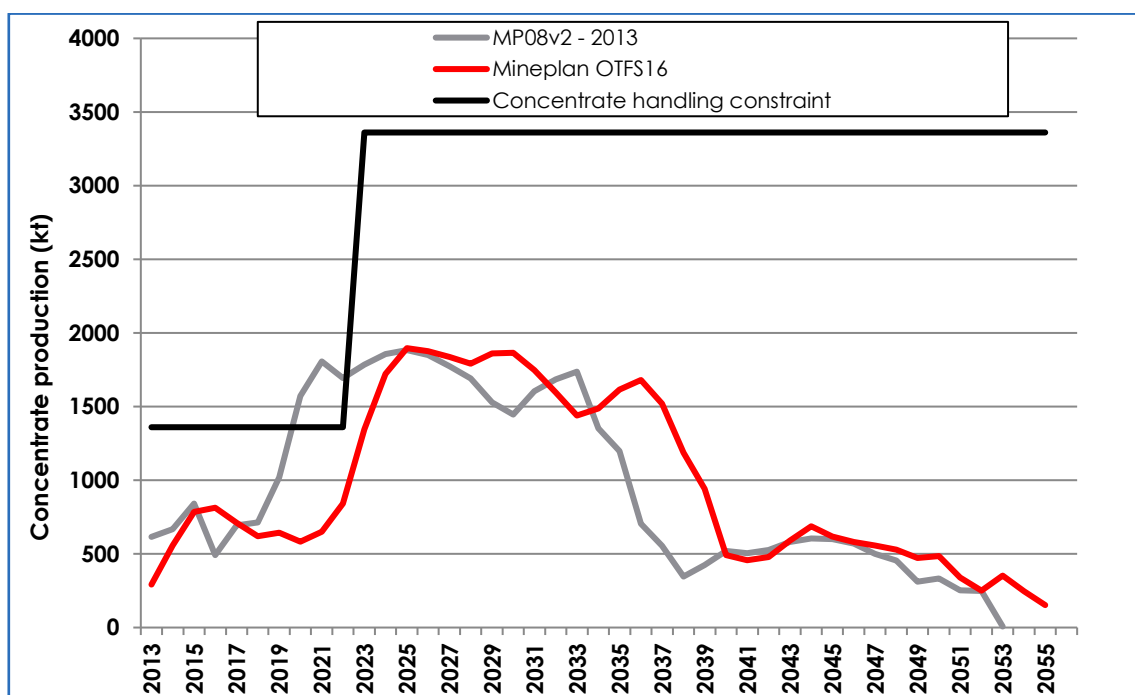


Figure 17.5 Comparison of MP08v2 and OTFS16 – Concentrate Production



In addition, a contingency plan exists for configuration of the fifth mill in series rather than in parallel, should additional SAG mill capacity be developed by upstream debottlenecking. Flotation feed top size is controlled more effectively by series operation than by parallel operation, especially when the new feed to the circuit is pre-classified at a size coarse enough to allow additional grinding effort to be directed only to the coarsest 20 wt% of particles with the lowest recovery. The net effect is a steeper flotation feed particle size distribution.

The regrind circuit has ample capacity to maintain the target grind for efficient concentrate cleaning at 110 kt/d capacity.

The Phase 1 design criteria specify a 30% pebble generation for each ore type. This is believed to be conservative with operating data indicating a lower median pebble generation rate. The softer underground and Central zone ores are expected to generate a lower percentage of pebbles than Southwest, and no pebble crushing circuit upgrades are planned for the concentrator conversion.

Flotation cell criteria are compared in Table 17.5 at the peak head grade condition. Retention times are specified per cell, with eight cells in line in roughing, four cells in cleaning, and four cells in cleaner scavenging.

Table 17.5 Flotation Cell Design Criteria

Parameter	Rougher	Cleaner	Scavenger	Column
Cu concentrate grade (%)	15	30.8	15	42.9
Stage Cu recovery (%)	96	87	87	60
Carrying capacity, maximum (t/h/m ²)	1.5	2	2	1.75
Retention time (minutes/cell)	2.5	2.5	2.5	–

Flotation circuit design is constrained by layout, available area, cell froth-carrying capacity limits, and minimum residence time requirements. Because of the change to much higher grade Hugo North ore, eight additional 160 m³ rougher bank cells have been selected for installation in the flotation area reserved for expansion in the Phase 1 design. An additional rougher bank was selected over a cleaner and cleaner scavenger bank because the rougher circuit was approaching its carrying capacity limits, as well as the rougher stage recovery being lower than the cleaner stage recovery in operation on Southwest ore. Residence time considerations are limited to maintaining minimum per cell residence times of 2.5 minutes to minimize short-circuiting potential. The rougher and cleaner cell retention scale-up factors relative to the latest SGS bench testwork are 1.65 and 2.5, respectively.

The design cleaning circuit recovery, at 3% copper in the feed, is 96%. With rougher losses, this equates to 93% overall copper recovery at peak head grade. By virtue of the head grade-recovery relationship, recoveries are projected to fall to 91% at average Hugo North heads of 1.66% Cu and to 87% at the 2016 Reserves Case average head grade of 0.84%. For comparison purposes, Southwest zone ore currently demonstrates 90%+ recoveries at head grades in this range, but until 2016, has been unburdened by the higher pyrite content in a 20% admixture of high-pyrite Central zone ore.

Additional column cells have been included to process the expected increased volume of concentrate. The column cell dimensions are identical to Phase 1 (5.5 m diameter x 16 m high). The concentrator conversion will require six new columns.

The concentrate thickener sizing parameters were based on testwork performed by FLSmidth; it was determined that one additional 23 m diameter thickener, identical to the two supplied in Phase 1, will be required. Although the Phase 2 concentrate volumes are almost triple the concentrate volumes in Phase 1, the Phase 1 concentrate thickeners were so significantly over-sized that the addition of a third similar thickener will suffice for the conversion. The intent is to run two units and have one on standby, compared to current operation with one unit operating and one on standby.

The concentrate thickening and storage design criteria (Table 17.6) assume the following:

- Use of Magnafloc 5250 or equivalent anionic flocculant such as Magnafloc 338, which is currently being used for concentrate thickening
- Specific settling rate of 0.055 m²/t/d at flocculant dosage of 4–6 g/t and 0.04 m²/t/d at flocculant dosage of 15–20 g/t
- Feed auto-dilution to 10%–15% solids.

Table 17.6 Concentrate Thickening and Storage Design Criteria

Design Parameter	Unit	Phase 1	Phase 2
Concentrate Flow	t/h	118.7	240
Concentrate Flow	t/d	2,849	5,291
Unit Thickening Rate	m ² /t/d	0.055	0.055
Area Required	m ²	157	291
Thickeners Provided	ea	2	3
Diameter	m	23	23
Area Provided	m ²	830	1,245
Concentrate Tanks	ea	1	2
Concentrate Tank Volume	m ³	4,200	8,400
Storage Capacity	h	42.8	26.6
Desired Capacity	h	>28	>24

The Phase 1 concentrate filtration sizing parameters were based on Southwest zone concentrate testwork performed by Larox at a 25 µm P₈₀. Existing operations data are used for Phase 2 design. The filtration rate for Southwest at 35 µm is taken as the midpoint between the peak and median three-day moving average filtration rates from May to December, 2013, scaled-up to the expected Phase 2 P₈₀ of 40 µm. They result in the addition of two more 144 m² horizontal plate filters, identical to the two supplied in Phase 1.

The design criteria include:

- Concentrate storage requirement minimum of 28 hours
- Filtration rate of 0.75 t/h/m²
- Cake moisture of 8.5%
- Cake bulk density of 1.99 t dry solids/m²
- Filter cycle time of 10.5 minutes

Concentrate bagging design is based on existing operations data. Operations reports that the maximum bagging rate, without circuit upgrades, is 3,934 t/d (wet) based on three of four bagging modules operating simultaneously. For Phase 2 the following design upgrades are required to increase the bagging plant capacity:

- Automation of the sampling, sealing, and scanning stages to allow the use of four modules simultaneously, providing a 33% increase in capacity.
- Installation of four additional modules. It is conservatively estimated that shared use of the existing roller conveyors will result in the four new modules providing a capacity increase of 50% instead of 100% because of queuing interference.
- Increase of bag capacity from 2.0–2.5 t. This would reduce the number of bags being filled by 20% and should result in an almost 25% increase in capacity (note that actual bag filling time is not capacity limiting). Rather than increase bag height, it is recommended that the vibration cycle be extended and the cross-section of the base of the bag be increased to improve stability.

With these upgrades the bagging plant will have a product design bagging rate of 411 t/h (wet) (378 t/h (dry)). At a planned usage rate of 7,100 h/a, the bagging capacity would be 2.7 Mt/a compared to a peak planned production of 1.9 Mt/a. This may obviate the need to automate, although this may be desirable from the perspective of operating cost reduction.

No additional tailings thickening capacity is planned for Phase 2 as the conversion is based upon the existing tailings capacity.

The design of the tailings storage facility (TSF) is as originally presented and externally audited in DIDOP. Variances in early operation from the design criteria have been noted and the site water balance has been updated to reflect a long-term reduction in water efficiency from the DIDOP balance.

Reagent consumption is based on laboratory flotation testwork.

17.4.1 Design Factors

A design factor is an additional safety allowance applied to nominal capacities to account for short-term surges and variability in plant operation. Design factors are not applied uniformly across the plant but instead target equipment that typically must operate in a surge mode, such as feeders, conveyors, and pumps. A 10% allowance (1.1 design factor) has been adopted for the new equipment of this type.

For process equipment such as grinding mills and flotation cells, which are sized from design criteria, a design factor of 1.0 is applied. Process equipment can generally tolerate normal process variations with minor changes in performance.

The design factors adopted for new equipment are shown in Table 17.7. Some existing equipment will fall below its design factors as a result of the expansion.

Table 17.7 Design Factors for Concentrator Conversion Equipment

Item	Factor	Basis
General		
Anticipated Capacity Increase (C)	1.21	100 / 121 kt/d
Normal Process Variation (N)	1.10	Estimated
Process Equipment	1.00	Sizing based upon design criteria
General Pumps and Piping	1.10	Normal process variation
Thickener U/F and O/F Piping	1.50	Two of three thickeners operating ($3/2 = 1.5$)
Material handling (concentrate)	1.10	Normal process variation
Special		
Gravity Flow Launderers Fill Fraction	0.5	Industry practice
Reagent Supply (Flow)	2.0	Industry practice
Gland and Seal Water Supply	1.5	Industry practice
Froth Factor – Rougher Concentrate	1.5	Industry practice
Froth Factor – Cleaner Concentrate	3.0	Industry practice
Froth Factor – Column Concentrate	4.0	Industry practice

17.4.2 Equipment Supply

The selection of process equipment suppliers for the concentrator was based on sole-sourcing from existing Phase 1 suppliers. Along with the benefits of operator familiarity and commonality of spares, sole-sourcing was particularly appropriate in the following circumstances:

- To the same manufacturer as used in Phase 1 when long-term, high-level technical support will be necessary for the life of the operation.
- To the same manufacturer and equipment model / size as used in Phase 1 when larger equipment has not been proven up and when Phase 1 design has made provision for certain models and model dimensions, e.g., Phase 1 ball mill aisle, Phase 1 flotation cells, and Phase 1 pressure filter suppliers.
- To suppliers with which Rio Tinto Procurement has negotiated strategic agreements, e.g., supplier of general electrical equipment and variable-speed drives.

17.5 Process Description

This section describes the flowsheet and general arrangement of the expanded processing facilities, starting at reception of ore from the overland conveying system and continuing through the concentrator to concentrate load-out and the distribution of tailings at the storage facility.

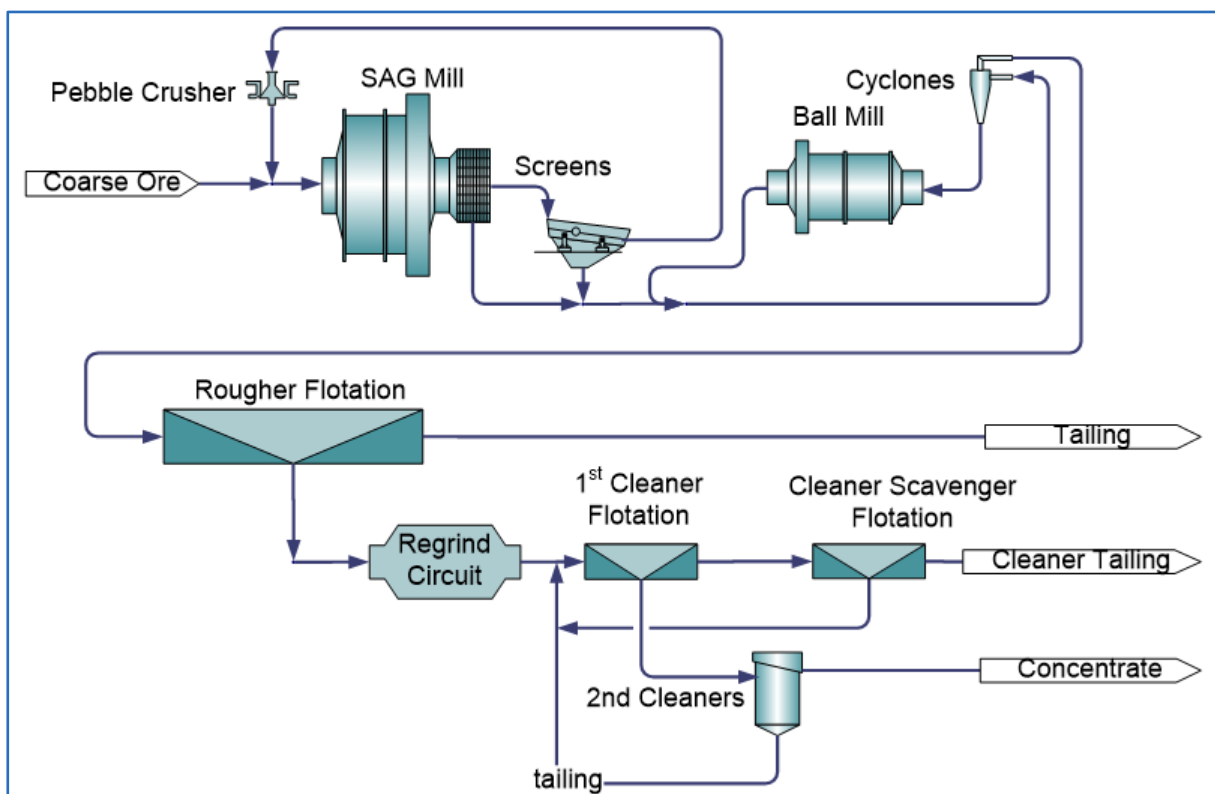
The description includes the modifications to be made to process Lines 1 and 2 to accept higher milling rates and head grades following the first three years after ore delivery from Hugo North Lift 1.

17.5.1 Overview

Oyu Tolgoi employs a conventional SAG mill / ball mill / grinding circuit (SABC) followed by flotation, as shown in the basic flowsheet (Figure 17.6).

Coarse ore is slurried and ground to approximately 2 mm in semi-autogenous grinding (SAG) mills. Screening of the discharge separates out +15 mm particles, which are transferred to pebble crushing for size reduction and then returned to the SAG mills. About 10%–15% of the feed circulates from the SAG mills to the pebble crushers, depending on ore type and grate condition. SAG mill screen undersize is ground further in ball mills operating in closed circuit with cyclones.

Figure 17.6 Basic Oyu Tolgoi Flowsheet – Phase 1



The cyclone underflow returns to the ball mills, while the overflow with an 80% passing size of 140–180 µm is distributed by gravity to the rougher flotation cells. The rougher concentrate is then reground in vertical tower mills to 35 µm before delivery to the first stage cleaners. The concentrate from the first stage cleaners is pumped to the column cells, which produce the final grade concentrate.

Tailings from the cleaner-scavenger and rougher flotation cells are combined, thickened, and pumped to the tailings storage facility (TSF), where they settle to their terminal density, allowing the recycling of process water to the concentrator. The column cleaner concentrate is thickened, filtered, bagged, and shipped to market.

Phase 1 uses two grinding lines, each consisting of a SAG mill, two parallel ball mills, and associated downstream equipment to treat up to 121 kt/d of ore from the Southwest and Central pits, which represents the tailings handling capacity of the plant. The Phase 2 concentrator development programme optimizes the concentrator circuit to enable it to maximize recovery from the higher grade Hugo North ore.

In 2017, Cell 2 of the TSF will become available for use, and the tailings pumping system will be upgraded to feed TSF Cells 1 and 2.

In 2021, before the underground mine reaches full capacity, Lines 1 and 2 will be expanded to allow them to handle both higher tonnage and higher grade material. The ball milling, rougher flotation, flotation columns, concentrate filtration, thickening, bagging and bagged storage facilities will be upgraded to accommodate the gradual introduction of ore from underground. In general, the augmentations embody the operational and maintenance philosophies guiding the design of Phase 1 and provide, as much as practicable, a seamless production environment.

The intent of Phase 2 is to treat all of the high-value Hugo North ore delivered by the mine, supplemented by open pit ore to fill the mill to its capacity limit. The open pit ores have different optimal processing conditions than does the Hugo North ore, and the concentrator operation will target capturing maximum value from the higher NSR of the underground ore. These conditions approximate those for Southwest zone ore but will not be optimal for Central zone ore, and the concentrate grade and recovery from the Central zone ore has been corrected accordingly. The high-grade of Hugo North ore will generate high tonnages of concentrates, which will beneficially dilute impurities, particularly arsenic from the Central zone ore.

Figure 17.7 is a block diagram of the process on completion of Phase 2. As shown in Figure 17.8, most of the new processing facilities will be located within the existing concentrator building. Installation of a new ball mill, column cells, and concentrate thickening will require the concentrator building to be expanded to the north.

Figure 17.7 Overall Block Diagram on Completion of Phase 2

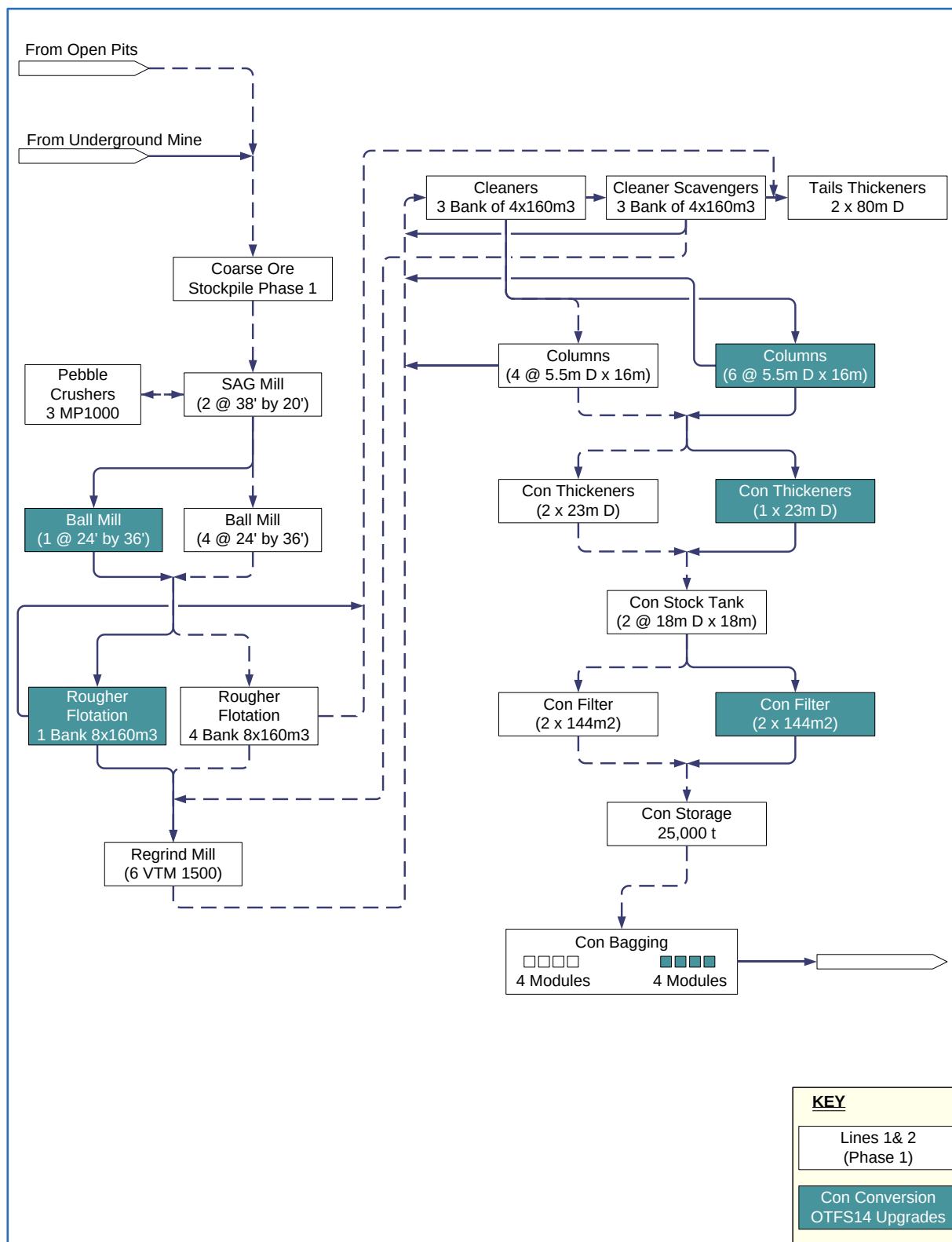
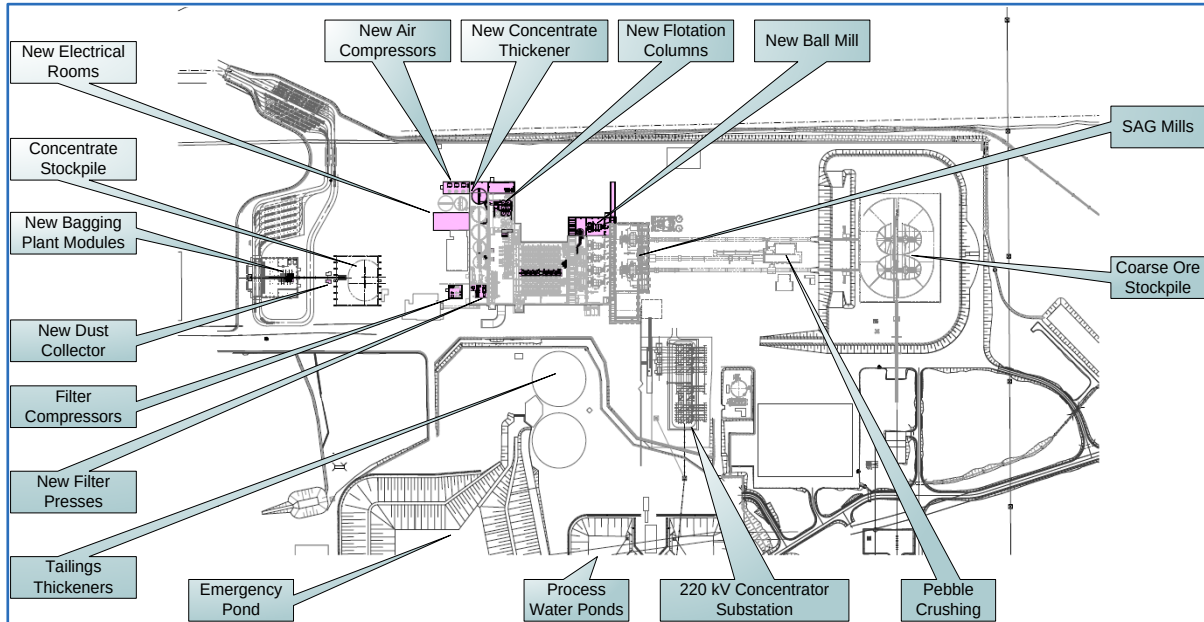


Figure 17.8 Location of New Facilities Relative to Phase 1 Installation



The existing concentrator substation to the south will be expanded to supply the additional electrical loads. The Phase 1 bagging plant will be expanded by the addition of four more bagging modules. This expansion was anticipated in the Phase 1 design, and ample room was provided for the new equipment.

The primary crushing and overland conveying systems that deliver crushed ore to the coarse ore stockpile do not need to be modified for Phase 2. The underground provides for the delivery of ore to the existing coarse ore storage gantry via an additional parallel conveyor, which was allowed for in the Phase 1 design.

17.5.2 Reagent and Grinding Media Storage and Supply

The conversion will share facilities with the Lines 1–2 reagent supply systems. The modifications to the reagent system are described below. The recommended inventories and method of delivery for the reagents are summarized in Table 17.8. In general, the aim is to have 45 days of reagent inventory on hand at or near the plant site.

- Lime – No additional lime storage capacity, beyond the four 1 kt silos installed in Phase 1 is required. An additional metering station will be required at the new rougher bank and the column cells.
- Primary Collector – The primary collector will be Aerophine 3418A (sodium di-isobutyl dithiophosphinate). Consumption will peak at nearly 1,700 kg/d, approximately 65% more than the Phase 1 usage. The Phase 1 system has ample dilution capacity to supply the conversion. An additional metering station will be required at the new rougher bank.

- Secondary Collector – The proposed on-site inventory for Phase 1 is 40 tonnes, which has not been increased for the conversion. An additional metering station will be required at the new rougher bank. No secondary collectors are currently added in Phase 1.
- Frother – Frother distribution in Phase 1 provides for the use of two frothers, methyl isobutyl carbinol added neat, and a secondary frother (polyglycol ether or similar) added as a low concentration solution in water. Primary frother consumption in Phase 2 will be roughly equal to the Phase 1 design. No additional frother tankage will be required.
- Tailings Flocculant – The major flocculant will be a non-ionic type such as Magnafloc 5250. Tailings flocculant use will increase to 2,400 kg/d, proportionate to tonnage. No new flocculant preparation equipment will be installed. The proposed reagent inventory is considered adequate for Phase 2. Recent testing of an alternate flocculant has led to higher underflow densities at significantly reduced consumption.
- Concentrate Flocculant – The flocculant used for concentrate thickening is an anionic variety, such as Magnafloc 338. Concentrate flocculant demand will increase to 110 kg/d, but the Phase 1 preparation capacity is sufficiently under-utilized that expansion will not be necessary. An additional flocculant metering pump and dilution system will be installed. Reagent inventory will be increased to five bulk bags.
- Water Treatment Chemicals – The existing anti-scalant and corrosion inhibitor supply systems will be adequate for both the process and raw water systems. The reagent inventory is also adequate for Phase 2. Some premature corrosion has been observed in Phase 1 systems exposed to raw and process water, and it is likely that additional reagents and addition points will be considered in the future.
- Grinding Media – No additional inventory is required for SAG milling.

For ball milling, the new Ball Mill 5 will use the existing 1.6 kt ball storage system for 75 mm balls and the ball conveying system will be modified to deliver to it. An additional inventory of 192 tonnes of 75 mm media in quarter-height isotainers is provided.

Using Phase 1 regrind media consumption estimates, the regrind mills will consume about 22 t/d of 16 mm media, reducing on-site inventory to eight days of operation. However, actual operating data indicate a large decrease in consumption, from the design 2013 plan of 130 g/t to 60 g/t for Southwest zone ore for OTFS16. Long-term consumptions in regrind milling are budgeted in terms of g/kWh for the various ore types.

Table 17.8 Recommended Reagent Inventories and Delivery Methods

Consumable	Unit	Design Usage (units/mo)	Design Usage (per day)	Proposed Inventory (d)	Proposed Inventory (t)	Phase 1 In-Plant Storage (t)	Phase 1 Warehouse Inventory (count)	Phase 2 In-Plant Storage (t)	Phase 2 Warehouse Inventory (t)	Total Inventory (d)	Description of Additional Phase 2 Storage	Additional Number Required (count)
125 mm balls ¹	t	1,809	59	30	1,784	1,800	–	–	–	44	–	–
75 mm balls ¹	t	2,171	71	30	2,141	1,440	–	–	192	23	24 t isotainers	8
16 mm balls ¹	t	70.9	22	15	324	165	–	–	–	8	–	–
Lime	t	3,000	100	45	4,500	4,000	–	–	–	50	1,000 t silo	–
3418A	kg	50,647	1,665	45	75	40	40	–	80	72	18 m ³ isotainers	3
Collector-2	kg	161,293	5,302	45	239	40	40	–	–	11	–	–
Frother-1	kg	54,265	1,784	45	80	41	12	–	40	45	FCL (18 m ³ isotainers)	2
Tails Flocculant	kg	72,353	2,378	45	107	3	140	–	105	45	750 kg bag	140
Conc Flocculant	kg	3,169	104	45	5	0.3	1	–	4	41	750 kg bag	6
Anti-scalant	m ³	9	0.28	45	17	–	19	–	–	67	–	–
Anti-corrosive	m ³	9	0.28	45	17	–	14	–	–	49	–	–

¹ 125 mm, 75 mm, and 16 mm media based upon 90% bin fill.

17.5.3 Raw Water Supply

Raw water is used for:

- Cooling
- Gland seal
- Domestic and fire water
- Column and filter wash water, and
- Total water inventory make-up to replace water lost to evaporation and to the settled tailings.

Raw water is delivered by pipeline from the lagoon to the raw water tank, from where it is pumped through cartridge filters to the grinding and air compressor cooling systems. Spent cooling water will supply a second gland seal water tank interconnected with the Phase 1 gland seal water tank. Excess spent cooling water will flow by gravity to the tailings collection box and make its way to the process water tank via the tailings thickener overflow; any shortfall in gland seal water requirement will be made up directly from the cooling water supply.

The concentrator conversion equipment will be serviced by the existing water system with minimal modification. The gland seal water storage capacity will be expanded and appropriate connections added to the existing network.

17.5.4 Process Water

The bulk of the process water is added to the SAG mill feed chutes and the cyclone feed pump boxes in high volumes at low pressure. The ball mills are secondary addition points. The rest of the process water is circulated around the mill at higher pressure for sprays, utility hoses, and other miscellaneous uses. A booster pump is provided for high-pressure washing of the mill liners. The increased tonnage in Phase 2 will require additional process water but no system modifications.

17.5.5 Concentrator Water Balance

The water demand estimate for Oyu Tolgoi assumes a peak design processing rate of 110 kt/d and 64% tailings density. Although the need for mine dewatering at a rate of up to 90 L/s is predicted, this value is uncertain and may not be realized. OTFS16 has made a conservative assumption that there will be no water provided from mine dewatering into the water supply system.

The total site design water demand ranges from 588–785 L/s, with an average of 696 L/s, to support the nominal production rate of 110 kt/d. The peak water demand (excluding recycling) in OTFS16 is:

• Concentrator (including TSF)	670 L/s
• Other (Camp, dust suppression, etc.)	75 L/s
• Underground mine	30 L/s
• Total	775 L/s

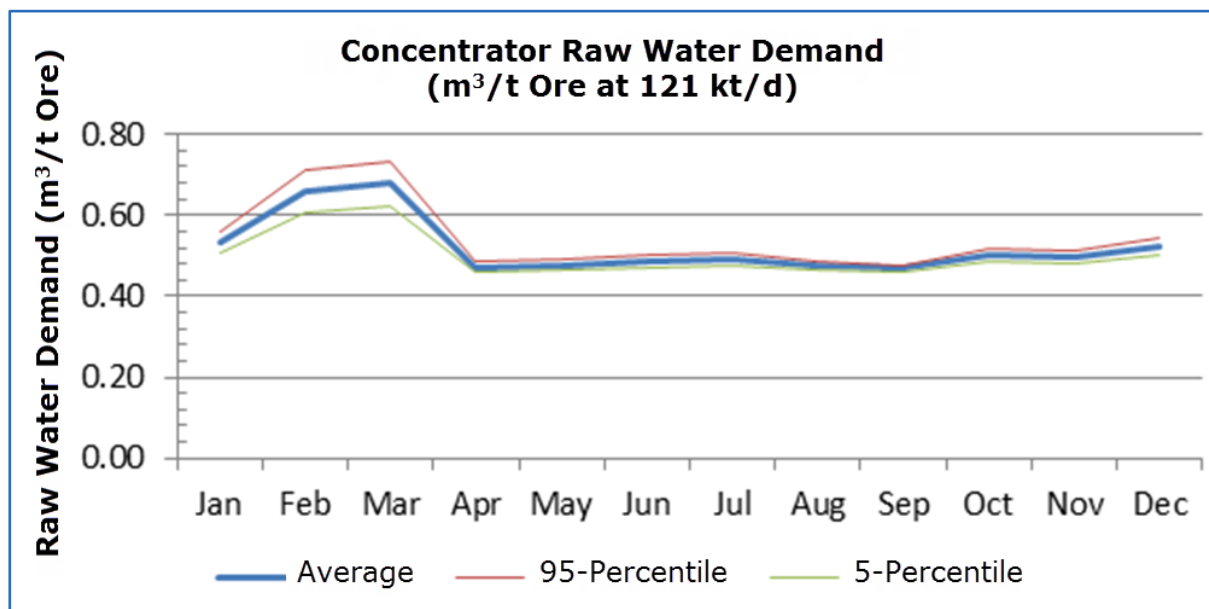
Water consumption performance to date has been better than predicted, with continuous improvements evident over the first three years of operation. Average water consumption in 2015 was 469 L/s, representing a usage rate of 433 L/t (ore). Water consumption is expected to increase to approximately 550 L/s as a result of underground mine development (up to 30 L/s) and ongoing improvements in concentrator production capacity (up to 20%). This rate remains well below the permitted usage rate of 918 L/s and the capacity of the raw water supply system.

The design groundwater pumping capacity is 900 L/s. Utilising drawdown of the lagoons will slightly reduce the lagoon recharge rate, but the current projection is that the peak instantaneous raw water demand could exceed 900 L/s at the Phase 2 volumetric limit of 121 kt/d, and approach it at the average of 117 kt/d in the peak Phase 2 year. This compares with the currently permitted extraction rate of 918 L/s. The largest water loss, 564 L/s, is the entrained water in the settled tailings. The Phase 1 design specified a final tailings settled density of 73.5%. That value has not been realized to date and a value of 70% has been used in the water balance model.

Figure 17.9 illustrates the projected raw water demand for OTFS16 under different weather assumptions. Based on the estimated raw water demand and the available 400,000 m³ surge capacity provided by the two raw water lagoons, the nominal raw water pumping rate of 900 L/s appears adequate. Up-rating of the raw water pipeline capacity by installation of an intermediate pump station would be required for capacity beyond the current mine plan. Normally the cooling water demand is lower than the raw water demand. However, the significant increase in reclaim rate and subsequent reduction in raw water demand during the spring thaw means that cooling water must be recirculated to balance the demands.

Studies continue per defined ongoing monitoring and socio-economic programmes developed by OT LLC, specifically with regard to water resource management. 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 if any on 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.

Figure 17.9 Seasonal Raw Water Demand



17.5.6 Concentrator Power

With the addition of the concentrator conversion loads, the peak operating load demand from the existing 220 kV concentrator substation will increase by an estimated 20 MW (from 116–136 MW), and the nominal operating (diversified) load will increase by an estimated 19 MW (from 106–125 MW). The operating power demand includes the diversity, demand, and percent duty factors specific to the type of equipment and process.

Total demand for Phase 1 and the concentrator conversion combined during normal operating conditions is estimated at 150 MW peak operating load and 144 MW nominal operating (diversified) load. This includes the peripheral 35 kV ring loads to the concentrator account. This nominal operating load results in an estimated annual power consumption of 1,094 GWh for the combined concentrator, an incremental increase of 161 GWh for the concentrator conversion.

The existing concentrator 35 kV line will distribute power through cable feeders to the following:

- One 16 MVA, 35 kV–10.5 kV Ball Mill 5 oil-filled transformer, and
- One 16 MVA, 35 kV–6.3 kV oil-filled transformer from a new 35 kV GIS switchgear section to be added.

The modifications will provide power for all of the new conversion equipment, in addition to the power demands of the relocated air compressors and the new column cells.

17.5.7 Future Process Work

Additional work is required in the following areas to advance and optimize the design of the process plant:

- Debottlenecking studies to enhance Phase 1 capacity beyond initial design at minimum cost
- Gravity or flotation recovery of gold from the regrind circuit
- Ongoing metallurgical testing programme for Central zone ores
- Smelter evaluations
- Heap leach studies
- Ongoing metallurgical testing programme for Hugo North infill holes
- Coarse flotation and split flotation testwork to determine whether the performance gain from the fifth ball mill can be achieved at lower cost
- Evaluation of series versus parallel configuration for the 5th ball mill
- Evaluation of separating out concentrate from the first rougher cell and sending it directly to the column cells, rather than passing it through the rest of the rougher cell bank
- Concentrator conversion feasibility study update
- Phase 2 detailed engineering
- Prefeasibility programme for additional resources (Heruga, Hugo South).

18 INFRASTRUCTURE

18.1 Introduction

Most of the infrastructure facilities required for the Oyu Tolgoi project were completed during Phase 1. Certain infrastructure buildings and services will be expanded or added during Phase 2. The facilities are summarized in Table 18.1.

Table 18.1 Summary of Infrastructure Facilities by Phase

Facility	Initial	Expansion
Power Supply	From China	No change
220 kV Substations	Central substation	No change
	Concentrator substation	Additional 35 kV switchgear for concentrator conversion.
	Shaft farm substation	Shaft farm substation to be modified
Power distribution	220 / 35 / 10.5 kV	Upgrade existing 35 kV and 10.5 kV systems
Standby-power	2 x 20 MW diesel power stations	No change
Access Roads	Internal access roads	Expanded internal access
	OT Road	
	Concentrate logistics facilities	Expanded logistics facilities
	North gatehouse	–
Airport	Regional airport	No change
Camp Accommodation	Oyut camp (1,952 beds)	Oyut camp expansion 18 Accommodation blocks providing 1,836 rooms with adjoining ensuites. Maximum additional beds 5,508 beds.
	Manlai camp	No additional construction camp facilities required
	Erchim camp	Overhaul to add 900 beds for peak requirements.
	Javkhlant camp	–
	Khanbogd Camp	–
Maintenance Facilities	Gobi Maintenance Complex	UG facilities
	Open Pit Truck Shop	–
	Toyota Workshop	–
Administration Building	410-person office	–
Central Mine Dry	1,200 lockers	–
Water Systems	Undai River diversion	No change
Water Distribution	Raw water	Expansion of existing systems
	High pressure firewater	Expansion of existing systems

Facility	Initial	Expansion
	Domestic water	Expansion of existing systems
	Sewer	Expansion of existing systems
	Return water	No change
Raw Water Supply from Borefield	900 L/s	No change
Water Treatment	Water treatment & bottling plant	No change
Wastewater Treatment	4,200 equivalent person	No change
ICT	Distributed control system	Expansion of existing systems
	Electrical monitoring system	Expansion of existing systems
	Local area network / voice-over internet protocol (VoIP)	Expansion of existing systems
	Fire alarm system	Expansion of existing systems
	Access control system	Expansion of existing systems
	Closed-circuit television	Expansion of existing systems
	Cable television	Expansion of existing systems
Operations Warehouse	8,100 m ² + 4,500 m ²	6,600 m ²
Medical Centre	Medical centre	No change
Fire Station	Built during early construction	No change
Central Heating	2 x 7 + 2 x 29 MW central heating plant	2 x 29 MW expansion
Waste Disposal	One landfill cell	Two landfill cells
	Two leachate cells	One leachate cell
	Incinerator	No change
Fuel Storage	–	–
Light Vehicle Fuel Depot	100 kL gasoline	No change
Light Vehicle Fuel Depot	800 kL diesel	No change
Mine Fleet Fuel Depot	2,800 kL diesel	No change
Diesel Power Station	200 kL diesel	No change
Underground Diesel Storage	–	In underground design
Core Management	Core Management Centre	No change
Construction Water	Supplied by 19 site bores	By raw water system
Batch Plants	Batch Plant No. 1 – 60 m ³ /hr	No change
	Batch Plant No. 2 – 240 m ³ /hr	–
	Batch Plant No. 3 – 40 m ³ /hr	–

18.2 Phase 2 Project Execution

OTLLC and Rio Tinto projects has developed a project execution plan that outlines the methods and project management elements to be used to manage the execution of the Oyu Tolgoi Expansion Project for Phase 2. The project execution plan establishes the baseline execution philosophy, with details on the management plans of each of the functions to complete and hand over the permanent facilities associated with:

- the Hugo North Lift 1 underground mine, including shafts, mine development, and mine infrastructure to enable mine production of 95 kt/d of underground ore
- extensions of and additions to the existing surface infrastructure required to support the expanded site operations and comply with the Investment Agreement between Oyu Tolgoi and the Mongolian Government
- conversion of the existing concentrator and bagging facilities to process the higher grade underground ore.

The PEP sets out the organization structure, division of responsibilities, risk management plan, work process, and systems necessary for the management of the project. Also, it lays out how the construction and operational readiness activities will be achieved, with an emphasis on interfaces within and between OT LLC, and the Rio Tinto Groups.

Applicable to all strategies and implementation plans are the broader goals of safety, transparency, Mongolian content, training and development, and other shared values as detailed in The Way We Work, Oyu Tolgoi's Code of Business Conduct.

The development of project management plans and project procedures is underway to provide specific direction on how to implement the requirements identified in the project execution plan. As the Oyu Tolgoi Mine is a fully functioning operation, it has a full suite of operating policies and procedures forming the basis for many of the areas that would generally be covered by an execution plan.

The project execution plan is intended to be a living document and will be continually updated to reflect specific events and timings together with evolving requirements as the project progresses.

The high-level project scope for Phase 2 is summarized as follows:

- underground lateral development
- underground mass excavation
- vent raises and orepasses
- Shaft 2, 3, 4, and 5 sinking and equipping
- Shaft logistics for material and labor, including supply and operation of associated equipment and facilities
- Development of conveyor decline and box cut
- Construction of permanent underground facilities: crushing and materials handling, workshops, services, and infrastructure
- Construction of mine enabling facilities, including Shaft 1 crusher and workshop

- Construction of surface mine buildings and infrastructure
- Upgrading the existing concentrator and ancillary services, including power, services, buildings, and stockpile feed conveying system to efficiently process underground ore.

18.3 Power Demand, Distribution, and Supply

18.3.1 Introduction

The Oyu Tolgoi project is energy-intensive, with requirements of a peak of 145 MW since start-up, increasing to approximately 250 MW in the longer term after completion of the underground development. A reliable and stable power supply is essential for operations and safety.

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. In terms of power, the IA includes an overarching commitment from the GOM and OT LLC to work together to determine the most optimal and reliable solutions for power supply.

Under the IA, OT LLC is required to secure its power requirements from within Mongolia within four years of commencement of production. However, this timeframe is currently suspended pursuant to the Southern Region Power Sector Cooperation Agreement, entered into by OT LLC and the GOM on 14 August 2014, that proposes an independently funded and operated coal fired power plant at Tavan Tolgoi (TTPP Project).

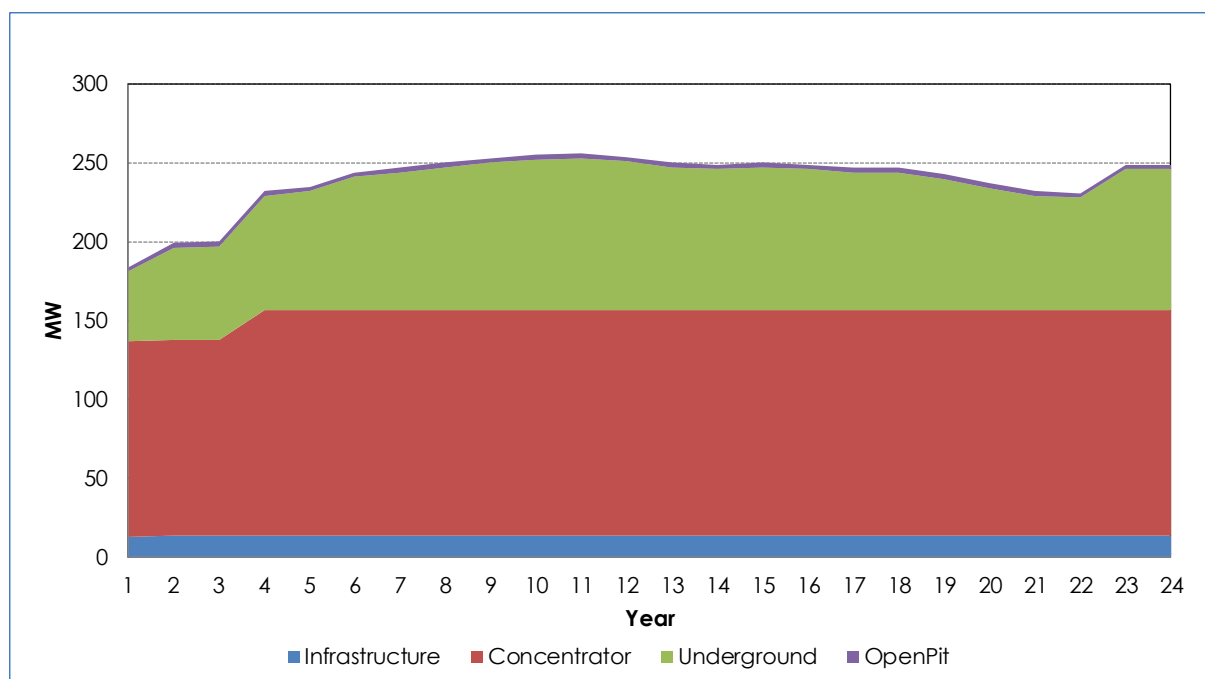
So long as OT LLC continues to participate in the TTPP Project the four-year timeframe for sourcing Mongolian power will be suspended. Upon a withdrawal from the TTPP Project by either OT LLC or the GOM, the four-year timeframe will be reinstated and recommence from the date of withdrawal. A request for proposals from potential investors in the TTPP coal fired power plant led to a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

OT LLC sources its present power under a four-year contract with a Chinese provider, the Inner Mongolia Power International Cooperation Company Ltd. (IMPIC) via the Mongolian National Power Transmission Grid (NPTG) authority. In May 2016 the parties agreed via a non-binding Memorandum of Understanding (MoU), which captured key agreed principles of the new Power Purchase Agreement (PPA), to extend the power supply agreement to at least 2021. OT LLC and the GOM have agreed under the Power Sector Cooperation Agreement that the GOM will assume responsibility for securing the extension of the power import arrangements through its national grid company NPTG. The agreed MoU includes comparable power pricing to the current agreement. OT LLC is endeavoring to execute binding agreements with the GOM and IMPIC within 2016.

18.3.2 Power Demand

Estimated annual peak power demand for the site over next 24 years is shown in Figure 18.1. Year 1 is 2017. This is based on a combination of actual site loads and load factors, and, where applicable, these factors are used for the future loads. These levels will be reviewed progressively as the project develops. The numbers indicate the maximum load estimated for that year.

Figure 18.1 Diversified Peak Demand Growth



18.3.3 Power Distribution

The central substation, approximately 500 m south of the concentrator facility, currently receives power from IMPIC and distributes it to the various facilities at either 220 kV or 35 kV. The substation consists of an outdoor, double bus 220 kV switchyard, two 31.5/41.5/51.5 MVA–220 kV / 35 kV transformers, and an indoor 35 kV substation.

Four 220 kV overhead transmission lines radiate from the central substation: two feed the concentrator substation 500 m to the north, and two feed the shaft farm substation 800 m to the south-east.

Two 220 kV / 35 kV transformers provide power to the indoor 35 kV substation, which uses 35 kV GIS to supply the concentrator ring loads, including primary crushing, conveying, and tailings pumping, some infrastructure loads, and the borefield loads.

The 35 kV infrastructure power distribution system consists of:

- Three 35 kV rings (north, south, and concentrator), 3-phase, 3-wire, single-circuit ring fed overhead power line. The north and south rings are fed from the shaft farm substation, and the concentrator ring is fed from the central substation.
- Two radial 35 kV, 3-phase, 3-wire, single-circuit overhead power lines to supply the borefield, fed from the concentrator substation. This feeder extends approximately 70 km to the north-east of the site.
- 35 kV feed to the underground mine.
- Temporary supply to community loads (Khanbogd and Bayan Ovoo).

The north ring, south ring, and concentrator ring loads are configured as three separate open-ring arrangements, where each ring receives power from the 35 kV overhead distribution lines through two circuit breakers. One circuit breaker will be mounted at each end of the open ring, and the two breakers will be isolated from each other through a normally open sectionalizer switch.

The raw water borefield includes a break tank pump station, five collector tank pump stations, and 28 bore pump stations. Collection tank and break tank pump stations are supplied at 35 kV by a double overhead line from the central substation.

Both medium- and low-level voltages are used for power distribution. Medium voltages are 35 kV, 10.5 kV, 6.3 kV, 3.3 kV, and 1 kV, all 3-phase and 50 Hz. Low voltages are 690 V and 400 V, both 3-phase, and 220 V, single phase at 50 Hz.

Welding outlets are 400 V, 4-pole, 3-wire, 50 Hz, solidly earthed. Lighting and general power supply are rated at 400 / 230 V, 3-phase, 4-wire, 50 Hz, solidly earthed. Lighting and power outlets are 230 V, 1 phase, 50 Hz.

18.3.4 Current Power Supply

Oyu Tolgoi is supplied with electricity from China in accordance with three agreements with Inner Mongolia Power International Cooperation Company Ltd. (IMPIC):

- An Electricity Purchase and Sale Agreement for Oyu Tolgoi between OT LLC, IMPIC, and NPTG (Power Purchase Agreement, or PPA).
- An Operation and Maintenance Agreement between OT LLC and IMPIC (O&M Agreement).
- A Dispatch Agreement between OT LLC and IMPIC (Dispatch Agreement).

Supply is physically obtained from a 220 kV double-circuit transmission line from Inner Mongolia. Either circuit is capable of supplying approximately 400 MW and thus Oyu Tolgoi's load can be met entirely from one circuit. To date the reliability of the electricity supply from IMPIC has been very good, with no full outage of the transmission line recorded.

IMPIC built, owns, operates, and maintains the section of transmission line that operates in Inner Mongolia. As part of the O&M Agreement, OT LLC agreed to pay IMPIC's costs for the operation and maintenance of this section. The capital cost of the line is recovered through the electricity tariff in the IMPIC PPA, but there is a requirement for OT LLC to make a final payment for the undepreciated capital at the end of the term of the IMPIC PPA.

Oyu Tolgoi constructed the section of transmission line that operates in Mongolia. On 19 October 2015, in accordance with the new Power Sector Cooperation Agreement (PSCA, 14 August 2015), the Mongolian section of the IMPIC 220 kV line was formally transferred to GOM. Oyu Tolgoi will continue to be accountable for operation and maintenance of the Mongolian section of the transmission line until an alternative local supply is in place.

The diesel power station at the Oyu Tolgoi site consists of two plants, each with 20 MW capacity, connected to the site 35 kV grid to supply standby and emergency power in the event of loss of power from external sources. The diesel power station provides standby power. Emergency loads are supplied either by the diesel power station via the 35 kV system or by dedicated generators such as those located at the administration building and central substation. No additions will be made to the DPS during the Phase 2 Project.

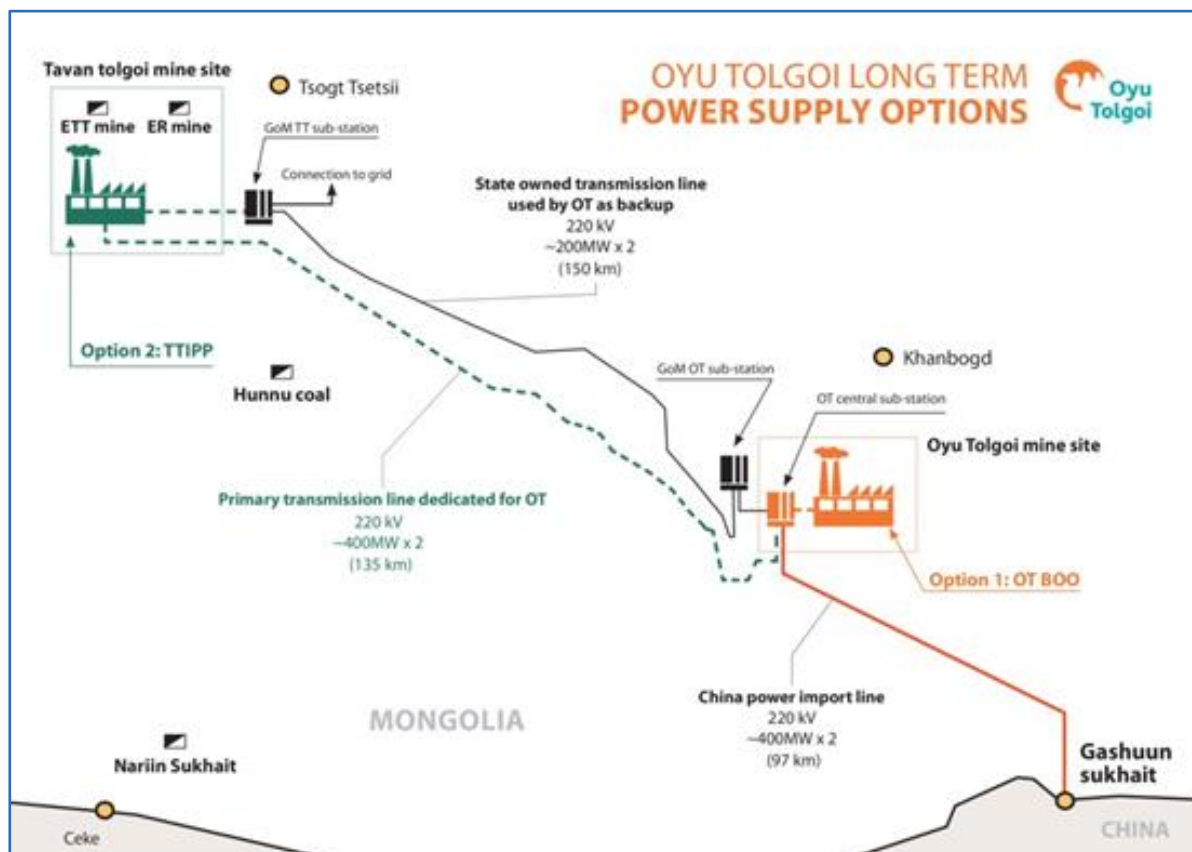
18.3.5 Power Supply Optionality

Phase 2 is based on extension of the existing supply arrangements. In parallel, OT LLC is developing opportunities to support the Oyu Tolgoi power demand. Oyu Tolgoi is currently focusing on maintaining two long-term power supply options: Tavan Tolgoi (TT) with an IPP, and Oyu Tolgoi Build Own Operate (OT BOO). Figure 18.2 shows the locations and supply routes for the available power supply options.

OT LLC has completed a feasibility study for a power plant at site with an initial installed capacity of 450/382.5 MW (gross/net), sufficient to meet the near-term net peak load demand of the Oyu Tolgoi project. OT LLC has obtained and maintains a power plant construction license from the Energy Regulatory Commission of Mongolia. The power plant ESIA has been completed and is ready for circulation to potential project financiers. Study work on the OT BOO option is ongoing to maintain both the currency of the option and the ready-to-execute status.

A request for proposals from potential investors in the TTPP coal fired power plant led to a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

Figure 18.2 Power Supply Options



18.4 Site Access

18.4.1 Airport

A permanent domestic airport has been constructed at Oyu Tolgoi to support the transportation of people and goods to the site from Ulaanbaatar. It further serves as the regional airport for Khanbogd soum. The facility has sufficient capacity to fully support the Phase 2 Project. The airport is 11 km north of the Oyu Tolgoi camp area. It is a non-precision approach, visual flight rules (VFR) facility. The surface is concrete, with a concrete apron at the terminal building. The runway has been aligned to the prevailing north-west-south-east wind direction to minimize cross-wind conditions and facilitate optimal landing and takeoff conditions. The design criteria are set to service commercial aircraft up to the Boeing 737-800 series aircraft.

18.4.2 Access Roads

18.4.2.1 Site Access Roads

All internal access roads are constructed of graded gravel. The road base is built on scarified/compacted existing ground where suitable; where the ground surface is unsuitable for use as a wear course, it has been replaced with well-graded gravel and sandy fines. The top elevation of the shoulders of the gravel pavement surface is approximately the same level as the surrounding surface except at pipeline crossings. Side-drain ditches are provided parallel to the road for stormwater drainage. During the expansion additional access roads will be constructed for the expanded operations camp, the shotcrete and concrete storage facility for the underground, the conveyor-to-surface service portal and Shaft 4.

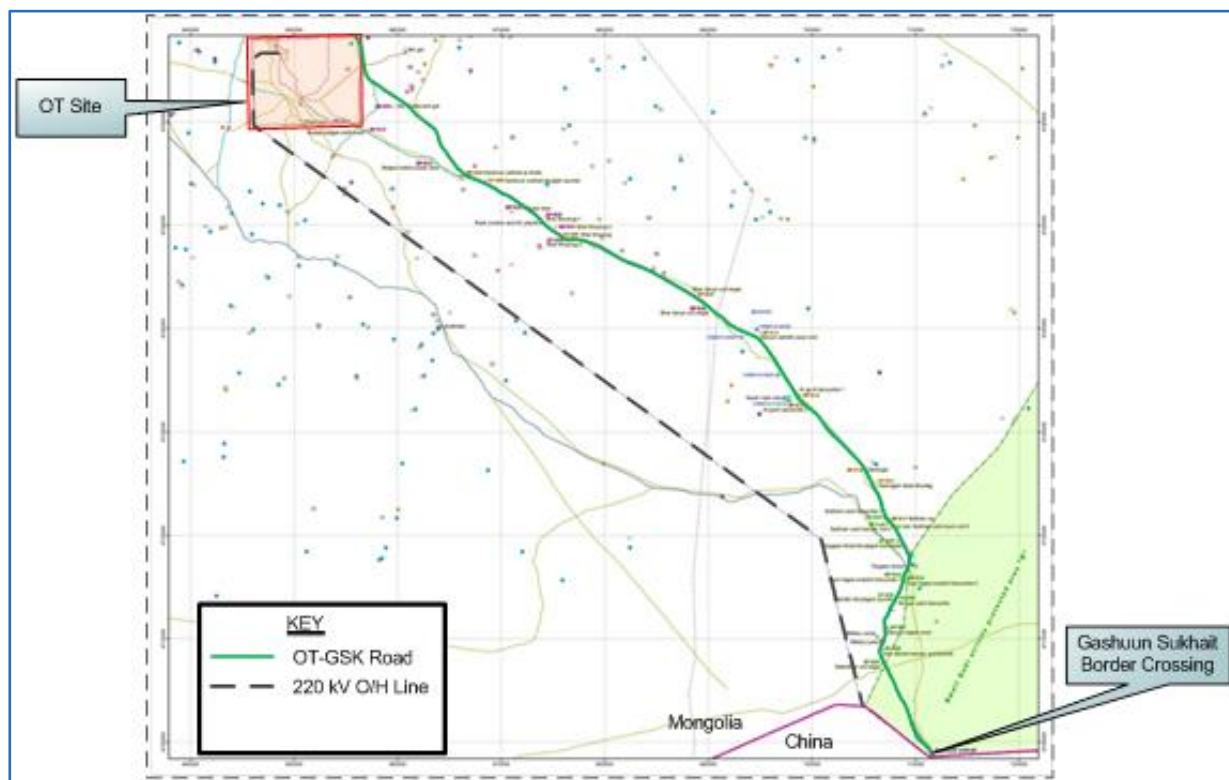
Access to the water borefield is over a gravel service road from the plant site, across the northern lease boundary, and following the pipeline route to Gunii Hooloi. Traffic loading for the borefield road is limited to light vehicles and occasionally heavy equipment and trucks for routine inspections and maintenance. The road is constructed to Mongolian standards in accordance with a resolution issued by the Mongolian Department of Roads in 2002. This requires that topsoil be removed and the road surface be levelled and compacted only. The road is formed with a cross-fall and table drains and has a gravel wear surface. The running surface is 3 m wide and the shoulders 1 m wide. Signage has been added as required. Because the Phase 2 Project does not change the borefield use, no project related cost increase is expected.

The entire site boundary is surrounded by a mine lease perimeter fence with security gates at entrance / exit points. The fence is a conventional post-and-chain mesh, wide-type, approximately 2 m high. Supplementary security fencing will be required at individual infrastructure facilities for the expansion activities.

18.4.3 Oyu Tolgoi – Gashuun Sukhait Road

The access road from Oyu Tolgoi to Gashuun Sukhait (OT-GSK Road) and the Chinese border crossing has been upgraded. The detail design was performed by a Mongolian design company, Mongolian Construction and Project Consultant (MCPC), based in Ulaanbaatar. The total length of 105 km from the North gatehouse to the Mongolia–China border (Figure 18.3) is a public road. The final sections of Zone 3 and Zone 5 of the Gashuun Sukhait road, which total around 23 km, are yet to be completed.

Figure 18.3 OT-GSK Road Oyu Tolgoi Site to Mongolia-China Border

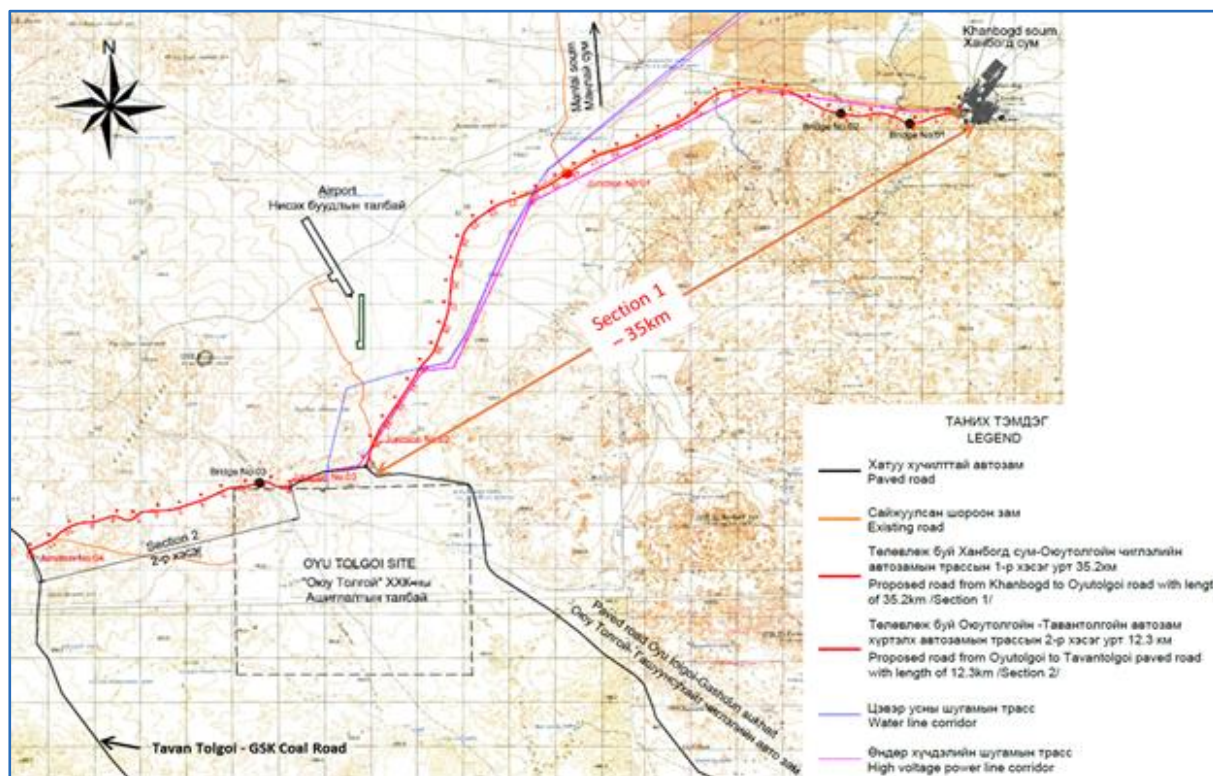


The OT LLC border improvement scope will involve upgrading the road link between the existing Mongolian customs facility at the Mongolian border in Gashuun Sukhait and the existing Chinese customs facility at the Chinese border in Ganqimaodao. The inter linking roads between the borders will be upgraded, subsequent to discussions with the Mongolian Government, to meet the expected project-related traffic needs at the border crossing, including the upgrade and dedication of two border-crossing checkpoints. In addition, a secured bonded yard is to be developed at Gashuun Sukhait to accommodate the concentrate fleet, should there be any unanticipated border closures, as well as a laboratory to test incoming chemicals and reagents.

18.4.3.1 Oyu Tolgoi to Khanbogd Road

In accordance with the Investment Agreement, the existing 35.1 km-long gravel road between Oyu Tolgoi site and Khanbogd soum will be upgraded as part of the Phase 2 Project. The road alignment, shown in Figure 18.4, will follow the existing road. The road will be of asphalt concrete construction and include two reinforced concrete bridges, 65 m and 32 m long, along with four causeways for a total length of 400 m. Initial project ramp-up is expected in late-2016 and the road will be completed in 2018.

Figure 18.4 Oyu Tolgoi to Khanbogd Road



18.4.4 Customs Bonded Zone and Marshalling Yard

A bonded yard with a security fenced area and administration building has been established at the Oyu Tolgoi site for import of equipment and material to the site. A secured customs bonded zone and marshalling yard has been added at the Oyu Tolgoi site. It serves as a marshalling yard to assemble convoys for the outbound concentrate fleet, a storage area for extra bags of concentrate, and a customs bonded zone approved by the Mongolian General Custom Administration. The marshalling yard will be expanded to address the increase in concentrate exports following ramp-up of underground production.

18.4.5 Access Road Through China

A reasonable quality provincial road, S212 connects the border town of Ganqimaodao with the Jingzang Expressway (G6) via the towns of Hailiutu and Wuyuan. Concentrate will be transported by road to the Jiayou Hua Fang terminal, 7 km beyond the Gashuun Sukhait-Ganqimaodao border crossing within China. Concentrate bags will be unloaded at Hua Fang, where Oyu Tolgoi will initially construct 100,000 m² of laydown area. Mobile gantry cranes will unload and reload the bags. Oyu Tolgoi will construct an additional 100,000 m² of laydown when concentrate production increases to 1.9 Mt/a.

18.4.6 Rail Considerations

A private Chinese rail operator, Shenhua, has recently expanded its rail network connecting Baotou to Gashuun Sukhait at the border. Shenhua has also concluded negotiations with Mongolian authorities to extend the line into Mongolia, to the vicinity of Tsaagan Khad. On completion, this rail line will technically enable Mongolia to export minerals to seaborne markets via the Chinese Port of Tianjin. This would require exports to originate in and traverse China using the same transport mode (rail gauge) in order to be classified as bonded cargo and therefore exempt from 17% Chinese VAT.

The GOM may construct or facilitate the construction and management of a railway in the vicinity of the project to the China-Mongolia border. The GOM will consult with OT LLC on the location and route of the railway, and, if the railway is constructed, then it will be made available to OT LLC on commercial and non-discriminatory terms. Energy Resources is currently constructing a single-track heavy-haul rail from its Ukhua Khudag coal mine (approximately 120 km to the north-west of Oyu Tolgoi) to Gashuun Sukhait, ultimately to be interconnected with the Chinese rail network at Ganqimaodao on the Chinese side of the border. Once constructed, the South Gobi Rail alignment would pass within 10 km of the Oyu Tolgoi project area and therefore represents an opportunity for eventual connection of the mine to the rail network.

Some consideration has been given to establishing a rail link to the site for the transport of:

- Concentrate (outbound to various Chinese smelters).
- Coal for the power plant (inbound from Mongolian coal mines).
- Diesel fuel (inbound from Russia).
- Other inbound equipment and consumables.

18.5 Mine Site Infrastructure

18.5.1 General Site Development

The Phase 2 Project activities are essentially within the footprint of the existing Oyu Tolgoi mine site. While some general site development is involved, the scope is described by OT LLC as relatively minor, primarily involving site finishing rather than general development.

18.5.2 Accommodation Strategy and Camp Management Plan

Oyu Tolgoi has two on-site camps, Oyut and Manlai. The Manlai Camp is about a kilometre to the south-east of Oyut. The Oyut camp will continue to be expanded during 2017 to meet the ramp-up of project personnel.

OT LLC has several existing accommodation options, ranging from single-bed rooms with an adjoining ablution facility to rooms with four beds and non-adjointing dormitory-style ablution facilities; personnel accommodation on site is based on their position banding.

In late-2015 OT LLC changed its accommodation policy so rooms would have a maximum of three beds, and all rooms, regardless of the number of beds, would have an adjoining ablution facility. This change was part of a number of initiatives implemented on site to

reduce fatigue-related incidents and accidents. Long-term, the operation will transition to ensuring that each person sleeps alone in a room, albeit sharing the room with another person on the opposite shift, i.e. one person day shift and one on night shift.

In assessing camp capacity requirements for Phase 2, including the number of beds and the number of rooms needed to house these beds, OT LLC considered the total number of operational and construction personnel during the project phase, their position bandings, and the recent changes in accommodation policy. The personnel and camp requirements outlined below represent the combined open pit and Phase 2 underground development staffing. The total number of people on site peaks at approximately 6,254 in 2018, including 3,519 project personnel and 2,735 operations personnel.

Other camp facilities are the Khanbogd Camp, the Erchim Camp, and the Javkhlant Camp.

The Khanbogd Camp is located in the town of Khanbogd, 45 km away from the mine site. It will be used during construction of the Oyu Tolgoi-Khanbogd road.

The Erchim Camp was originally located north of the site but has been relocated to an area south-west of the central heating plant. It has approximately 900 beds. The Erchim Camp has been recommissioned to provide accommodation in the short-term while the Oyut Camp is completed and has also been identified to provide accommodation for major international contractors when a stand-alone, segregated camp is most practical for management of the construction workforce. The Javkhlant Camp has been decommissioned.

18.5.3 Open Pit Truck Shop Complex

The open pit truck shop complex is approximately 1 km north-west of the primary crusher, within the maintenance complex, adjacent to the bonded customs storage area to the north-east and the main fuel storage facility to the south-east. It covers a land area of 500 m x 350 m, or 17.5 ha and incorporates outdoor facilities and four self-contained structural steel, pre-engineered buildings designed to accommodate the required facilities for repair, maintenance, and rebuild of the open pit mining equipment. The area also has storage space for staged spare parts and consumables and administration offices. No mine personnel change facilities are included in this complex.

Open pit trucks enter via the entry gate at the south corner of the facility and are placed on the “dead” line awaiting maintenance. Trucks already repaired are held on the “go” line along the south side of the complex. Repaired vehicles exit back to the open pit through the south gate. The complex includes buildings for the truck shop, washing shop, lubricant storage building, and welding and machine shop.

18.5.4 Central Heating Plant

The central coal-fired boiler plant, was completed in 2012. The plant provides hot water heating for the concentrator building, open pit truck shop, construction and operations warehouses, administration building, Oyut and Manlai camps, electrical substations, and all other surface facilities, as well as mine air heating systems for Shafts 1, 2, and 3. Hot water from the central boiler plant is supplied and returned through a primary circulation loop to the various secondary circulation and heating loops, which are complete with dedicated hot water / glycol heat exchangers to provide heating to the end users. The heat distribution system begins at the boiler house mechanical annex, from where the hot water is pumped to

two heating distribution stations through pre-insulated pipelines. The pipe is buried most of the way to the distribution stations except near the beginning, where it crosses the river on a pipe rack. Water from the distribution stations is then circulated to heat exchangers at the end-user facilities for building heating in compliance with health and safety standards and regulations.

The CHP will service the new facilities. Lines will run to Shaft 1 to service the larger mine air heaters that will replace the temporary units; to supply the air heaters at Shafts 2 and 3, and to meet the needs of the concentrator conversion, the new operations warehouse, the underground mine dry facility and the operations camp expansion. The two 29 MW units are planned to be operational in late-2018.

18.5.5 Underground Utility Services

The underground utility services include raw water, firewater, domestic water, sewage, and return water.

18.5.6 Waste Disposal Facilities

Waste generated during the development and operation of the Oyu Tolgoi mine is collected and disposed of in accordance with Mongolian and international laws. The waste management centre is a focal point of waste management practice at site. Principal features of the waste management centre include a non-hazardous waste landfill, leachate treatment, and a waste incinerator / oil burner. A recycling and composting building will be constructed as part of the sustaining capital plan.

18.5.7 Fuel Storage

Diesel fuel is delivered by the supplier in a tank truck. The fuel is unloaded and stored in storage tank farms. No changes to the open pit or diesel power stations are required for OTFS16. Separate diesel storage for underground use is included at the shaft farm. The underground storage and fuel distribution system will be supplied from three surface storage tanks, batch transferred to underground storage day tanks by means of 50 mm piping installed down a dedicated borehole.

18.5.8 Core Storage Facility

The existing core storage is capable of servicing the existing operation and the expansion operation, no changes are required. The core storage facility has gravel road access, an outdoor core storage area with compacted soil floor, and a 35 m x 24 m prefabricated steel building for core management facilities. The facility includes:

- Core logging work area
- Saw room
- Core test rooms
- General storage room
- Break room
- Washroom facilities.

18.5.9 Toyota Workshop

Toyota has constructed a light-vehicle maintenance facility for contract maintenance of all light vehicles at the site. The facility is complete and fully functional but is now being used for general purpose maintenance storage. This shop will be used to provide light / medium vehicle maintenance for the construction fleet.

18.5.10 Information and Communications Technology (ICT) Systems

A state-of-the-art information, security, data, and voice communications system is installed to ensure that operational needs are met. A fibre optic communications backbone extends through the entire mine site and out to the borefield to support the following principal components:

- Distributed Control System (DCS)
- Programmable Logic Controller (PLC)-based control systems
- Electrical monitoring system (EMS)
- Local Area Network (LAN)
- Voice over Internet Protocol (VoIP) system
- Security system, including closed-circuit television (CCTV) and access control system (ACS)
- Fire alarm system (FAS)
- Digital trunk radio system (DTR)
- Cable television (CATV) for operations personnel entertainment.

Each of the components will be expanded to support modifications to the surface concentrator, underground development, power distribution system, and operations camp. Components and suppliers will be similar to those used for the existing facilities. This will provide continuity in support services and parts inventories of the systems and make use of the training and experience gained by site personnel. The network backbone infrastructure will continue to provide connectivity at the expanded and new facilities.

18.5.11 Batch Plants

There are two batch plants that are complete and currently under care and maintenance. No further batch plants are planned for Phase 2. One batch plant is rated for 120 m³/h with an optimal rate of 90 m³/h and will provide for the Phase 2 requirement. The other batch plant optimally operates at 40 m³/h and currently provides a dedicated supply of concrete for the underground mine.

OT LLC owns eight 10 m³ concrete mixer trucks and plans to augment the delivery fleet by six 10 m³ units for the expansion activities. Any further short-term mixer truck capacity will be provided by contractors as required.

Concrete will be delivered from the surface batch plants through slicklines in Shaft 1 and a dedicated facility at Shaft 2 with two slicklines.

18.5.12 Other Support Facilities and Utilities

Other facilities include the warehouses, medical facilities and fire station.

The operations warehouse is a 45 m x 180 m heated building adjacent to the construction warehouse that provides 8,100 m² for the storage of process equipment parts, spares, critical piping and valves. Ongoing operation and development of the underground mine will increase the demand for heated warehouse storage space by approximately 6,600 m². A new 52.5 m x 126 m operations warehouse will therefore be constructed adjacent to the existing one. Similar to the existing operations warehouse, it will be fitted with shelves, offices, loading/unloading areas, and roll-up doors.

The existing surface medical and fire station infrastructure is capable of servicing the construction and operational needs of the existing operation and the OTFS16 expansion activities.

A 90 m x 50 m construction warehouse has been built and is in daily use. As the function of the warehouse has evolved into an operations warehouse, more warehouse facilities will be required for expansion construction. Therefore, the operations warehouse expansion will be built early in the expansion programme for use as a construction warehouse. This will provide an interim space of 52.5 m x 126 m for the expansion construction needs.

18.6 Water Management

18.6.1 Introduction

This section supplements the information on surface water and groundwater in the project area that has been collected in previous studies. It includes updated meteorological data, water balances, and the results of additional groundwater exploration and analysis. It also summarizes the current understanding of water resource use in the South Gobi Region (SGR), including the effects of anticipated increases in the local population resulting from activities associated with mine development. It provides a detailed description of the Gunii Hooloi aquifer and proposed borefield and presents Oyu Tolgoi's plans for water conservation, on-site water management, water distribution and treatment, and a diversion of the Undai River to accommodate project facilities. The information is based on the work of several consultants retained by Oyu Tolgoi over the years.

Water resource development in the SGR is part of the Mongolian national water resources strategy, and its management is embedded in national legislation and the institutional framework.

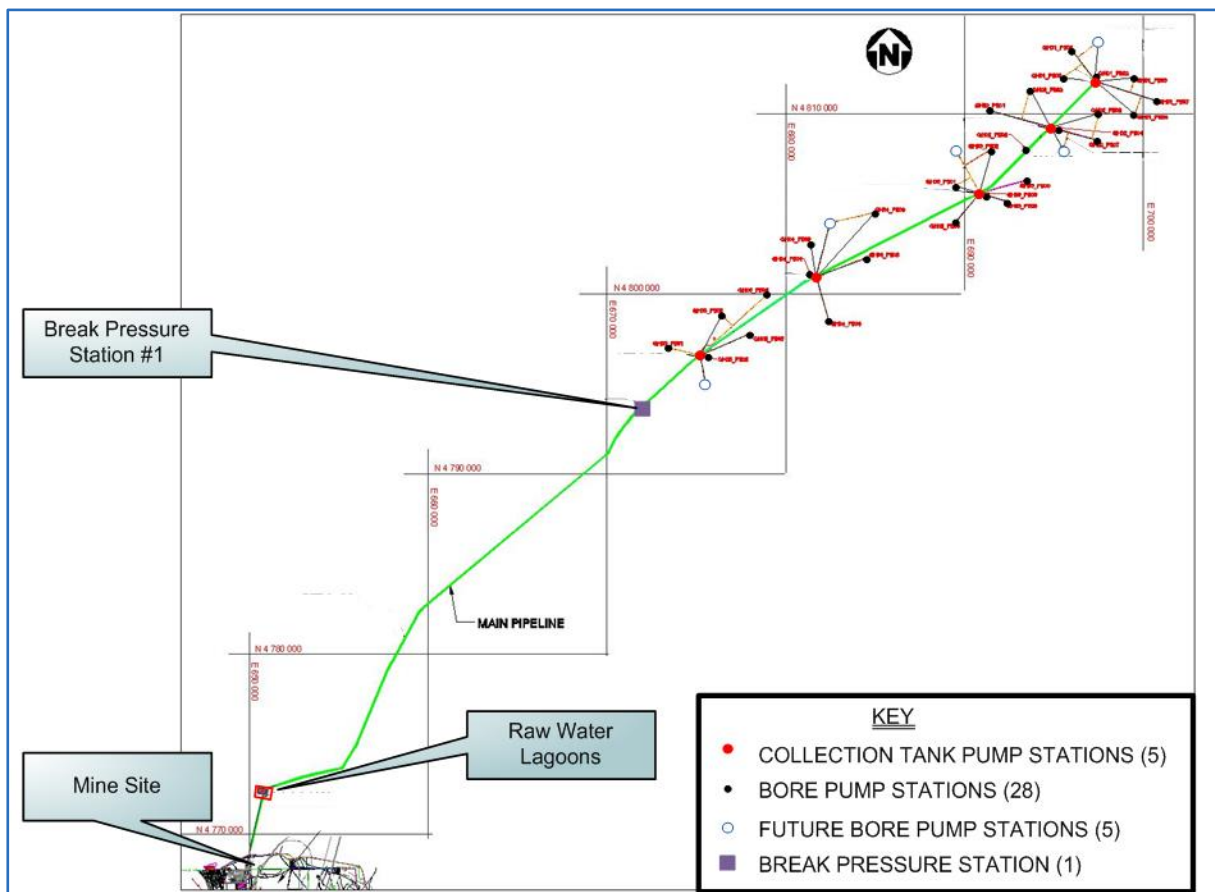
Rainfall and the presence of surface water normally occur during storm events in the summer season. Given the scarcity of surface water, groundwater is the main water supply resource in the SGR and is the source of water for the mine. Near-surface aquifers recharged by the occasional rains are used traditionally for domestic purposes and raising livestock. Oyu Tolgoi will not use the surficial aquifers to supplement its mine water requirements. Oyu Tolgoi is aware of the importance of water in the SGR, and the project has implemented a wide range of measures to promote water conservation and to minimize the amount of water the project will use. OT LLC believes that Oyu Tolgoi is among the most water-efficient mines of its kind in the world.

18.6.2 Raw Water

Actual water usage performance has exceeded expectations, with a maximum usage of 622 L/s recorded January 2014 (winter, driven by temporary water loss due to freezing at the TSF). Average water consumption in 2015 was 469 L/s.

OTFS16 water demand for the Oyu Tolgoi facilities, including the Phase 2 expansion, has been calculated at between 588 L/s and 785 L/s, with an average yearly demand of 696 L/s, to meet a production rate of 110 kt/d. The primary source of raw water to meet these requirements is the Gunii Hooloi basin, which extends 35 km to 70 km north of the Oyu Tolgoi site. The raw water supply pipeline system is shown in Figure 18.5

Figure 18.5 Raw Water Supply Pipeline



Bores have been developed in two distinct areas of the Gunii Hooloi borefield, south-west and north-east. It is anticipated that the 10 water bores in the nearer, south-west part of the borefield will provide approximately 30 L/s per bore (300 L/s in total) and that the 15 bores in the higher transmissivity, north-east part of the borefield will be able to provide another 40 L/s per bore (600 L/s in total). The raw water pipeline has a design peak capacity of 900 L/s, providing an additional 115 L/s margin that can be used to refill the project emergency storage lagoons after emergency use, without affecting water availability to site.

Water from groups of individual bores accumulate into five centrally located collection tank pump stations, from which water is pumped into the main water line leading to the Oyu Tolgoi site. A break tank pump station decreases the line pressure to atmospheric pressure and provides the additional pumping energy to bring the water to the site.

Water is pumped into a 400,000 m³ emergency storage lagoon (two cells, 200,000 m³ each) situated on elevated ground approximately 5 km north of the Oyu Tolgoi site. The lagoon provides an approximately one-week emergency supply of water as a contingency in the event of a pipeline / borefield breakdown and need for maintenance. Water is gravity-fed to the site through two pipelines from the two cells.

All equipment in the Gunii Hooloi borefield and pipeline is remotely controlled by the site distributed control system (DCS) linked by redundant telecommunications networks. All equipment in the borefield is powered with electricity through a high-voltage transmission line routed adjacent to the pipeline, with power drops to each pump station and to the borefield. A light-duty access road runs along the pipeline to each pump station and bore.

18.6.3 Site Water Systems

Domestic water is only used for washing, shower, and eye-wash stations, not for drinking. The raw water for domestic use is treated in the water treatment and bottling plant and then delivered from the plant to end users by HDPE SDR 13.5 pipe. Use areas include the central heating plant, warehouse, open pit truck shop, concentrator, primary crusher, 220 kV central substation, diesel power station, North gatehouse, operation camp, and administration building. The system will be extended to include the operations warehouse expansion and the operations camp expansion.

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. The plant consists of two areas, the water storage and pump station, and the water treatment and bottling plant.

The existing wastewater treatment plant is adequate to manage the Phase 2 Project requirements. This facility, installed within the main camp area, was upgraded from 800 equivalent people (EP) capacity to 4,000 EP in 2006 and is fully operational to accommodate the sewage production of the construction and operations labor force. All sewage generated on site will either be pumped directly to the plant or transported by truck to an unloading bay. The treatment plant technology is mechanical biological activation using sequencing batch reactors (SBRs). The sanitary sewer system collects sanitary wastewater from the central heating plant, warehouse, open pit truck shop, Shafts 1 and 2, concentrator, North gatehouse, operations camp, and administration building and delivers it to the wastewater treatment plant. Sanitary wastewater from remote areas is collected in local holding tanks and trucked to the wastewater treatment plant. The system will be expanded to include the operations camp expansion, operations warehouse expansion, Shaft 3, shaft farm substation, and new mine office control room and dry changerooms.

The return water system is used to transport dewatering water from Shaft 1 and Shaft 2, effluent from the wastewater treatment plant, and emergency water from the central heating plant to tailings storage. It does not include the existing plant discharge water, which is transferred to a holding tank to be used for dust suppression. The system will be expanded

to include the operations warehouse expansion, operations camp expansion, Shaft 3, conveyor-to-surface facilities, and the new mine office, control room, and dry changerooms.

Six fire pump stations are available on-site to provide firewater for end users. The pump stations are at the following locations:

- Raw water treatment and bottling plant
- Construction warehouse (former)
- Concentrator
- Shaft 1
- Shaft 2
- Central heating plant

The fire protection pump zones cover the majority of the on-site facilities. Remote on-site and off-site facilities are not connected to the high-pressure firewater system, but instead rely on local manual fire protection. Fire detection and alarm systems are installed at key facilities and report to the mill area control room in the process plant or to the North gatehouse, which is manned 24 h/d.

Firewater systems are to be provided for the underground mine and associated maintenance and explosives storage areas. These systems are self-contained and serviced from the raw water distribution system via the shafts.

18.6.4 Water Conservation

Minimizing water use throughout all the operational aspects has been a key focus of attention during mine planning and design. As examples of water conservation planning, the following initiatives have been implemented:

- Reuse of cooling water – The process plant is the largest consumer of water. Within the plant, all water discharged from the cooling systems, still categorized as clean water, is sent to the process water pond for reuse in the concentrator.
- High-efficiency tailings thickeners – The tailings thickener at Oyu Tolgoi uses advanced techniques and is able to achieve a tailings solids content of 60%–64%, which significantly reduces the amount of water sent to the TSF. These design modifications help to greatly reduce the amount of reclaim water released and evaporative losses from the TSF.
- High-efficiency TSF reclaim – The TSF has been designed so that tailings are deposited in discreet cells, rather than broadly across the facility, to reduce evaporative losses. The entire base of the TSF rests on natural or installed clay and includes a comprehensive seepage collection system to minimize seepage losses. The TSF reclaim system has been designed to ensure that all supernatant water and collected seepage is returned to the process plant for reuse.
- 100% mine water recovery – All water encountered in the underground and open pit mines is recovered for use as process water or for dust suppression. Recovery of mine water helps to reduce site demand for raw water from the Gunii Hooloi aquifer. From a water balance perspective, this mine recovery water has conservatively not been included as inflow.

- 100% treated wastewater reuse – All treated wastewater produced in the site wastewater treatment plant will be reused in the process plant or for dust suppression.
- 100% truck wash water reuse – A comprehensive water treatment system has been installed at the project mine truck washing facility to allow all truck wash water to be continuously recycled and reused.
- Lagoon floating cover – A floating cover is placed over the entire raw water storage lagoon to eliminate evaporative water losses and to limit the accumulation of dust within the lagoon.
- Selection of low or zero water use equipment – Examples of this include the selection of air cooling systems at the hazardous waste incinerator and central heating plant.

Ongoing attention to water conservation will be maintained during operation through the continuous review of key performance indicators for water use and implementation of additional water conservation measures.

18.6.5 Water Balance

A site water balance has been established that identifies water supply, recovery loops and losses at Oyu Tolgoi during routine 110 kt/d operations. Notable features of the water balance with regard to water recovery and reuse include:

- Tailings thickener recovery loop – 76% (2,356 L/s of 3,088 L/s) of water is recovered for reuse within the concentrator.
- TSF reclaim loop – A further 6.6% (205 L/s of 3,088 L/s) of water is recovered for reuse within the concentrator.
- Cooling water recovery – Raw water supply (163 L/s) for concentrator equipment cooling is recovered to the process water pond and reused in the concentrator.
- Treated wastewater recovery – Recoverable treated domestic wastewater (anticipated 3.4 L/s) is returned to the process water pond and reused in the concentrator (or reused for dust suppression).
- Mine water recovery – Recoverable water from mining operations is returned to the process water pond and reused in the concentrator (or reused for dust suppression).
- Zero discharges – Water is fully recovered and reused within the system and there are no planned discharges.

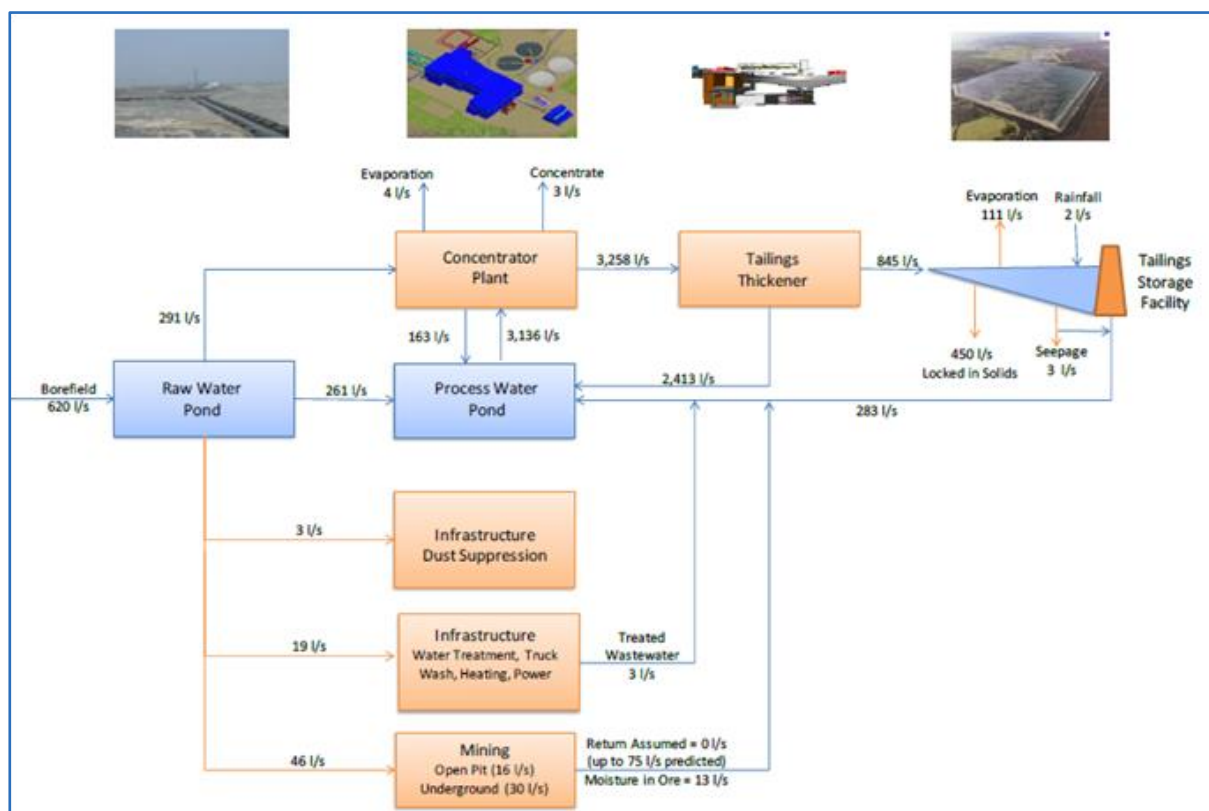
A simplified site water balance is shown in Figure 18.6.

As a result of the water conservation and reuse measures, more than 80% of the water used in production consists of recycled water; water will typically be used more than five times before being lost. Oyu Tolgoi is expected to consume less than 550 L/t of ore, which is less than half the typical water usage rate (1,220 L/s of ore) for copper–gold concentrating mines worldwide.

Most water losses will be from the TSF, primarily associated with water that is locked in solids (retained in the tailings soil matrix), but also due to evaporation. Additional water losses will result from dust control in the underground mine and the roads associated with the open pit and infrastructure. Minor water losses relate to construction activities (e.g., concrete

production), infrastructure maintenance (e.g., heating system make-up), and unrecoverable water from domestic water use.

Figure 18.6 Simplified Site Water Balance



Source: Oyu Tolgoi ESIA

18.6.6 Water Demand

The water demand estimate for Oyu Tolgoi assumes a peak design processing rate of 110 kt/d and 64% tailings density. Although the need for mine dewatering at a rate of up to 90 L/s is predicted, this value is uncertain and may not be realized. OTFS16 has made a conservative assumption that there will be no water provided from mine dewatering into the water supply system.

The total site design water demand ranges from 588–785 L/s, with an average of 696 L/s, to support the nominal production rate of 110 kt/d. The peak water demand (excluding recycling) in OTFS16 is:

- Concentrator (including TSF) 670 L/s
- Other (Camp, dust suppression, etc.) 75 L/s
- Underground mine 30 L/s
- Total 775 L/s

Water consumption performance to date has been better than predicted, with continuous improvements evident over the first three years of operation. Average water consumption in 2015 was 469 L/s, representing a usage rate of 433 L/t (ore). Water consumption is expected to increase to approximately 550 L/s as a result of underground mine development (up to 30 L/s) and ongoing improvements in concentrator production capacity (up to 20%). This rate remains well below the permitted usage rate of 918 L/s and the capacity of the raw water supply system.

18.6.7 Groundwater Resources

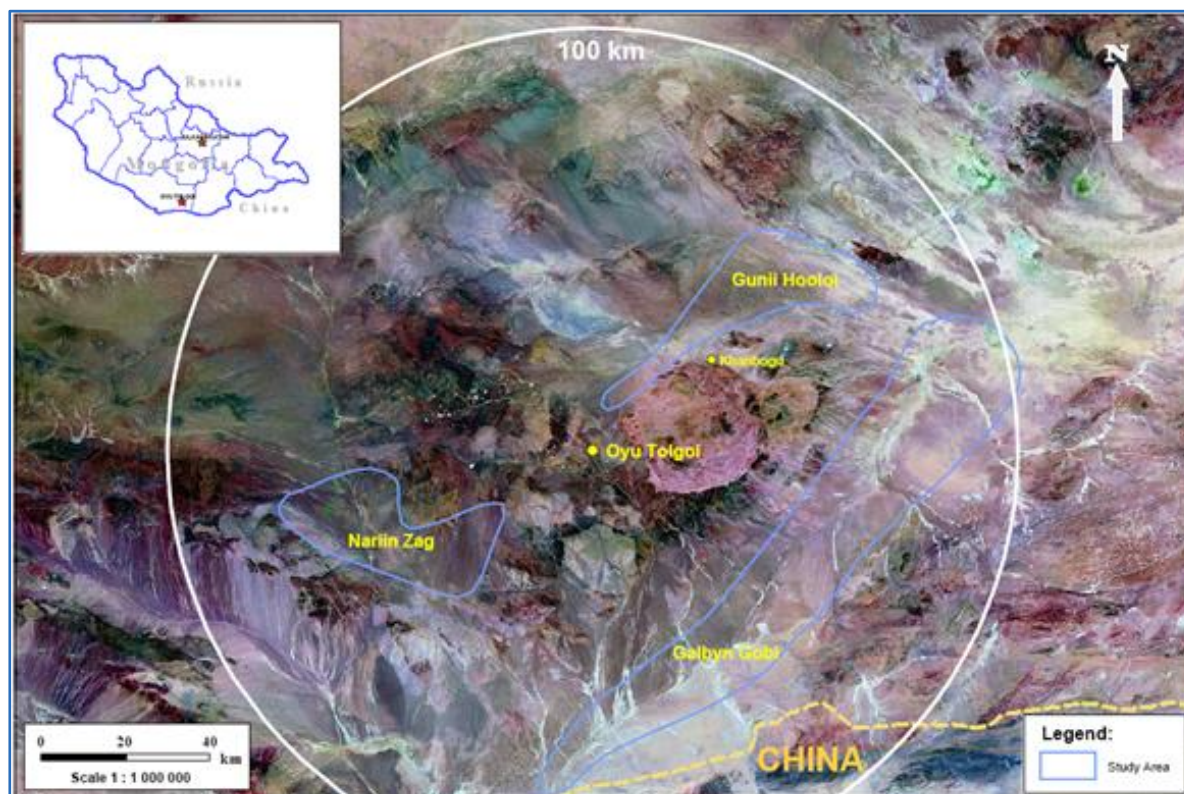
Project water supply investigations began in early-2002 with geophysical surveys of known sedimentary basins within a feasible distance of the project area. This early exploration indicated the presence of at least six substantial groundwater deposits within the region, all within 100–200 km from Oyu Tolgoi:

• Tsagaan Tsav	800 L/s
• Balgasyn Ulaan Nuur	150 L/s
• Galbyn Gobi	300 L/s
• Tavan Ald	130 L/s
• Nariin Zag	50 L/s
• Zairmagt	30 L/s

Based on initial assessment, Galbyn Gobi was selected for further work, considering its high potential for aquifer development and relatively close proximity to the project site.

Figure 18.7 shows the locations of the three regional aquifers initially selected for consideration: Galbyn Gobi, Gunii Hooloi, and Nariin Zag.

Figure 18.7 Location of Regional Aquifers



Source: Oyu Tolgoi Integrated Development and Operations Plan, May 2011.

18.6.8 Gunii Hooloi Aquifer

The suitability of the Gunii Hooloi resource has been determined through three key groundwater exploration and testing programmes undertaken for OT LLC by RPS Aquaterra, one 2003 / 2004, the second from January to September 2007 and the third using the significant additional body of data generated with completion of the borefield construction in 2011 / 2012 and the 2011 drilling and testing programmes.

The results from the three key field investigation programmes were integrated to promote a detailed understanding and a detailed hydrogeological model has been developed to represent the natural behaviour and dynamics of the aquifer system.

The geology in the Gunii Hooloi area is highly faulted and structurally complex, giving rise to aquifer features that include shallow alluvial aquifers, shallow bedrock with weathering-enhanced permeability, and deeper, structurally controlled aquifers.

The shallow alluvial aquifers occur in near-outcrop colluvial deposits, alluvial outwash plains, and alluvium in modern creek beds. Alluvial deposits cover approximately 70% of the concession area. Recharge to the shallow aquifers is via infiltration along the interface between bedrock outcrop and colluvium, direct infiltration from rainfall, and creek flow. These shallow aquifers are typical of water supplies used by local people for domestic and livestock purposes along drainage features in the surrounding area.

Bedrock lithologies within the project area are typically of very low permeability, although secondary permeabilities resulting from fracturing and weathering are not uncommon. The upper weathering and transported horizons are generally of low permeability.

Gunii Hooloi is a fault-controlled sedimentary basin within a basement graben structure. The sedimentary in-fill sequence includes undifferentiated Quaternary sediments, underlain by the Upper Cretaceous Bayanzag and Bayanshiree formations. The surface topography of the sedimentary basin is relatively flat and subdued. In contrast, the bedrock topography that defines the basin profile outcrops around the margins of the basin and forms pronounced topographic highs.

The basin contains two main aquifer systems: a major, deep, largely confined aquifer, and a localized, unconfined streambed aquifer associated with active drainage channels. The key characteristics of these systems are outlined below.

18.6.8.1 Main (Deep) Aquifer

The deep aquifer in the Gunii Hooloi basin is the source of raw water for the project. The main water-bearing unit typically consists of a stratified sequence of sands, gravel, sandstones, siltstones, and gravelstones interbedded with lesser clayey aquitard horizons. The unit is aerially extensive (>550 km²) and flows continuously for over 40 km to the north-east. It ranges in thickness from 30 m in the south-west to >260 m in the large north-eastern sector. An extensive, clayey, confining aquitard overlies the main aquifer and separates it from the near-surface system; the clay layer ranges in thickness from 10 m in the south-west to approximately 100 m in the north-east.

18.6.8.2 Streambed Aquifer System

The streambed aquifer system is a discrete river alluvial feature associated with the major surface water drainage networks. The shallow water is usually found in alluvium materials less than five metres thick.

As with the regional deep Gunii Hooloi aquifer, surface water systems extend farther north-east, but these appear to be fed from seasonal rainfall and there is no evidence of linkage to the regional aquifer. Neither is there seen to be any connection between the deep aquifer and the shallow herders' wells in near-surface streambed aquifers.

18.6.8.3 Aquifer Water Quality

Field measurements from the 2007 test programme indicated that water in the deep Gunii Hooloi aquifer was generally saline, with total dissolved solids (TDS) in the range of 1,500–5,300 ppm and averaging 2,760 ppm. Other testing has found salinity levels ranging from 1,300–3,200 ppm with an average of 2,400 ppm. The water is also slightly alkaline, with an average pH of 7.46 as measured in 2007.

Water in the shallow streambed aquifers was found in the 2007 field measurements to have TDS mostly below 800 mg/L and averaging 690 mg/L. Water quality in shallow herders' wells is usually better, often having a salinity of less than 350 ppm. Most wells are associated with surface drainage features that recharge seasonally.

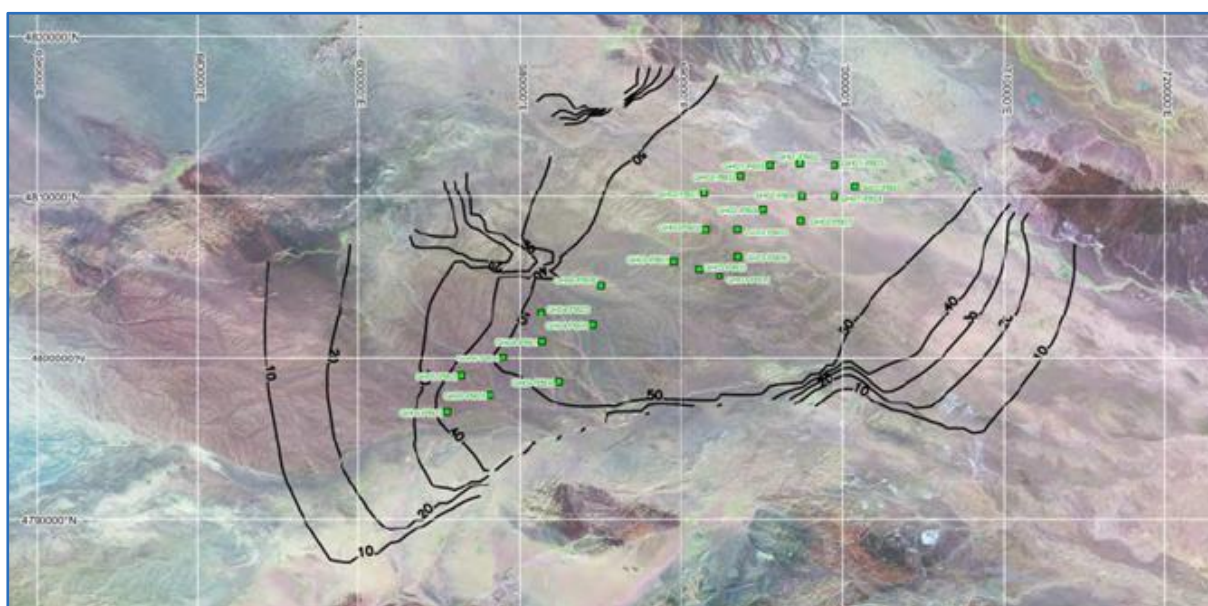
18.6.8.4 Aquifer Drawdown and Yield

The GOM awarded a water utilization contract to OT LLC until 2040, which may in turn be extended for 20-year periods beyond 2040, in accordance with the Law on Water. OT LLC is currently entitled to utilize water at a rate of 918 L/s. Updated hydrogeological modelling, completed in 2013, and based on all three hydrogeological investigation programmes, demonstrates that the Gunii Hooloi aquifer is capable of providing 1,475 L/s, based on the same time and drawdown conditions. Given the minor amount of recharge to the main aquifer system, abstraction of groundwater at the rates proposed will result in drawdown of the aquifer water levels. Assessments were based on the predicted aquifer drawdown limits of up to 75 m over 40 years at Gunii Hooloi. It is recognized that the depletion of these aquifers will take many hundreds of years to recover. However, the groundwater model of Gunii Hooloi predicts that even after 40 years of extraction, the deep aquifer system will remain confined and is likely to have no effect on the streambed aquifers. Predicted aquifer drawdown is illustrated in Figure 18.8.

Environmental studies for the project (see Section 20, Environment) have found that the drawdown would have very low potential for affecting the environment or shallow water resources currently accessed by herders. The shallow water systems present in some parts of the Gunii Hooloi area are stream-fed shallow groundwater within riverbeds that are replenished following summer rainfall; water levels within the riverbeds are highly variable. The deep layer of clay above the deep aquifer is expected to limit leakage from the shallow riverbed systems that could result from the drawdown. Considering the seasonal and annual variations in water levels within the river sediments, any leakage would be insignificant.

Drawdown will be monitored, and the aquifer model will be updated and refined as more data become available. As necessary, abstraction from the boreholes will be adjusted to optimize drawdown characteristics and protect the shallow groundwater resources.

Figure 18.8 Aquifer Drawdown (40 years, base case conditions)



Source: Gunii Hooloi Groundwater Model Report, RPS Aquaterra, December 2013.

18.7 Tailing Storage Facility

18.7.1 Introduction

Several options for the design of the Oyu Tolgoi Tailings Storage Facility (TSF) have been investigated since 2002. Knight Piésold (KP), Klohn Crippen Berger (KCB), and Golder have been engaged at various times to identify and develop alternative sites, deposition strategies, and layouts. Most recently, an updated specific TSF Feasibility Study and accompanying Project Execution and Management Plan, realize a number of design improvements and provide clear definition of the overall management and project execution planning for ongoing development of the TSF.

Site selection was based on consideration of such aspects as local topography, location relative to other project facilities, required storage capacity, potential environmental impacts, water conservation, and the potential for future TSF expansion. Central or perimeter discharge, paste tailings, and conventional thickened tailings deposition methods were all evaluated. Because of the flat topography, the design requires the construction of a perimeter embankment to retain the tailings within a basin.

The selected site for the TSF is 2 km east of the open pit, 5 km south-east of the process plant, and is based on deposition of conventional thickened tailings. The concept is to construct cells using perimeter embankments that are raised and expanded over the life of the mine.

OTFS16 requires the construction of up to four independent tailings cells of the same design. The current feasibility study fully defines the development of the first two cells, which are sufficient for the first 20 years of mine operation, and identifies conceptual locations for additional cells that can be used over the remaining life of the mine.

The impoundment layout for the TSF (see Figure 18.9) comprises two 2 km x 2 km cells (Cell 1 and Cell 2), each subdivided into four sub-cells (A through D). Cell 1 has already been put into operation. Tailings are spigotted from the west embankment into the sub-cells, where they form a beach that extends toward a reclaim pond at the downstream corner. Supernatant water recovered from the reclaim pond is returned to the concentrator for reuse.

This section provides a brief introduction to the updated TSF Feasibility Study and the Project Execution and Management Plan, including historical design studies, design criteria and engineering controls, geotechnical and geochemical assessments, the approach to construction and operations, and ongoing / future work.

18.7.2 Design Studies

The Oyu Tolgoi tailings storage facility has been under feasibility study development since 2004, involving a number of phases and consultants. The various preliminary and detailed studies undertaken to date are listed below.

- 2004 – Prefeasibility Study by KP, with thickened tailings options by Golder
- 2006 – Feasibility Study by KCB
- 2007 – Detailed design commenced by KCB
- 2008 – Detailed design put on hold; interim report issued

- 2010 – Draft Feasibility Study by KCB
- 2011 – Final Feasibility Study by KCB (August 2011), MTOs (October 2011), and detailed design of starter embankment by KCB (May 2011)
- 2013 – Design and quality control consulting responsibility transitioned to Golder. KCB retained for provision of quality assurance reviews.
- 2014 – Detailed design of first embankment raise completed by Golder. Independent Technical Review Team re-established to provide expertise oversight on ongoing design development, construction, and operations.

In 2015, Golder updated the feasibility study design for the planned ongoing development of Cells 1 and 2 of the TSF, and prepared a supporting paper on TSF disposal options over the life of the mine. Key improvements to previous studies include:

- Downstream Embankment Construction. The embankment construction method was changed from centreline / upstream to downstream to improve constructability and provide future flexibility.
- Reduced Filter Layer Widths. The widths of the key filter layers were optimized to reduce earthworks construction volumes, reduce the amount of material handling, and minimise the use of scarce sand and gravel.
- Steepened Embankment Exterior Slope. More-efficient embankment exterior slopes on clay, reducing from 10H : 1V to 4H : 1V, are being evaluated based on a detailed review of instrumentation to date and rigorous geotechnical analysis that considers the development of strain and pore water pressures during embankment construction.

The improvements related to downstream embankment construction and reduced filter layer widths are supported and have been endorsed by the Independent Technical Review Team (ITRT). The ITRT also supports the strategy for optimizing exterior slopes, subject to ongoing characterisation of the Cretaceous clay foundation.

At the time of OTFS16, the design of future facility expansion beyond Cell 2 has not been completed to a Feasibility Study level. Ongoing tailings management opportunities post Cells 1 and 2 (i.e. from ~2030) include tailings cells to the east (baseline), additional cells to the south/south-west, further raising of the established embankments, and/or in-pit disposal. These possibilities will need further definition and assessment. For this study, the baseline case of developing two additional cells to the east has been adopted.

18.7.3 Current Status

The starter dam construction was completed in 2013 in time for the concentrator start-up. The QC reports indicate that the work was largely completed in accordance with the starter dam design drawings issued by KCB. However, a number of issues arose shortly after start-up, including:

- The tailings were observed to be much more fluid than expected and did not create a well-formed beach.
- The tailings beach slope was observed to be about 0.6% initially and then flattened to 0.3% versus the design assumption of 1%.
- Tailings density was estimated to be about 1.26 t/m³, significantly lower than the design assumption of 1.45 t/m³.
- Large ponds needed to be generated within each sub-cell to achieve a degree of clarity in the decant pond.

All of these factors resulted in more water being stored with the tailings, higher rates of rise, poorer return water quality, and the need for higher embankment raises. However, the ultimate storage volume of the design remains overwhelmingly a function of the settled tailings density.

It was subsequently determined that many of the tailings thickener rake ploughs had sheared off, resulting in lower concentrations of solids than the design assumption of 62%–64%. Due to these depositional properties, a number of embankment modifications were implemented:

- All upstream and modified centreline construction designs in the KCB design were changed to downstream construction.
- Embankment zonation and specifications were not modified, but the embankment zones were kept relatively wide to facilitate construction.
- Because of poor trafficability on the west beach slope, an intermediate bench was added to each stage in this area for distribution pipeline raises.
- The east dike needed to be raised quickly to accommodate the flatter beach slopes and the additional tailings reporting to the east side of the embankment. This was accomplished immediately before winter 2013–2014, and the tailings surface has since been equilibrating at a slope of about 0.45% (May 2014 results). Tailings flocculation and rheology testwork has indicated the potential for steepening the slope to 0.5% and beyond with a closer approach to design deposition densities.

Golder re-evaluated the foundation shear strength and concluded that embankment quantities could possibly be reduced during the next few years. These assumptions remain to be fully modelled, but initial results indicate that quantities can be maintained at or below the KCB projections.

18.7.4 Design Criteria and Design Basis

KCB prepared a design criteria report as part of the feasibility study (KCB, 2011). Parameters include site climatic and hydrology conditions, design throughputs and total storage requirements, operating requirements, and environmental considerations. Minimum standards for geotechnical and hydro-technical design include return periods for design precipitation events, required factors of safety for static conditions and seismic events, and allowable deformations under seismic loading conditions.

18.7.5 Tailings Characteristics

The tailings characteristics are summarized as follows:

- Gradation testing of the tailings indicates the material is a typical copper tailings sandy silt with 45%–65% fines (less than 74 μm) and clay-sized material (less than 2 μm) ranging from 6%–16%.
- The design density of the tailings was estimated to be 1.45 t/m^3 for initial placement in the starter dam and consolidating to 1.5 t/m^3 for the ultimate facility. Tailing densities during initial start-up were found to be about 1.26 t/m^3 , however, having been adversely affected by poor thickener performance and limited drainage with few sub-cells in use. A density of 1.35 t/m^3 is being used in current projections to reflect anticipated improvements in sub-cell management and thickener performance.
- Hydraulic conductivity for consolidated tailings at shallow depth is estimated to be 1×10^{-7} m/s, decreasing to 5×10^{-9} m/s at the base of the tailings impoundment due to increased consolidation pressure.
- Geochemical testwork predicts that tailings from Central zone, the Wedge zone, Hugo South, and Heruga will be strongly potentially acid forming (PAF). Non-acid-forming (NAF) tailings are predicted from Southwest zone, Hugo North, the Shaft Farm area, and Hugo North Extension (Rio Tinto, 2011a).
- The original design assumed that tailings would be pumped at a target density of 64% solids, forming an assumed overall beach slope of 1%. Performance to date suggests that densities at discharge may be in the range of 58%–60% solids. With the lower solids concentration, beach slopes of about 0.5%–0.7% are anticipated, with short-term beach slopes much shallower.
- Based on laboratory testwork, the tailings were expected to form a hard crust as they dried that would aid in inhibiting dust formation in periods of inactivity. Performance data have shown the tailings beaches remain wet during tailings deposition, with little dust emission. Estimates of evaporative losses have therefore been updated in the water balance to reflect the larger wetted area.

18.7.6 Embankment Borrow Material

The TSF is being developed by constructing engineered earth fill and rock fill embankments. Locally borrowed general fill and clay materials will be used for the starter dam. The clay borrow material will also be used for lower permeability liners where Cretaceous clay is not present and within core zones.

Two categories of waste have been designated: potentially acid forming (PAF) and non-acid-forming (NAF). Most of the embankment will be constructed of various mine waste rock materials and will include:

- Oxide waste rock – This material will act as the processed filter zones between the fine tailings and the coarser waste. The oxide units are NAF and can be segregated for construction; the remaining oxide would be classified as PAF and placed in waste rock storage.
- Sedimentary rock – This material will be placed at the base of the downstream embankment shells as a NAF filter and drainage system.
- Random waste rock – Both NAF and PAF waste rock from the open pit will be placed in the embankment shells of the TSF where water cannot come into contact with the PAF, while NAF-only material will be used where these materials could be in contact with seepage.

The mine plan indicates that NAF rock types comprise about 30% (547 Mt) of the total waste rock to be produced. The timing and ability to effectively separate the NAF rock will affect the potential to use all of it for embankment construction. Dam zoning to incorporate some PAF rock into the dry sections of the embankment shells may be required. OT LLC has developed a draft strategy to identify and segregate waste products by chemistry and physical characteristics (gradation). All filter zones and the basal blanket layer will be constructed of NAF oxide and sedimentary rock units.

18.7.7 Description of Feasibility Design

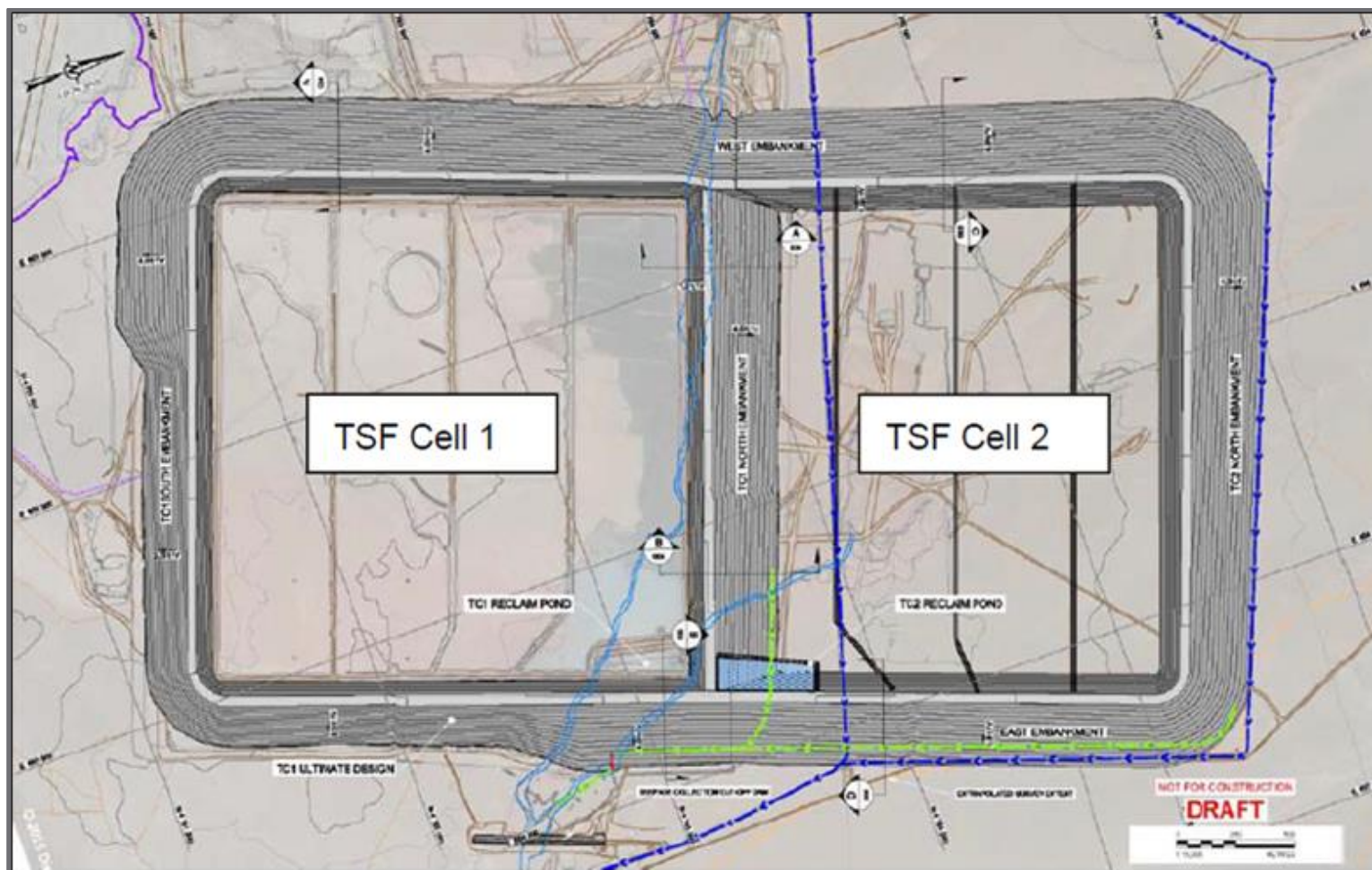
The TSF consists of two adjacent square 2 km x 2 km cells with up to 70 m high embankments enclosing the four sides of the impoundment. Construction of Cell 1 started in February 2011, beginning with a starter embankment that was capable of providing storage for the first 18-months of production. Tailings deposition commenced in March 2012 along with ongoing construction to provide the required tailings storage volumes. The general arrangement of the cells is shown in Figure 18.9.

The TSF embankments are constructed by means of downstream methods, with an inner inclined clay core liner and filter layers and a main body NAF / PAF waste rock shell. The inner and base layers of selected NAF waste rock enable any seepage through the liner / filter layers to be collected without contacting PAF waste rock, and to be conveyed to the perimeter seepage collection system.

The permanently wet section of the TSF around the reclaim pond includes an inner erosion protection facing, a clay liner, filter layers, a NAF waste rock seepage collection zone, the main body NAF / PAF waste rock embankment core, and a base seepage drainage blanket.

The dry sections of the TSF include a general fill inner liner, filter layers, a NAF waste rock seepage collection zone, the main body NAF / PAF waste rock embankment core, and a base seepage drainage blanket.

Figure 18.9 General TSF Arrangement of Cell 1 and Cell 2



18.7.7.1 Dam Structures

The earth and rock fill embankments constructed to store the tailings will be up to 70 m high and will generally vary in form based on the varying tailings deposition and water management regimes around the perimeter of each cell.

Golder re-evaluated foundation strengths to see if it was possible to permit steeper final embankment slopes that would use less material within the embankment at a given elevation. Because these design input assumptions have not been reviewed by the ITRB, the more conservative slopes and embankment quantities developed by KCB are being used at this time. Golder's design assumptions will be evaluated further in a monitoring programme during operations, which will include monitoring piezometers, surveying beaches, surface observations, inclinometer measurements, and embankment surveys.

Currently, Golder has been tasked to design the impoundment for flatter beach slopes of 0.6% for the initial years of operation. This redesign will affect short-term embankment quantities, bringing additional material forward. Once stable beach slopes are established, embankment quantities are expected to be as forecast.

Due to the low tailings density and low deposition gradients observed during start-up and the resulting high raise rates, shorter-term design modifications have been necessary. All the KCB design sections were modified to a downstream construction approach because a stable beach was not available for use in either the centreline or modified upstream construction method. This approach is able to impound the soft tailings without requiring support from the tailings themselves.

The current embankment sections follow the original design of the KCB decant pond cross-section, which was applied around the entire perimeter of the Cell 1 embankment. No modifications were made to the filter and clay layers previously developed and approved by the ITRB.

A preliminary quantity estimate provided by Golder compares favourably to the KCB quantity estimate in terms of total volume of materials. Planning for the project will continue with the current, more conservative KCB estimate until a design review is completed and the ITRB has approved the design modifications to implement the Golder design.

Three types of embankment sections have been developed for construction:

- Reclaim pond section – this section is in the vicinity of the reclaim pond area where a minimum three-metre-deep water pond will accumulate against the upstream face of the embankment during operations. This section was originally and continues to be designed as a downstream-constructed embankment with a low hydraulic conductivity zone (Cretaceous clay) to limit seepage. Figure 18.10 shows the similar Golder design for the typical reclaim pond section and the various material types used to construct this portion of the embankment.
- Dry section – this section will be used at the upstream end of the TSF where seepage will only occur during periods of active tailings deposition. Golder has added operational benches to support the spigot discharge line. This section is now also being downstream constructed with filter zones incorporated to prevent migration of tailings into the embankment. Figure 18.10 (lower image) shows a typical dry section.

- Wet section – this section will also be raised using the downstream approach and will incorporate filter zones to prevent migration of tailings into the embankment. The operational benches for placement of spigot pipelines have been removed; otherwise the section is the same as the dry section. Figure 18.10 (upper image) shows a typical wet section.

The foundation characterization has identified brittle clay deposits that will require varying downstream buttress berms according to the thickness of clay. Buttresses may range in width up to 500 m, and the resulting overall downstream slope of the embankments may range from 2.5H : 1V to 9.2H : 1V (horizontal to vertical).

In general, the steeper slopes are applicable to the southern/south-east part of the embankment, where foundation clays become discontinuous and bedrock is present, and the flatter slopes apply to the north-west part of Cell 2. Given that the stability of the dam on the clay foundation is dependent upon the rate of consolidation and pore pressure dissipation, an observational and monitoring approach will be used to monitor the embankment foundation and test fills to validate and refine the final geometry.

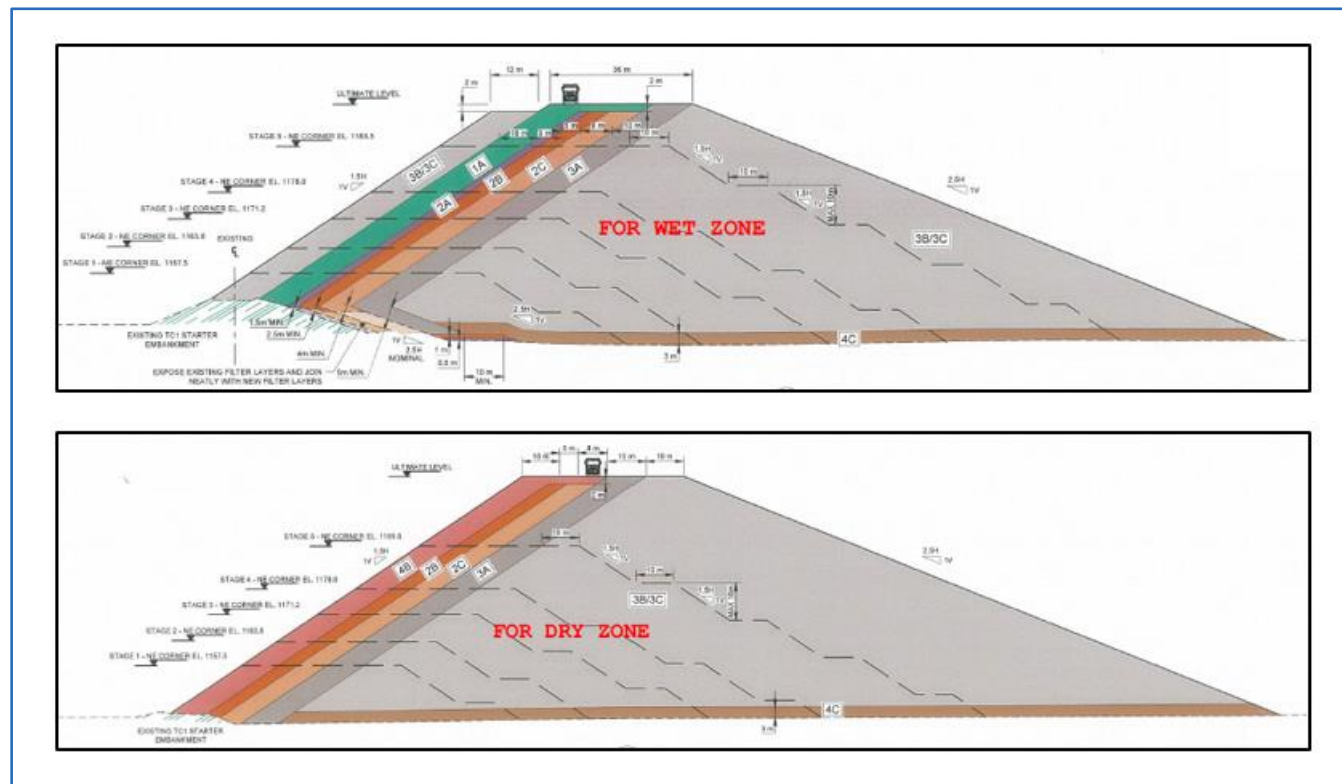
18.7.7.2 Material Quantities and Take-offs

The material quantities are based on the current plan for downstream embankment construction with a 4H:1V exterior embankment profile.

Minimum annual earthworks material quantities vary from 3.93 Mt/a to 5.52 Mt/a, decreasing during the construction of Cell 2 (year 2022) because of the reduction from four to three embankments, and other annual variations due primarily to the amount of ancillary earthworks (such as perimeter road construction) to be done.

Minimum annual waste rock material quantities vary from 7.7 Mt/a to 43.98 Mt/a, incorporating a general increase associated with increasing embankment height in each cell and a general reduction during construction of Cell 2 due to the reduction from four to three embankments.

Figure 18.10 Typical Cross-Section of TSF Embankment



18.7.8 Future Work

A routine annual work cycle has been established for ongoing construction of the tailings storage facility that covers planning, engineering, permitting, construction, and monitoring activities. In general, the work cycle commences in July of each year with a detailed review of established rates of rise and tailings beach slope angles, which information is used to predict embankment elevation requirements for the following year. Based on the review, a planning model is established that determines earthworks and waste rock quantities for the following year and which forms the basis for budgeting and detailed engineering design.

Earthworks are undertaken during the summer months (1 April to 1 December), during which time temperatures are above freezing and supportive of the required material preparation, placement, and compaction effort. Waste rock placement is year-round based on the continuing operation of the open pit mine.

Key development milestones over the life of TSF Cells 1 and 2 include:

- 2016 – installation of an additional booster pump at the tailings booster pump station to ensure pumping head capacity for tailings discharge as Cell 1 starts to exceed 1,190 m elevation. Costing and detailed planning rests with the OT Concentrator team, but this is a key milestone for ongoing TSF development. Installation of updated Cell 1 geotechnical monitoring instrumentation.
- 2020 – Cell 2 site investigation and installation of geotechnical instrumentation.
- 2021 – commencement of Cell 2 construction inclusive of embankment construction, perimeter road, and seepage drain. Start of Cell 1 rehabilitation including placement of topsoil and hydro-seeding.
- 2022 – commencement of Cell 2 operation and completion of rehabilitation and closure of Cell 1.
- 2030 – completion of Cell 2 construction and operation. Commencement of construction of future tailings disposal location (location to be determined, not yet at feasibility study level of design).
- 2031–2032 – rehabilitation and closure of TSF Cell 2.

18.8 Innovation and Technology Opportunities

OT LLC plans to investigate and implement projects in these areas: project monitoring, process technology, and underground technology. The innovations and possible applications outlined below are not exhaustive, nor definitive, but rather are currently viewed as having the most significant potential impact for Oyu Tolgoi. OT LLC plans to make use of the services of the Rio Tinto Groups' innovation and technology experience and facilities to provide expertise to take advantage of opportunities and add value to Oyu Tolgoi.

OT LLC plans a longer-term view to developing its operations management capability to maximize performance through evaluation and implementation of advanced technologies. This approach would involve the strategic evaluation and collaboration with technology partners before further developing and implementing system capabilities in a phased and prioritized manner. Experience from a variety of industries has shown this approach to be crucial in achieving an integrated system that maximizes the potential of the various technologies and the benefit to Oyu Tolgoi. The innovation and technology opportunities that Rio Tinto is currently examining are:

- Project Monitoring and Optimization
- Process Technology
- Underground Technology
- Operating Systems and Technologies
- Data Management
- Geotechnical Research
- Extraction Level Construction
- Cave Production
- Cave Monitoring

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

Oyu Tolgoi sells copper concentrates into the international export market. Rio Tinto has been appointed as the exclusive provider of marketing services to Oyu Tolgoi in the Oyu Tolgoi Management Agreement.

Shipment of Oyu Tolgoi concentrates commenced in July 2013 and reached a milestone of 1 Mt in March 2015. At present, sales are made on a 'delivered at place' (DAP) basis at the warehouse facility at Huafang in China, approximately 7 km from the Mongolia-China border, where customers pay for the copper concentrate by means of a letter of credit and take responsibility for product delivery by truck or train to the respective smelter.

Sales contracts have been agreed for approximately 95% of Oyu Tolgoi's expected 2016 concentrate production. Opportunities to place the balance of production will include additional trial shipments to new Chinese smelter customers as well as additional sales to existing customers. In 2017, close to 97% of production has been committed with existing customers.

Oyu Tolgoi's marketing priorities over the next two years include:

- Entering into agreements with buyers for production beyond 2017.
- Improving interactions between production, logistics, and marketing to optimize mine to-market alignment.
- Diversifying the customer base to prepare for increased volumes from the underground mine
- Executing trial shipments by alternative logistics routes and different modes of transport to other potential customers, including non-Chinese markets.

19.2 Product Specifications

The forecast specifications for Oyu Tolgoi concentrate are listed in Table 19.1. Product specifications and volume forecasts are updated regularly in accordance with the planned production for a given period. The Sales and Marketing group communicates and discusses any specification updates with Oyu Tolgoi's customers.

Table 19.1 Specifications for Major Components of Oyu Tolgoi Concentrate

Element	Unit	Typical
Cu	%	28
Au	g/dmt	9
Ag	g/dmt	60
As	ppm	1,267
F	ppm	586
Moisture	%	8.5

19.3 Supply and Demand Forecasts

Based on analysis by copper industry analysts, demand will continue to be supported by further, albeit slower expansion in China, continued solid growth in other emerging markets, such as India and ASEAN and modest growth in more industrialized regions such as the USA and Europe. The scope for further growth in China and elsewhere is driven by continued urbanization, industrial and infrastructure upgrades, and rising household incomes in the medium to long-term.

Scrap is expected to increase as a proportion of global copper requirements over the next 15 years. Substitution will be another important factor particularly in a lower price environment for aluminum, which is a key competing material. While easy areas for substitution have already taken place e.g. PVC for plumbing, there is a downside risk of further substitution in the future. Examples include using aluminum for automotive wiring harnesses and some industrial building wire and cable applications. It is worth noting there are also upside areas, in the form of higher copper intensity for more energy efficient electric motors, non-traditional energy sources and electric vehicles.

Medium-term supply is expected to remain constrained despite new project capacity coming online in the near term. Production at existing mines will continue to decline due to decreasing grades and resource depletion. Meanwhile, bringing on new projects (particularly greenfield projects) is increasingly hampered by the locations and characteristics of undeveloped ore deposits. Typical conditions include remoteness and high altitudes, lower grade deposits, high capital intensity and upfront capital expenditure, and social and environmental requirements and risks.

The combination of ongoing attrition at existing mines and further demand growth means the industry will need to find significant new capacity in the medium to long-term.

19.3.1 Global Copper Smelting Capacity

In line with solid demand fundamentals for mined copper, global refined copper consumption is expected to see further growth in the coming years. China and the other Asian regions are expected to continue to dominate this market.

According to copper industry analysts, global smelting capacity is forecast to increase through 2030. Asia is expected to account for more than half of global smelting capacity in 2030 and onwards in line with an increase in China's share. Asian smelting capacity is dominated by China, Japan, India, and Indonesia.

China is forecast to see the majority of growth in next few years and contribute to a significant proportion of the additional global smelting capacity by 2030. In 2016, China had 49 smelters in operation with a total capacity of 7 Mt of copper. The total capacity in China is estimated to be 8 Mt. Planned smelter developments and expansions will increase Chinese capacity to 10 Mt/a, in 2030. However, raw material constraints have historically led to low utilization rates of this capacity, which has exacerbated the regional Chinese demand for concentrate. Given that this trend is forecast to continue, the issue for China in the years ahead will be the availability of concentrates for its custom smelters as national capacity continues to grow.

Custom, or traded, concentrates are those that are mined by one company and processed by a different company, as opposed to being smelted and refined in-house at a particular copper mine. The market for custom concentrates now accounts for half of all copper concentrates produced, driven in recent years by the rapid growth of the custom smelting industry in China and to a lesser extent, India. Despite limited domestic copper resources, Chinese companies have invested heavily in smelting capacity and are highly dependent on the custom market for raw materials, followed by Japanese and Indian smelters in 2030 onwards.

The custom concentrate market is highly centralized. However, the market will show opposite trends with the emergence of new smelting capacity. The number of buyers will increase, encouraging major miners to increase their share of the custom concentrates market by investing in new mine capacity. The vast majority of near-term global production increases are expected by various sources to be in China.

19.4 Marketing Plan

The Oyu Tolgoi five-year marketing plan is endorsed by the board of OT LCC on an annual basis. It considers the following elements:

- Available production – Sales and Marketing works closely with operations to plan sales in accordance with changing production.
- Customer selection and allocation.
- Term / spot contract allocation – Portfolio management of the contract book allows for a mix of spot and term sales, depending on market conditions, production factors, and other relevant matters.
- Commercial terms – Oyu Tolgoi's commercial sales terms will continue to be in line with terms and conditions on the international concentrates market.
- Delivery basis – Currently, Oyu Tolgoi concentrates are sold DAP Ganqimaodao.

19.5 Concentrate Logistics and Export Process

Concentrate is transported in two-tonne bags. Mongolian custom clearance occurs at the marshalling yard at the Oyu Tolgoi site. Chinese customs clearance occurs at the bonded warehouse.

The current logistics model involves road transport from the mine to the bonded warehouse, located in China, 14 to 21 days' storage at the warehouse for customs clearance, and then a mix of road and rail from Ganqimaodao to the customer. Oyu Tolgoi is responsible for transportation up to the bonded warehouse, and customers are responsible for transportation thereafter.

At present, border operating hours and schedule are limited, and the border is subject to sporadic closure. Weather, communications, and energy failures all contribute to this situation. Efforts are underway on both sides of the border to increase its capacity and efficiency.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Impact Summary

OT LLC has completed a comprehensive Environmental and Social Impact Assessment (ESIA) for Oyu Tolgoi. The ESIA undertaken as part of the project finance process was publicly disclosed in August 2012. The culmination of nearly 10 years of independent work and research carried out by both international and Mongolian experts, the ESIA identifies and assesses the potential environmental and social impacts of the project, including cumulative impacts, focusing on key areas such as biodiversity, water resources, cultural heritage, and resettlement.

The ESIA also sets out measures through all project phases to avoid, minimize, mitigate, and manage potential adverse impacts to acceptable levels established by Mongolian regulatory requirements and good international industry practice, as defined by the requirements of the Equator Principles, and the standards and policies of the International Finance Corporation (IFC), European Bank for Reconstruction and Development (EBRD), and other financing institutions.

Corporate commitment to sound environmental and social planning for the project is based on two important policies: TRQ's Statement of Values and Responsibilities, which declares its support for human rights, social justice, and sound environmental management, including the United Nations Universal Declaration of Human Rights (1948); and The Way We Work 2009, Rio Tinto's Global Code of Business Conduct that defines the way Rio Tinto manages the economic, social, and environmental challenges of its global operations.

OT LLC has implemented and audited an environmental management system (EMS) that conforms to the requirements of ISO 14001:2004. Implementation of the EMS during the construction phases to focus on the environmental policy; significant environmental aspects and impacts and their risk prioritization; legal and other requirements; environmental performance objectives and targets; environmental management programmes; and environmental incident reporting.

The EMS for operations consists of detailed plans to control the environmental and social management aspects of all project activities following the commencement of commercial production in 2013. The Oyu Tolgoi ESIA builds upon an extensive body of studies and reports, and Detailed Environmental Impact Assessments (DEIA's) that have been prepared for project design and development purposes, and for Mongolian approvals under the following laws:

- The Environmental Protection Law (1995);
- The Law on Environmental Impact Assessment (1998, amended in 2001); and
- The Minerals Law (2006).

These initial studies, reports, and DEIA's were prepared over a six-year period between 2002 and 2008, primarily by the Mongolian company Eco-Trade LLC, with input from RPS Aquaterra on water issues.

The original DEIA's provided baseline information for both social and environmental issues. These DEIA's covered impact assessments for different project areas, and were prepared as separate components to facilitate technical review as requested by the GOM. The DEIA's were in accordance with Mongolian standards and while they incorporated World Bank and IFC guidelines, they were not intended to comprehensively address overarching IFC policies such as the IFC Policy on Social and Environmental Sustainability, or the EBRD Environmental and Social Policy. Following submission and approval of the initial DEIA's, the GOM requested that OT LLC prepare an updated, comprehensive ESIA whereby the discussion of impacts and mitigation measures was project-wide and based on the latest project design. The ESIA was also to address social issues, meet GOM (legal) requirements, and comply with current IFC good practice.

For the ESIA, the baseline information from the original DEIA's was updated with recent monitoring and survey data. In addition, a social analysis was completed through the commissioning of a Socio-Economic Baseline Study and the preparation of a Social Impact Assessment (SIA) for the project. The requested ESIA, completed in 2012, combines the DEIA's, the project SIA, and other studies and activities that have been prepared and undertaken by and for OT LLC.

This section provides a summary of the legal and policy framework and social impact and mitigation measures; environmental design; construction management plan; and closure and rehabilitation plans with respect to compliance with Mongolian National Standards (MNS) and conformance with international guidelines and requirements, specifically those for the IFC, World Bank (WB), and EBRD.

Where applicable, the text refers to relevant information from the updated ESIA for the project completed in 2012.

20.2 Legal and Policy Framework

The Mongolian Constitution (1992) sets out the personal rights and freedoms of the people of Mongolia, including the right to a healthy and safe environment and protection against environmental pollution and ecological imbalance (Article 16.1.2). It also describes the system of government and allocates powers and responsibilities to each branch of government.

In general, national laws are introduced and enforced by the central government. The Ministry of Environment and Green Development (MEGD) has legal authority for environmental protection legislation and regulations. The Ministry of Natural Resources and Mines (MNRM) has legal authority for mining legislation and regulations. The central government delegates some powers to provincial (aimag) and regional (soum) levels.

The ESIA (OT LLC, 2012) provides a good overview of the applicable environmental legislation and regulations currently applicable to the project.

20.3 Health, Safety, and Environmental Management System

Oyu Tolgoi has developed a comprehensive Health, Safety, Environment and Communities Management System (HSEC MS) that meets the requirements of the Rio Tinto HSEC Management System Standard and Health, Environment, Safety and Community Performance Standards. The management system is designed on the principles of continual improvement and adopts the methodology of Plan, Do, Check and Review, which comprises 17 discrete elements for implementation. The Oyu Tolgoi HSEC MS has been audited and is certified to ISO14001 and OHSAS18001.

20.4 Environmental and Social Baseline

The baseline assessment was prepared by Citrus, for OT LLC, by drawing upon the wide range of internal and independent studies that have been prepared for the Oyu Tolgoi project since 2003. The existing information was reviewed and assessed for accuracy, consistency, and validity. Where additional baseline data became available in 2010 and 2011 prior to the completion of the ESIA, these were incorporated in the publicly disclosed ESIA.

Plans for ongoing data collection and studies are set out in the corresponding impact assessment chapters and management plans of the ESIA, as well as supplementary Operational Management and Monitoring Plans, to ensure baseline data continues to improve and that the results of ongoing monitoring and improved knowledge are integrated into updated and revised management plans and procedures.

In each baseline chapter of the ESIA, the following issues were considered in order to draw together the data and provide an overview of the sources, robustness, and validity of the original data:

- Clarity of data sources with key references cited, ensuring a clear appreciation of the origin of information used.
- Identification of third-party data verification undertaken by Oyu Tolgoi.
- Clear descriptions of the methodologies used to develop the baseline data.
- Identification of where Oyu Tolgoi has commissioned further data collection, or where data collection remains ongoing.

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 covering the biophysical environment and human environment.

The biophysical environment baseline addresses the following topics:

- Climate and climate change
- Air quality
- Noise and vibration
- Topography, geology, and topsoils
- Water resources

- Biodiversity
- Ecosystem services

The human environment baseline addresses the following topics:

- Population and demographics
- Employment and livelihoods
- Land use
- Transport and infrastructure
- Cultural heritage
- Community health, safety, and security

20.5 Environmental and Social Impact Assessment

OT LLC is committed to sustainable development through effectively managing its environmental, social, and economic responsibilities. Throughout development, OT LLC's primary intent is to assess and prevent the potentially adverse effects that mining activities can have on the natural environment, and to plan appropriately for future mining operations at the Oyu Tolgoi project.

OT LLC's environmental work for the Oyu Tolgoi project is compliant with Mongolian regulatory requirements, internal policies and procedures, and external agreements. The environmental management plans for the project are designed to ensure that key environmental factors are monitored and protected.

The Oyu Tolgoi ESIA of 2012 built upon an extensive body of studies, reports, and DEIAs prepared since 2002 for project design and development purposes and for Mongolian approvals under the following laws:

- The Environmental Protection Law (1995)
- The Law on Environmental Impact Assessment (1998, amended in 2001)
- The Minerals Law (2006)

The various earlier studies and reports have been updated regularly by a number of certified Mongolian DEIA consultants, with oversight by Sustainability East Asia and input from RPS Aquaterra on water issues.

The DEIAs provided baseline information for both social and environmental issues and were generally structured under a core categorization of Mining and Processing, Transport and Infrastructure Corridor, Gunii Hooloi Water Supply, Coal Fired Heating Plant, and Airport. In addition, a number of specific DEIAs were required to address arising/updated facilities and requirements not covered under the broader categories. Examples are specific assessments for the fuel depot, use of chemicals, and the Undai River diversion.

All DEIAs were prepared and approved in accordance with Mongolian standards, but, although developed to meet international standards, they were not specifically intended to comprehensively address overarching IFC or EBRD social or environmental sustainability policies.

In preparing the ESIA, the baseline environmental and social information, assessments, and monitoring and mitigation requirements were updated with recent data and management plans that meet IFC and EBRD performance standards/requirements. In addition, a social analysis was completed through the commissioning of a Socio-Economic Baseline Study and SIA for the Oyu Tolgoi project.

Table 20.1 summarizes the previous key baseline studies and core DEIAs prepared for Oyu Tolgoi, and Table 20.2 summarizes of the supplementary DEIAs and studies.

Table 20.1 Baseline and Core DEIA Studies for the Oyu Tolgoi Project

EIA Study Title	Description	Date	Status
Environmental Baseline Study for Oyu Tolgoi	Covers geography, geological, hydrology, hydrogeology, soil, climate, air quality, flora and fauna, the socio-economic status, and infrastructure of the Oyu Tolgoi site and its surrounding areas.	2002	No approval required
Environmental Baseline Study for Town Planning	Covers geography, geological, hydrology, hydrogeology, soil, climate, air quality, flora and fauna, the socio-economic status and infrastructure of potential development and interconnecting infrastructure areas for Khanbogd town developments.	2012	No approval required
Oyu Tolgoi EIA Volume I: Transport and Infrastructure Corridor from Oyu Tolgoi to Gashuun Sukhait	DEIA of the road and power line proposal from Oyu Tolgoi to the Gashuun Sukhait border crossing. Provides approval for access through the South Gobi Strictly Protected Area (SGSPA). A number of amendments have been undertaken to address changing alignments.	2004 2006 2010 2012	Approved Approved Approved Approved
Oyu Tolgoi EIA Volume II: Water Supply from the Gunii Hooloi Aquifer	DEIAs for the proposed aquifer and water supply system for the provision of a sustainable water supply for Oyu Tolgoi. A number of amendments have been completed to capture developments in the groundwater resource assessment and water supply pipeline design.	2004 2009 2010 2012	Approved Approved Approved Approved
Oyu Tolgoi Volume III: Oyu Tolgoi Mining and Processing Facilities	DEIA of the open pits, underground mine, concentrator, tailings, and all facilities and support infrastructure located within the Oyu Tolgoi mining license Area. The assessment was largely based on IDP05, but reflected the general permitting layout of May 2006. The maximum production rate was assumed to be 85 kt/d.	2006 2012 2016	Approved Approved Approved
Oyu Tolgoi Volume IV: Coal Fired Power Plant	EIA documentation drafted for a coal-fired power plant at the Oyu Tolgoi site. An amendment has been undertaken to reflect updates in design for three 150 MW generating units.	2006 2011	Not submitted Approved

Table 20.2 Supplementary DEIA studies for Oyu Tolgoi Project

Project EIA Component	Description	Date	Status
Fuel Station Facility	DEIA for the fuel facility built in 2004 within the mining license Area. Amendment completed for extension of the fuel depot.	2005 2010	Approved Approved
Shaft 1	DEIA for Shaft 1, including headframe facilities, waste rock, and water disposal.	2005	Approved
Shaft 2	DEIA for Shaft 2, including headframe facilities, waste rock, and water disposal.	2006	Approved
Diesel Power Station	DEIA for the diesel power station located within the Mine License Area.	2007	Approved
Waste Water Treatment Plant	Supplementary DEIA for the construction camp waste water treatment plant expansion to 4,000-person equivalent capacity.	2007	Approved
Quarry Batch Plant and Quarry	DEIA of hard rock quarry, concrete batching plant, and crusher located at the northern boundary of the mining license Area.	2007	Approved
20 MW Diesel Power Plant	The assessment included the initial development of six 2 MW diesel power stations followed by a Stage 2 addition of four 2 MW diesel generators.	2007	Approved
Chemicals	Covers the importation, storage, use, and disposal of chemicals. Amendments have been undertaken to update chemicals being used in construction, commissioning, and operation.	2008 2011 2012	Approved Approved Approved
Javkhlant EJV Area	DEIA for future project facilities, infrastructure, and Heruga underground mine located within the Javkhlant EJV area.	2009	Approved
Shivee Tolgoi EJV Area	DEIA for project facilities, infrastructure, and portion of the Hugo Dummett underground mine located within the Shivee Tolgoi EJV area.	2009	Approved
Main Fuel Storage Facility	DEIA for the main fuel storage facility within the OT License.	2011	Approved
Permanent Airport	DEIA for permanent airport	2011	Approved
Undai River Diversion Detailed Environmental Impact Assessment	DEIA for diversion of the Undai River.	2011	Approved
Oyu Tolgoi to Khanbogd Power Line	Covers the development of a 35 kV power line connecting Oyu Tolgoi to Khanbogd. DEIA has been developed, but approval was obtained based on a DEIA screening submission.	2012	Approved (based on screening submission)
Waste Management Centre	DEIA for OT waste management centre	2015	Approved
Tailings Storage Facility	DEIA for Tailings Storage Facility	2015	Approved
Gashuun Sukhait Road Zone 3 and Zone 5	DEIA Zone 3, Zone 5, and Diversion of GSK road	2012 2015	Approved
Khanbogd to Oyu Tolgoi to Javkhlant Road	DEIA for paving road	2016	Approved

20.5.1 Future Project Elements Not Directly Addressed in the ESIA

In addition to the project elements identified above, certain other activities and facilities are expected to be developed over time, either as part of or in support of the project, which do not constitute part of the project for the purposes of the ESIA. These include:

- Project expansion to support an increase in ore throughput from 100–160 kt/d.
- Long-term project power supply. Under the terms of the IA, OT LLC will source electricity from within Mongolia. OT LLC may develop a coal-fired power plant within the Oyu Tolgoi mining license area to provide the required power from Mongolian sources.

While the impacts and management of these future project elements are not directly addressed in the ESIA, they are considered in the cumulative impact assessment.

20.5.2 Scope of Impact on Communities and Community Members

The potential impacts of the project on the human environment may extend farther than the direct physical impacts of the project on the biophysical environment. As a result, the Project Area of Influence (for the purposes of the ESIA) also extends to include:

- Communities and community members that will be directly affected by the project in ways that are foreseeable and within the reasonable control of the project during construction, operations, and closure. This includes economically and physically displaced persons such as herder families whose winter camps and access to traditional summer pastures may be affected by the project.
- Communities and community members that may be directly affected by population influx, e.g., effects on water supply, wastewater, solid waste, housing, and other public services or facilities, including Khanbogd soum and Khanbogd soum centre.
- Communities and herder households that may be affected by potential changes to local and regional groundwater supplies in the Gunii Hooloi basin, downstream of the Oyu Tolgoi site, and along transportation corridors where access to, and water supplies in, established herder wells may be affected.

20.5.3 Cumulative Impacts

Cumulative impacts are defined by the IFC as the combination of multiple impacts from existing projects, the proposed project, and/or anticipated future projects that could result in significant adverse and/or beneficial impacts that would not be expected in the case of a stand-alone project.

Geographical areas, communities, and regional stakeholders could be subject to cumulative impacts from further developments at Oyu Tolgoi together with other existing or planned projects, trends, and developments within the South Gobi region. These will include:

- Macro-economic impacts across the Mongolian economy.

- Impacts on communities and infrastructure in the South Gobi region related, for example, to influx, economic changes, and pressure on infrastructure. Specifically, within Ömnögovi aimag, this includes the soums of Khanbogd, Bayan Ovoo, Manlai, and Tsogttsetsii and the aimag capital, Dalanzadgad.
- Biodiversity impacts related to the fragmentation of ecosystems by roads and other infrastructure.
- Impacts on water resources in terms of both shallow aquifers for herder water supplies and deep aquifers for potential industrial water supplies.

The ESIA addresses a project with a 27-year design life. It is anticipated that the project will continue in operation well after that date, possibly at higher production rates. Such plans are still at an early-stage, so while they are referred to in the ESIA, they are not evaluated in the ESIA because of the limited amount of information available.

Similarly, a number of future developments of project-associated facilities are still under evaluation, and no clear decision has yet been made as to the preferred approach to be adopted by the project. These include:

- the rail link that is currently being constructed from the South Gobi into China for the export of coal from the Tavan Tolgoi area. Oyu Tolgoi may propose the construction of a spur line from the Oyu Tolgoi site to join the rail link. This would have significant impacts in terms of reducing the volume of heavy vehicles transporting concentrate from Oyu Tolgoi and coal from other mines into China.
- The future power supply for the project. OT LLC sources its present power under a four-year contract with a Chinese provider, the IMPIC via the NPTG authority. In May 2016 the parties agreed to extend the power supply agreement to at least 2021, after which Oyu Tolgoi is required to source electricity from within Mongolia.

20.6 Environmental Impacts and Mitigation Measures

20.6.1 Climate and Air Quality

The key issues in terms of potential impact to air quality include:

- Dust emissions, together with their impact on human health and their potential to cause nuisance to those exposed.
- Emissions of potentially polluting gases: sulfur dioxide (SO₂), oxides of nitrogen (NO₂), and carbon monoxide (CO), and their potential impact on human health.
- Emissions of other potentially hazardous species, including hydrochloric acid (HCl), dioxins and furans, cadmium (Cd), lead (Pb), mercury (Hg), hydrogen fluoride (HF), and their potential impact on human health.
- Emissions of greenhouse gases (GHGs) (principally CO₂).

20.6.2 Noise and Vibration

Those receptors considered relevant to project noise include:

- Worker accommodation facilities within the mining license, which will be used during the operational and decommissioning phases of the project.
- Established and permanent winter herder camps, although these are farther than 10 km from the mining license. The herder winter camp resettlement programme undertaken to move herders away from the construction activities associated with the project was completed in 2004. Oyu Tolgoi has since developed new winter camp locations and wells for relocated herder families that previously had established nomadic camps within 10 km of the project site.
- Local population in the vicinity of project activities outside the mining license area, including the infrastructure corridor to the south, the airports, and the borefield and its associated infrastructure, although the borefield operation is not expected to be a significant source of noise.

Khanbogd, the nearest soum centre, is 35 km to the north-east of the mining license and is therefore not expected to be affected by noise during operations. Should this position change with, for example, future developments connected to the project, all necessary studies and approvals will be undertaken in accordance with the Project Standards.

Through a review of the project design basis, construction schedule, scoping and baseline assessment, the key noise and vibration issues considered to be associated with the project include:

- Noise impacts on herders from the operation of the Oyu Tolgoi to Gashuun Sukhait road.
- Noise impacts on herders from the use of the permanent airport.
- Noise impacts on workers within the mining license.

The screening assessments and acoustic modelling undertaken for the ESIA have demonstrated that, with appropriate mitigation, the noise and vibration impacts of the project will be negligible.

Aircraft noise will be noticeable for several kilometres from the airport. Modelling and monitoring of noise at the airport have shown that these impacts are minor, of short duration, and limited to daytime when a flight is landing or taking off.

20.6.3 Topography, Geology, and Topsoils

Actual and potential impacts on the topography, geology, and topsoil arising from the construction, operation, and closure of the project are as follows:

- Construction of mine infrastructure, including the tailing storage facilities (TSF) and waste rock dump (WRD) areas.
- Impacts associated with the open pit.
- Block caving mining activities, resulting in a surface subsidence zone.

- Creation of structures such as the camps and shaft headframes.
- Diversion of the Undai River and other ephemeral watercourses.
- Losses of topsoil from erosion by wind and water around earthworks, topsoil stockpiles, and rehabilitated areas.
- Potential subsidence impacts of the area overlying the Gunii Hooloi aquifer.

Impacts during closure will relate to legacy issues associated with the open pit, block caving, TSF, WRD, and potential settlement associated with drawdown of the deep aquifer that will have been used to supply the project's operational water requirements. While the scope here discusses closure at the end of the project, Oyu Tolgoi also considers the potential for early or forced closure every year. The Closure Plan, which was prepared based on the June 2012 design status was updated in 2014 to reflect the OTFS scope.

By means of careful design and planning processes, Oyu Tolgoi aims to prevent and mitigate, as far as practically possible, impacts on topography, landscape, geology, and topsoil. The key design measures taken to avoid impacts are listed below.

- As areas are decommissioned, e.g., the closure of the first cell of the TSF, progressive rehabilitation and landscaping will be undertaken while under OT LLC control, allowing vegetation to become established without any impacts from grazing by herd animals, which will be kept out by the mining license fence.
- The WRD areas will be rehabilitated as soon as feasible. Stored topsoil will be used to rehabilitate the lower slopes of the waste rock facilities where the risk of losing the topsoil through windblown erosion is limited. The WRD area will not be used by underground mining operations; at the end of the operational life of the open pit, the remaining active areas will be stabilized and rehabilitated. This rehabilitation will be carried out concurrently with the underground block caving, with the aim of having the WRD rehabilitation completed a significant time (decades) before the end of the mine life.
- Mitigation of the impacts on the topography and landscape will focus on the final design of the WRD, preferably resulting in a similar profile to the Khanbogd Mountain or the steep-sided Javkhlant Mountain.
- Landscaping will be considered to lessen the visual impact of buildings. Per Oyu Tolgoi's obligation under the Land Law, this will include assigning over 10% of the built areas within the mining license as a green zone. Plants are currently being grown at a nursery in Khanbogd near a local well. These saplings will be transplanted to the mining license area and watered until established.

20.6.4 Water Resources

Based on an appraisal of the baseline conditions and sensitivities discussed in Chapter B6: Water Resources of the ESIA, the following key issues were identified:

- The impact of the various elements of the project on surface water systems, including ephemeral watercourses and ephemeral and permanent springs. These impacts could affect water quantity, quality, or the length of time an ephemeral watercourse sustains surface or groundwater flows over the course of a year.

- The impact of the project's water demand on the deep aquifer water resources of Gunii Hooloi and its significance to the overall water resources of the region and impacts on potential future water use.
- The impact of the project on shallow aquifer resources across the Project Area of Influence.

These key aspects of surface water systems, deep groundwater resources, and shallow groundwater resources are influenced by a variety of interactive impacts, including:

- Impacts on flora and fauna due to potential disruptions to ephemeral surface water and groundwater flows and springs, and access to these water sources by wildlife.
- The impact of the diversion of the Undai River on downstream springs and water users and risks to the Undai diversion from the adjacent WRD and other infrastructure, including the quality of the water that drains from these features.
- The impact on surficial aquifers and local ephemeral watercourses of water abstraction for construction water supply and of dewatering associated with excavation and operation of the open pit and underground workings.
- The impact of water abstraction for construction purposes from locations outside the mining license.
- The impact of the abstraction of deep aquifer water resources, which are generally non-potable, on shallower potable water aquifers, including the surficial aquifers along the ephemeral watercourses used by herders and wildlife.
- Impacts on herder water supplies, particularly at their winter camps, which are critical for the maintenance of their livelihoods.
- The impact of the increased water demand due to the known and predicted future increase in population of Khanbogd soum centre.

The impacts on the aquifers in the Gunii Hooloi basin are influenced by the project water demand. Minimizing water demand is a KPI that has been at the forefront of the design of the project. OT LLC is engendering a culture of water conservation throughout the company, its employees, and contractors and has set targets for water conservation and recycling at all levels of the project. Examples include choosing sanitary ware that reduces water consumption through to using treated and recycled wastewater in the concrete batch plant and concentrator.

20.6.5 Biodiversity and Ecosystem Services

In 2011, OT LLC implemented a Regional Biodiversity Assessment (RBA) as part of the risk assessment process for impacts on various identified biodiversity features. The risks were categorized as critical, high, medium, and low. Proposed mitigation options for the potential impacts on specific biodiversity features are addressed in Chapter D6: Flora and Fauna Construction Management Plan of the project ESIA.

The detailed hydrological and hydrogeological investigations undertaken to date have given OT LLC a good appreciation of where the most likely impacts on surface hydrology and surficial aquifers will occur and of how to develop mitigation measures to address them (Chapter C5: Water of the ESIA).

There are, however, a number of uncertainties relating to the likelihood and magnitude of project-induced impacts on biodiversity features. These uncertainties relate to the likelihood for impacts to shallow water systems and associated biodiversity features resulting from drawdown of the deeper regional aquifer and from mine dewatering. Although the RBA process categorized the likelihood of impacts as low, and thus a risk assessment of medium/low, the following hydrological items are still considered to retain some degree of inherent uncertainty in impact risk assessment:

- Effect of dewatering on the mine surface area.
- Maintaining surface water and groundwater flow in the Undai River.
- Connectivity between the deep Gunii Hooloi aquifer and overlying surficial and alluvial aquifers and the deep Galbyn Gobi aquifers.
- Connectivity between the deep Durulj Mount Southern aquifer and overlying surficial aquifers.

Ecosystem services is a concept that cuts across conventional approaches of looking at issues from a subject-specific basis and enables the interactions between the biophysical and human environments to be identified. Most, if not all, of these issues, impacts, and associated mitigation actions are addressed within either the Biodiversity Mitigations or other chapters of the ESIA.

20.6.5.1 Mitigation Measures

Biodiversity Impacts

Mitigation actions have been developed for all potential critical and high risk impacts to priority biodiversity features.

Actions OT LLC will take to mitigate low and moderate risk impacts to priority biodiversity features, or impacts to lower priority biodiversity features, are described in Chapter D6: Flora and Fauna Construction Management Plan of the ESIA, and in various other management plans, including:

- Water Resources Management Plan
- Atmospheric Emissions Management Plan
- Land Use Management Plan and related Land Disturbance Procedures
- Non-Mineral Waste Management Plan
- Hazardous Materials Management Plan

Oyu Tolgoi's Biodiversity Offset Strategy

The Oyu Tolgoi project has committed to a goal of Net Positive Impact (NPI) on biodiversity. As such, residual impacts on priority biodiversity features will be offset to achieve this goal. The Biodiversity Offset Plan for the project outlines the agreed Offset programmes. This is supported by the Biodiversity Monitoring and Evaluation Plan, which outlines the monitoring, evaluation and adaptive management approach being adopted to manage Biodiversity and deliver on commitments.

Oyu Tolgoi will have unavoidable residual impacts on biodiversity. Residual impacts were predicted for 15 priority biodiversity features, two ecosystems, and three priority habitats known or likely to occur in the Project Area of Influence. Predicted residual impacts include direct habitat loss, indirect habitat loss, direct mortality (from collision with and electrocution by powerlines), indirect mortality (from increased hunting and collecting and increased numbers of natural predators) and fragmentation owing to linear infrastructure. Conservation of the Asiatic Wild Ass is recognized as the highest priority for Oyu Tolgoi, given the international importance of the Southern Gobi region to this rapidly declining, globally endangered species and the potential residual project impacts in the region. OT LLC, in consultation with stakeholders, has selected four offset programmes to respond to these impacts:

1. Reduced illegal hunting and collecting.
2. Improve rangeland conditions through sustainable cashmere initiative.
3. Reduced impacts of non-project powerlines through introduction of National Mongolian powerline standard and powerline insulation.
4. Railway fence removal.

20.6.6 Land Use and Displacement

Displacement impacts arising from the construction, operation, and closure phases of the project are as follows:

- Total physical displacement of herder households from the mining license area and displacement of winter camps from a 10 km residential exclusion zone around the mining license.
- Economic displacement of herders affected by reduced access to and/or loss of summer pastures due to land taken for the airport.
- Division of pastures caused by the construction of linear project components, including the Oyu Tolgoi-to-Gashuun Sukhait Road and the water supply pipeline (construction corridor).
- Disruption to herding activities.
- Loss of wells and other impacts to water availability/quality, e.g., impeded access to wells.
- Overall reduction of pastureland in Khanbogd soum, leading to increased competition for grazing and over-use of remaining grazing land.

20.6.6.1 Project Land Requirements

OT LLC will require approximately 10,500 ha of land to construct and operate the mine and ancillary facilities. This includes land for the mining license area that was granted in 2009; and additional land required for the TSF, concentrator, batch plant, airports, Gunii Hooloi borefield and water pipeline, and transport/infrastructure corridor between Oyu Tolgoi and Gashuun Sukhait.

Land will also be temporarily disturbed during the construction phase for activities such as the installation of worker construction camps, excavation of borrow pits, and soil stripping along the water pipeline and transmission line corridors.

20.6.6.2 Summary of Residual Impacts to Land Use

Significant residual land use and displacement impacts after the implementation of mitigation measures by Oyu Tolgoi will include:

- Herder resettlement and associated changes to herding activities and livelihoods as a result of physical and economic displacement.
- Reduction in the overall quantity of grazing land available for local pastoralists.
- Division of pastures for some herder families, affecting seasonal migration routes and access to grazing land and other resources.
- Increases in mining-related employment and new and diversified land- and non-land-based income opportunities and associated increases in income and household wealth.
- Improved pastureland management within the soum, including positive changes to livestock raising and production, and thus rural livelihoods.

The overall impact of physical and economic displacement of herders is significant and will require resettlement with well-designed and -implemented livelihood restoration and pastureland management programmes.

The loss of pastures and other land use changes is likely to be adverse for local herders in the short term, but effective implementation of the Resettlement Action Plan is expected to potentially result in benefits for affected herders in the long-term from increased income generation opportunities, education, training assistance, and other regional community development programmes being implemented by OT LLC.

20.6.7 Heritage

From early in the EIA process for the various project elements, OT LLC has attempted to design out impacts to archaeological heritage. Wherever possible, changes have been made to the location of fixed project elements and the design of project linear features, such as the roads and the water pipeline, in consideration of archaeological findings.

Mitigation measures are summarized below.

- Realignment of the Oyu Tolgoi-to-Gashuun Sukhait Road – Archaeological investigations for the original (2004) Oyu Tolgoi-to-Gashuun Sukhait transport corridor identified significant archaeological findings, which was one of several factors leading to the realignment of the road. Further surveys were conducted for revised routing options in 2006 and again in 2010 and 2011.
- The 2010 investigations discovered five burial sites that were subject to rescue excavations in early-2011. No additional sites along the corridor were found to be at risk of road construction impacts during the most recent survey. As a result, potential impacts are considered to have been avoided.
- Alignment of the Gunii Hooloi Water Supply Pipeline – a number of possible grave sites were identified close to the Gunii Hooloi borefield water-gathering pipelines and production boreholes (OTS-1, OTS-2, and OTS-3). As a result, the main pipeline alignment and connection network were designed to avoid damage to the graves.

No significant finds were encountered within the footprint of either the temporary, or permanent airports, and therefore no design changes were made on the basis of archaeological significance. Further, as part of land disturbance activities, a cultural heritage assessment is required as part of the approval process.

20.6.7.1 Summary of Impacts to Cultural Heritage

Considering that the project is at an early-stage of operation, many of the impacts on cultural heritage have already been realized in relation to land disturbance, topsoil stripping, and construction of new access roads and associated borrow pits. Existing and possible future impacts are listed below:

- Physical loss of tangible heritage (physical resources) from physical land disturbance associated with the construction phase of the project.
- Indirect disturbance of tangible heritage through the operation of construction vehicles and machinery, operations vehicles, dust deposition, and vibration effects.
- Damage and/or deliberate disturbance of heritage by project workers and/or incomers to the region.
- Loss of intangible heritage over time as the patterns of work, kinship, worship, and sources of income change; this includes the loss of "traditional livelihoods" as herders transition from subsistence to wage-based employment.

During the operations phase, the potential for permanent physical disturbance of archaeological and paleontological sites is expected to be low, since the scale and intensity of additional earthworks and engineering activities will be limited in comparison to the construction phase. Similarly, further impacts during the decommissioning phase are expected to be limited because any further major ground disturbance is unlikely.

However, some levelling, landscaping, contouring, and other land-based activities do have the potential to result in further minor archaeological impacts. Should these activities remain within the existing disturbance footprint, no significant impacts are expected. Nevertheless, predicted changes to traditions and the traditional way of life are considered long-term effects.

20.7 Environmental and Social Management Plans

The development of environmental and social management plans for the project has been a two-stage process:

- Stage 1: Construction Phase Management Plans to control the health, safety, environment, and social (HSES) management aspects of day-to-day construction activities, were initially developed based on the outcomes of the ESIA and were used as the basis of management during the project construction phase, which ended in December 2012.
- Stage 2: Operations Phase Management Plans have been developed and implemented that control the HSES management aspects of all project activities since the completion of construction in December 2012 and the commencement of mining operations.

The management plans address the management of health, safety, environment, and social aspects associated with the project. The management plans form part of the OT Integrated Health, Safety, Environment and Community Management System (HSEC MS). The scope of coverage for these plans is defined as follows:

- The physical and biological environment as may be affected by activities associated with on-site and off-site project facilities, including biodiversity, air quality, water resources, waste management, transportation, emergency response, and mine closure and rehabilitation.
- Occupational health and safety (OHS) issues that are managed under the contractor engagement, transport management and health and safety plans.
- Social and community matters, which are managed under the Oyu Tolgoi HSEC MS. This is aligned with the Rio Tinto Communities Standard. Social matters include those related to community relations, consultation and stakeholder engagement, labour/worker management, community health and safety, influx management, resettlement, cultural heritage, regional development, and other potential socio-economic impacts of the project on third parties.

20.8 Progressive Rehabilitation and Closure Planning

Progressive rehabilitation and planning for closure is a critical and integral part of the business process and demonstrates a commitment to sustainable development. Progressive rehabilitation involves the continuous technical and biological rehabilitation of disturbed areas following completion of works activities. Closure planning involves the development of strategies to avoid or mitigate potential environmental and social impacts associated with closure to the extent that is financially appropriate.

20.8.1 Progressive Rehabilitation

The progressive rehabilitation planning that forms part of the Oyu Tolgoi Closure Plan adheres to all regulatory requirements of the GOM and industry best practices as stated in IFC and EBRD performance standards and the Rio Tinto Closure Standard (Rio Tinto, 2009).

Progressive reclamation will be performed on any areas of the mine site where it is deemed practical to do so and with consideration of the need to preserve future mine expansion options. Disturbed areas that are no longer used in the active operation will be technically and biologically rehabilitated concurrently with ongoing mining operations, as practicable.

Significant progressive rehabilitation opportunities exist for the technical and biological rehabilitation of disturbed land following completion of construction. Examples include:

- Historically used temporary airport.
- Historically used borrow pits and quarries used to support construction activities (e.g., of airport, Oyu Tolgoi to Gashuun Sukhait road).
- Land disturbed areas following completion of installation of underground pipelines (e.g., for the Gunii Hooloi raw water supply pipeline).
- Land disturbed areas following completion of drilling activities (e.g., from exploration).

- Historically used tracks and access roads (e.g., the diversion road used during construction of the Oyu Tolgoi to Gashuun Sukhait road, 220 kV transmission line and temporary roads).
- Any other areas that have been subject to land disturbance during construction, but which are no longer used.

Opportunities for further progressive reclamation related to the underground and open pit mines or surface facilities and infrastructure are more limited. The main ones presented in the Closure Plan relate to the following:

- Any facility that is either redundant or no longer in use will be decommissioned and closed.
- Landfill cells and lagoons at the waste management centre will be progressively maintained during operations by placing a rock cover over the waste every day. The waste disposal grounds have a life of up to 30 years, after which the cell will be closed and a second phase initiated if necessary.
- The waste rock dumps are currently planned to be progressively reclaimed, as practicable, over the mine operating life. The dumps will be constructed with set-back berms so that the slopes can adhere to the final stable configurations during operations. Completed parts of the dump will be covered with a suitable thickness of NAF material during operations, followed by progressive placement of topsoil/growth media and revegetation, where possible.
- Cells 1 and 2 of the TSF will be the first to be completed and will be allowed to desiccate over time. Alternatively, a NAF cover may be constructed progressively over PAF tailings surfaces, taking into account the potential to re-use those cells for tailings deposition in the future. In the meantime, to mitigate environmental impacts due to inactive and dry tailings surfaces, interim reclamation measures may include the placement of a thin layer of NAF cover to protect against wind erosion and maintain surface water run-off quality.

There are potential opportunities for local communities and herder groups to participate in the implementation of several progressive rehabilitation measures that could result in economic benefits and capacity development for those involved. The Closure Plan outlines some of these opportunities.

20.8.2 Closure Planning

The Oyu Tolgoi Mine Closure Plan for OT LLC, was completed in June 2012 and updated in 2014, based on the design status at that time. The Closure Plan documents the outcomes of an order-of-magnitude closure study conducted with the following objectives:

- Compliance with the Rio Tinto corporate Closure Standard.
- Compliance with relevant Mongolian national standards.
- Compliance with relevant international guidelines and directives.
- Documentation of closure vision, objectives, and targets.
- Early development of strategies to meet closure objectives and targets.
- Early identification of likely site-specific closure issues and assessment of risks.

- Identification of action items that should be conducted to manage and mitigate risks and enable efficient and effective closure methods and technologies in the future.
- Preparation of a preliminary closure schedule based on current information.
- Estimation of costs associated with the closure, developed to an intended accuracy of an order-of-magnitude study.
- Development of a multi-disciplinary information resource.

20.8.3 Post-closure Monitoring

A large number of parameters will be monitored during the closure and post-closure phases of the mine, to characterize both physical and chemical stability of the project area and the environmental impact of the project.

Physical stability monitoring at the site will cover the following facilities, structures, and features:

- The open pit and future subsidence area
- Mine site and disturbed areas
- Waste rock dumps and TSF
- Undai River diversion
- Site security features

The Closure Plan describes the post-closure chemical stability monitoring at specific facilities, such as the following, in more detail:

- Open pit and subsidence area
- Mine site and disturbed areas
- Waste rock dumps
- Tailings storage facility
- Undai River diversion
- Gunii Hooloi borefield (water levels)

20.9 Water Management

20.9.1 Water Conservation

Minimizing water use throughout all the operational aspects has been a key focus of attention during mine planning and design. As examples of water conservation planning, the following initiatives have been implemented:

- Reuse of cooling water – The process plant is the largest consumer of water. Within the plant, all water discharged from the cooling systems, still categorized as clean water, is sent to the process water pond for reuse in the concentrator.
- High-efficiency tailings thickeners – The tailings thickener at Oyu Tolgoi uses advanced techniques and is able to achieve a tailings solids content of 60%–64%, which significantly

reduces the amount of water sent to the TSF. These design modifications help to greatly reduce the amount of reclaim water released and evaporative losses from the TSF.

- High-efficiency TSF reclaim – The TSF has been designed so that tailings are deposited in discrete cells, rather than broadly across the facility, to reduce evaporative losses. The entire base of the TSF rests on natural or installed clay and includes a comprehensive seepage collection system to minimize seepage losses. The TSF reclaim system has been designed to ensure that all supernatant water and collected seepage is returned to the process plant for reuse.
- 100% mine water recovery – All water encountered in the underground and open pit mines is recovered for use as process water or for dust suppression. Recovery of mine water helps to reduce site demand for raw water from the Gunii Hooloi aquifer. From a water balance perspective, this mine recovery water has conservatively not been included as inflow.
- 100% treated wastewater reuse – All treated wastewater produced in the site wastewater treatment plant will be reused in the process plant or for dust suppression.
- 100% truck wash water reuse – A comprehensive water treatment system has been installed at the project mine truck washing facility to allow all truck wash water to be continuously recycled and reused.
- Lagoon floating cover – A floating cover is placed over the entire raw water storage lagoon to eliminate evaporative water losses and to limit the accumulation of dust within the lagoon.
- Selection of low or zero water use equipment – Examples of this include the selection of air cooling systems at the hazardous waste incinerator and central heating plant.

Ongoing attention to water conservation will be maintained during operation through the continuous review of key performance indicators for water use and implementation of additional water conservation measures.

20.9.2 Ongoing Work Programmes

Ongoing work programmes for developing water resources include:

- Gunii Hooloi Resource – Updated hydrogeological model based on monitoring response of the initial three years of operation.
- Khanbogd Resource – Clarification of aquifer reserve approval for the eastern part of the aquifer.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

21.1.1 Capital Cost Summary

The capital costs in 2016 OTTR are from the estimates in OTFS16. The capital cost estimate represents the overall development and includes the Hugo North underground mine, supporting shafts, the concentrator conversion project, and the infrastructure expansion project. The capital estimate also includes the costs associated with the EPCM and Owner's project execution teams.

The total estimated capital cost to design, procure, construct, and commission the complete expansion, inclusive of an underground block cave mine, supporting shafts, concentrator conversion, and supporting infrastructure expansion, is US\$4.635b. All costs in 2016 OTTR are expressed in real Q4'16 US dollars. Timing of expenditures in 2016 OTTR assume a start date of 1 January 2017 and that 2016 expenditures are sunk costs. The 2016 OTTR estimates include contingency and constant real exchange rates and do not include escalation. The original OTFS16 cost estimate was based on nominal Q4'15 US dollars.

The estimates for each major component cover:

- The direct field cost of executing the project
- Indirect cost associated with the design, construction, and commissioning of the new facilities
- Mongolian customs duties, Mongolian VAT
- Some allowances for contingency.

The total 2016 Reserves Case capital costs are shown Table 21.1, these cost are the costs from 1 January 2017. Total OTFS16 Reserves Case direct capital costs, including the estimated costs for 2016, are shown in Table 21.2.

Table 21.1 Total Project Capital Cost – 2016 Reserves Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.564	1.564
Underground	2.240	3.065	5.304
Concentrator	0.145	0.149	0.293
Infrastructure	0.354	0.230	0.585
Tailings Storage Facility (TSF)	–	0.912	0.912
Subtotal	2.739	5.920	8.659
Construction Indirect	0.858	–	0.858
Contractor Execution – EPCM	0.310	–	0.310
Owner Execution	0.429	0.168	0.597
GOM Fees & Charges – Mongolian VAT	0.298	0.669	0.967
Total	4.635	6.756	11.391

Notes:

1. Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. The 2016 Reserves Case total capital cost excludes capital costs for the year 2016. Expansion capital for 2016 excluded is US\$0.46b.

Table 21.2 Total Project Capital Cost – OTFS16 Reserves Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.601	1.601
Underground	2.442	3.065	5.506
Concentrator	0.145	0.161	0.306
Infrastructure	0.404	0.234	0.638
Tailings Storage Facility (TSF)	–	0.938	0.938
Subtotal	2.991	5.999	8.990
Construction Indirect	0.902	–	0.902
Contractor Execution – EPCM	0.374	–	0.374
Owner Execution	0.501	0.181	0.682
GOM Fees & Charges – Mongolian VAT	0.326	0.678	1.004
Total	5.093	6.858	11.952

Notes:

1. Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. The OTFS16 Reserves Case total capital cost above Includes capital costs for the year 2016. The additional total 2016 expansion and sustaining capital costs are as follows: Direct Costs US\$0.331b; Construction Indirect US\$0.044b; Contractor Execution – EPCM US\$0.063b; Owner Execution US\$0.085b; and GOM Fees & Charges – Mongolian VAT US\$0.037b. The total of 2016 additional capital cost is US\$0.561b.
3. Costs shown are real costs not nominal costs.

21.1.2 Scope of Estimate

21.1.2.1 Project Execution Plan

The project execution plan key outputs include:

- Project management and delivery strategies.
- Contracting plan and list of major installation packages.
- Level 1 Project Master Schedule.

In summary the PEP management plan entails the following strategy:

- The Owner's team will be directly responsible for the overall programme management and will establish the project governance, overall execution plan, systems and procedures to be adopted across the project to ensure the overall business drivers are delivered.
- The Owner's team will manage overall project interfaces between the project and external stakeholders along with internal interfaces between the mining contractors, EPCM, and existing site operations.
- An Owner's team will focus on the execution of the underground mine development, conveyor-to-surface (C2S) decline development, and shaft excavation. The Owner's team

will comprise Owner's team personnel (OT LLC and RT) and service providers.

- OT LLC Operations will provide common services to Owner's where capability exists, such as IT Infrastructure, Finance, Procurement, HR, HSE and Training.
- An internationally recognized EPCM company will be engaged to deliver the capital portion of Owner's, excluding the underground development, C2S decline, and shaft sinking activities.

21.1.2.2 Underground Mining and Shafts

The scope in this area covers the following:

- Surface Construction – This includes the design and construction of underground mine surface support facilities such as the mine dry, overland conveyors, and supporting utilities, but not shaft-sinking or equipping of the shafts.
- Shafts 2, 3, 4, and 5 – The scope of work for the shafts is defined largely by Issued-for-Construction design, and pricing is from awarded contracts and purchase orders or firm bids. Capital costs for the shafts include the detail design and construction of all structures, utilities, materials, equipment, shaft-sinking as well as all associated indirect and management costs, and contractor and engineering support to commission the facilities.
- Underground Construction – This includes design and construction of all underground facilities including crushing, materials handling to the surface portal transfer station, underground workshops and offices, and supporting utilities.
- Underground Development – This includes the horizontal and vertical development for underground mine access and ventilation as well as the mass excavations for receiving the constructed facilities. Shaft logistics, waste rock handling, drawpoint construction, and haul road construction are also included. Mine development crew numbers will increase over time as the constructed underground ventilation system is progressively commissioned.
- Capitalized operating costs – This includes capital construction and development proceeding to first ore production. As OT LLC owns the development equipment, the capitalized operating costs include maintenance as well. There will also be capitalized operating costs for mine management, technical services groups, administration, safety, and training activities, hoisting, haulage, equipment and other costs prior to first underground ore production.

21.1.2.3 Concentrator Conversion

Conversion of the Phase 1 concentrator to efficiently process underground ore includes the following:

- One ball mill
- One rougher flotation line
- Six flotation columns
- One concentrate thickener
- Two concentrate filters
- Four concentrate bagging modules
- Associated minor equipment
- Engineering and other indirect services to support the conversion.

21.1.2.4 Infrastructure Expansion

The scope in this area covers the following:

- Central heating plant expansion: two 29 MW coal-fired boilers and two 7 MW diesel fired backup boilers
- Operations camp expansion
- Operations warehouse expansion
- Expansion of three logistics centres at Oyu Tolgoi, Gashuun Sukhait, and Hua Fang
- Extensions of related backbone utilities
- Engineering and other indirect services to support the above scope.

21.1.2.5 EPCM Services

The scope of EPCM services covers the following:

- Refurbishment of existing site concrete batch plant to operate throughout Phase 2
- Construction warehouse mobile equipment
- Project management of the surface and underground facilities (excludes shaft-sinking and lateral development activities), including:
 - Engineering management
 - Project control services
 - Contract administration
 - Materials management
 - Construction management
 - No-load commissioning.

21.1.2.6 Owner's Costs

The scope in this area covers the following:

- Overall programme management of the complete Phase 2 works
- Government permit applications
- Customs / border management
- Construction insurances
- Interface management with the Operations group
- Overall engineering and construction management of the underground lateral and vertical development, including underground mass excavations and shaft-sinking.

21.1.3 Estimate Assumptions

The following estimate assumptions are excluded from schedule contingency analysis, and if they cannot be achieved, the project schedule may be delayed and/or execution duration extended. On a project of this magnitude, time-dependent costs, e.g., overheads, equipment rental etc., may be considerable:

- All permissions required to initiate the project on time will be received without incurring additional cost or affecting the schedules.
- Transportation access from point of manufacture to the project site, including the border crossing, will be unrestricted.

21.1.4 Currency and Commodity Rates

For consistency of estimating and conversion of native currency costs to the US dollar reporting currency, fixed rates of currency exchange and key project commodities were established and applied across all source estimates. Major currency exchange rates used were MNT 2,037 / US\$ and RMB 6.56 / US\$. The commodity rate assumptions are shown in Table 21.3. The estimate does not provide for variation of exchange rate.

Concrete and shotcrete rates were updated for OTSF16. Rates were based on the engineered mix designs, Owner's batch plant operation, OT LLC quarry operation, and use of imported bulk cement. Updated pricing for supply of imported cement, aggregate, and concrete additives was applied to the design mixes and recalculated concrete supply / delivery costs.

Backfill rate is based on average costs for Phase 1 construction. Surface rock handling rates were derived from first principles using OT LLC open pit fleet equipment and local decline and shaft waste stockpiles. Camp and catering costs allow for a combination of site services to be provided by the OT LLC Operations group in support of the Phase 2 capital works construction, including airport handling, employee site busing, bottled drinking water, and camp and messing services.

Table 21.3 Major Commodity Pricing

Major Commodity	Unit	Value
Diesel Fuel	L	1.27
Power	kWh	0.12
Concrete – Surface works up to 35 MPa	US\$/m ³	125.00
Concrete – UG works up to 35 MPa	US\$/m ³	150.00
Concrete – UG works 35–80 MPa (High Strength)	US\$/m ³	270.00
Shotcrete (40 MPa Fibrecrete)	US\$/m ³	270.00
Backfill	US\$/m ³	25.40
Surface rock hauling (up to 1.6 km haul)	US\$/t	1.30
Charter Flights UB–OT	US\$/return trip	229.00
Site Support Services	Camp Man-Day	25.00

21.1.5 Sustaining Capital

21.1.5.1 Tailings Storage Facility Construction

PAF mine waste is used for construction of the major tailings embankment structure, the downstream shell. Of the total amount of embankment material, 70%–80% is composed of mine waste that will be placed by the mine fleet, and so is included in the open pit haulage estimate. Allowance is made for dozing mine dumped material to achieve the final contour. Other mine waste requiring controlled placement will be delivered to a stockpile located between the pit and Cell 1 and will then be reloaded and hauled to the TSF by a fleet of 60-tonne trucks.

21.1.5.2 Concentrator

Costs are included for replacement of the concentrator support mobile equipment, and mobile equipment supporting the construction of the tailings dam. Replacement is based on the operating life of each piece of equipment, which varies from 10–15 years.

Costs are included to replace the fixed processing plant equipment after it is no longer feasible to maintain its designed function. Replacement costs are based on 0.5% per annum of the initial capital value.

An allowance is included to modify the expanded process streams after commencement of their operation. Most process plants typically require some minor changes to the initial design to attain design or optimum capacity.

21.1.5.3 Underground Sustaining capital

All mine development, lateral or vertical, is capitalized. This includes development associated with the material handling system, off-footprint ventilation infrastructure, permanent shafts, and main shops, undercut drill and blast, associated swell mucking, and drawbell drill and blast costs, and equipment replacement.

For Hugo North, sustaining capital costs fall under four main categories:

- Ongoing Development: All mine development, lateral or vertical, is capitalized until after first ore (May 2020). All development not directly associated with the final material handling system, off-footprint ventilation infrastructure, permanent shafts, and main shops will be considered sustaining capital after that time.
- Undercutting and Caving: All undercut drill and blast, associated swell mucking, and drawbell drill and blast are considered sustaining capital. The only exception is the portion of this work done prior to first ore.
- Ongoing Construction: Construction activities included under the category of sustaining capital are projects that are considered routine and are an integral part of the mine operations. The mine schedule provides the information required to determine how many of each type of installation was required during each schedule period. The following work is included in this category:
 - Drawpoint lintels
 - Grizzlies
 - Truck-loading chutes
 - Ventilation control doors
 - Gathering sumps
 - Power stations (for portable substations)
 - Stations for portable refuge stations
 - Concrete road construction
 - Ventilation controls and bulkheads
 - Service doors
- Mobile Equipment Rebuild and Replacement: The annual cost of mobile equipment replacement is based on estimated operating hours. Mobile equipment rebuild and replacement schedules are a product of the mining schedule. The following methodology was used to determine the annual cost of mobile equipment rebuild and replacement:
 - Rebuild life is estimated as 60% of the initial life of the equipment.
 - Rebuild cost is assessed at 40% of the base unit cost.
 - Replacement cost is assessed at 100% of the base unit cost plus development allowance and freight.
 - No replacement costs are provided for any of the mobile equipment during the final four years of mine operations, and no rebuild costs are provided for any of the mobile equipment during the final two.

21.1.5.4 Infrastructure Sustaining Capital

Sustaining capital includes the following:

- Replacement of information and communication (ICT) equipment at a rate of 10% per annum of initial capital value.
- Refurbishment / replacement of the central heating plant boiler every 10 years.
- Refurbishment of process and non-process buildings approximately every 10 years.
- Expansion of the waste management centre to provide additional capacity.

21.1.6 Mine Closure

The mine closure estimate has been prepared using quantities and installation hours from existing capital cost estimates and the closure plan. The hours required for demolition have been assumed to be 20% of the original install and construct hours for most of the surface infrastructure. No residual or salvage values are included. The closure expenditures commence 10 years prior to the completion of mining and processing after which there is a 10-year post-closure monitoring programme, which is followed by a long-term monitoring and inspection programme. The estimate contains direct costs consisting, among others, of the following:

- Costs prior to closure.
- Demolition and disposal of permanent facilities.
- Rehabilitation and revegetation.
- Collection, treatment, and disposal of hazardous wastes.
- Human resources.
- Community and socioeconomic initiatives.
- Post-closure monitoring and ongoing obligations.

The estimate also includes indirect costs, including the following:

- Closure support facilities.
- Catering costs.
- Closure management (EPCM) services.
- Owner's costs.
- Contingency, currently evaluated at 25% of all direct and indirect costs.

The following summarizes the scope of the estimate and briefly describes the proposed main closure and rehabilitation activities:

- Underground Mine – Flush, decontaminate, clean, and leave underground equipment in place. Remove any hazardous wastes for disposal and cap all surface openings.
- Surface Facilities – Flush, decontaminate, and clean all equipment and dismantle for disposal within the open pit mine. Demolish existing structure internals and dispose of in the open pit. Demolish existing structures and dispose of in open pit with the exception of structural steel,

which will be sorted and cut to manageable pieces for off-site disposal for recycling by a third-party contractor. Demolish existing concrete buildings and foundations to 1 m below final grade, remove scrap metal and rebar, and dispose of rubble in the open pit.

- Underground Mine Subsidence Area – Establish a safe perimeter and setback, construct safety berm, install proper signage, and conduct ground monitoring at regular intervals for up to 10 years of post-closure and 40 years of long-term monitoring.
- Low-Grade Stockpiles – All material to be processed before closure. Clean, scarify, and regrade the areas.
- Access Roads – Selectively scarify, regrade, and re-contour local access roads. Leave in place and maintain only roads that are critical for either monitoring, maintenance operation programmes, or emergency access.
- Gunii Hooloi Water Supply – Demolish and remove all debris and dispose of in open pit. Cut well casings below ground level and plug openings.
- Borrow Areas and Quarries – Regrade and re-contour slopes to a stable condition for the long-term. Redirect runoff to minimize erosion and revegetate where and if applicable.
- Disturbed Lands within Property Lines – Remove debris, scarify, regrade, and re-contour the area to blend into the surrounding topography. Revegetate where and if possible.
- Affected Soils (metals, hydrocarbons) – Remove, treat, and dispose of off-site or on site as applicable.
- Wastewater Treatment and Waste Lagoons – Remove sludge to the open pit, demolish and remove facility, and dispose of in the open pit. Remove and dispose of any liners and flatten the lagoon berms to ensure positive drainage of the area.
- Hazardous Waste Management – Remove and dispose of off-site to an approved facility.
- Above-ground Mining Equipment – Flush, decontaminate, clean, and dispose of any remaining equipment in the open pit and provide a rock cover.
- Tailings Storage Facility – Regrade top of tailings to attain desired grades and construct a store-and-release cover using well-graded (assumed crushed) non-acid forming (NAF) material, with the aim of reducing infiltration into the tailings and to minimize and control dust emissions. Regrading and reshaping the side slopes and the perimeter tailings dikes are not included in the cost estimate, assuming perimeter dike slopes are built to final geometry during operations. Remove and dispose of all tailings discharge structures, build closure spillway, and revegetate where and if applicable.
- NAF Waste Rock Dump – Reshape and regrade side slopes; revegetate where and if applicable. The NAF waste dump will act as reinforcement to the Undai River diversion.
- PAF and SOM Waste Rock Dumps – Reshape and regrade side slopes; it is assumed that PAF slopes will be overlain by a suitable thickness of NAF material during operations, and so the cost associated with covering the slopes of the waste dumps is not included in the estimate. A store-and-release cover will be constructed over flat surfaces, using well-graded (assumed crushed) NAF material with the aim of reducing infiltration. Revegetate where and if applicable.
- Closure and Post-Closure Monitoring – Includes physical and chemical stability monitoring of the site and structures, ecological monitoring, and socioeconomic activities. Assume five years of pre-closure engineering period, five years of closure implementation work, 10 years of post-closure monitoring, and 40 years of long-term monitoring.

- Long-term Closure Monitoring and Maintenance – Includes physical and chemical stability monitoring of the site and structures, ecological monitoring, and socioeconomic activities. Assume 40 years of monitoring based on the creation of a sinking fund for self-funding of long-term activities.

21.1.7 Contingency

In general, the base estimate has been developed on the following principles:

- The project will be implemented in accordance with the project execution plan assuming typical site conditions known for the project location without undue interruptions from abnormal weather, civil unrest, and the like.
- Neat quantity take-offs were prepared from the available developed design, with the addition of design growth allowances to represent conditions anticipated at design completion.
- Equipment and bulk material pricing rates are taken from a combination of formal quotes, budget quotes, informal quotes, and historical experience. Quoted rates were adjusted, where deemed appropriate, to include specific project terms and conditions, wastage, freight components, and the like.
- Installation pricing rates are based primarily on pre-suspension awarded data or Phase 1 project experience.
- Costs are escalated to the anticipated time of expenditure based on projected pricing indices.
- Exchange rates are fixed.

The amount of contingency included is based on a risk analysis of the quality and maturity of the major estimate input variables plus the identified discrete risk events, with consideration to the level of allowances and provisions included in the base estimate. The capital contingency added to the OTFS16 estimate was approximately 14%.

Major estimate input variables include scope definition, pricing rates, and implementation methodology. Discrete risk events address the issues that cannot be included in the development of the base estimate because of uncertainty over the likelihood of occurrence and cost impact; that is these events are possible not probable. The amount of contingency is calculated as the difference between the mean value from a Monte Carlo simulation and the base estimate value plus the outcome of the discrete risks analysis.

21.2 Operating Costs

21.2.1 Summary

Mine operations at Oyu Tolgoi commenced in July 2011 with the start-up of the open pit mine. First mill feed to the concentrator was in August 2012, and the first concentrate was produced in September 2013. Feed from the underground mine is planned to commence in mid-2020 and to ramp-up to the full underground design tonnage of 95 kt/d. The mill operating rate at that time will be a nominal 110 kt/d, with supplementary ore feed provided from the open pit mine. Once the Hugo North Lift 1 underground mine reserve is mined out, the operation will revert back to

feed from open pit mining until 2050. The mill will continue to operate on open pit feed from the low- and medium-grade stockpile until closure.

All costs in 2016 OTTR are expressed in real Q4'16 US dollars. Timing of expenditures in 2016 OTTR assume a start date of 1 January 2017 and that 2016 expenditures are sunk costs. The 2016 OTTR estimates include contingency and constant real exchange rates and do not include escalation. The original OTFS16 cost estimate was based on nominal Q4'15 US dollars.

The estimate includes all expenses to operate and maintain the Oyu Tolgoi plant, Oyut open pit and Hugo North Lift 1.

The following battery limits are noted:

- All costs are in Q4'16 US dollars, based on fixed exchange rates.
- Costs pre-31 December 2016 are excluded.
- Escalation is excluded from the operating costs.
- Joint venture fees are included.
- Power has been treated as a purchased utility from a third-party provider.

The 2016 Reserves Case operating costs are shown in Table 21.4.

Table 21.4 Operating Costs – 2016 Reserves Case

Operating Cost	US\$b	US\$/t Ore Milled		
	Total 2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Mining (all sources)	8.43	4.85	5.55	5.82
Processing and Tailings	11.91	7.45	7.77	8.22
G&A and Operations Support	2.77	2.52	2.44	1.91
Infrastructure and Other	1.76	1.83	1.69	1.22
Government Fees & Charges	2.83	1.98	2.06	1.95
Management and JV Payments	2.75	2.74	2.54	1.90
Total Site Operating Costs	30.44	21.36	22.05	21.02

21.2.2 Estimate Methodology

All operating cost estimates were prepared on a first principles basis wherein all expenses have been quantified as much as possible and unit cost rates applied. Estimates were prepared by major operating cost function, as follows:

- Open pit mine.
- Underground mine.
- Process plant – inclusive of concentrator and bagging plant.

- Tailings – inclusive of tailings pumping, tailings dewatering, and tailings storage facility.
- Infrastructure.
- General and Administrative (G&A) – Inclusive of operations support.
- Mine closure.

Operating cost estimates were also developed by expense type classified into the following categories:

- Labor – Cost to employ all direct OT LLC staff.
- Fixed overheads – Accommodation and meals, travel, business overheads, insurances, and general expenses.
- Utilities – Power and water charges.
- External services – Third-party contracted services.
- Materials and supplies – Fuels, equipment spares and parts, process consumables and reagents, other maintenance materials.

Pricing is predominantly based on operating experience and market conditions.

21.2.3 Open Pit

The open pit operating costs were developed by the OT LLC open pit Technical Services team based on the OTFS16 mine plan. The operating costs are driven mainly by the hauling unit requirement, which will increase as the pit deepens and hauling distances increase. The open pit operating costs include all activities to mine ore and waste from the pit phases, including direct mining, maintenance, supervision and technical support.

The following boundaries are noted for the scope of the open pit operating cost estimate:

- The operating cost for any ore rehandle at the concentrator stockpile (coarse ore) is carried by the concentrator.
- The primary crusher and conveyors designed to size and transfer open pit ore from the truck dump to the coarse ore stockpile are within the scope of the concentrator for both capital and operating cost estimates.
- The open pit provides material for construction of the tailings dam, either at a dump or directly to the dam wall. Costs for rehandle and any placement costs beyond dumping are carried in the tailings sustaining cost estimate.

21.2.4 Underground Operating Costs

Mine Underground operating costs are the costs of operating the underground mine, including all indirect costs. The costing for operations is based on the following assumptions:

- Labor is either a fixed value (i.e. one hoist man on duty, two electricians on duty, etc.) or on equipment productivity (one driver per truck).
- Labor rates are standard rates provided by Oyu Tolgoi.

- Material costs are the same as used in the capital estimate. For specific replacement costs, such as skips, the costs used in the initial purchase costs are considered to be valid for the life of the mine.
- Indirect costs are similar to those for the capital phase of the project.

The estimate is derived on the following key headings:

- Mine Management: this includes the mine management and technical services groups, administration, safety, and training activities.
- Mine Production: this includes all direct costs associated with moving production ore from the drawpoints to the primary crushers. This includes mucking, secondary blasting, grizzly/raise/chute operations, truck haulage, and ground repair.
- Costs are developed in the cost model from time-phased equipment performance data from the production simulation, as well as predictions of oversize material projected in the geotechnical models, both provided by OT LLC.
- Rock Handling: this includes all crushing and conveying of rock from the truck dump at the crusher to the coarse ore stockpile at the concentrator. It does not include power costs or shaft-hoisting operations.
- Costs are based on ore throughput rates from the mine plan and estimated equipment maintenance costs on a per tonne basis. Labor costs are based on fixed crew sizes developed from the equipment in service during a given period.
- Shaft and Logistics Operations: this includes the operating costs of all shafts for service and rock hoisting and the operation of the logistics system to deliver supplies from the surface to designated delivery points underground. This specifically includes logistics support for all initial construction activities prior to first ore.
- Maintenance Overhead: this includes all maintenance overhead for the life of the mine: maintenance management, planning, indirect labor support, vendor reps, and other similar support functions. This covers both OT LLC and mine contractor functions. Maintenance labor costs for fixed plant are included here for estimating purposes.
- Electrical and Instrumentation: this includes costs for electrical and instrumentation and communication maintenance labor.
- Mine Services: this includes labor and equipment costs to perform routine mine-wide services such as road maintenance, drainage and pumping, and maintenance of utility piping.
- Utility Back-Charges: this carries all charges for utilities, the largest being electrical power.

21.2.5 Concentrator

The concentrator operating costs were developed by the OT LLC based on OT LLC Phase 1 operating cost data derived from OTFS16. Concentrator costs include the following elements:

- Power
- Grinding Media
- Labor

- Maintenance Materials
- Mill Liners and Crusher Liners
- Reagents and Lime
- Contractors / Consultants
- Mobile Equipment
- Bagging Supplies
- Accommodation / Catering
- Water Costs
- Concentrator G&A
- Safety Supplies

The following battery limits apply to the concentrator costs:

- Operating costs for any ore rehandle before gyratory crushing are carried by the open pit.
- The conveyors used to transfer underground ore from the shafts to the coarse ore stockpile are within the underground mine capital cost scope but within the concentrator operating cost scope.
- Tailings operating labor costs are included in the concentrator area, with OT LLC operating and construction equipment labor integrated with the overall concentrator labor plan. Incremental costs to move benign waste from the open pit to the local stockpile at the TSF are included in the open pit mine estimate.
- Tailings operating costs for fuel and other non-labor construction equipment costs are presented separately to the concentrator.

Annual operating costs are projected to vary primarily as a function of unit power cost and percentage of expatriate personnel. Unit costs vary primarily as a function of capacity and thus average ore hardness. Phase 2 annual costs are not significantly changed from Phase 1 costs.

21.2.6 Infrastructure Operating Costs

The infrastructure operating cost estimate covers the costs directly attributable to operational activities of the infrastructure department. The main responsibilities of this department are to operate and maintain all Phase 1 and Phase 2 site infrastructure, including:

- Central heating plant (CHP)
- Raw water supply from the borefields north of the site
- Heavy mobile equipment (HME) facility
- Warehouse (buildings only)
- Water bottling plant
- Electrical utilities other than the power plant and 220 kV distribution
- Camp facility (buildings only)

- Airport
- Light vehicle facility
- Other building maintenance, including the waste management centre

The cost estimate adopted the cost element groups utilized by the OTFS16 study team at the time. These include:

- Labor
- Fixed overheads
- Utilities
- External services
- Materials

21.2.7 General and Administrative

The General and Administrative (G&A) costs encompass costs not directly attributable to operational output such as the mining and processing operations. G&A costs are divided into two main areas: Operations Support, comprising departments based at Oyu Tolgoi, and General and Administrative, covering those departments principally operating from OT LLC'S Ulaanbaatar offices.

The Operations departments are:

- Asset Management
- Health, Safety, and Environment
- Value Chain Process& Improvement
- Office of COO

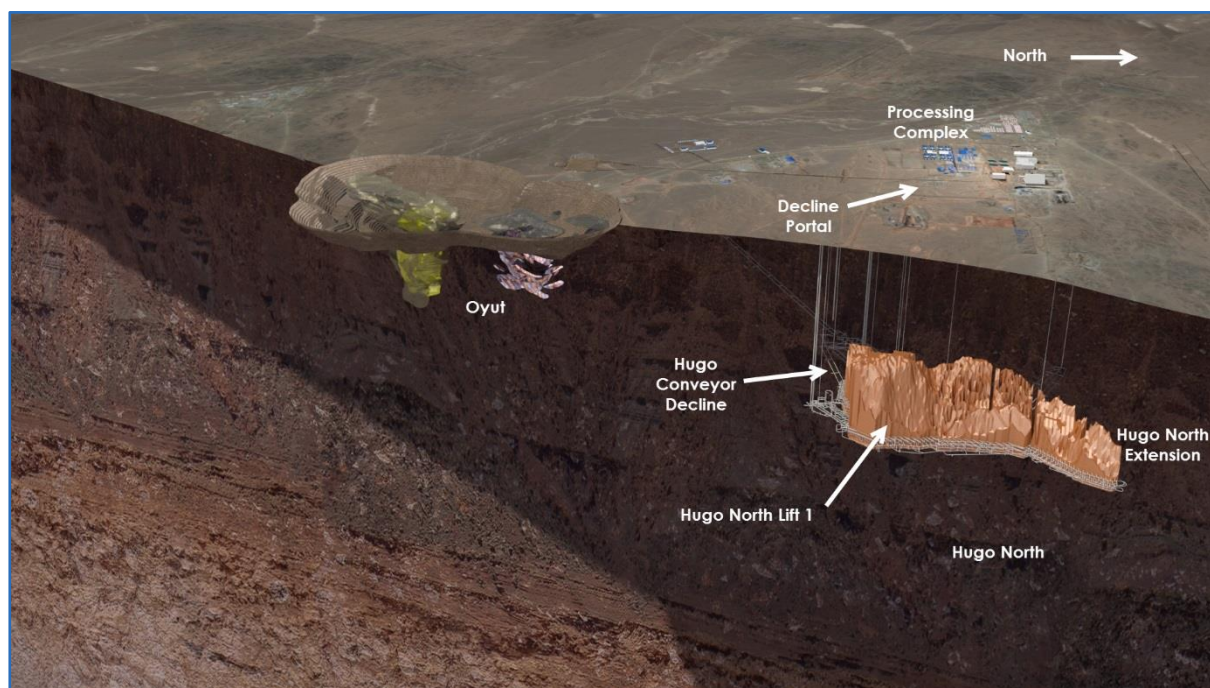
The General and Administrative departments are:

- Finance
- IT
- Procurement
- Office of CEO
- Regional Development
- Communications and Media Relations
- People and Organization
- Resource Strategy
- Power Strategy
- Admin and Facilities
- Construction Engineering Group

22 ECONOMIC ANALYSIS

The 2016 Reserves Case assumes processing of 1.4 bt of ore, mined from the Oyut open pit and the first lift in the Hugo North underground block cave. The mining areas included in the 2016 Reserves Case are shown schematically in Figure 22.1.

Figure 22.1 2016 Reserves Case Mining Areas



Over time, there are expected to be multiple investment decisions made for Oyu Tolgoi and an evaluation of each development option, as and when it is required, ensuring that the commitments made for the project represent the optimum use of capital to develop Oyu Tolgoi for Mongolia.

The initial investment decision was made in 2010 to construct Phase 1 of Oyu Tolgoi. Phase 1 consisted of the Oyut open pit mine, concentrator and supporting infrastructure. These facilities are complete and the operation has commenced. Processing operations have been in production since December 2012, commercial production was achieved in September 2013, and first concentrate exported in October 2013.

Part of the initial investment decision included continued investment into the development of the Hugo North underground mine in parallel with mining the open pit. Lift 1 of Hugo North is the most significant value driver for the project. The Phase 2 scope, which includes the Hugo North underground block cave, supporting conveyor decline and shafts, concentrator conversion, and supporting infrastructure expansion has now commenced development.

The 2016 OTTR project scope from OTFS16 that has been used for the Mineral Reserve evaluation is the 2016 Reserves Case. A summary of the production and financial results for the 2016 Reserves Case are shown in Table 22.1.

Table 22.1 Summary Production and Financial Results – 2016 Reserves Case

Description	Units	2016 Reserves Case
Total Processed	bt	1.4
Cu Grade	%	0.86
Au Grade	g/t	0.30
Ag Grade	g/t	1.95
Copper Recoverable	blb	23.9
Gold Recoverable	Moz	10.4
Silver Recoverable	Moz	74.3
Life	Years	38
Expansion Capital	US\$b	4.63
NPV8% After Tax	US\$b	6.94
IRR After Tax	%	21
Payback Period	Years	8

Notes:

1. NPV8% is Net Present Value (NPV) at a discount rate of 8% for all years.
2. IRR is Internal Rate of Return.
3. For mine planning the metal prices used to calculate block model Net Smelter Returns (NSR) were copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t and the underground costs are based on US\$15.34/t.
4. 2016 OTTR financial analysis long-term metal prices used are: copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
5. For the underground block cave, all Mineral Resources within the shell have been converted to Mineral Reserves. This includes low grade Indicated Mineral Resources. It also includes Inferred Mineral Resources, which have been assigned a zero grade and treated as dilution.
6. The Oyut open pit Mineral Reserves are the Mineral Reserves in the pit at 31 December 2015. The Mineral Reserves do not include stockpiles as at that date.
7. For Oyut, only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
8. For Hugo North, Measured and Indicated Mineral Resources were used to report Probable Mineral Reserves.
9. The Mineral Reserves reported above are not additive to the Mineral Resources.
10. Economic analysis has been calculated from the start of 2017 and excludes \$0.46b expansion capital from 2016. Costs shown are real costs not nominal costs. Expansion capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

22.1 Economic Assumptions

Estimates of cash flows have been prepared on a real basis as at 1 January 2017 and discounted to a Net Present Value (NPV) at a rate of 8% for all years (NPV8%). The NPV results have been calculated starting from January 2017.

The results of the 2016 Reserves Case show an after tax NPV8% of US\$6.94b. The case exhibits an after tax Internal Rate of Return (IRR) of around 20% and a payback period of around eight years.

The key economic assumptions for the analyses are shown in Table 22.2. Metal prices were selected based on reviews of long-term consensus estimates and metal prices reported in public reports. The metal prices used in the initial years and the long-term prices are shown in Table 22.3. The discount rate of 8% was selected for the base case after a review of public reporting for base metal projects. Smelter terms for treatment and refining charges were selected from published long-term contract terms and forecasts.

Table 22.2 Economic Assumptions

Parameter	Unit	Long-Term Financial Analysis Assumptions	Average Financial Analysis Assumptions
Copper Price	US\$/lb	3.00	2.97
Gold Price	US\$/oz	1,300	1,300
Silver Price	US\$/oz	19.00	19.00
Treatment Charges	US\$/dmt conc.	85.00	85.45
Copper Refining Charge	US\$/lb	0.085	0.085
Gold Refining Charge	US\$/oz	4.50	4.50

Table 22.3 Metal Price Assumptions – 2016 Reserves Case

Metal	Unit	Year				
		1	2	3	4	5 Onwards
Copper	US\$/lb	2.15	2.36	2.58	2.79	3.00
Gold	US\$/oz	1,300	1,300	1,300	1,300	1,300
Silver	US\$/oz	19.00	19.00	19.00	19.00	19.00

22.2 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 Oyu Tolgoi. 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 IA is effective for an initial term of 30 years and an extension of a further 20 years. The term of a mining license under the Minerals Law is for 30 years with two 20-year extensions.

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 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 TRQ 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.

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.

Pursuant to the ARSHA, a Management Services Payment (MSP) equal to 3% of total operating and capital costs prior to commencement of production and 6% of operating and capital costs during production is payable by OT LLC to the Management Team (or other TRQ or Rio Tinto group entities to whom the Management Team may direct payment). Under the terms of the Underground Mining Development and Financing Plan (UDP) and Management Agreement (MA), OT LLC shareholders have agreed that in calculating the MSP, the rate to be applied to the capital costs of developing the Underground Stage shall be 3% instead of 6% as provided in the ARSHA. Underground Stage is defined by the UDP to mean the construction of the Hugo North Lift 1 underground mine and modification of the concentrator and infrastructure to handle underground ore.

The MA imposes a cap on the amount of capital costs that can be used to calculate the MSP and clarifies the categories of capital costs and operating costs that are to be included and excluded in calculation of the MSP. The MA clarifies the calculation of the MSP during the periods between 31 March 2010 and 1 September 2013, from 1 September 2013 and 4 June 2015 (the date of execution of the MA), and from 4 June 2015 onwards, and sets out a mechanism for OT LLC and RTOTM to carry out a reconciliation of the amounts of MSP payable against MSP payments already made. The MA further sets out the amount of chargeable costs to be borne by OT LLC, being costs incurred by the Management Team in providing Operational Management to OT LLC.

OT LLC was required to achieve commencement of production within seven years of the effective date of the IA and commercial production was achieved in 2013.

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 as they applied as at the date of the IA. The taxes and fees payable to the GOM and their rates, include:

- Corporate income tax 25%
- Mineral royalties 5% (sales value)
- Value added tax 10%
- Customs duties 5%
- Withholding tax 20%

In accordance with the Windfall Profits Tax (WPT) invalidating law, as from 1 January 2011, OT LLC will not be subject to the WPT.

OT LLC is also only subject to those taxes listed in the General Taxation Law as at the date of the IA and not taxes introduced at any future date.

In 2009, the GOM enacted amendments to the legislation governing the carry-forward of income tax losses. These terms are incorporated into the IA. The loss carry-forward period has been extended to eight years and, if sufficient, can be applied to offset 100% of taxable income. The IA also provides OT LLC with the benefit of a 10% tax credit for all capital investment made during the construction period.

By signing the UDP and related documents, OT LLC and its shareholders have agreed on key outstanding taxation matters including:

- (a) calculation and the amount of OT LLC's investment tax credits and losses carried forward;
- (b) the calculation of royalties (to be calculated on a gross rather than a net sales proceeds basis without deduction of costs for processing including treatment and refining charges), freight differentials, penalties, or payables; and
- (c) double taxation treaty stabilization and stabilized rate of the withholding tax.

Pursuant to the UDP, the OT LLC shareholders and the GOM have agreed that international market prices shall be used in determining sales values to calculate the mineral royalties¹ payable. The UDP sets out the applicable international markets for determining the sales value of copper, gold, and silver and makes provision if these market pricing mechanisms cease to be available.

The following points provide additional commentary on taxes from OTFS16:

- In accordance with the Investment Agreement, OT LLC is not subject to the excess profits tax or any similar windfall tax.
- The copper concentrate does not qualify as a "finished product" for VAT purposes, and so OT LLC is unable to claim back-credits / refunds for the 10% VAT incurred by it on its purchases.

- OT LLC is provided a 10% investment tax credit in respect of its investment in depreciable non-current assets made during the lesser of the construction period (set as the period to reach 58 Mt/a) or seven years from the IA effective date (i.e. until first quarter 2017). This credit is carried forward on a year-by-year basis for three profitable tax years and then expires. It is noted in the IA that if VAT payments, which are currently non-refundable, become refundable in the future, then the availability of the investment tax credit will cease from that point. However, brought-forward credits can still be utilized.
- The IA allows for the tax loss carry-forward period in relation to Corporate Income Tax losses in respect of tax years occurring from 1 January 2010 to be extended to eight years and can be applied to offset 100% of taxable income.
- Customs duty of 5% is payable on all imported goods and associated inbound logistics costs incurred outside of Mongolia, which includes ocean freight, air freight, and land transportation. For purposes of the OTFS16 study, those portions of the project operating costs to which VAT applies are also assumed to be subject to customs duties. Customs duties on capital costs were calculated more precisely for each scope item.

22.3 Operating Assumptions

Although it has a requirement to make its self-discovered water resources available to be used for household purposes, the IA confirms that OT LLC holds the sole rights to use these water resources for the project. On 17 October 2014, a water use permit for 25 years was issued to OT LLC. In June 2016, OT LLC entered into a utilization agreement with a GOM water agency for 25 years (until June 2040). Together with water use conclusions issued annually and the approved water reserve rate, these arrangements enable OT LLC to utilize the water required to develop the project. As the Law on Water and IA provide that the term of water use permits for exploiting mineral deposits of strategic importance shall be the same as the term of mining licenses, OT LLC considers that it is entitled to extensions of its water permit and water utilization agreements for subsequent 20-year periods as its mining licenses are renewed.

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. The IA includes an overarching commitment from the GOM and OT LLC to work together to determine the optimal and most reliable solutions for power supply.

Under the IA, OT LLC is required to secure its power requirements from within Mongolia within four years of commencement of production. However, this timeframe is currently suspended pursuant to the Southern Region Power Sector Cooperation Agreement entered into by OT LLC and the GOM on 14 August 2014 that proposes an independently funded and operated coal fired power plant at Tavan Tolgoi (TTPP Project).

So long as OT LLC continues to participate in the TTPP Project the four-year timeframe for sourcing Mongolian power will be suspended. Upon a withdrawal from the TTPP Project by either OT LLC or the GOM, the four-year timeframe will be reinstated and recommence from the date of withdrawal. A request for proposals from potential investors in the TTPP coal fired power plant led to a consortium, led by Marubeni Corporation and MCS Energy (local infrastructure investor), being selected as the preferred bidder for the project in February 2016. A final decision on the TTPP's construction is expected in 2017. Should the terms of the response to the TTPP arrangement prove unattractive to OT LLC, OT LLC retains the option to meet its power needs and domestic sourcing obligation under the IA through the construction and financing of its own power plant at the project site.

OT LLC sources its present power under a four-year contract with a Chinese provider, the Inner Mongolia Power International Cooperation Company Ltd. (IMPIC) via the Mongolian National Power Transmission Grid (NPTG) authority. In May 2016 the parties agreed via a non-binding Memorandum of Understanding (MoU), which captured key agreed principles of the new Power Purchase Agreement (PPA), to extend the power supply agreement to at least 2021. OT LLC and the GOM have agreed under the Power Sector Cooperation Agreement that the GOM will assume responsibility for securing the extension of the power import arrangements through its national grid company NPTG. The agreed MoU includes comparable power pricing to the current agreement. OT LLC is endeavoring to execute binding agreements with the GOM and IMPIC within 2016.

OT LLC has the right to construct, manage, and use an aerodrome in connection with the project, based on permits issued in accordance with Mongolian law. A permanent domestic airport, capable of servicing Boeing 737-800 series aircraft, has been constructed at Oyu Tolgoi to support the transportation of people and goods to the site from Ulaanbaatar. It further serves as the regional airport for Khanbogd soum.

The GOM may construct or facilitate the construction and management of a railway in the vicinity of the project to the China-Mongolia border. The GOM will consult with OT LLC on the location and route of the railway, and, if the railway is constructed, then it will be made available to OT LLC on commercial and non-discriminatory terms. Energy Resources is currently constructing a single-track heavy-haul rail from its Ukhua Khudag coal mine (approximately 120 km to the north-west of Oyu Tolgoi) to Gashuun Sukhait, ultimately to be interconnected with the Chinese rail network at Ganqimaodao on the Chinese side of the border. Once constructed, the South Gobi Rail alignment would pass within 10 km of the Oyu Tolgoi project area and therefore represents an opportunity for eventual connection of the mine to the rail network.

OT LLC also has the right to construct roads for the transport of its product. A gravel road has been constructed to the town of Khanbogd and is being maintained. OT LLC intends to construct a paved road from the mine site to the town of Khanbogd. A 105 km sealed road is being constructed to the Chinese border crossing at Gashuun Sukhait, with sealing of the entire road expected to be completed in 2017. On the Chinese side of the border a provincial road connects the border town of Ganqimaodao with the Jingzang Expressway via the towns of Hailiutu and Wuyuan.

Pursuant to the ARSHA, a Management Services Payment (MSP) equal to 3% of total operating and capital costs prior to commencement of production and 6% of operating and capital costs during production is payable by OT LLC to the Management Team (or other TRQ or Rio Tinto group entities to whom the Management Team may direct payment). Under the terms of the Underground Mining Development and Financing Plan (UDP) and Management Agreement (MA), OT LLC shareholders have agreed that in calculating the MSP, the rate to be applied to the capital costs of developing the Underground Stage shall be 3% instead of 6% as provided in the ARSHA. Underground Stage is defined by the UDP to mean the construction of the Hugo North Lift 1 underground mine and modification of the concentrator and infrastructure to handle underground ore.

The MA imposes a cap on the amount of capital costs that can be used to calculate the MSP and clarifies the categories of capital costs and operating costs that are to be included and excluded in calculation of the MSP. The MA clarifies the calculation of the MSP during the periods between 31 March 2010 and 1 September 2013, from 1 September 2013 and 4 June 2015

(the date of execution of the MA), and from 4 June 2015 onwards, and sets out a mechanism for OT LLC and RTOTM to carry out a reconciliation of the amounts of MSP payable against MSP payments already made. The MA further sets out the amount of chargeable costs to be borne by OT LLC, being costs incurred by the Management Team in providing Operational Management to OT LLC.

22.4 Financing Assumptions

The Project Financing package closure was completed in 2016 with 100% of project finance net proceeds and operating cash flow from the Oyut open pit used to fund underground development. Based on the study assumptions it is anticipated that additional supplemental debt, up to the senior debt cap of US\$6b, will be required to complete development. All project finance debt is forecast to be repaid by 2030.

The financing package includes a completion guarantee underwritten by Rio Tinto. For taking the risk of this completion guarantee, Rio Tinto will receive a fee based on the average outstanding annual debt until project completion, projected to be in the mid 2020's. This fee will be serviced from project cash flows.

The full US\$4.4b of project finance funds available were drawn down upon closing with net funds (post payment of fees) used to temporarily repay shareholder loans and deposited with Rio Tinto. All funds deposited will be invested back into the project to fund underground development as required. This recycling mechanism enables Oyu Tolgoi to reduce financing costs by repaying higher cost shareholder loans and replacing them with less-expensive project financing, thus avoiding paying commitment fees on any undrawn project finance facilities. In addition, the Rio Tinto guarantee fee will be waived for all amounts on deposit, reducing the effective rate of the guarantee.

22.5 Project Results – 2016 Reserves Case

A summary of the 2016 Reserves Case project financial results is shown in Table 22.4 and the mining production statistics are shown in Table 22.5. The estimates of cash flows of the project have been prepared on a real basis based at 1 January 2017 and discounted to an NPV at a rate of 8% (NVP8%). Long-term metal prices used for the analysis are copper US\$3.00/lb, gold US\$1,300/oz, and silver US\$19.00/oz.

Table 22.4 Financial Results – 2016 Reserves Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	25.31	23.00
	5%	11.83	10.95
	6%	10.15	9.43
	7%	8.71	8.10
	8%	7.45	6.94
	9%	6.35	5.92
	10%	5.39	5.03
IRR (%)	–	21	21
Project Payback Period (Years)	–	8	8

The 2016 Reserves Case processing, and concentrate and metal production are summarized in Figure 22.2 and Figure 22.3 respectively.

Table 22.5 Mining Production Statistics – 2016 Reserves Case

		2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Quantity Ore Treated	Mt	1,448	38.2	38.9	38.1
Copper Feed Grade	%	0.86	0.51	0.97	0.86
Gold Feed Grade	g/t	0.30	0.32	0.39	0.30
Silver Feed Grade	g/t	1.95	1.35	2.18	1.95
Copper Recoveries	%	87	81	88	87
Gold Recoveries	%	75	73	77	75
Silver Recoveries	%	82	77	82	82
Copper Concentrate	Mt (dry)	38.6	0.6	1.1	1.0
Copper Concentrate Grade	%	28	25	30	28
Contained Metal in Concentrate					
Copper	Mt	10.8	0.2	0.3	0.3
Copper	blb	23.9	0.3	0.7	0.6
Gold	Moz	10.4	0.3	0.4	0.3
Silver	Moz	74.3	1.3	2.3	2.0

Figure 22.2 Processing – 2016 Reserves Case

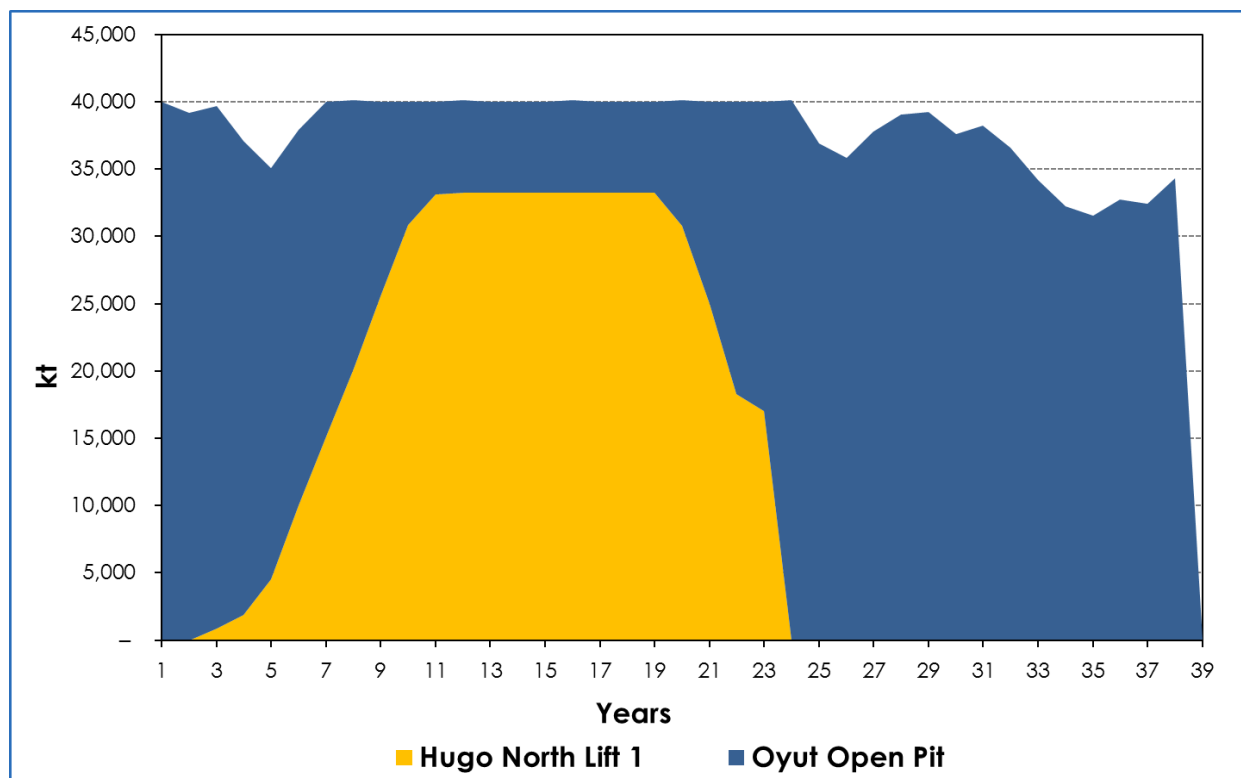
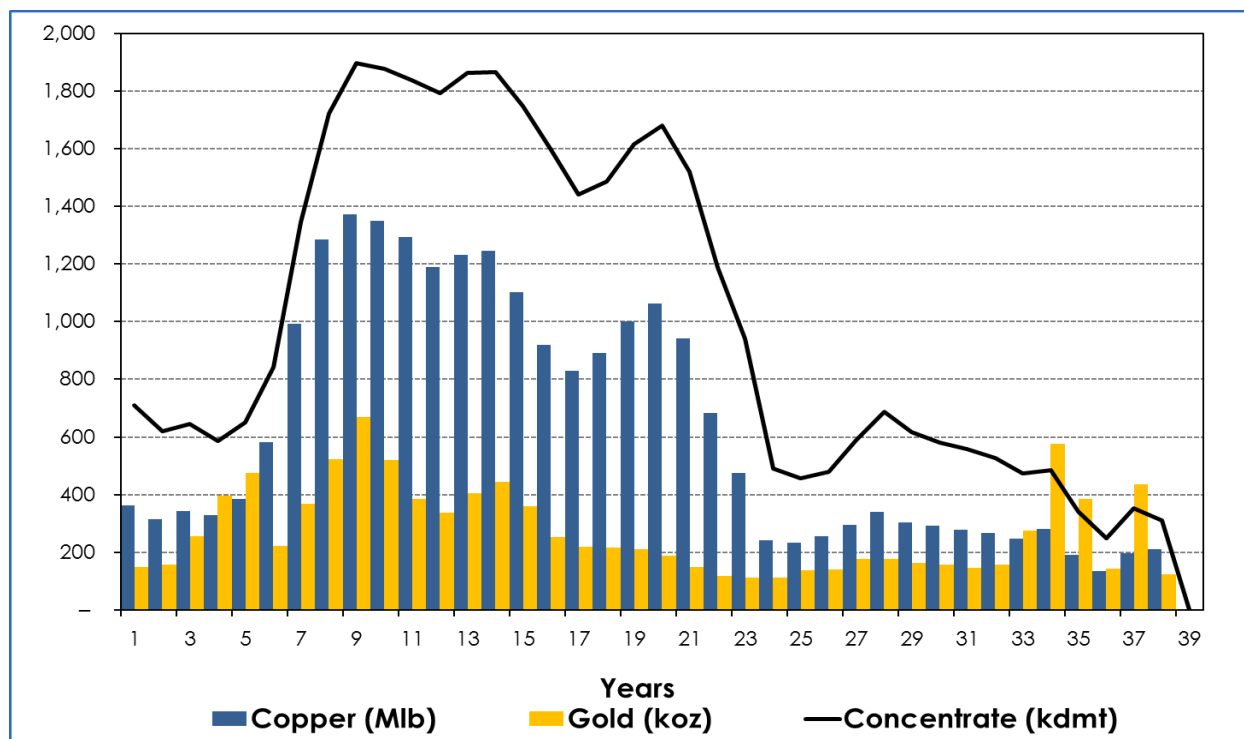


Figure 22.3 Concentrate and Metal Production – 2016 Reserves Case



Mine site cash costs are shown in Table 22.6. Cash costs are those costs relating to the direct operating costs of the mine site, namely:

- Mining (all sources)
- Processing and Tailings
- G&A and Operations Support
- Infrastructure and Other
- Government Fees & Charges
- Management and JV payments.

Table 22.6 C1 Cash Costs – 2016 Reserves Case

Description	US\$/lb Payable Copper			
	2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Mine Site Cash Cost	1.86	3.01	1.73	1.55
By-product Credit	0.62	1.15	0.71	0.53
C1 Cash Costs (Net of By-product Credit)	1.24	1.86	1.01	1.02

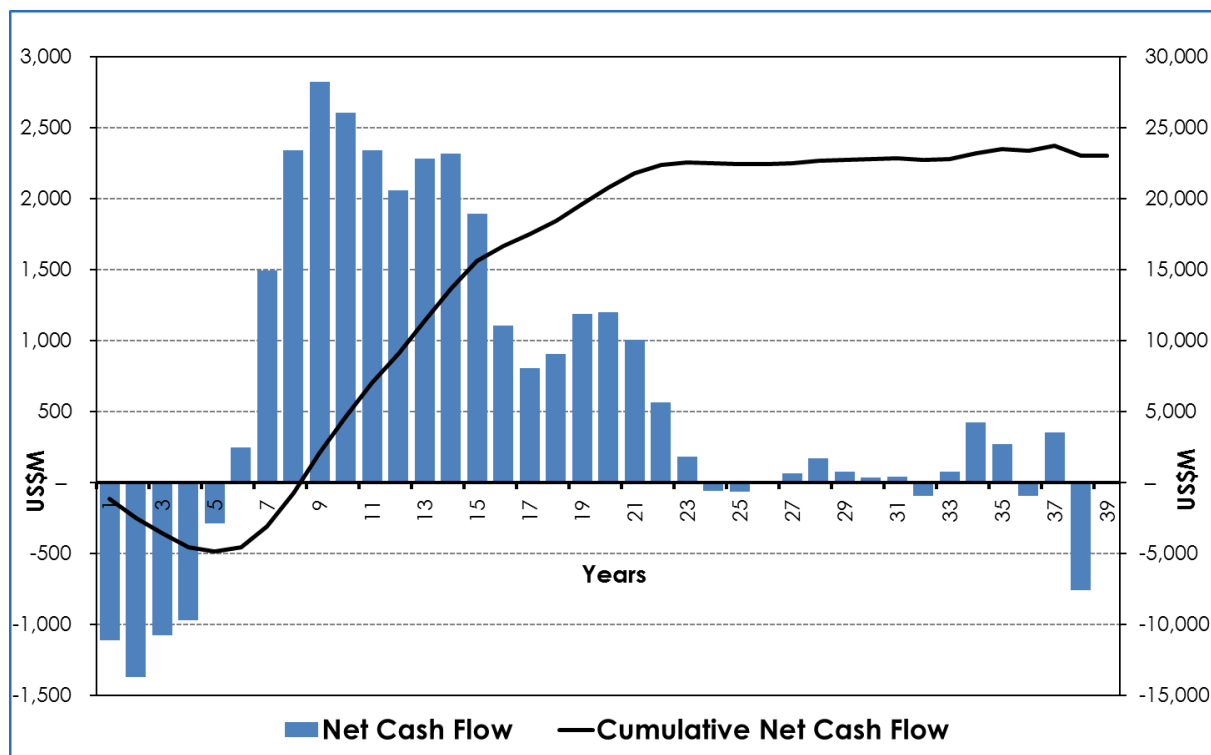
The revenues and operating costs are presented in Table 22.7, along with the net sales revenue value attributable to each key period of operation. Interest charges from shareholder loans and project financing are not included in the operating and capital costs but are allowed for in the calculation of tax.

The total project direct capital costs required are shown in Table 22.8. The changes in financial results for a range of copper and gold prices, calculated at a silver price of US\$19.00/oz, are shown in Table 22.9. The changes in financial results for a range of silver prices, calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz, are shown in Table 22.10.

A cumulative cash flow for the 2016 Reserves Case is depicted in Figure 22.4 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and a complete cash flow is provided in Table 22.11. The project indirect costs include GOM VAT and duties on operating costs, Management and JV Fees, VAT on Management Fees, and property tax.

Table 22.7 Operating Costs and Revenues – 2016 Reserves Case

	US\$b	US\$/t Ore Milled		
	Total 2016 Reserves Case	5-Year Average	10-Year Average	2016 Reserves Case Average
Revenue				
Gross Sales Revenue	82.81	32.48	65.59	57.18
Less: Realization Costs				
Realization Costs	9.46	4.49	7.15	6.53
Government Royalty	4.31	1.69	3.41	2.97
Total Realization Costs	13.77	6.18	10.56	9.51
Net Sales Revenue	69.04	26.30	55.03	47.67
Less: Site Operating Costs				
Mining (all sources)	8.43	4.85	5.55	5.82
Processing and Tailings	11.91	7.45	7.77	8.22
G&A and Operations Support	2.77	2.52	2.44	1.91
Infrastructure and Other	1.76	1.83	1.69	1.22
Government Fees & Charges	2.83	1.98	2.06	1.95
Management and JV Payments	2.75	2.74	2.54	1.90
Total Site Operating Costs	30.44	21.36	22.05	21.02
Operating Margin	38.59	4.93	32.97	26.65

Figure 22.4 Cumulative Cash Flow – 2016 Reserves Case

Table 22.8 Total Project Capital Cost – 2016 Reserves Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.564	1.564
Underground	2.240	3.065	5.304
Concentrator	0.145	0.149	0.293
Infrastructure	0.354	0.230	0.585
Tailings Storage Facility (TSF)	–	0.912	0.912
Subtotal	2.739	5.920	8.659
Construction Indirect	0.858	–	0.858
Contractor Execution – EPCM	0.310	–	0.310
Owner Execution	0.429	0.168	0.597
GOM Fees & Charges – Mongolian VAT	0.298	0.669	0.967
Total	4.635	6.756	11.391

Notes:

- Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
- The 2016 Reserves Case total capital cost excludes capital costs for the year 2016. Expansion capital for 2016 excluded is US\$0.46b.

Table 22.9 After Tax Metal Price Sensitivity – 2016 Reserves Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.89	-2.54	-2.20	-1.86	-1.51	-1.17	-0.31
2.00	-1.48	-1.14	-0.79	-0.45	-0.11	0.24	1.10
2.50	2.04	2.39	2.73	3.07	3.42	3.76	4.62
2.80	4.16	4.50	4.84	5.19	5.53	5.87	6.73
3.00	5.56	5.91	6.25	6.59	6.94	7.28	8.14
3.50	9.09	9.43	9.77	10.12	10.46	10.80	11.66
4.00	12.61	12.95	13.29	13.64	13.98	14.32	15.18
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	7
2.00	–	–	5	6	8	9	11
2.50	13	14	14	15	16	17	18
2.80	16	17	18	18	19	20	21
3.00	19	19	20	20	21	22	23
3.50	23	23	24	25	25	26	27
4.00	27	27	28	28	29	30	31
Project Payback After Tax (Years)							
1.80	14.4	13.7	13.1	12.7	12.2	11.7	10.6
2.00	12.2	11.8	11.4	11.0	10.7	10.3	9.7
2.50	9.7	9.5	9.3	9.2	9.0	8.9	8.6
2.80	9.0	8.8	8.7	8.6	8.5	8.4	8.2
3.00	8.6	8.5	8.4	8.3	8.3	8.2	8.0
3.50	8.1	8.0	7.9	7.8	7.7	7.7	7.5
4.00	7.6	7.5	7.5	7.4	7.4	7.3	7.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.35	1.31	1.27	1.23	1.18	1.14	1.04
2.00	1.36	1.32	1.28	1.24	1.19	1.15	1.05
2.50	1.38	1.34	1.30	1.26	1.22	1.18	1.08
2.80	1.40	1.36	1.32	1.27	1.23	1.19	1.09
3.00	1.41	1.37	1.33	1.28	1.24	1.20	1.10
3.50	1.43	1.39	1.35	1.31	1.27	1.23	1.12
4.00	1.46	1.42	1.37	1.33	1.29	1.25	1.15

[†] Calculated at a silver price of US\$19.00/oz

Table 22.10 After Tax Silver Price Sensitivity – 2016 Reserves Case

After Tax Values [†]	Silver (US\$/oz)						
	10.00	12.00	15.00	17.00	19.00	24.00	30.00
NPV8% (US\$b)	6.73	6.78	6.85	6.89	6.94	7.05	7.19

[†] Calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz

Table 22.11 Cash Flow – 2016 Reserves Case

Cash Flow Statement (US\$M)	Year													Total
Year Number	1	2	3	4	5	6	7	8	9	10	11	21	31	
Year To											20	30	40	
Gross Revenue	954	923	1,189	1,402	1,735	1,993	3,380	4,434	4,879	4,622	35,422	13,698	8,175	82,806
Realization Costs	263	237	222	212	246	311	513	664	728	709	5,955	2,502	1,206	13,770
Net Sales Revenue	691	686	967	1,190	1,489	1,682	2,867	3,770	4,151	3,912	29,467	11,196	6,969	69,036
Site Operating Costs														
Mining	191	182	177	188	188	221	220	250	287	254	3,072	2,158	1,038	8,427
Processing and Tailings	285	295	297	279	266	292	326	328	329	327	3,248	3,193	2,445	11,911
G&A and Operations Support	100	93	94	96	97	96	96	96	92	88	851	586	387	2,771
Infrastructure and Other	65	84	91	41	69	57	69	83	69	30	373	359	371	1,761
Total Site Operating Costs	641	655	659	604	620	665	710	756	776	700	7,544	6,297	4,241	24,869
Operating Surplus / (Deficit)	51	31	307	585	869	1,016	2,157	3,013	3,375	3,213	21,923	4,899	2,728	44,167
Indirect Costs	171	182	180	187	182	178	182	188	178	164	1,746	1,190	848	5,576
Net Profit Before Income Tax	-121	-151	128	398	688	838	1,974	2,825	3,197	3,049	20,177	3,708	1,880	38,591
Income Tax	-	-	-	-	-	-	-	-	-	-	1,496	557	254	2,307
Net Profit After Income Tax	-121	-151	128	398	688	838	1,974	2,825	3,197	3,049	18,681	3,151	1,627	36,284
Capital Expenditure														
Expansion Capital	874	1,071	1,080	831	387	92	-	-	-	-	-	-	-	4,336
Sustaining Capital	82	101	58	351	424	373	397	430	320	350	1,912	866	424	6,088
VAT & Duties	79	82	66	102	75	44	43	47	35	38	209	99	47	967
Subtotal	1,035	1,254	1,205	1,285	886	509	440	477	354	388	2,121	964	472	11,391
Working Capital, Capitalised Operating Costs and Closure	-47	-34	-2	80	87	76	37	6	18	49	411	187	937	1,805
VAT & Duties (Capex)	3	1	-	5	2	3	1	-	1	5	44	21	-	86
Total Capital Expenditure	992	1,221	1,203	1,369	975	588	479	483	374	442	2,576	1,172	1,408	13,282
Net Cash Flow After Tax	-1,112	-1,372	-1,075	-971	-287	250	1,496	2,342	2,823	2,607	16,105	1,979	218	23,003

22.6 Comparison to 2014 OTTR

An analysis comparing the 2014 OTTR and 2016 OTTR financial models was carried out. The major changes between the two studies are the impact of the delay between the 2014 OTTR analysis and the project restart and the changes in metal prices and terms. Capital and operating costs were re-estimated for OTFS16.

Open pit mining has continued during the delay and there has been 81 Mt of mining depletion from the Oyut open pit. The underground mining scenario including development and production has remained the same.

22.7 TRQ May 2016 News Release

In the News Release dated 5 May 2016 TRQ announced the Oyu Tolgoi Notice to proceed. The NPV8 described in the News Release was US\$4.6b using metal prices of US\$2.86/lb Cu, US\$1,201/oz Au and US\$17.38/oz Ag. The 2016 Reserves Case base case NPV8% is US\$6.94b using metal prices of US\$3.00/lb Cu, US\$1,300/oz Au and US\$19.00/oz Ag. The major differences in NPV8% are due to the change in metal prices differences (NPV8% US\$0.9b), the change in discounting period to the start of 2017 (NPV8% US\$0.5b). Other differences are due to changes in operating cost (US\$0.5b) and other changes (NPV8% US\$0.4b) that were changed to make the 2016 Reserves Case consistent with the final OTFS16 analysis.

23 ADJACENT PROPERTIES

This section is not used.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Alternative Production Cases

Oyu Tolgoi is a very large project that includes five separate deposits. The long-term development of Oyu Tolgoi would involve the development of the resources on all deposits. Alternative Production Cases have been developed to provide early stage analysis of the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second Lift of Hugo North). Development of these deposits will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi.

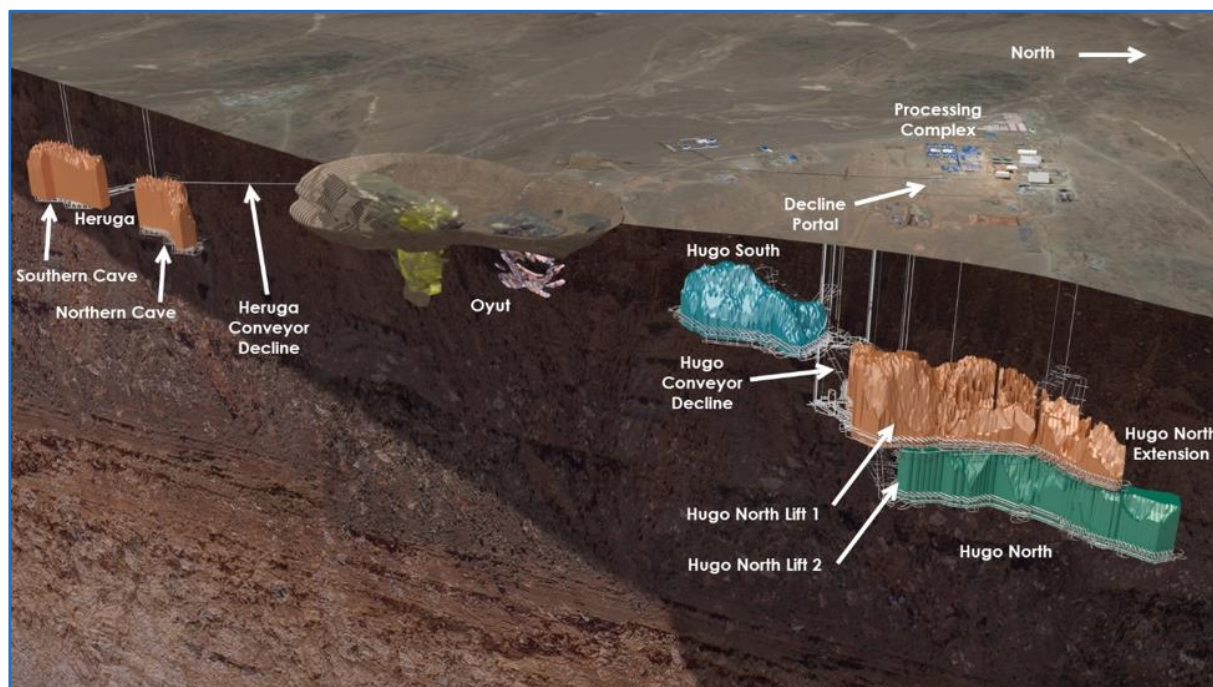
Accordingly, the analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

The 2016 Resources Case was prepared based on the OTFS16 Resources Case. The OTFS16 incorporates matters resolved between the shareholders and has been approved by the OT LLC board of directors and shareholders. The 2016 Resources Case, which is a baseline of the expansion analysis, assumes that the plant capacity remains at the planned OTFS16 average production rate of 110 kt/d (40 Mt/a).

The mine designs used for the Alternative Production Cases are shown schematically in Figure 24.1. The designs are:

• Oyut open pits	(2016 Mineral Reserves)
• Hugo North Lift 1 block cave (Panels 0–2)	(2016 Mineral Reserves)
• Hugo North Lift 1 block cave (Panels 3–5)	(Inferred)
• Hugo North Lift 2 block cave	(Inferred)
• Hugo South block cave	(Inferred)
• Heruga block cave	(Inferred)

Figure 24.1 Alternative Production Case Mine Designs



Three expansions to the 2016 Resources Case have been examined in order to confirm the long-term development strategy and timing. The analyses of the plant capacity expansions for Alternative Production Cases were prepared by OreWin with OT LLC and TRQ using the costs and assumptions from the 2016 Resources Case and from Phase 1 of Oyu Tolgoi. The costs include capital for the underground mines, plant and infrastructure. The plant capacities of these Alternative Production Cases are the production schedule averages not nominal capacity. The Alternative Production Cases and plant capacity assumptions are shown in Table 24.1.

Table 24.1 Alternative Production Cases and Plant Capacity Assumptions

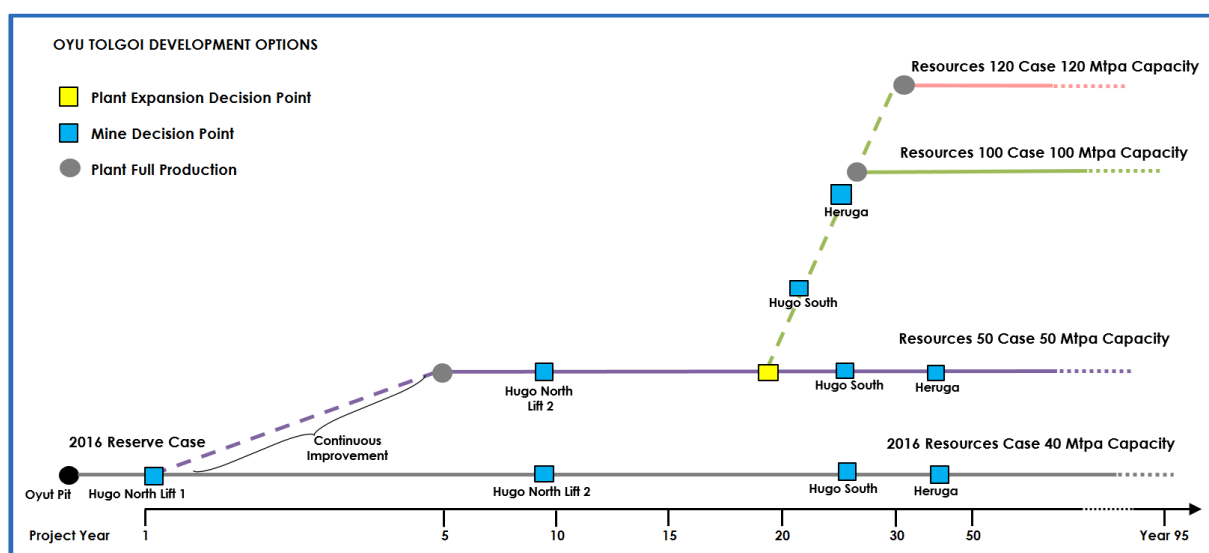
Alternative Production Case	Plant Capacity Assumptions
2016 Resources Case	Plant capacity 40 Mt/a for life.
Resources 50 Case	Plant capacity 40 Mt/a with a 5% improvement in throughput capacity per year for five years to 125% of initial capacity. The average production is 50 Mt/a.
Resources 100 Case	Resources 50 followed by an expansion to 100 Mt/a.
Resources 120 Case	Resources 50 followed by an expansion to 120 Mt/a.

The 2016 Resources Case assumes that the plant capacity remains constant and that the Oyut open pit and Hugo North Lift 1 are followed by Hugo North Lift 2, Hugo South, and Heruga.

The Alternative Production Cases present development options shown in the potential decision tree in Figure 24.2. There are decision points for each of the mines and plant expansions. The development options shown represent possible scenarios. The actual decisions will consider the results of mine and plant performance over the intervening years and will need to identify the optimum timing and size of both mine and any plant expansions that may be applicable.

Following the commencement of development of Hugo North Lift 1, the next decision point for the Oyu Tolgoi Project is the development of the Hugo North Lift 2 block cave, shown in Figure 24.2 at Year-10. Optimization and utilization of the installed underground haulage capacity of 140 kt/d (50 Mt/a) will need to be considered along with the results of the Hugo North Lift 1 development and cave performance when determining the optimal Hugo North Lift 2 scenario. If the performance of Hugo North Lift 1 is such that it has a faster ramp-up and/or greater final production rate than predicted in OTFS16 and if that performance was to apply to Hugo North Lift 2, then the decision point for plant expansions (shown in in Figure 24.2 at Year-20) would be brought forward.

Figure 24.2 Oyu Tolgoi Development Options



24.1.1 Mining Assumptions – Alternative Production Cases

A summary of the assumptions for the additional mines follows. The mine designs used for the Alternative Production Cases are:

- Oyut open pits (2016 Mineral Reserves)
- Hugo North Lift 1 block cave (Panels 0–2) (2016 Mineral Reserves)
- Hugo North Lift 1 block cave (Panels 3–5) (Inferred)
- Hugo North Lift 2 block cave (Inferred)
- Hugo South block cave (Inferred)
- Heruga block cave (Inferred)

24.1.1.1 Open Pit

The open pit designs, equipment selection assumptions, and production assumptions for the Alternative Production Cases are the same as those used for the 2016 Reserves Case. The work is based on the latest resource model ('mpst15.v9') and Base Data Template 31 (BDT31) metallurgical response parameters, unchanged from the 2014 Technical Report.

The optimization, designs, and production scheduling were completed using Measured and Indicated resources only, with Inferred Resources treated as waste. In the Alternative Production Cases the Inferred Resources within the pit designs were also treated as waste.

24.1.1.2 Hugo North Lift 1 – Panels 0, 1 and 2

The assumptions for Hugo North Lift 1 are the same as for 2016 Reserves Case. The Hugo North Lift 1 production and cost is the underlying (base) case for each of the Alternative Production Cases.

The overall layout of the Hugo Dummett deposits is shown in oblique projection view in Figure 24.3. The Hugo North Panel configuration is shown in Figure 24.4.

Figure 24.3 Hugo Dummett Mine Layout Plan (Oblique Projection)

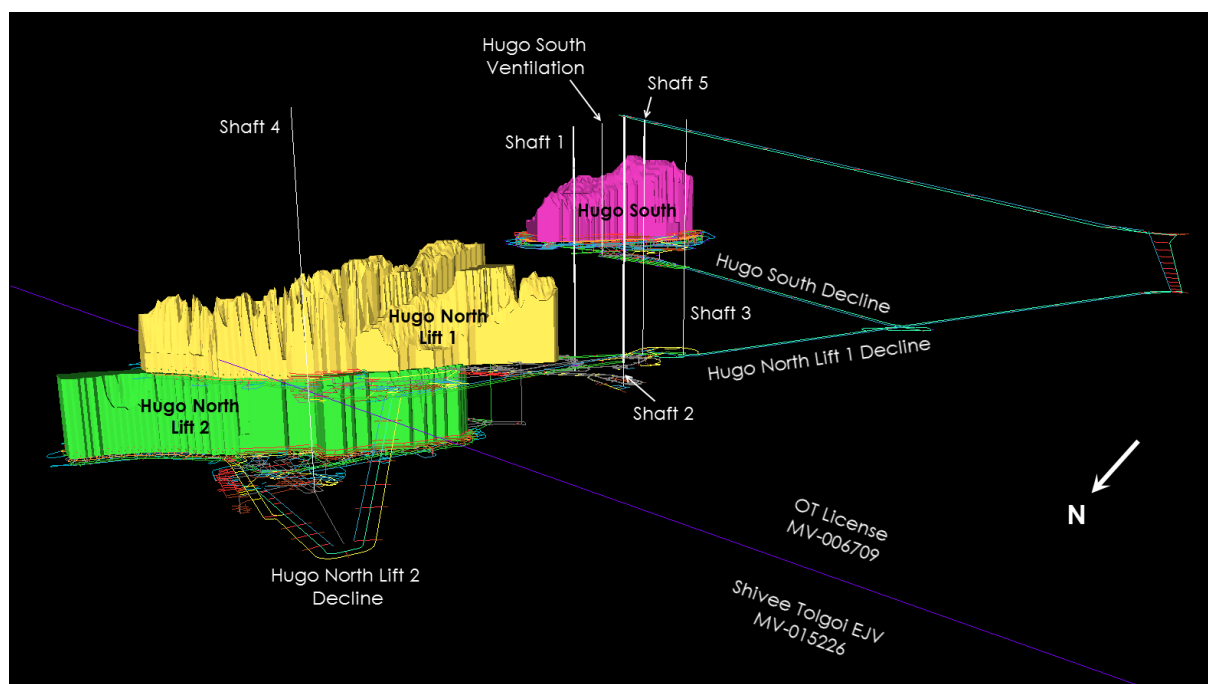
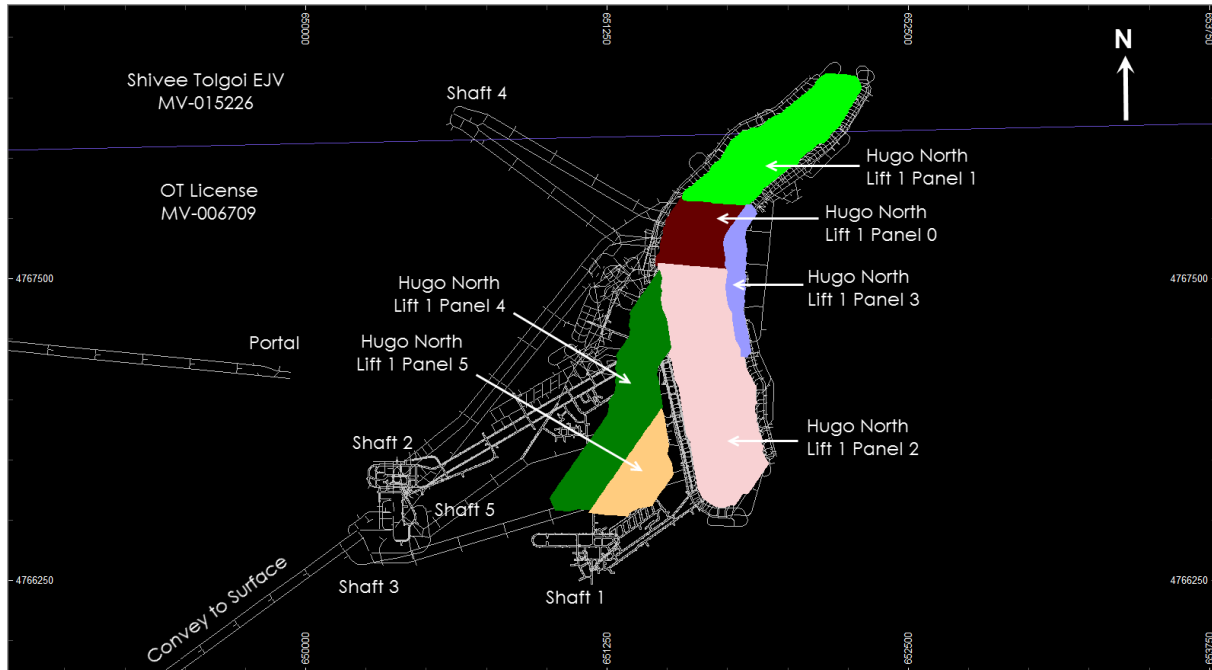


Figure 24.4 Hugo North Mine Panel Configuration Plan



24.1.1.3 Hugo North Lift 1 – Panels 3, 4, and 5

Hugo North Panels 3, 4, and 5 are on the same level as the higher grade Hugo North Panels 0 to 2. The production schedule is based on the Hugo North resource block model, with cut-off and shut-off grades determined by a project options evaluation finalized in mid-2014.

Panel 3 is a narrow panel on the eastern side of Panels 0 to 2, consisting of 178 drawpoints. Despite a mining width of only 60 m, the panel is anticipated to achieve the relatively low average height of draw of 220 m due to the favorable caveability predictions for the rock mass. Panel 3 is planned to ramp up to a maximum production rate of 20 kt/d, and to be completed before the underlying Hugo North Lift 2 area is mined.

Panels 4 and 5 are located on the western side of Lift 1 Panel 2 and consist of 1,111 drawpoints. The area has been split into two panels to maintain an undercut face length of less than 350 m. A 100 m-wide pillar is planned to be maintained between Panel 5 and Panel 2. At their closest point, Panels 4 and 5 are 120 m to the west of the underlying Hugo North Lift 2 Panel 2 boundary. While the latter is not expected to affect Lift 1 Panels 4 and 5, it is acknowledged in OTFS16 that further work is required to better understand and mitigate this risk.

Panels 4 and 5 are planned to be mined at a maximum production rate of 45 kt/d. The mine design includes an additional 70 km of lateral development and assumes use of the existing materials handling system, haulage level, crushers, workshops, main access ways, ventilation systems and other infrastructure.

24.1.1.4 Hugo North Lift 2

Hugo North Lift 2 aims to mine the vertical continuation of the high-grade core of the Hugo North deposit. During the mine planning phase, three extraction level elevations were tested: 300 m, 400 m, and 500 m below the Lift 1 elevation of –100 m. The schedule evaluations concluded that a block height of 400 m provided substantially higher value than the 300 m option because the production rate of 95 kt/d could be maintained with only minor impact on overall mined copper grade. The 400 m block height also improves the resource extraction compared with a 300 m lift. A cut-off and shut-off value of US\$17/t NSR was established during the options analysis by developing a range of schedules and evaluating the results.

Hugo North Lift 2 has been broken into three panels. The mine design, paneling strategy, and production strategy mirror those for Lift 1 because of the similar geometry and geotechnical conditions. Access to the Lift 2 mining area is expected to be via a decline from Lift 1 with ventilation extensions to Shafts 2, 3, 4, and 5. Ore will pass from trucks through one of two primary crushers off-footprint at the Lift 2 horizon and be transported to surface via a two-leg extension to the Lift 1 main incline conveyor. The Lift 2 mine design incorporates 229 km of lateral development and 1,340 m of shaft extensions. The Lift 2 work is based on the Hugo North resource block model.

24.1.1.5 Hugo South

During study work in 2013, both open pit and block cave mining methods were considered for the Hugo South deposit because the top of Hugo South is approximately 330 m below surface. If developed as an open pit, however, there would be a high risk of open pit slope failure being induced by block cave mining of the adjacent Hugo North deposit. Block caving was therefore selected as the most appropriate mining method for Hugo South. An extraction level of 520 mRL was selected in conjunction with US\$17/t NSR cut-off grade and US\$20/t NSR shut-off grade.

Hugo South will consist of four panels with more than 1,700 drawpoints and an average height of draw of 300 m. The panel configuration is designed to allow for initiation in the highest grade zone with lateral extensions into the more marginal areas later in the mine life. The mine is planned to ramp up to a maximum rate of 50 kt/d.

The production method is based on the same principles as the Hugo North caving areas, with production LHDs tipping to central passes and haulage-level trucks transporting from there to a single primary crusher. An additional 1.5 km long conveyor is required to transfer material onto the Hugo North main incline conveyor to surface. A drive parallel to the conveyor will provide the primary access to the mining area. A single 694 m long exhaust shaft is required to support the initial construction effort. In the longer term, parallel lateral ventilation drives connecting Hugo South with Shafts 3 and 5 will provide the remaining ventilation capacity as the Hugo North mine ramps down. The completed mine design includes an additional 131 km of lateral development.

24.1.1.6 Heruga

The Heruga deposit is also suitable for block cave extraction due to its comparatively low grades and large extent. Given the 500 m depth to the top of the resource, Heruga is considered too deep for open pit mining. The layout of the Heruga deposits is shown in Figure 24.5.

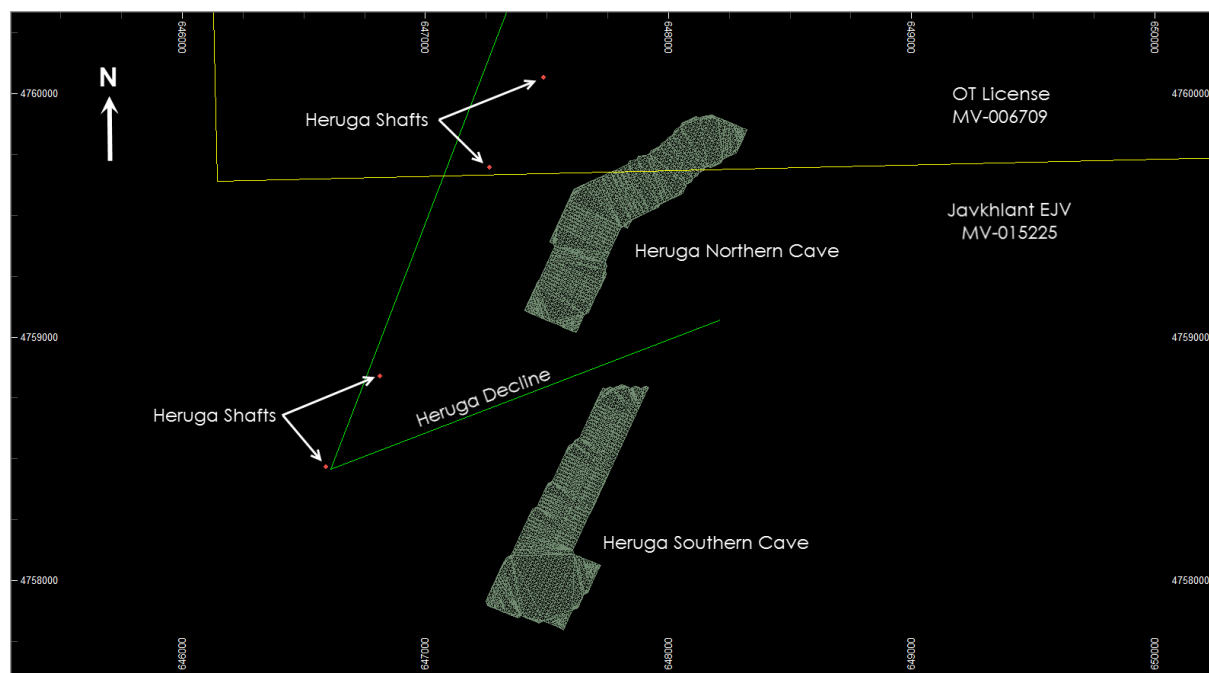
The Heruga footprint size and elevation were reviewed in an NSR model that included value and costs from the addition of a molybdenum recovery circuit and secondary crushing capability to support processing of harder Heruga plant feed at the desired production rate. In each of the Alternative Production Cases the molybdenum circuit is added when Heruga production commences to take advantage of the higher molybdenum grades of the Heruga resources.

Given the dip of the mineralization at Heruga, combined with its relatively low grade, a two-level split cave was considered. The footprint elevations selected are: -20 mRL for the southern panel and 350 mRL for the northern panel.

Drawpoints for each layout were determined by the need for each to cover the incremental cost of footprint expansion of US\$600,000 per drawpoint and an estimated operating cost of US\$17/t. The production rate from Heruga in this schedule is 95 kt/d, which requires that both panels be in operation at the same time.

Ore will be withdrawn by LHDs, dumped in central passes, and fed to trucks for delivery to a large gyratory crusher station, one for each panel. An incline conveyor will be used between the lifts. The southern panel will have more than 1,300 drawpoints and the northern panel more than 1,200 drawpoints, for a total Heruga inventory exceeding 2,600 drawpoints. The cost estimate for Heruga is based on data from the OTFS16 Hugo North Lift 1 costs.

Figure 24.5 Heruga Mine Layout Plan



24.1.2 Costs – Alternative Production Cases

Separate capital and operating costs were prepared for the plant, infrastructure, and individual mines for the OTFS16 Resources Cases. Costs from the 2016 Reserves Case and the 2016 Resources Case were used to prepare the costs for the Resources 50, Resources 100, and Resources 120 Cases.

24.1.2.1 2016 Resources Case Cost Estimation

The 2016 Resources Case costs were based on the cost estimates developed for the OTFS16, as described below.

The production and development quantities for Hugo North Lift 1 Panels 3 to 5, Hugo North Lift 2, and Hugo South were based on preliminary designs and block cave analysis. Heruga production was based on block cave production modelling, and development quantities for the two-level split cave layout were calculated from the anticipated development required for the number of drawpoints, plus an additional allowance for the increased depth of the lower lift.

Capital costs were based on the OTFS16 costs for Lift 1, with adjustments to build the additional facilities currently thought to be required for each mining area.

Detailed Engineering, EPCM / PMC, and Owner's Project Management costs are included at the same percentages of the value of works covered in OTFS16.

The development and development costs were factored for differences associated in the designs for Hugo North Lift 1 Panels 3 to 5, Hugo North Lift 2, Hugo South, and Heruga. Development costs for Heruga were reduced to reflect productivity improvements associated with expected better ground conditions and less ground support compared to Hugo North Lift 1.

Power costs were adjusted using the Hugo North Lift 1 assumptions for differences in ventilation and vertical haulage. Fixed costs for the underground operations were based on expected annual costs for the later stages of Hugo North Lift 1.

The infrastructure and G&A costs are based on the OTFS16 annual costs for tailings storage facilities and the concentrator variable costs.

In the 2016 Resources Case, open pit mining costs are based on a unit cost of US\$2.26/t for mining at the rate of 40 Mt/a in 2025. This cost was then increased by 1% per year to allow for extra costs associated with hauling from deeper benches while maintaining the larger footprint of the pit.

The mine closure costs are factored up for the number and areas of tailings cells requiring remediation and for the surface facilities that need to be demolished together with surface rehabilitation.

24.1.2.2 Resources 50, Resources 100, and Resources 120 Cases Costs

For the Alternative Production Cases, separate capital costs estimates were prepared using the 2016 Reserves Case (base case) and OTFS16 Resources Case capital costs for equipment and installation of the plant and infrastructure, factored for each expansion. The mining costs for the underground caves in the Alternative Production Cases were taken from the OTFS16 Resources Case and shifted to suit the timing for the production schedule in each case. Underground capital and operating costs in the Alternative Production Cases are the same as for the OTFS16 Resources Case, except where the underground production rate in the Hugo Dummett deposits is greater than 140 kt/d, in which case costs for additional shafts were added to allow for the increase in the hoisting capacity.

Open pit costs, based on the Alternative Production Cases open pit production, were prepared using the unit cost rates and a similar method as the base case. G&A and Operations Support costs were estimated separately for each Alternative Production Case using the assumptions from the OTFS16 Resources Case and making allowances for changes in personnel numbers and other inputs.

24.1.3 Sensitivity Analysis – Alternative Production Cases

A comparison was made of the 2016 Reserves Case (base case) with the Alternative Production Cases. Four cost sensitivity options were analyzed. Each sensitivity assumes an improvement in the costs and productivities. The improvements could be the result of optimization and efficiencies from the experience that will be gained over the years of developing and operating the plant and mines at Oyu Tolgoi. The cost assumptions are:

- Underground construction capital costs reduced by 30%. This excludes the surface works, shafts but does include the underground construction costs for development and construction.
- Operating costs reduced by 15%.
- G&A costs are assumed to reach a long-term average annual cost of US\$50M from Year-7. This cost is based on a review of costs from studies of other copper projects.
- Rail freight available to the project after 2020 and the concentrate freight cost is reduced to US\$25/t.

The after tax NPV8% results of the comparisons are shown in Table 24.2. Option 'A' compares the results using the base case assumptions and Options 'B' to 'E' show the results of applying the sensitivities cumulatively. The results indicate that for the Option 'A' (base case assumptions) there is an improvement in after tax NPV8% for the 2016 Resources Case and the Resources 50 Case has the highest value. When each of the options is applied cumulatively there is an increase in value and the Resources 100 Case has the highest value for Options 'B' and 'C' and the Resources 120 Case has the highest value for Options 'D' and 'E'. The after tax NPV8% results based on US\$3.50/lb copper and US\$1,400/oz gold is shown in Table 24.3.

Table 24.2 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.00/lb Copper and US\$1,300/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	6.94	8.37	9.32	8.88	8.80
B	Underground Construction Capital Reduced by 30%	US\$b	7.85	9.64	10.57	10.59	10.51
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	8.97	10.20	11.86	12.00	11.98
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	9.14	10.43	12.20	12.50	12.57
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	9.62	11.02	13.15	13.58	13.69

Note: Based on US\$3.00/lb copper, US\$1,300/oz gold, US\$19.00/oz silver, and 8% discount rate.

Table 24.3 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.50/lb Copper and US\$1,400/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	10.80	12.95	14.36	14.59	14.69
B	Underground Construction Capital Reduced by 30%	US\$b	11.72	14.21	15.62	16.30	16.40
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	12.83	14.78	16.91	17.71	17.87
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	13.00	15.01	17.25	18.21	18.46
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	13.48	15.59	18.20	19.29	19.58

Note: Based on US\$3.50/lb copper, US\$1,400/oz gold, US\$19.00/oz silver, and 8% discount rate.

24.1.4 Project Results – 2016 Resources Case

The 2016 Resources Case reflects the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second lift of Hugo North), which will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi. Accordingly, the analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

Processing, and concentrate and metal production, for the 2016 Resources Case are summarized in Figure 24.6 and Figure 24.7 and the production schedule is shown in Table 24.6.

Figure 24.6 Processing – 2016 Resources Case

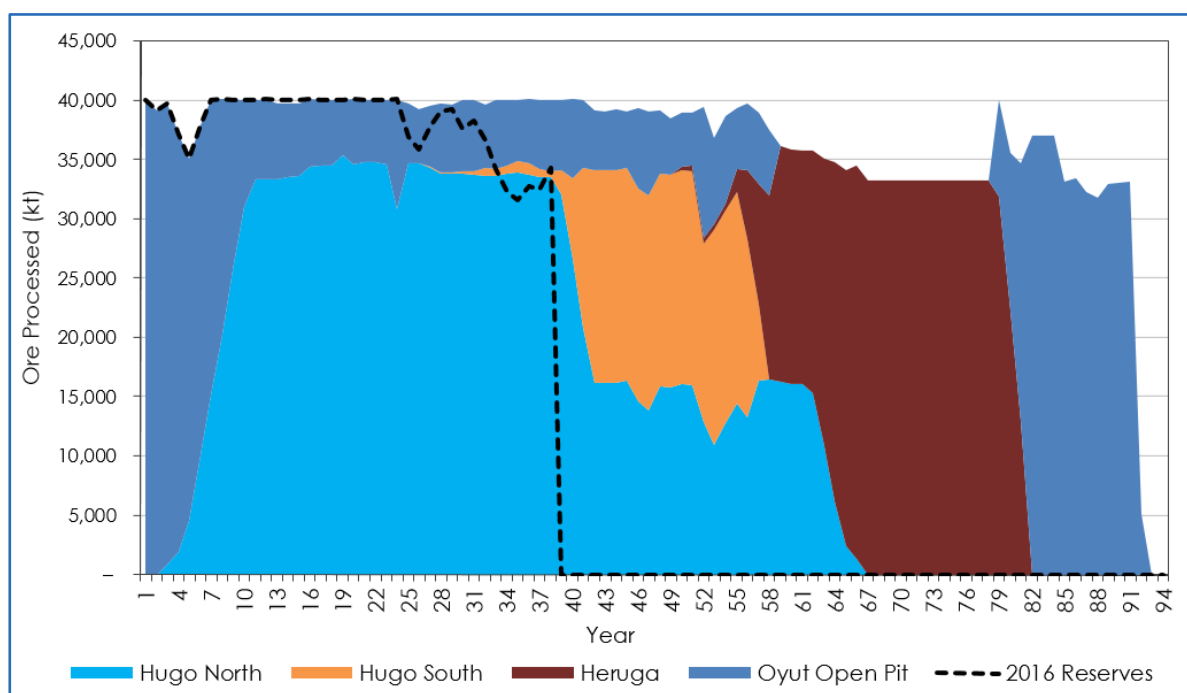
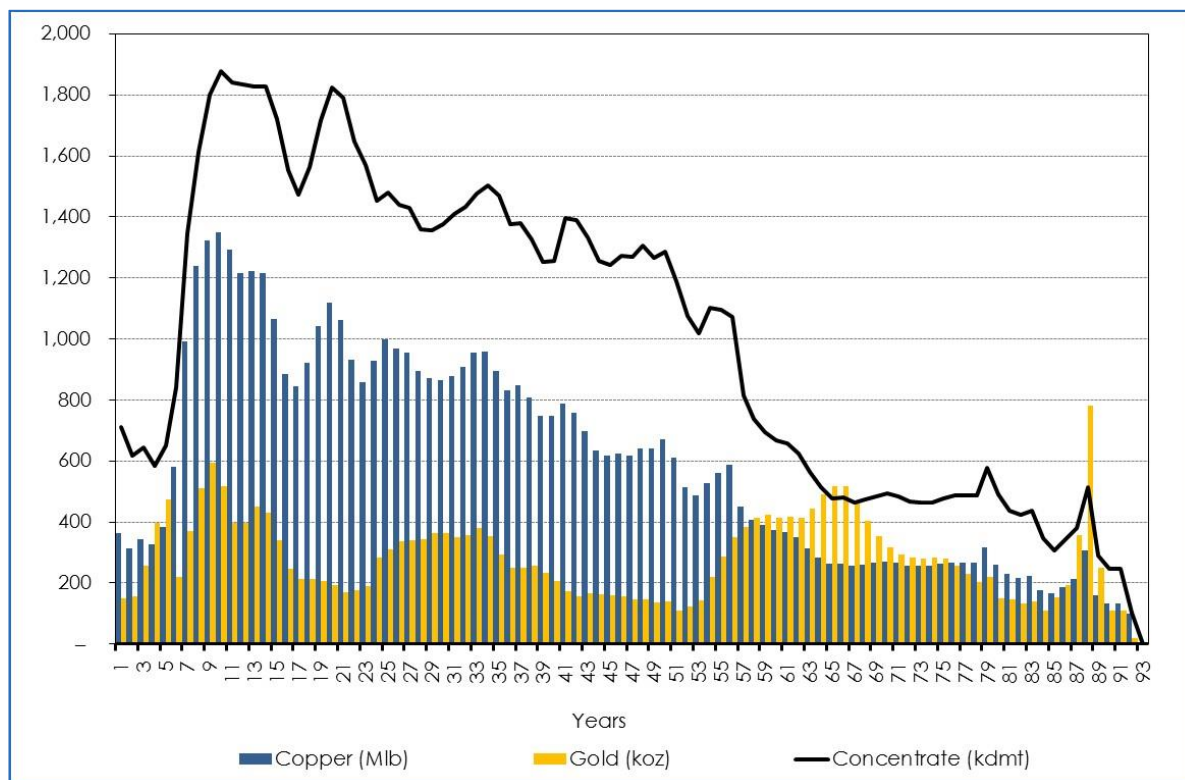


Figure 24.7 Concentrate and Metal Production – 2016 Resources Case



The underground tonnages and grades from Hugo North, Hugo South, and Heruga are summarized in Table 24.4.

Table 24.4 Underground Tonnage and Grades

Deposit	Mined (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Recovered Metal		
					Cu (Mlb)	Au (koz)	Ag (koz)
Hugo North	1,477	1.23	0.33	2.95	36,309	12,828	119,169
Hugo South	298	1.07	0.06	2.07	6,284	448	16,815
Heruga	699	0.42	0.41	1.52	5,544	7,265	27,929
Total	2,474	0.98	0.32	2.44	48,137	20,541	163,914

A summary of the production and financial results for the 2016 Reserves Case are shown in Table 24.5.

Table 24.5 Summary Production and Financial Results – 2016 Resources Case

Description	Units	2016 Resources Case
Total Processed	bt	3.4
Cu Grade	%	0.83
Au Grade	g/t	0.31
Ag Grade	g/t	2.09
Copper Recoverable	blb	55.3
Gold Recoverable	Moz	26.3
Silver Recoverable	Moz	191.1
Life	Years	92
Expansion Capital	US\$b	9.73
NPV8% After Tax	US\$b	8.37
IRR After Tax	%	21
Payback Period	Years	8

Notes:

1. The 2016 Resources Case reflects the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second Lift of Hugo North), which will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of the project. Accordingly, the 2016 Resources Case is effectively a Preliminary Economic Assessment under NI 43-101.
2. Insofar as the 2016 Resources Case includes an economic analysis that is based, in part, on Inferred Mineral Resources, the 2016 Resources Case does not have as high a level of certainty as the 2014 Reserves Case. The 2016 Resources Case is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the 2016 Resources Case will be realized.
3. Metal prices used for calculating the financial analysis are as follows: long term copper at US\$3.00/lb; gold at US\$1,300/oz; and silver at US\$19.00/oz. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
4. For mine planning the metal prices used to calculate block model Net Smelter Returns were copper at US\$3.01/lb; gold at US\$1,250/oz; and silver at US\$20.37/oz.
5. For the open pit processing and general administration, the following operating costs have been used to determine cut-off grades: Southwest at US\$8.37/t, Central Chalcocite, Central Covellite, and Central Chalcopyrite at US\$7.25/t and the underground (including some mining costs) costs are based on US\$15.34/t.
6. For the underground block caves, all Mineral Resources within a design shell have been included.
7. Economic analysis has been calculated from the start of 2017 and excludes \$0.46b expansion capital from 2016. Costs shown are real costs not nominal costs. Expansion capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

Table 24.6 Production Schedule – 2016 Resources Case

		Year																		Total	
Year Number		1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81		91
Year To												20	30	40	50	60	70	80	90		100
Open Pit																					
Plant Feed	Mt	40	39	39	35	31	28	25	20	14	9	59	57	58	54	53	–	21	329	38	949
Waste	Mt	66	47	43	75	54	60	47	26	38	29	288	304	311	159	65	–	11	68	–	1,691
Total Open Pit	Mt	106	86	82	110	85	88	72	46	52	38	347	362	368	213	118	–	33	397	38	2,640
Underground	Mt	–	–	1	2	5	10	15	20	26	31	340	341	342	337	329	343	320	13	–	2,474
Processed	Mt	40	39	40	37	35	38	40	40	40	40	399	398	400	391	381	343	342	342	38	3,423
	NSR	25.33	23.03	28.51	34.14	42.99	45.96	73.98	94.22	102.09	101.65	78.34	68.45	63.37	45.32	39.84	35.33	28.58	22.44	13.73	49.45
	Cu %	0.53	0.47	0.49	0.48	0.57	0.79	1.27	1.55	1.65	1.68	1.36	1.18	1.09	0.89	0.68	0.45	0.41	0.34	0.24	0.83
	Au g/t	0.17	0.18	0.28	0.45	0.56	0.24	0.36	0.49	0.57	0.49	0.30	0.28	0.28	0.16	0.30	0.50	0.29	0.31	0.17	0.31
	Ag g/t	1.29	1.25	1.26	1.39	1.60	2.02	2.68	3.21	3.40	3.43	2.85	2.75	2.83	1.99	1.75	1.73	1.42	1.16	1.03	2.09
	As ppm	103.60	139.21	82.65	47.97	25.33	52.14	75.41	59.57	54.21	55.79	73.62	197.55	67.04	95.74	72.49	1.72	5.18	53.04	41.54	72.83
	Mo ppm	55.85	57.83	45.72	50.18	55.04	38.62	53.11	47.77	35.80	28.80	35.28	30.40	36.54	51.76	59.00	104.67	122.99	54.34	45.20	58.40
Bulk Concentrate	Conc. kt	710	619	644	585	650	842	1,346	1,613	1,800	1,878	17,183	14,902	13,883	13,021	9,469	5,228	4,884	3,727	284	93,268
	Conc. Cu %	23.24	23.05	24.13	25.42	26.86	31.33	33.41	34.87	33.35	32.59	28.58	28.41	28.02	23.32	23.54	25.13	24.89	24.43	24.42	26.90
	Conc. Au g/t	6.53	7.85	12.37	21.13	22.73	8.16	8.53	9.83	10.28	8.56	5.58	6.00	6.57	3.68	9.41	25.83	15.81	19.82	13.87	8.76
	Conc. Ag g/t	54.23	58.59	59.75	68.66	69.62	74.70	66.92	67.94	64.67	62.76	56.47	62.27	68.82	49.52	57.55	92.70	81.12	79.51	99.04	63.73
	Conc. As ppm	2,597	4,114	2,458	1,207	320	601	994	741	706	728	1,123	3,510	1,250	1,903	1,838	75	114	1,198	519	1,600
	Conc. Mo ppm	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	3,944	5,514	3,198	3,905	649
	Conc. F ppm	613	612	588	550	494	499	509	536	553	563	618	572	534	836	597	62	32	491	463	551
Recovered Metal	Copper blb	0.4	0.3	0.3	0.3	0.4	0.6	1.0	1.2	1.3	1.3	10.8	9.3	8.6	6.7	4.9	2.9	2.7	2.0	0.2	55.3
	Gold Moz	0.1	0.2	0.3	0.4	0.5	0.2	0.4	0.5	0.6	0.5	3.1	2.9	2.9	1.5	2.9	4.3	2.5	2.4	0.1	26.3
	Silver Moz	1.2	1.2	1.2	1.3	1.5	2.0	2.9	3.5	3.7	3.8	31.2	29.8	30.7	20.7	17.5	15.6	12.7	9.5	0.9	191.1

24.1.4.1 Economics – 2016 Resources Case

The base case price assumptions for the financial evaluation are the same as those used for the 2016 Reserves Case. The metal prices used in the initial years and the long term prices are shown in Table 24.7. A summary of the financial results is shown in Table 24.8 and the mining production statistics are shown in Table 24.9. Mine site cash costs, revenues, operating costs, and capital costs are shown in Table 24.10 to Table 24.12.

Table 24.7 Metal Price Assumptions – 2016 Resources Case

Metal	Unit	Year				
		1	2	3	4	5 Onwards
Copper	US\$/lb	2.15	2.36	2.58	2.79	3.00
Gold	US\$/oz	1,300	1,300	1,300	1,300	1,300
Silver	US\$/oz	19.00	19.00	19.00	19.00	19.00
Molybdenum	US\$/lb	7.50	7.50	7.50	7.50	7.50

Table 24.8 Financial Results – 2016 Resources Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	59.59	48.22
	5%	17.07	14.75
	6%	13.93	12.15
	7%	11.45	10.07
	8%	9.46	8.37
	9%	7.83	6.97
	10%	6.49	5.80
IRR (%)	–	21	21
Project Payback Period (Years)	–	8	8

Table 24.9 Mining Production Statistics – 2016 Resources Case

		2016 Resources Case	5-Year Average	10-Year Average	2016 Resources Case Average
Quantity Ore Treated	Mt	3,423	38.19	38.89	37.21
Copper Feed Grade	%	0.83	0.51	0.96	0.83
Gold Feed Grade	g/t	0.31	0.32	0.38	0.31
Silver Feed Grade	g/t	2.09	1.35	2.17	2.09
Copper Recoveries	%	88	81	88	88
Gold Recoveries	%	78	73	77	77.74
Silver Recoveries	%	83	77	82	83
Copper Concentrate	Mt (dry)	93.3	0.6	1.1	1.0
Copper Concentrate Grade	%	27	25	31	27
Contained Metal in Concentrate					
Copper	Mt	25.1	0.2	0.3	0.3
Copper	blb	55.3	0.3	0.7	0.6
Gold	Moz	26.3	0.3	0.4	0.3
Silver	Moz	191.1	1.3	2.2	2.1

Table 24.10 Unit Operating Costs by Copper Production – 2016 Resources Case

Description	US\$/lb Payable Copper
Mine Site Cash Cost	1.84
By-product Credit	0.69
Cash Costs (Net of By-product Credit)	1.15

Table 24.11 Operating Costs and Revenues – 2016 Resources Case

	US\$b	US\$/t Ore Milled		
	Total 2016 Resources Case	5-Year Average	10-Year Average	2016 Resources Case Average
Revenue				
Gross Sales Revenue	196.11	32.48	64.59	57.28
Less: Realization Costs				
Realization Costs	23.00	4.49	7.03	6.72
Government Royalty	10.21	1.69	3.36	2.98
Total Realization Costs	33.21	6.18	10.39	9.70
Net Sales Revenue	162.90	26.30	54.21	47.58
Less: Site Operating Costs				
Mining (all sources)	17.24	4.87	5.29	5.03
Processing and Tailings	26.21	7.57	7.94	7.66
G&A and Operations Support	6.77	2.52	2.44	1.98
Infrastructure and Other	4.82	1.84	1.67	1.41
Government Fees & Charges	5.75	2.00	2.00	1.68
Management and JV Payments	6.96	2.72	2.51	2.03
Total Site Operating Costs	67.76	21.52	21.85	19.79
Operating Margin	95.14	4.78	32.35	27.79

**Table 24.12 Total Project Expansion and Sustaining Capital Cost –
2016 Resources Case**

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.42	1.42
Underground	7.12	15.89	23.01
Concentrator	0.14	0.42	0.57
Infrastructure	0.35	0.63	0.98
Tailings Storage Facility (TSF)	–	2.62	2.62
Subtotal	7.62	20.98	28.60
Construction Indirect	0.86	–	0.86
Contractor Execution – EPCM	0.31	–	0.31
Owner Execution	0.43	0.44	0.87
GOM Fees & Charges – Mongolian VAT	0.51	1.88	2.39
Total	9.73	23.31	33.03

Notes:

1. Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. The 2016 Resources Case total capital cost excludes capital costs for the year 2016.

The changes in financial results for a range of copper and gold prices, calculated at a silver price of US\$19.00/oz, are shown in Table 24.13.

The changes in financial results for a range of silver prices, calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz, are shown in Table 24.14.

A cumulative cash flow for the 2016 Resources Case is depicted in Figure 24.8 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and a complete cash flow is provided in Table 24.15.

Table 24.13 After Tax Metal Price Sensitivity – 2016 Resources Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-3.22	-2.84	-2.47	-2.09	-1.72	-1.34	-0.40
2.00	-1.54	-1.16	-0.79	-0.41	-0.04	0.34	1.28
2.50	2.67	3.04	3.42	3.79	4.17	4.54	5.48
2.80	5.19	5.57	5.94	6.32	6.69	7.06	8.00
3.00	6.87	7.25	7.62	8.00	8.37	8.75	9.68
3.50	11.07	11.45	11.82	12.20	12.57	12.95	13.89
4.00	15.28	15.65	16.03	16.40	16.78	17.15	18.09
Project After Tax IRR (%)							
1.80	-	-	-	-	-	-	6
2.00	-	-	-	6	8	9	11
2.50	13	14	15	15	16	17	18
2.80	17	17	18	19	19	20	22
3.00	19	19	20	21	21	22	23
3.50	23	24	24	25	25	26	28
4.00	27	27	28	28	29	30	31
Project Payback After Tax (Years)							
1.80	-	14.0	13.3	12.8	12.3	11.8	10.7
2.00	12.3	11.9	11.5	11.1	10.7	10.4	9.8
2.50	9.7	9.5	9.4	9.2	9.1	8.9	8.6
2.80	9.0	8.9	8.8	8.7	8.6	8.4	8.2
3.00	8.7	8.6	8.5	8.4	8.3	8.2	8.0
3.50	8.1	8.0	7.9	7.8	7.8	7.7	7.5
4.00	7.6	7.5	7.5	7.4	7.4	7.3	7.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.27	1.22	1.18	1.13	1.09	1.04	0.93
2.00	1.28	1.23	1.19	1.14	1.10	1.05	0.94
2.50	1.30	1.26	1.21	1.17	1.12	1.08	0.97
2.80	1.32	1.27	1.23	1.18	1.14	1.09	0.98
3.00	1.33	1.28	1.24	1.19	1.15	1.10	0.99
3.50	1.35	1.31	1.26	1.22	1.17	1.13	1.02
4.00	1.38	1.33	1.29	1.24	1.20	1.15	1.04

[†] Calculated at a silver price of US\$19.00/oz

Table 24.14 After Tax Silver Price Sensitivity – 2016 Resources Case

After Tax Values†	Silver (US\$/oz)						
	10.00	12.00	15.00	17.00	19.00	24.00	30.00
NPV8% (US\$b)	8.13	8.18	8.26	8.32	8.37	8.51	8.67

† Calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz

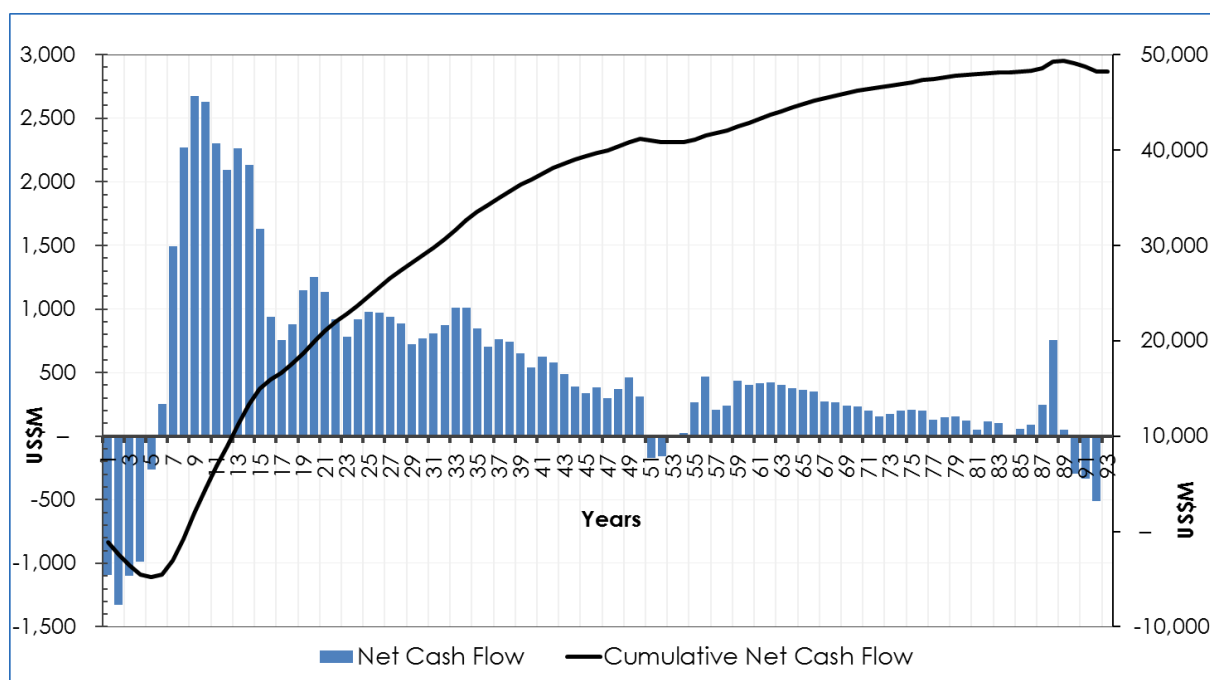
Figure 24.8 Cumulative Cash Flow – 2016 Resources Case


Table 24.15 Cash Flow – 2016 Resources Case

Cash Flow Statement (US\$M)	Year																			Total
Year Number	1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81	91	
Year To											20	30	40	50	60	70	80	90	100	
Gross Revenue	954	923	1,189	1,402	1,735	1,993	3,380	4,286	4,643	4,616	35,663	31,054	28,958	21,462	17,976	14,421	11,473	9,115	868	196,110
Realization Costs	263	237	222	212	246	311	513	632	693	709	6,022	5,589	4,854	4,180	3,213	2,019	1,787	1,374	132	33,211
Net Sales Revenue	691	686	967	1,190	1,489	1,682	2,867	3,654	3,950	3,907	29,641	25,464	24,104	17,282	14,763	12,402	9,686	7,740	735	162,899
Site Operating Costs																				
Mining	191	182	175	190	192	224	222	218	238	227	2,641	2,499	2,419	2,076	1,773	1,129	1,542	989	109	17,237
Processing and Tailings	281	302	304	286	273	299	333	337	337	336	3,330	3,282	3,251	3,119	2,586	2,442	2,430	2,416	271	26,213
G&A and Operations Support	100	94	94	96	97	96	96	96	92	88	853	839	838	838	775	525	525	524	105	6,770
Infrastructure and Other	64	85	91	42	70	57	69	73	50	50	567	533	596	785	701	379	379	192	38	4,820
Total Site Operating Costs	637	662	663	614	631	676	720	724	717	701	7,391	7,153	7,105	6,818	5,833	4,475	4,877	4,121	524	55,041
Operating Surplus / (Deficit)	55	23	303	575	858	1,006	2,147	2,930	3,232	3,206	22,250	18,311	16,999	10,463	8,930	7,927	4,809	3,620	212	107,858
Indirect Costs	169	180	182	190	181	176	180	178	163	155	1,454	2,227	1,921	1,378	1,236	913	910	778	146	12,717
Net Profit Before Income Tax	-114	-156	121	386	676	830	1,967	2,753	3,070	3,051	20,796	16,084	15,078	9,085	7,694	7,014	3,899	2,842	65	95,141
Income Tax	-	-	-	-	-	-	-	-	-	-	1,375	3,198	2,727	1,423	810	979	450	410	-	11,373
Net Profit After Income Tax	-114	-156	121	386	676	830	1,967	2,753	3,070	3,051	19,421	12,886	12,351	7,662	6,884	6,035	3,449	2,432	65	83,768
Capital Expenditure																				
Expansion Capital	874	1,072	1,080	831	387	92	-	9	18	24	1,553	247	877	173	1,978	-	-	-	-	9,216
Sustaining Capital	69	53	76	353	392	361	395	409	306	330	1,937	3,099	3,026	3,030	2,888	2,498	1,617	562	22	21,423
VAT & Duties	78	77	68	103	72	43	43	45	34	37	296	266	290	245	286	199	141	66	2	2,392
Subtotal	1,021	1,203	1,224	1,287	851	496	437	464	358	391	3,786	3,613	4,193	3,449	5,152	2,697	1,757	628	24	33,031
Working Capital, Capitalised Operating Costs and Closure	-44	-31	-3	80	88	75	37	15	36	29	217	240	189	-28	21	-1	-2	625	886	2,429
VAT & Duties (Capex)	3	1	-	5	2	3	1	1	3	3	24	27	20	-	-	-	-	-	-	92
Total Capital Expenditure	980	1,172	1,221	1,372	941	574	475	480	398	423	4,027	3,879	4,402	3,421	5,173	2,696	1,755	1,253	911	35,552
Net Cash Flow After Tax	-1,094	-1,328	-1,100	-986	-264	256	1,492	2,273	2,672	2,628	15,395	9,006	7,950	4,241	1,710	3,339	1,693	1,179	-845	48,216

24.1.5 Results – Alternative Production Cases

Oyu Tolgoi is a very large project that includes five separate deposits. The long-term development of Oyu Tolgoi would involve the development of the resources on all deposits. Alternative Production Cases have been developed to provide early stage analysis of the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second Lift of Hugo North). Development of these deposits will require separate development decisions in the future based on then-prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

24.1.5.1 Production – Resources 50 Case

A summary of the production for the Resources 50 Case is shown in Figure 24.9 to Figure 24.14. For reference the charts also show the 2016 Reserves Case and 2016 Resources Case. The production schedule is shown in Table 24.16. Total tonnes in the production schedule for the Resources 50 Case, the Resources 100 Case, and the Resources 120 Case are different from the 2016 Resources Case because they do not include a stockpile of 35Mt at 0.35% copper, 0.13 g/t gold, and 0.81 g/t silver.

Figure 24.9 Processing Production – Resources 50 Case

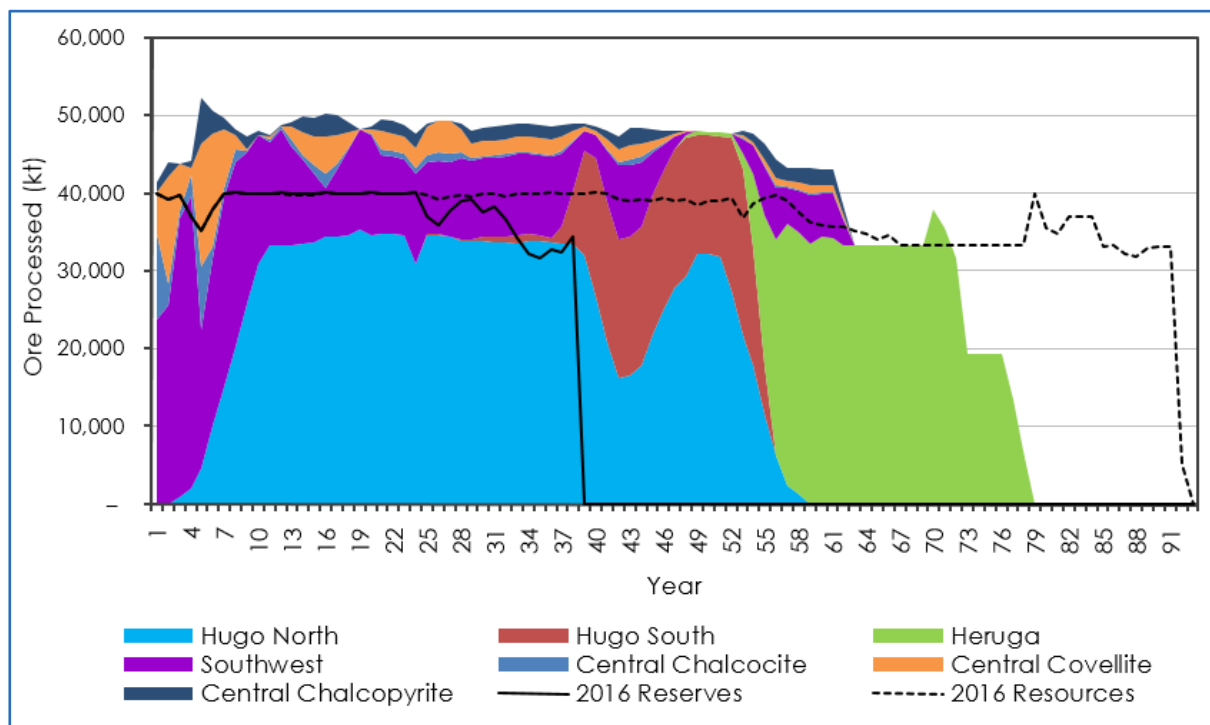


Figure 24.10 Concentrate Production – Resources 50 Case

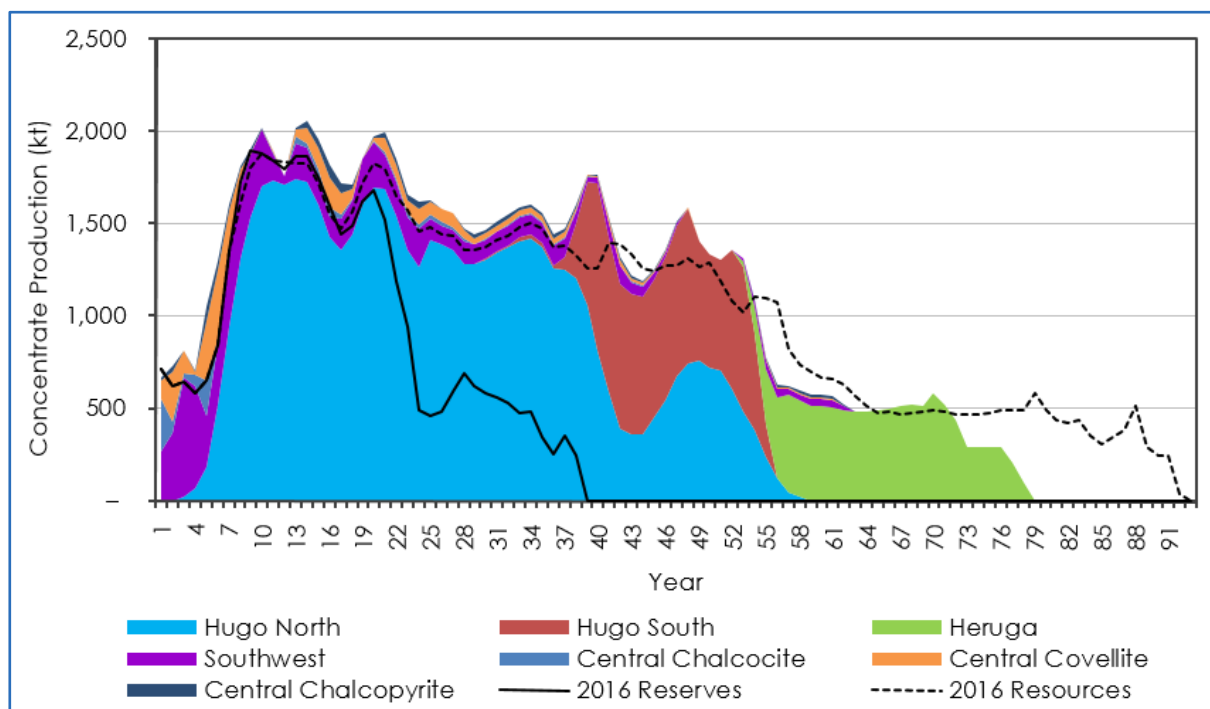


Figure 24.11 Recovered Copper Production – Resources 50 Case

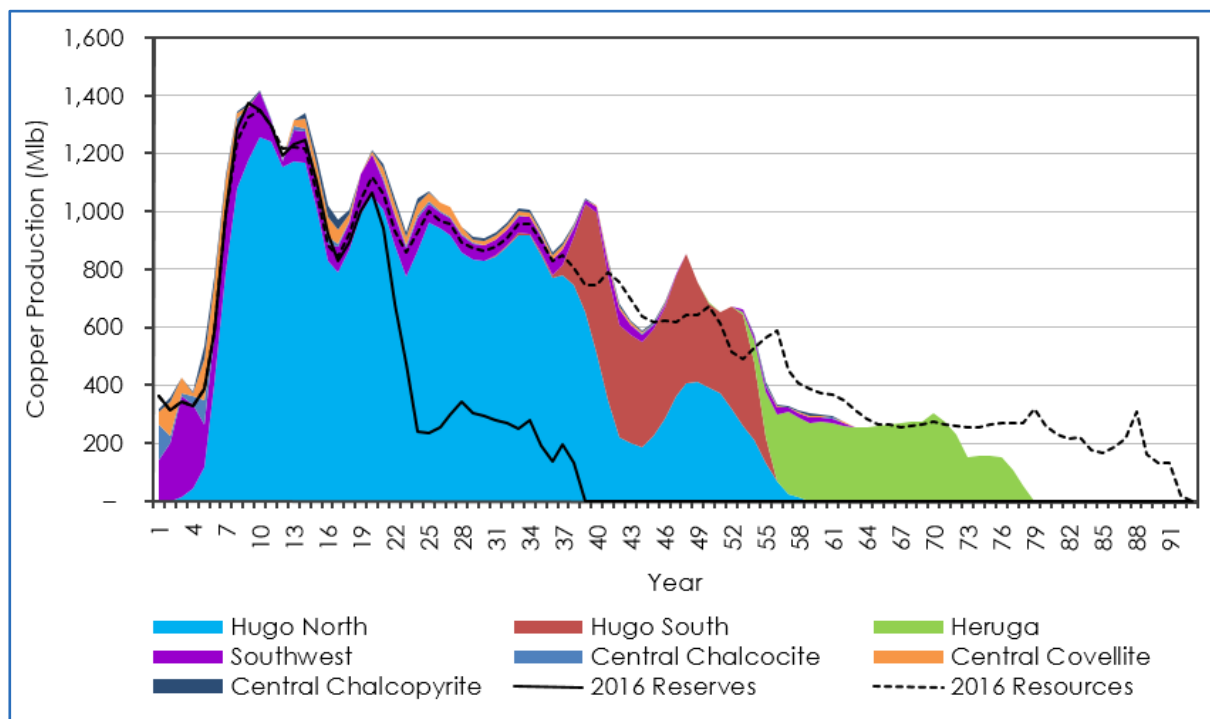


Figure 24.12 Recovered Gold Production – Resources 50 Case

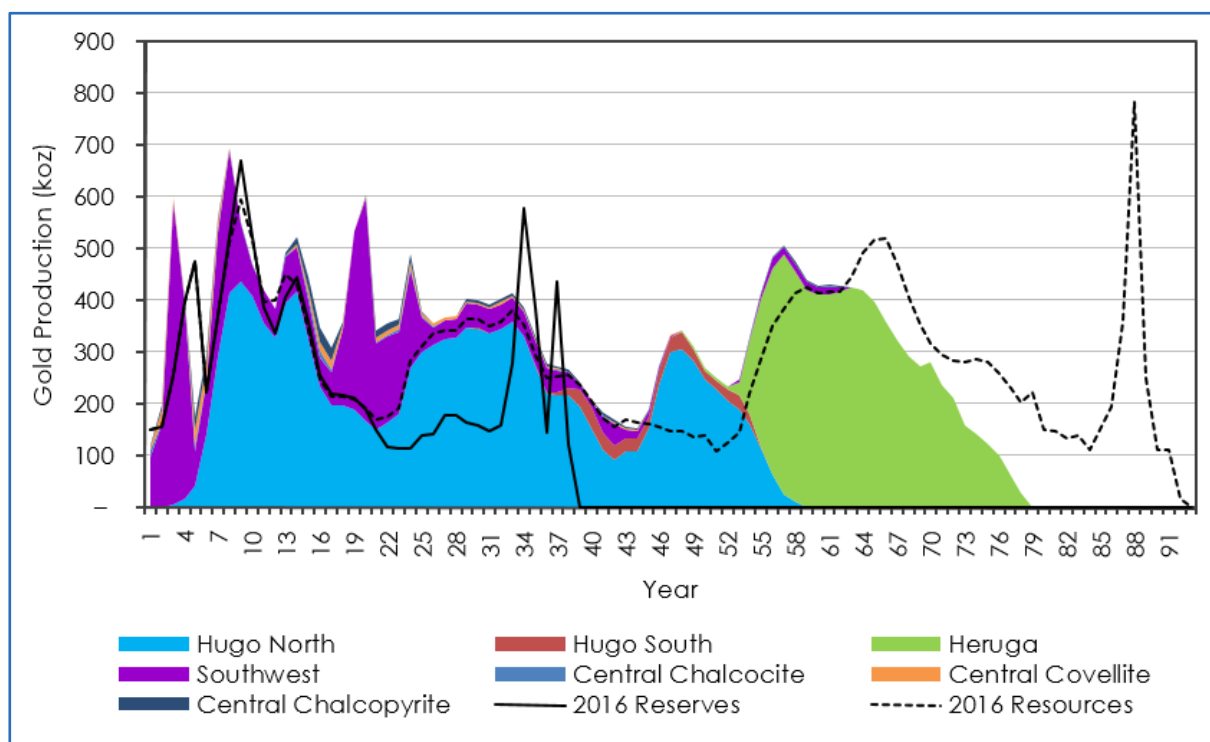


Figure 24.13 Recovered Silver Production – Resources 50 Case

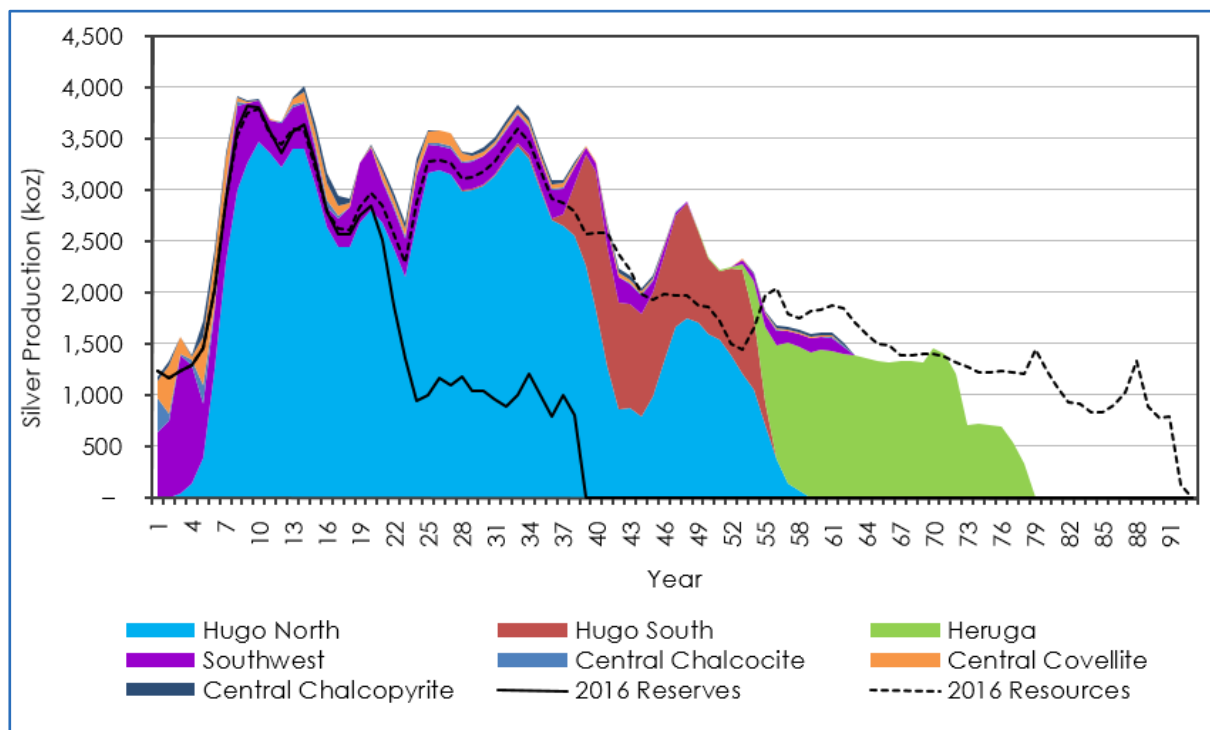


Figure 24.14 Recovered Molybdenum Production – Resources 50 Case

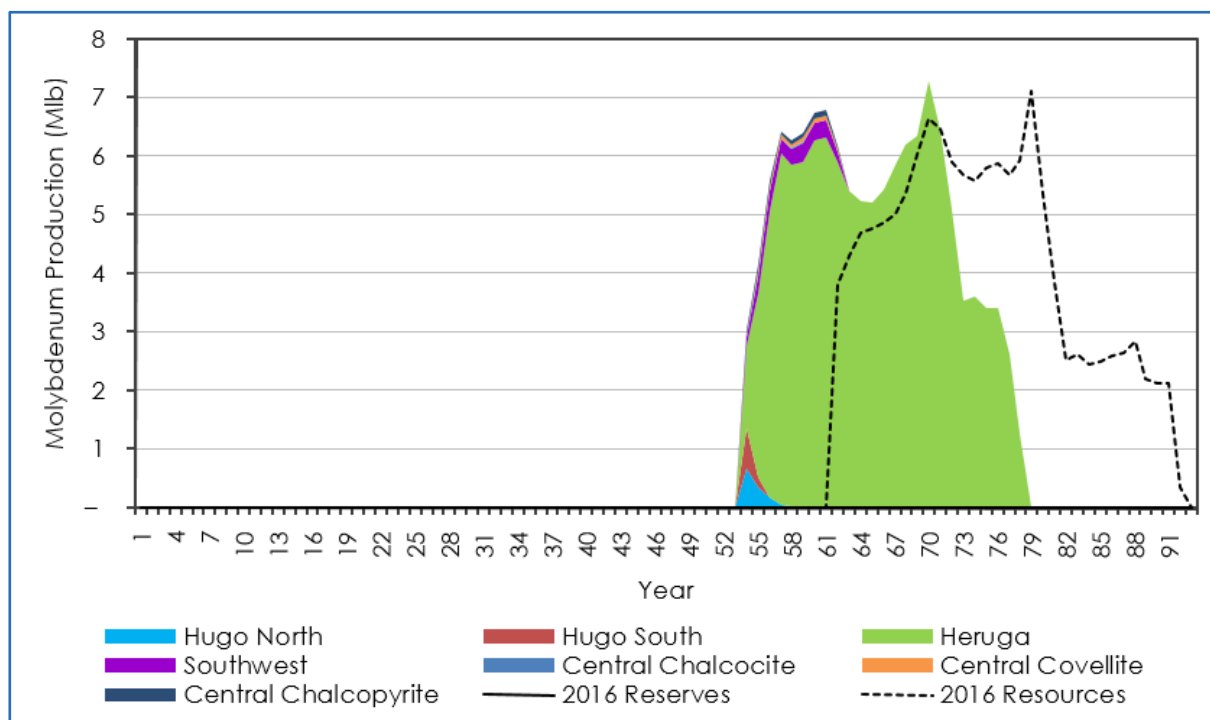


Table 24.16 Production Schedule – Resources 50 Case

		Year																		Total	
Year Number		1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81		91
Year To												20	30	40	50	60	70	80	90		100
Open Pit																					
Plant Feed	Mt	41	44	43	42	48	41	35	28	22	17	151	147	115	66	62	14	–	–	–	914
Waste	Mt	65	77	95	100	69	83	90	105	117	120	771	–	–	–	–	–	–	–	–	1,693
Total Open Pit	Mt	106	121	138	142	117	124	125	133	139	137	922	147	115	66	62	14	–	–	–	2,607
Underground	Mt	–	–	1	2	5	10	15	20	26	31	340	342	373	415	393	338	165	–	–	2,475
Processed	Mt	41	44	44	44	52	51	50	48	47	48	491	488	488	480	455	352	165	–	–	3,389
	NSR	22.15	25.35	40.17	31.81	29.18	45.45	70.91	88.23	88.11	87.28	71.43	61.78	57.58	41.53	33.77	31.19	27.62	–	–	49.97
	Cu %	0.48	0.48	0.53	0.47	0.60	0.82	1.16	1.40	1.44	1.47	1.20	1.05	1.01	0.77	0.53	0.41	0.41	–	–	0.84
	Au g/t	0.13	0.20	0.56	0.38	0.16	0.24	0.45	0.56	0.44	0.37	0.36	0.31	0.26	0.20	0.34	0.41	0.25	–	–	0.31
	Ag g/t	1.24	1.27	1.40	1.25	1.36	1.85	2.51	2.96	2.96	2.93	2.61	2.50	2.60	1.92	1.61	1.51	1.45	–	–	2.11
	As ppm	103.85	94.76	35.30	42.59	166.28	94.71	64.75	56.96	59.47	76.03	66.08	169.85	75.04	74.93	47.35	2.41	–	–	–	73.24
	Mo ppm	58.45	51.51	49.46	66.50	63.51	58.13	48.60	41.30	45.94	51.41	41.75	32.85	39.23	45.06	74.76	120.27	125.93	–	–	58.88
Bulk Concentrate	Conc. kt	669	727	812	707	1,051	1,292	1,599	1,806	1,898	2,015	18,745	16,254	15,837	13,690	8,829	5,150	2,414	–	–	93,495
	Conc. Cu %	21.62	22.40	23.81	24.05	22.95	27.25	31.89	33.74	32.84	31.93	28.26	28.10	27.54	23.58	23.36	23.97	24.00	–	–	26.72
	Conc. Au g/t	5.48	8.37	22.91	17.51	5.17	7.05	10.84	11.94	9.01	7.22	7.32	7.31	6.26	5.38	13.38	21.89	13.57	–	–	8.75
	Conc. Ag g/t	54.87	57.24	60.15	61.06	50.74	58.27	65.35	67.33	63.47	60.00	57.49	63.17	67.34	55.38	66.99	84.28	81.31	–	–	63.29
	Conc. As ppm	3,053	2,849	743	791	3,977	1,796	1,003	747	540	591	1,039	3,345	1,371	1,689	1,497	52	–	–	–	1,595
	Conc. Mo ppm	–	–	–	–	–	–	–	–	–	–	–	–	–	–	1,982	5,272	5,509	–	–	620
	Conc. F ppm	233	246	307	260	227	328	323	327	327	328	342	367	340	358	454	271	5	–	–	–
Recovered Metal	Copper blb	0.3	0.4	0.4	0.4	0.5	0.8	1.1	1.3	1.4	1.4	11.7	10.1	9.6	7.1	4.5	2.7	1.3	–	–	55.1
	Gold Moz	0.1	0.2	0.6	0.4	0.2	0.3	0.6	0.7	0.5	0.5	4.4	3.8	3.2	2.4	3.8	3.6	1.1	–	–	26.3
	Silver Moz	1.2	1.3	1.6	1.4	1.7	2.4	3.4	3.9	3.9	3.9	34.6	33.0	34.3	24.4	19.0	14.0	6.3	–	–	190.2

24.1.5.2 Economics – Resources 50 Case

The base case price assumptions for the financial evaluation for the Resources 50 Case are the same as those used for the 2016 Reserves Case.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

A summary of the financial results is shown in Table 24.17 and the mining production statistics are shown in Table 24.18.

Mine site cash costs, revenues, operating costs, and capital costs are shown in Table 24.19 to Table 24.21.

Table 24.17 Financial Results – Resources 50 Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	64.20	51.51
	5%	19.21	16.39
	6%	15.67	13.50
	7%	12.88	11.19
	8%	10.65	9.32
	9%	8.84	7.78
	10%	7.36	6.51
IRR (%)		23	23
Project Payback Period (Years)		8	8

Table 24.18 Mining Production Statistics – Resources 50 Case

		Resources 50 Case	5-Year Average	10-Year Average	Resources 50 Case Average
Quantity Ore Treated	Mt	3,389	45.12	46.95	43.44
Copper Feed Grade	%	0.84	0.51	0.90	0.84
Gold Feed Grade	g/t	0.31	0.28	0.35	0.31
Silver Feed Grade	g/t	2.11	1.31	2.00	2.11
Copper Recoveries	%	88	79	87	88
Gold Recoveries	%	78	72	76.73	78
Silver Recoveries	%	83	76	82	83
Copper Concentrate	Mt (dry)	93.6	0.8	1.3	1.2
Copper Concentrate Grade	%	27	23	29	27
Contained Metal in Concentrate					
Copper	Mt	25.0	0.2	0.4	0.3
Copper	blb	55.1	0.4	0.8	0.7
Gold	Moz	26.3	0.3	0.4	0.3
Silver	Moz	190.2	1.4	2.5	2.4

Table 24.19 Unit Operating Costs by Copper Production – Resources 50 Case

Description	US\$/lb Payable Copper
Mine Site Cash Cost	1.77
By-product Credit	0.69
Cash Costs (Net of By-product Credit)	1.08

Table 24.20 Operating Costs and Revenues – Resources 50 Case

	US\$b	US\$/t Ore Milled		
	Total Resources 50 Case	5-Year Average	10-Year Average	Resources 50 Case Average
Revenue				
Gross Sales Revenue	195.36	31.14	59.64	57.65
Less: Realization Costs				
Realization Costs	22.80	4.59	6.72	6.73
Government Royalty	10.17	1.63	3.10	3.00
Total Realization Costs	32.97	6.22	9.82	9.73
Net Sales Revenue	162.39	24.92	49.81	47.92
Less: Site Operating Costs				
Mining (all sources)	15.43	4.31	5.39	4.55
Processing and Tailings	25.35	7.20	7.51	7.48
G&A and Operations Support	6.86	2.50	2.41	2.02
Infrastructure and Other	4.01	1.34	0.78	1.18
Government Fees & Charges	5.53	1.80	1.88	1.63
Management and JV Payments	6.69	2.43	2.28	1.97
Total Site Operating Costs	63.88	19.58	20.24	18.85
Operating Margin	98.51	5.34	29.57	29.07

Table 24.21 Total Project Expansion and Sustaining Capital Cost – Resources 50 Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.19	1.19
Underground	7.12	15.89	23.01
Concentrator	0.14	0.33	0.47
Infrastructure	0.35	0.52	0.87
Tailings Storage Facility (TSF)	–	2.13	2.13
Subtotal	7.62	20.06	27.68
Construction Indirect	0.86	–	0.86
Contractor Execution – EPCM	0.31	–	0.31
Owner Execution	0.43	0.33	0.76
GOM Fees & Charges – Mongolian VAT	0.51	1.77	2.28
Total	9.73	22.16	31.89

Note: Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

The changes in financial results for a range of copper and gold prices, calculated at a silver price of US\$19.00/oz, are shown in Table 24.22.

The changes in financial results for a range of silver prices, calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz, are shown in Table 24.23.

A cumulative cash flow for the Resources 50 Case is depicted in Figure 24.15 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and a complete cash flow is provided in Table 24.24.

Table 24.22 After Tax Metal Price Sensitivity – Resources 50 Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-3.52	-3.07	-2.61	-2.16	-1.71	-1.26	-0.13
2.00	-1.68	-1.23	-0.78	-0.32	0.13	0.58	1.71
2.50	2.91	3.37	3.82	4.27	4.72	5.18	6.31
2.80	5.67	6.12	6.57	7.03	7.48	7.93	9.06
3.00	7.51	7.96	8.41	8.87	9.32	9.77	10.90
3.50	12.10	12.56	13.01	13.46	13.91	14.36	15.50
4.00	16.70	17.15	17.60	18.05	18.51	18.96	20.09
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	8
2.00	–	–	5	7	8	10	12
2.50	14	14	15	16	17	18	20
2.80	17	18	19	20	20	21	23
3.00	20	20	21	22	23	23	25
3.50	24	25	26	27	27	28	30
4.00	29	29	30	31	31	32	34
Project Payback After Tax (Years)							
1.80	–	–	–	13.9	13.1	12.5	10.8
2.00	13.2	12.6	12.1	11.3	10.8	10.4	9.6
2.50	9.5	9.3	9.1	9.0	8.8	8.6	8.3
2.80	8.7	8.6	8.5	8.3	8.2	8.1	7.8
3.00	8.4	8.2	8.1	8.0	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.4	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.20	1.15	1.11	1.06	1.02	0.97	0.86
2.00	1.21	1.16	1.12	1.07	1.03	0.98	0.87
2.50	1.23	1.19	1.14	1.10	1.05	1.01	0.90
2.80	1.25	1.20	1.16	1.11	1.07	1.02	0.91
3.00	1.26	1.21	1.17	1.12	1.08	1.03	0.92
3.50	1.28	1.24	1.19	1.15	1.10	1.06	0.95
4.00	1.31	1.26	1.22	1.17	1.13	1.08	0.97

[†] Calculated at a silver price of US\$19.00/oz

Table 24.23 After Tax Silver Price Sensitivity – Resources 50 Case

After Tax Values [†]	Silver (US\$/oz)						
	10.00	12.00	15.00	17.00	19.00	24.00	30.00
NPV8% (US\$b)	9.05	9.11	9.20	9.26	9.32	9.47	9.65

[†] Calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz

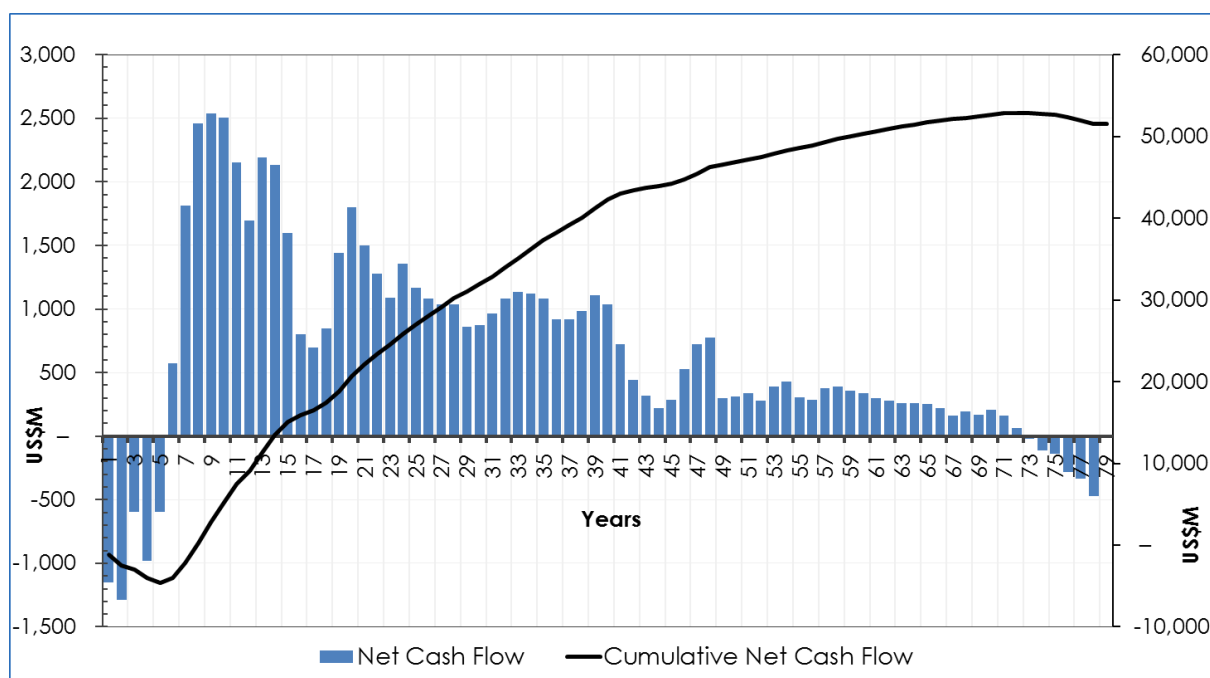
Figure 24.15 Cumulative Cash Flow – Resources 50 Case


Table 24.24 Cash Flow – Resources 50 Case

Cash Flow Statement (US\$M)	Year																			Total
Year Number	1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81	91	
Year To											20	30	40	50	60	70	80	90	100	
Gross Revenue	819	1,074	1,834	1,526	1,772	2,647	4,012	4,829	4,729	4,756	39,826	34,413	32,316	23,773	18,425	13,062	5,548	–	–	195,361
Realization Costs	251	252	295	244	361	445	602	702	716	743	6,565	6,038	5,459	4,415	3,066	1,928	891	–	–	32,972
Net Sales Revenue	568	822	1,540	1,282	1,411	2,202	3,410	4,126	4,013	4,013	33,262	28,375	26,857	19,359	15,359	11,135	4,657	–	–	162,389
Site Operating Costs																				
Mining	141	191	199	201	239	267	300	317	329	343	4,199	1,922	1,841	1,731	1,418	1,057	734	–	–	15,431
Processing and Tailings	288	328	326	322	360	367	388	384	380	382	3,856	3,796	3,744	3,608	2,930	2,488	1,407	–	–	25,353
G&A and Operations Support	103	105	104	114	140	125	117	114	107	104	1,040	998	992	998	900	539	260	–	–	6,860
Infrastructure and Other	65	66	48	36	87	22	2	19	14	4	506	633	560	556	542	500	352	–	–	4,013
Total Site Operating Costs	598	690	677	673	827	781	807	834	830	833	9,602	7,349	7,136	6,893	5,790	4,584	2,754	–	–	51,657
Operating Surplus / (Deficit)	–30	132	863	610	584	1,421	2,603	3,293	3,182	3,180	23,660	21,026	19,721	12,465	9,570	6,551	1,903	–	–	110,732
Indirect Costs	161	189	187	200	217	202	206	210	192	189	1,969	2,274	1,903	1,431	1,171	933	589	–	–	12,223
Net Profit Before Income Tax	–191	–57	676	410	367	1,219	2,396	3,082	2,991	2,991	21,691	18,752	17,818	11,034	8,399	5,617	1,313	–	–	98,509
Income Tax	–	–	–	–	–	–	–	–	–	–	1,705	3,838	3,496	1,841	997	757	49	–	–	12,683
Net Profit After Income Tax	–191	–57	676	410	367	1,219	2,396	3,082	2,991	2,991	19,986	14,915	14,322	9,193	7,401	4,861	1,265	–	–	85,826
Capital Expenditure																				
Expansion Capital	874	1,072	1,080	831	387	92	–	9	18	24	1,553	464	660	945	1,207	–	–	–	–	9,216
Sustaining Capital	57	103	75	369	433	385	424	485	323	338	2,405	2,937	3,024	3,423	2,402	2,374	835	–	–	20,391
VAT & Duties	76	83	68	104	76	45	46	53	36	38	345	262	271	256	260	192	65	–	–	2,277
Subtotal	1,008	1,258	1,224	1,304	897	523	469	547	377	399	4,303	3,663	3,956	4,623	3,869	2,566	900	–	–	31,885
Working Capital, Capitalised Operating Costs and Closure	–48	–28	43	79	68	120	106	68	70	76	305	–26	13	–49	37	–2	1,511	–	–	2,342
VAT & Duties (Capex)	3	3	4	5	0	7	9	7	7	8	32	–	–	–	–	–	–	–	–	85
Total Capital Expenditure	962	1,233	1,271	1,388	965	650	583	622	454	484	4,640	3,637	3,969	4,574	3,906	2,564	2,411	–	–	34,311
Net Cash Flow After Tax	–1,152	–1,290	–595	–979	–598	570	1,813	2,460	2,536	2,507	15,346	11,278	10,353	4,619	3,495	2,297	–1,146	–	–	51,515

24.1.5.3 Production – Resources 100 Case

A summary of the production for the Resources 100 Case is shown in Figure 24.16 to Figure 24.21. For reference the charts also show the 2016 Reserves Case and 2016 Resources Case. The production schedule is shown in Table 24.25. Total tonnes in the production schedule for the Resources 50 Case, the Resources 100 Case, and the Resources 120 Case are different from the 2016 Resources Case because they do not include a stockpile of 35Mt at 0.35% copper, 0.13 g/t gold, and 0.81 g/t silver.

Figure 24.16 Processing Production – Resources 100 Case

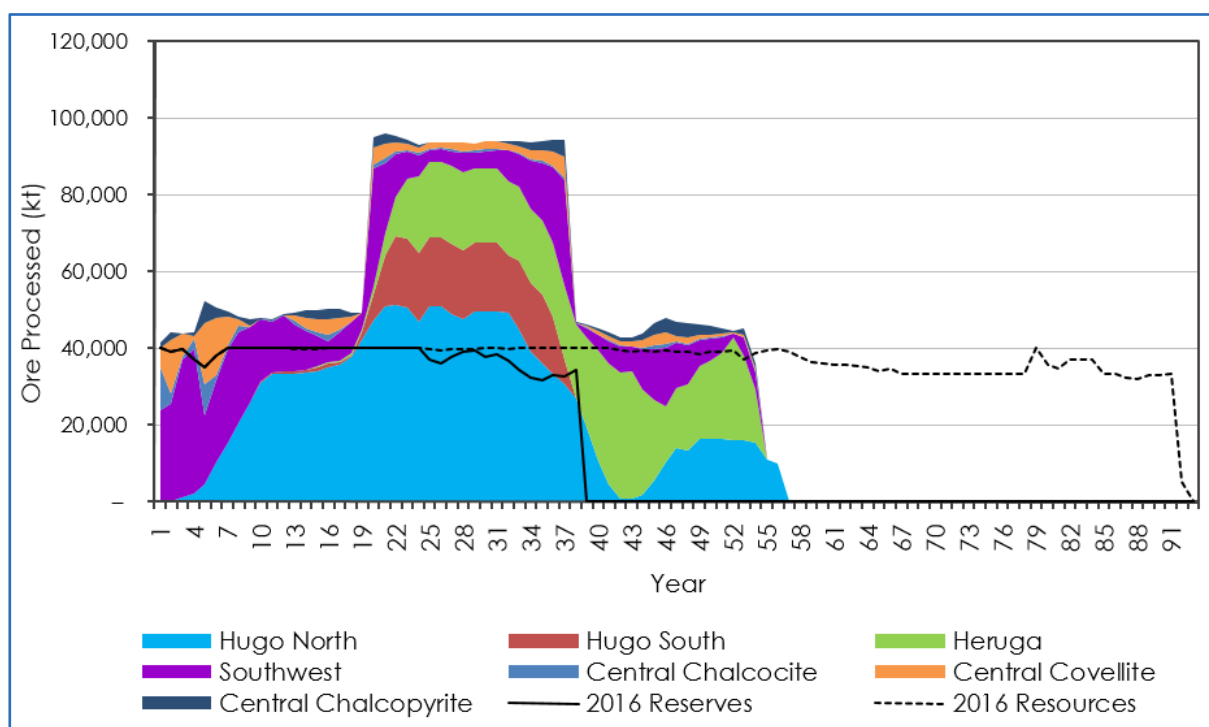


Figure 24.17 Concentrate Production – Resources 100 Case

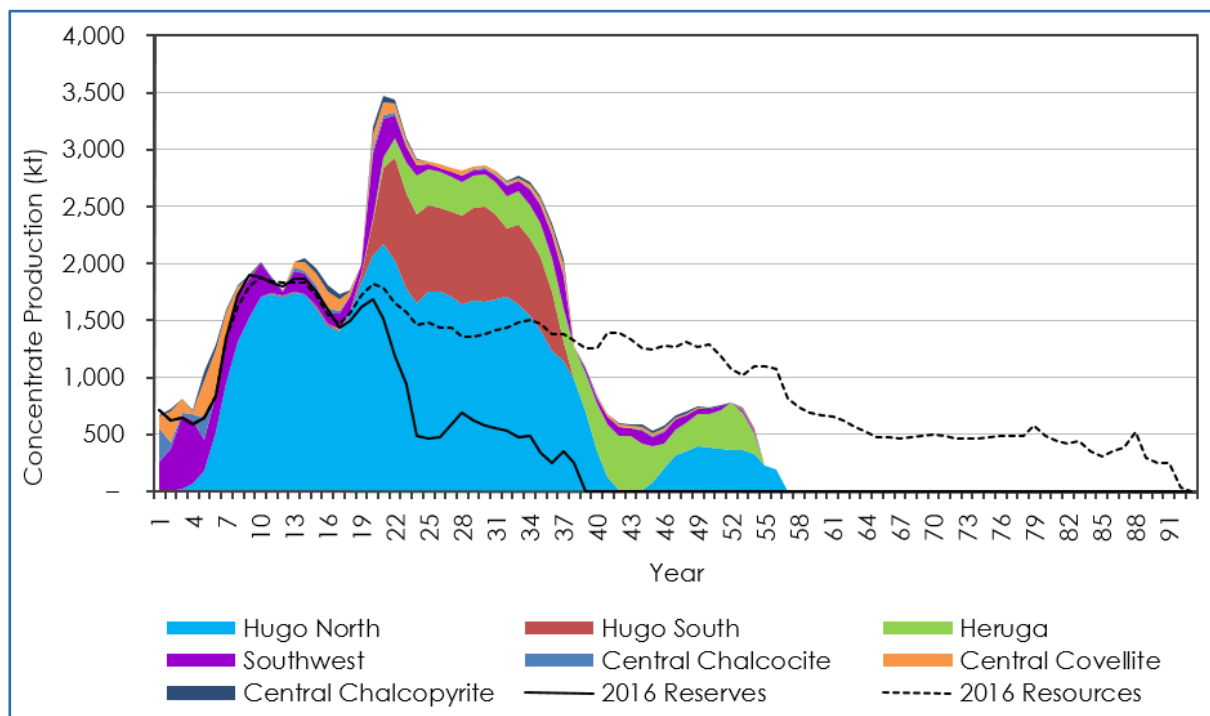


Figure 24.18 Recovered Copper Production – Resources 100 Case

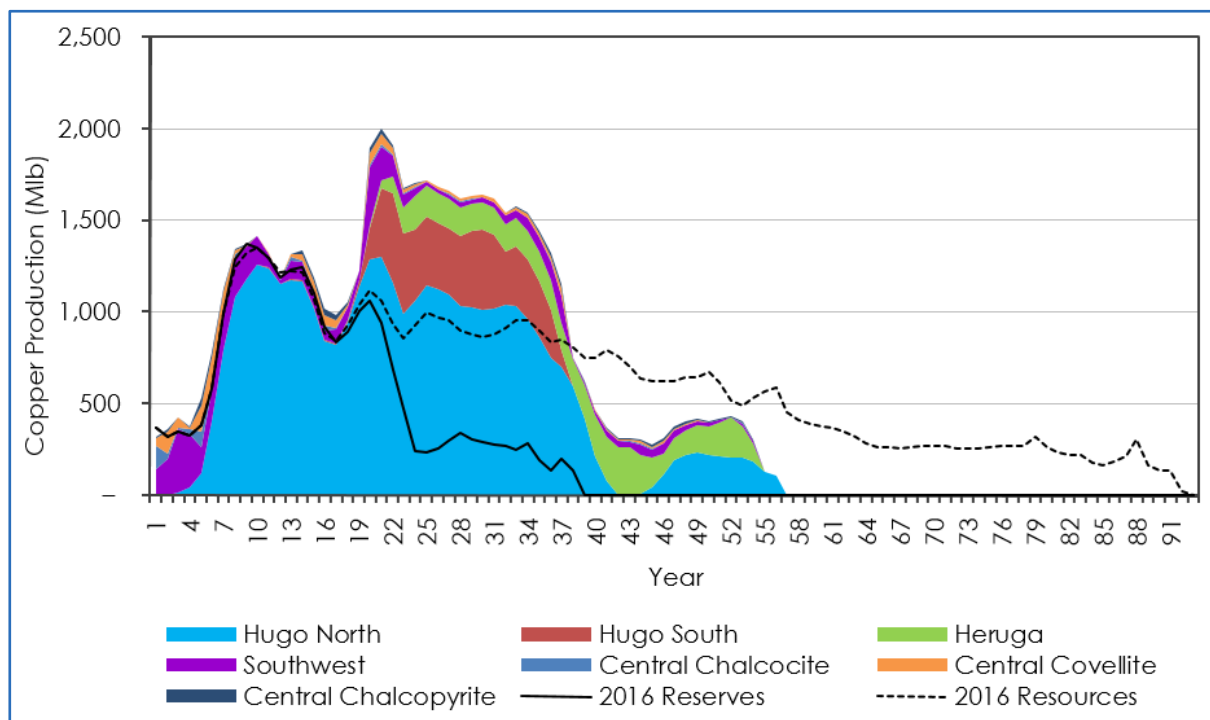


Figure 24.19 Recovered Gold Production – Resources 100 Case

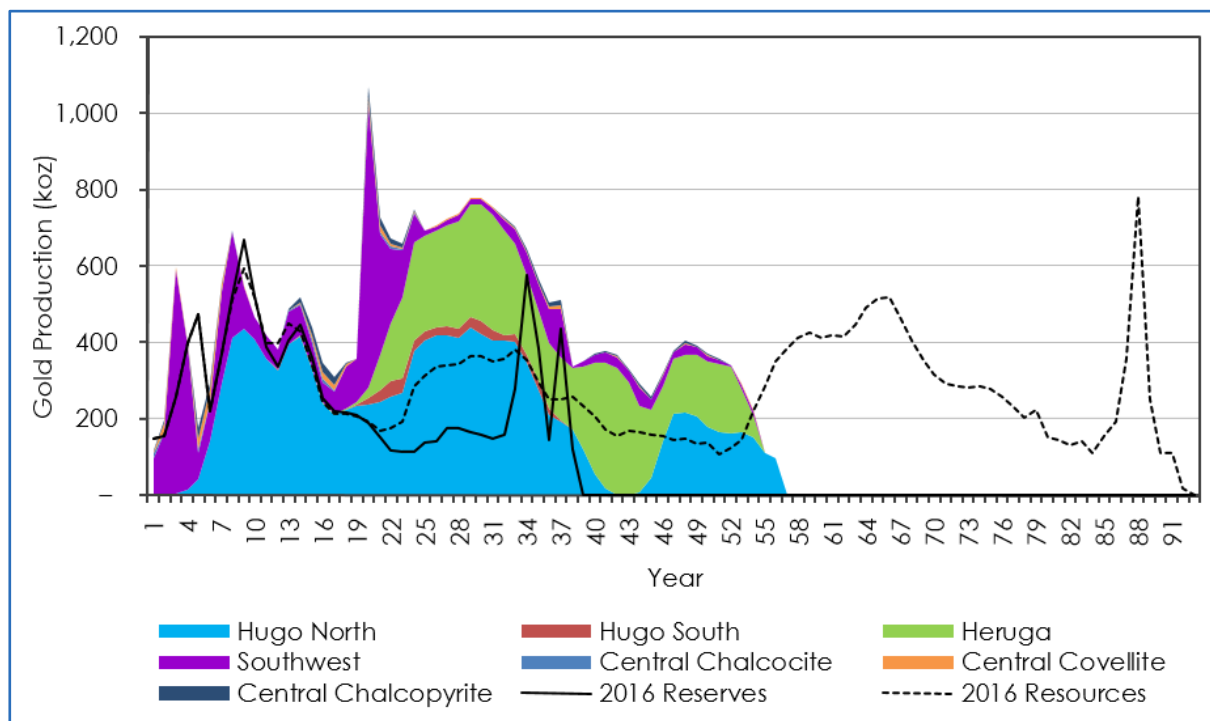


Figure 24.20 Recovered Silver Production – Resources 100 Case

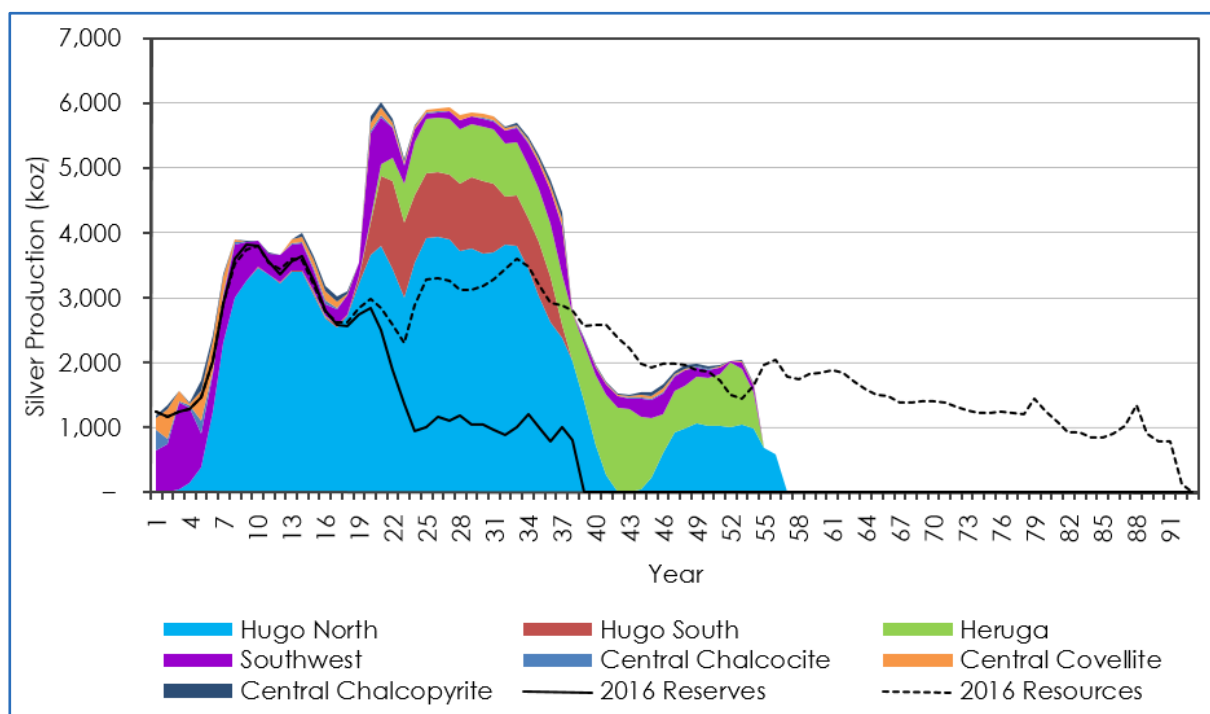


Figure 24.21 Recovered Molybdenum Production – Resources 100 Case

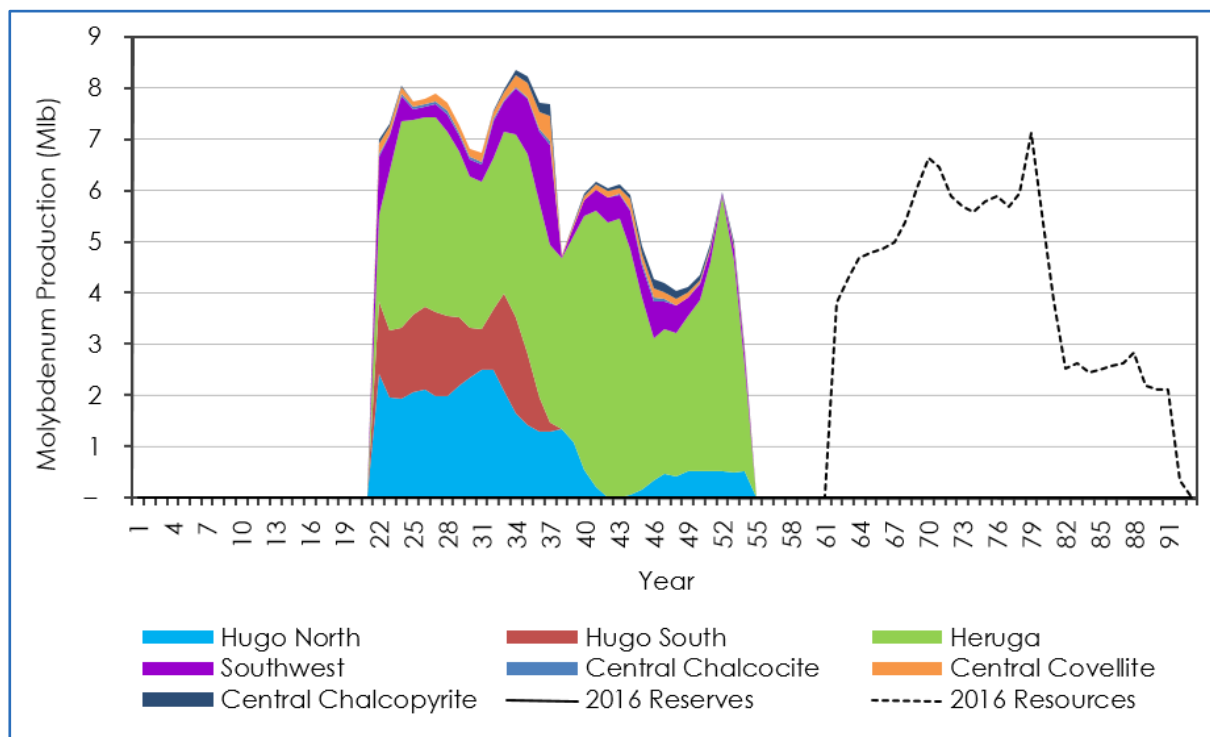


Table 24.25 Production Schedule – Resources 100 Case

		Year																		Total		
Year Number		1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81		91	
Year To												20	30	40	50	60	70	80	90		100	
Open Pit																						
Plant Feed	Mt	41	44	43	42	48	41	34	28	22	17	155	97	143	138	22	–	–	–	–	914	
Waste	Mt	65	77	95	100	69	83	90	105	117	120	771	–	–	–	–	–	–	–	–	1,693	
Total Open Pit	Mt	106	121	138	142	117	124	125	133	138	137	926	97	143	138	22	–	–	–	–	2,607	
Underground	Mt	–	–	1	2	5	10	15	20	26	31	383	843	654	315	170	–	–	–	–	2,475	
Processed	Mt	41	44	44	44	52	51	50	48	47	48	538	940	797	453	191	–	–	–	–	3,389	
	NSR	22.15	25.35	40.17	31.81	29.18	45.45	70.88	88.19	87.98	87.24	69.53	54.49	45.78	28.28	32.68	–	–	–	–	49.97	
	Cu%	0.48	0.48	0.53	0.47	0.60	0.82	1.16	1.40	1.44	1.47	1.17	0.94	0.78	0.42	0.49	–	–	–	–	0.84	
	Au g/t	0.13	0.20	0.56	0.38	0.16	0.24	0.45	0.56	0.44	0.37	0.34	0.30	0.27	0.31	0.30	–	–	–	–	0.31	
	Ag g/t	1.24	1.27	1.40	1.25	1.36	1.85	2.51	2.96	2.96	2.93	2.58	2.28	2.07	1.49	1.80	–	–	–	–	2.11	
	As ppm	103.85	94.76	35.30	42.59	166.28	94.71	64.59	56.87	59.40	75.12	64.73	115.40	66.56	26.01	12.15	–	–	–	–	73.24	
	Mo ppm	58.45	51.51	49.46	66.50	63.51	58.13	48.49	41.24	45.86	51.01	42.73	55.93	62.40	78.28	71.68	–	–	–	–	58.88	
Bulk Concentrate	Conc. kt	669	727	812	707	1,051	1,292	1,597	1,805	1,898	2,012	20,162	30,052	21,186	6,422	3,241	–	–	–	–	93,635	
	Conc. Cu %	21.62	22.40	23.81	24.05	22.95	27.25	31.90	33.75	32.84	31.94	28.15	26.03	25.76	24.44	24.87	–	–	–	–	26.68	
	Conc. Au g/t	5.48	8.37	22.91	17.51	5.17	7.05	10.84	11.94	9.00	7.23	7.21	7.48	8.02	16.88	13.63	–	–	–	–	8.74	
	Conc. Ag g/t	54.87	57.24	60.15	61.06	50.74	58.27	65.35	67.31	63.47	60.06	57.92	59.81	64.68	83.44	86.22	–	–	–	–	63.20	
	Conc. As ppm	3,053	2,849	743	791	3,977	1,796	1,002	747	540	591	1,041	2,460	1,505	654	339	–	–	–	–	1,592	
	Conc. Mo ppm	–	–	–	–	–	–	–	–	–	–	–	–	1,019	1,504	3,540	2,637	–	–	–	–	1,002
	Conc. F ppm	233	246	307	260	227	328	323	327	327	328	342	366	377	323	107	114	–	–	–	–	325
Recovered Metal	Copper blb	0.3	0.4	0.4	0.4	0.5	0.8	1.1	1.3	1.4	1.4	12.5	17.2	12.0	3.5	1.8	–	–	–	–	55.1	
	Gold Moz	0.1	0.2	0.6	0.4	0.2	0.3	0.6	0.7	0.5	0.5	4.7	7.2	5.5	3.5	1.4	–	–	–	–	26.3	
	Silver Moz	1.2	1.3	1.6	1.4	1.7	2.4	3.4	3.9	3.9	3.9	37.5	57.8	44.1	17.2	9.0	–	–	–	–	190.2	

24.1.5.4 Economics – Resources 100 Case

The base case price assumptions for the financial evaluation for the Resources 100 Case are the same as those used for the 2016 Reserves Case.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

A summary of the financial results is shown in Table 24.26 and the mining production statistics are shown in Table 24.27.

Mine site cash costs, revenues, operating costs, and capital costs are shown in Table 24.28 to Table 24.30.

Table 24.26 Financial Results – Resources 100 Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	61.04	49.31
	5%	19.18	16.06
	6%	15.57	13.12
	7%	12.71	10.78
	8%	10.41	8.88
	9%	8.55	7.33
	10%	7.03	6.06
IRR (%)	–	22	22
Project Payback Period (Years)	–	8	8

Table 24.27 Mining Production Statistics – Resources 100 Case

		Resources 100 Case	5-Year Average	10-Year Average	Resources 100 Case Average
Quantity Ore Treated	Mt	3,389	45.12	46.95	60.51
Copper Feed Grade	%	0.84	0.51	0.90	0.84
Gold Feed Grade	g/t	0.31	0.28	0.35	0.31
Silver Feed Grade	g/t	2.11	1.31	1.99	2.11
Copper Recoveries	%	88	78.79	86.71	88.10
Gold Recoveries	%	78	72.26	76.73	78.19
Silver Recoveries	%	83	75.87	81.81	82.93
Copper Concentrate	Mt (dry)	93.7	0.8	1.3	1.7
Copper Concentrate Grade	%	27	23	29	27
Contained Metal in Concentrate					
Copper	Mt	25.0	0.2	0.4	0.4
Copper	blb	55.1	0.4	0.8	1.0
Gold	Moz	26.3	0.3	0.4	0.5
Silver	Moz	190.2	1.4	2.5	3.4

Table 24.28 Unit Operating Costs by Copper Production – Resources 100 Case

Description	US\$/lb Payable Copper
Mine Site Cash Cost	1.76
By-product Credit	0.70
Cash Costs (Net of By-product Credit)	1.06

Table 24.29 Operating Costs and Revenues – Resources 100 Case

	US\$b	US\$/t Ore Milled		
	Total Resources 100 Case	5-Year Average	10-Year Average	Resources 100 Case Average
Revenue				
Gross Sales Revenue	195.88	31.14	59.61	57.81
Less: Realization Costs				
Realization Costs	22.51	4.59	6.72	6.64
Government Royalty	10.20	1.63	3.10	3.01
Total Realization Costs	32.72	6.22	9.82	9.65
Net Sales Revenue	163.17	24.92	49.79	48.15
Less: Site Operating Costs				
Mining (all sources)	15.92	4.31	5.39	4.70
Processing and Tailings	25.31	7.20	7.51	7.47
G&A and Operations Support	7.15	2.50	2.41	2.11
Infrastructure and Other	3.23	1.34	0.78	0.95
Government Fees & Charges	5.46	1.80	1.88	1.61
Management and JV Payments	6.58	2.43	2.30	1.94
Total Site Operating Costs	63.65	19.58	20.26	18.78
Operating Margin	99.52	5.34	29.53	29.37

Table 24.30 Total Project Expansion and Sustaining Capital Cost – Resources 100 Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.19	1.19
Underground	7.36	16.06	23.42
Concentrator	3.63	0.40	4.03
Infrastructure	0.35	0.39	0.75
Tailings Storage Facility (TSF)	–	2.45	2.45
Subtotal	11.34	20.49	31.83
Construction Indirect	0.86	–	0.86
Contractor Execution – EPCM	0.31	–	0.31
Owner Execution	0.43	0.29	0.72
GOM Fees & Charges – Mongolian VAT	0.52	1.82	2.34
Total	13.47	22.59	36.06

Note: Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

The changes in financial results for a range of copper and gold prices, calculated at a silver price of US\$19.00/oz, are shown in Table 24.31.

The changes in financial results for a range of silver prices, calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz, are shown in Table 24.32.

A cumulative cash flow for the Resources 100 Case is depicted in Figure 24.22 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and a complete cash flow is provided in Table 24.33.

Table 24.31 After Tax Metal Price Sensitivity – Resources 100 Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-5.65	-5.14	-4.62	-4.10	-3.59	-3.07	-1.78
2.00	-3.58	-3.06	-2.54	-2.03	-1.51	-0.99	0.30
2.50	1.62	2.14	2.65	3.17	3.68	4.20	5.49
2.80	4.74	5.25	5.77	6.28	6.80	7.32	8.61
3.00	6.81	7.33	7.85	8.36	8.88	9.40	10.69
3.50	12.01	12.52	13.04	13.56	14.07	14.59	15.88
4.00	17.20	17.72	18.23	18.75	19.27	19.78	21.07
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	–
2.00	–	–	–	–	-2	5	9
2.50	11	12	13	14	15	16	18
2.80	16	17	18	18	19	20	22
3.00	18	19	20	21	22	22	25
3.50	24	24	25	26	27	27	29
4.00	28	29	29	30	31	32	34
Project Payback After Tax (Years)							
1.80	–	–	–	–	–	13.4	11.5
2.00	–	13.4	12.7	12.1	11.3	10.7	9.8
2.50	9.7	9.5	9.3	9.1	8.9	8.7	8.3
2.80	8.8	8.7	8.5	8.4	8.3	8.1	7.8
3.00	8.4	8.3	8.2	8.1	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.18	1.13	1.09	1.04	1.00	0.95	0.84
2.00	1.19	1.14	1.10	1.05	1.01	0.96	0.85
2.50	1.21	1.17	1.12	1.08	1.03	0.99	0.88
2.80	1.23	1.18	1.14	1.09	1.05	1.00	0.89
3.00	1.24	1.19	1.15	1.10	1.06	1.01	0.90
3.50	1.26	1.22	1.17	1.13	1.08	1.04	0.93
4.00	1.29	1.24	1.20	1.15	1.11	1.06	0.95

[†] Calculated at a silver price of US\$19.00/oz

Table 24.32 After Tax Silver Price Sensitivity – Resources 100 Case

After Tax Values†	Silver (US\$/oz)						
	10.00	12.00	15.00	17.00	19.00	24.00	30.00
NPV8% (US\$b)	8.57	8.64	8.74	8.81	8.88	9.05	9.25

† Calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz

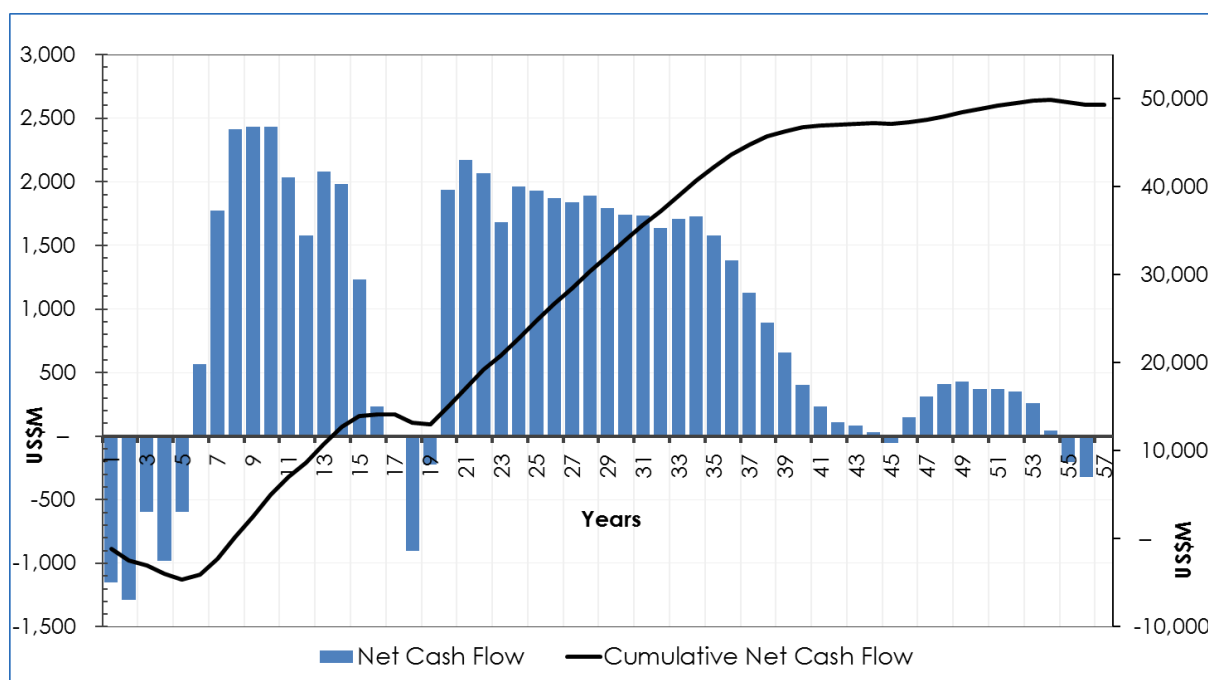
Figure 24.22 Cumulative Cash Flow – Resources 100 Case


Table 24.33 Cash Flow – Resources 100 Case

Cash Flow Statement (US\$M)	Year																			Total
Year Number	1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81	91	
Year To											20	30	40	50	60	70	80	90	100	
Gross Revenue	819	1,074	1,834	1,526	1,772	2,647	4,009	4,826	4,728	4,752	42,624	60,152	42,715	14,987	7,419	–	–	–	–	195,884
Realization Costs	251	252	295	244	361	445	602	702	716	742	7,045	10,302	7,260	2,320	1,179	–	–	–	–	32,715
Net Sales Revenue	568	822	1,540	1,282	1,411	2,202	3,408	4,124	4,012	4,010	35,579	49,850	35,455	12,668	6,240	–	–	–	–	163,169
Site Operating Costs																				
Mining	141	191	199	201	239	267	300	317	329	343	4,397	3,710	2,825	1,445	1,010	–	–	–	–	15,916
Processing and Tailings	288	328	326	322	360	367	388	384	380	382	4,183	6,900	5,852	3,455	1,399	–	–	–	–	25,313
G&A and Operations Support	103	105	104	114	140	125	117	114	107	104	1,136	1,920	1,619	941	403	–	–	–	–	7,152
Infrastructure and Other	65	66	48	36	87	22	2	19	14	4	525	817	685	545	292	–	–	–	–	3,229
Total Site Operating Costs	598	690	677	673	827	781	807	834	831	833	10,241	13,346	10,981	6,386	3,105	–	–	–	–	51,609
Operating Surplus / (Deficit)	–30	132	863	610	584	1,421	2,601	3,291	3,181	3,177	25,338	36,504	24,474	6,282	3,136	–	–	–	–	111,560
Indirect Costs	161	189	187	200	217	202	207	212	196	191	2,331	2,930	2,809	1,371	641	–	–	–	–	12,044
Net Profit Before Income Tax	–191	–57	676	410	367	1,219	2,393	3,079	2,985	2,986	23,006	33,574	21,664	4,911	2,494	–	–	–	–	99,516
Income Tax	–	–	–	–	–	–	–	–	–	–	1,462	5,777	3,787	473	232	–	–	–	–	11,731
Net Profit After Income Tax	–191	–57	676	410	367	1,219	2,393	3,079	2,985	2,986	21,545	27,797	17,878	4,438	2,262	–	–	–	–	87,786
Capital Expenditure																				
Expansion Capital	874	1,072	1,080	831	387	96	37	51	123	84	7,835	471	–	–	–	–	–	–	–	12,941
Sustaining Capital	57	103	74	369	434	385	424	485	323	337	3,055	7,657	4,652	2,164	252	–	–	–	–	20,772
VAT & Duties	76	83	68	104	76	46	48	55	42	41	495	629	382	178	20	–	–	–	–	2,344
Subtotal	1,008	1,258	1,223	1,304	898	527	508	591	488	462	11,386	8,757	5,035	2,342	272	–	–	–	–	36,057
Working Capital, Capitalised Operating Costs and Closure	–48	–28	43	79	68	119	100	67	60	84	180	98	–2	30	1,481	–	–	–	–	2,330
VAT & Duties (Capex)	3	3	4	5	0	7	9	7	7	8	32	–	–	–	–	–	–	–	–	85
Total Capital Expenditure	962	1,233	1,270	1,388	965	653	617	665	555	554	11,598	8,855	5,033	2,371	1,753	–	–	–	–	38,472
Net Cash Flow After Tax	–1,152	–1,290	–594	–979	–599	566	1,776	2,413	2,431	2,432	9,947	18,942	12,845	2,066	509	–	–	–	–	49,314

24.1.5.5 Production – Resources 120 Case

A summary of the production for the Resources 120 Case is shown in Figure 24.23 to Figure 24.28. For reference the charts also show the 2016 Reserves Case and 2016 Resources Case. The production schedule is shown in Table 24.34. Total tonnes in the production schedule for the Resources 50 Case, the Resources 100 Case, and the Resources 120 Case are different from the 2016 Resources Case because they do not include a stockpile of 35Mt at 0.35% copper, 0.13 g/t gold, and 0.81 g/t silver.

Figure 24.23 Processing Production – Resources 120 Case

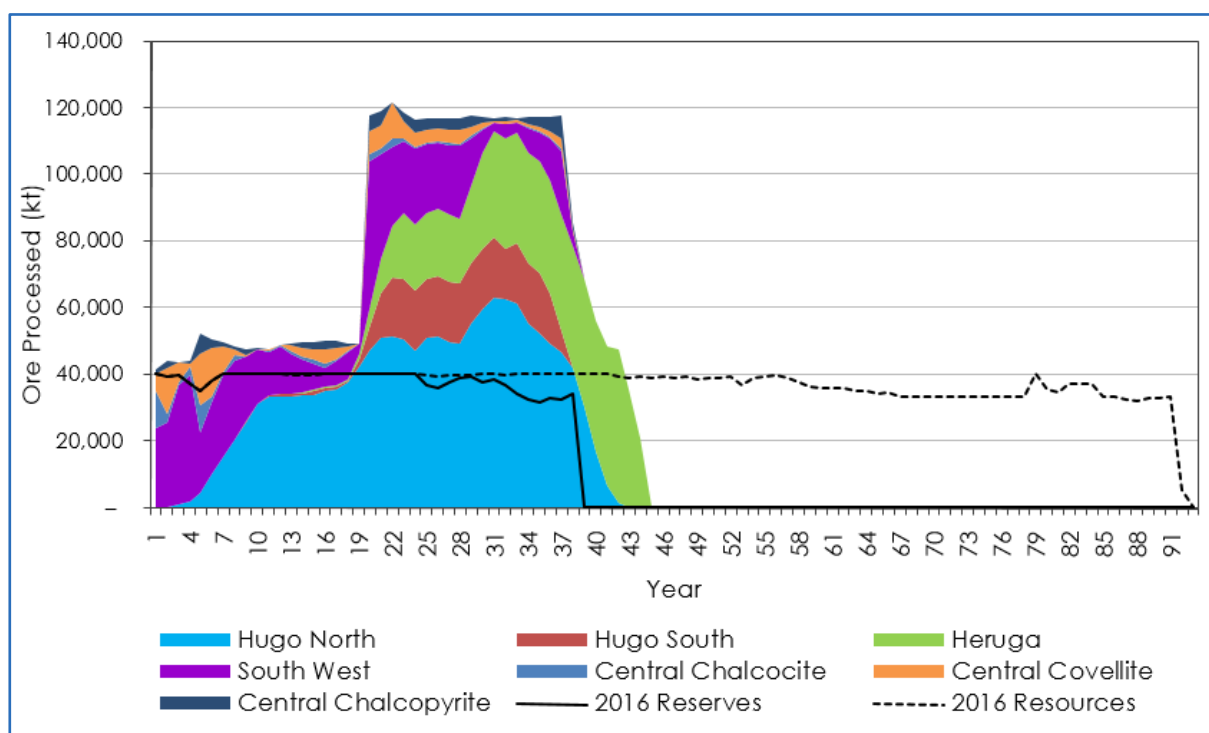


Figure 24.24 Concentrate Production – Resources 120 Case

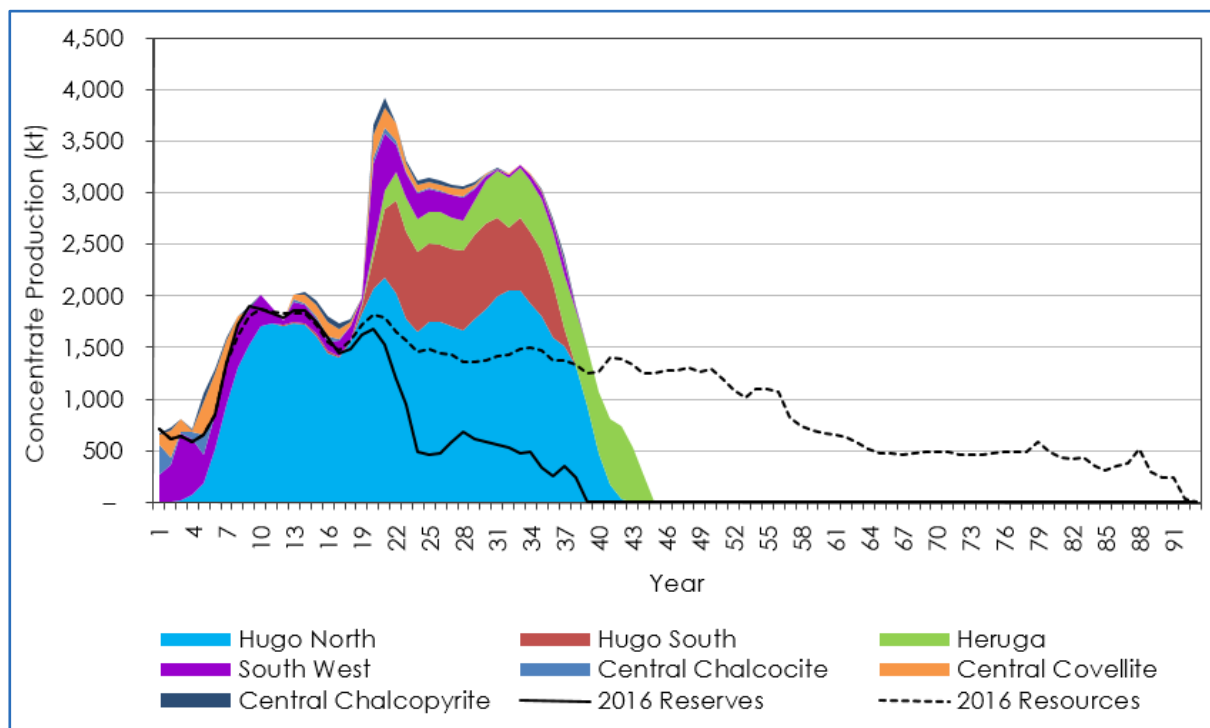


Figure 24.25 Recovered Copper Production – Resources 120 Case

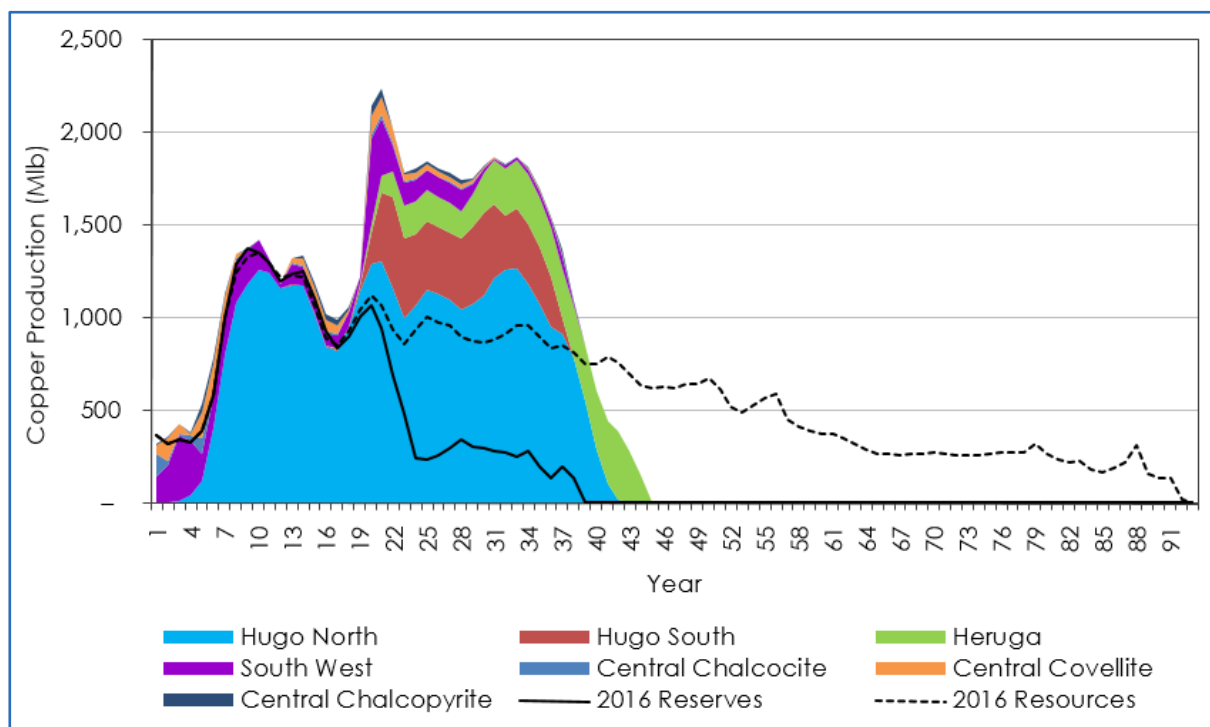


Figure 24.26 Recovered Gold Production – Resources 120 Case

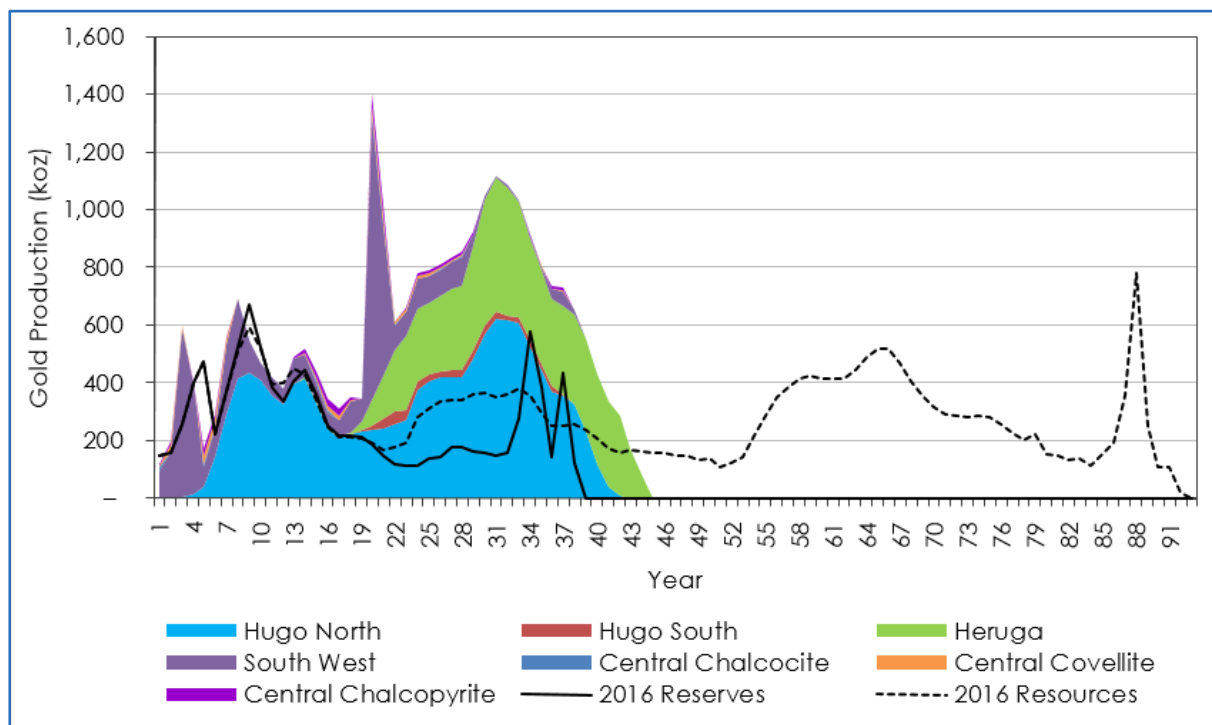


Figure 24.27 Recovered Silver Production – Resources 120 Case

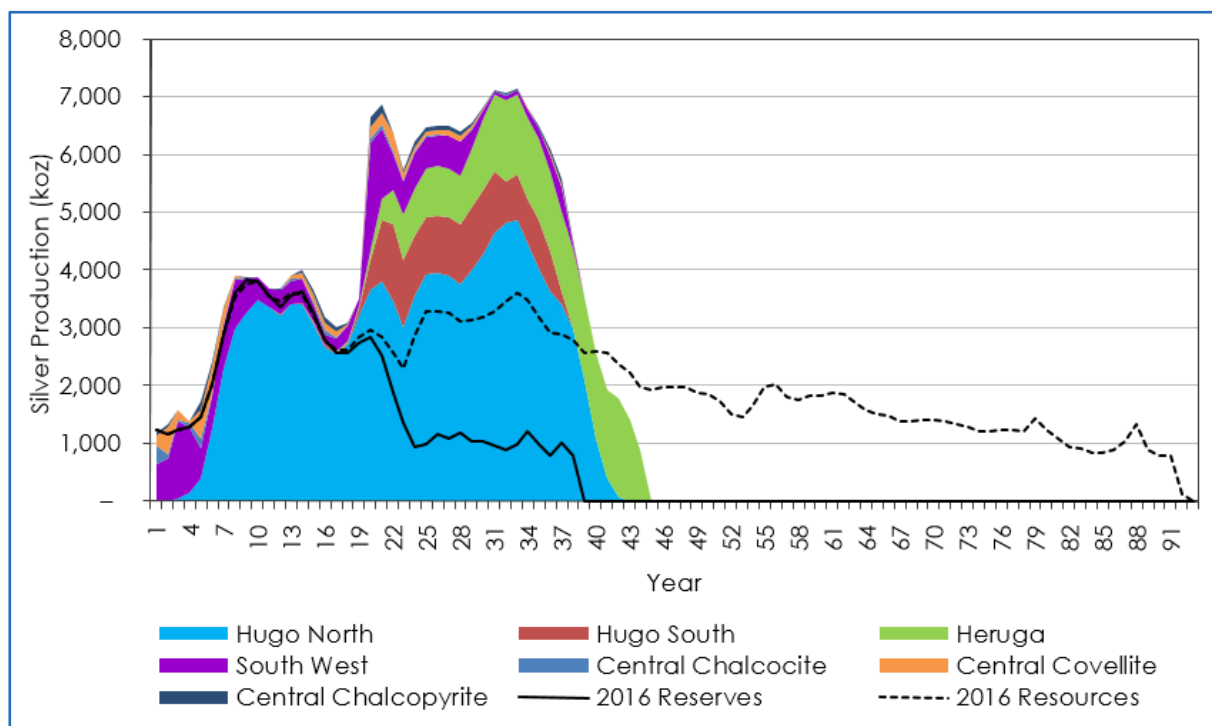


Figure 24.28 Recovered Molybdenum Production – Resources 120 Case

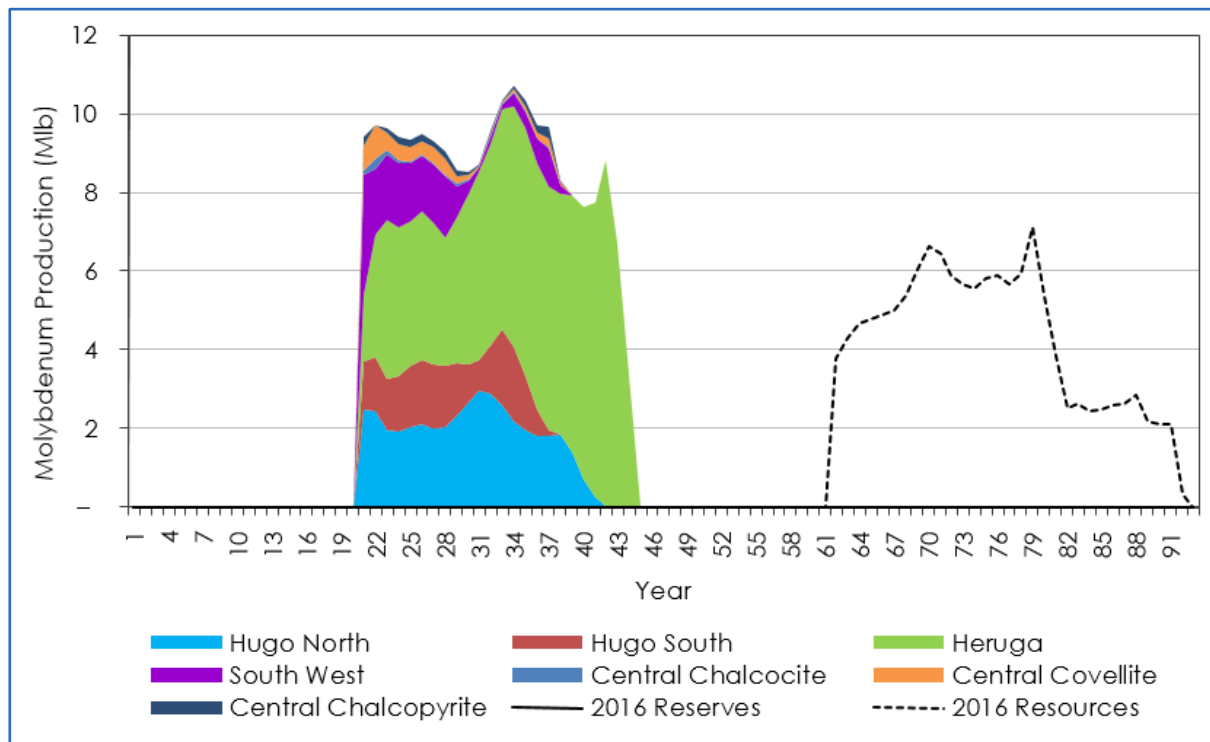


Table 24.34 Production Schedule – Resources 120 Case

		Year																		Total	
Year Number		1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81		91
Year To												20	30	40	50	60	70	80	90		100
Open Pit																					
Plant Feed	Mt	41	44	43	42	48	41	34	28	22	17	172	289	94	–	–	–	–	–	–	914
Waste	Mt	65	77	95	100	69	83	90	105	117	120	771	–	–	–	–	–	–	–	–	1,693
Total Open Pit	Mt	106	121	138	142	117	124	125	133	138	137	943	289	94	–	–	–	–	–	–	2,607
Underground	Mt	–	–	1	2	5	10	15	20	26	31	389	888	936	152	–	–	–	–	–	2,475
Processed	Mt	41	44	44	44	52	51	50	48	47	48	561	1,176	1,029	152	–	–	–	–	–	3,389
	NSR	22.15	25.35	40.17	31.81	29.18	45.45	70.88	88.19	87.98	87.24	68.39	47.38	44.45	28.31	–	–	–	–	–	49.97
	Cu %	0.48	0.48	0.53	0.47	0.60	0.82	1.16	1.40	1.44	1.47	1.15	0.81	0.73	0.43	–	–	–	–	–	0.84
	Au g/t	0.13	0.20	0.56	0.38	0.16	0.24	0.45	0.56	0.44	0.37	0.35	0.28	0.31	0.23	–	–	–	–	–	0.31
	Ag g/t	1.24	1.27	1.40	1.25	1.36	1.85	2.51	2.96	2.96	2.93	2.53	2.05	2.07	1.49	–	–	–	–	–	2.11
	As ppm	103.85	94.76	35.30	42.59	166.28	94.71	64.59	56.87	59.40	75.12	64.02	107.23	46.76	1.50	–	–	–	–	–	73.24
	Mo ppm	58.45	51.51	49.46	66.50	63.51	58.13	48.49	41.24	45.86	51.01	43.83	55.54	63.82	123.59	–	–	–	–	–	58.88
Bulk Concentrate	kt	669	727	812	707	1,051	1,292	1,597	1,805	1,898	2,012	20,606	32,712	25,531	2,347	–	–	–	–	–	93,766
	Conc. Cu %	21.62	22.40	23.81	24.05	22.95	27.25	31.90	33.75	32.84	31.94	28.05	25.73	25.73	24.17	–	–	–	–	–	26.64
	Conc. Au g/t	5.48	8.37	22.91	17.51	5.17	7.05	10.84	11.94	9.00	7.23	7.54	7.92	9.82	11.68	–	–	–	–	–	8.73
	Conc. Ag g/t	54.87	57.24	60.15	61.06	50.74	58.27	65.35	67.31	63.47	60.06	57.89	61.22	69.29	79.43	–	–	–	–	–	63.11
	Conc. As ppm	3,053	2,849	743	791	3,977	1,796	1,002	747	540	591	1,039	2,428	1,215	66	–	–	–	–	–	1,588
	Conc. Mo ppm	–	–	–	–	–	–	–	–	–	–	–	–	1,281	1,650	5,142	–	–	–	–	1,025
	Conc. F ppm	233	246	307	260	227	328	323	327	328	342	364	365	270	24	–	–	–	–	–	323
Recovered Metal	Copper blb	0.3	0.4	0.4	0.4	0.5	0.8	1.1	1.3	1.4	1.4	12.7	18.6	14.5	1.3	–	–	–	–	–	55.1
	Gold Moz	0.1	0.2	0.6	0.4	0.2	0.3	0.6	0.7	0.5	0.5	5.0	8.3	8.1	0.9	–	–	–	–	–	26.3
	Silver Moz	1.2	1.3	1.6	1.4	1.7	2.4	3.4	3.9	3.9	3.9	38.4	64.4	56.9	6.0	–	–	–	–	–	190.2

24.1.5.6 Economics – Resources 120 Case

The base case price assumptions for the financial evaluation for the Resources 120 Case are the same as those used for the 2016 Reserves Case.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

A summary of the financial results is shown in Table 24.35 and the mining production statistics are shown in Table 24.36.

Mine site cash costs, revenues, operating costs, and capital costs are shown in Table 24.37 to Table 24.39.

Table 24.35 Financial Results – Resources 120 Case

	Discount Rate	Before Taxation	After Taxation
NPV (US\$b)	Undiscounted	59.96	48.07
	5%	19.15	15.95
	6%	15.53	13.03
	7%	12.66	10.69
	8%	10.35	8.80
	9%	8.48	7.25
	10%	6.96	5.98
IRR (%)		22	21
Project Payback Period (Years)		8	8

Table 24.36 Mining Production Statistics – Resources 120 Case

		Resources 120 Case	5-Year Average	10-Year Average	Resources 120 Case Average
Quantity Ore Treated	Mt	3,389	45.12	46.95	77.02
Copper Feed Grade	%	0.84	0.51	0.90	0.84
Gold Feed Grade	g/t	0.31	0.28	0.35	0.31
Silver Feed Grade	g/t	2.11	1.31	1.99	2.11
Copper Recoveries	%	88.10	78.79	86.71	88.10
Gold Recoveries	%	78.19	72.26	76.73	78.19
Silver Recoveries	%	82.93	75.87	81.81	82.93
Copper Concentrate	Mt (dry)	93.8	0.8	1.3	2.1
Copper Concentrate Grade	%	27	23	29	27
Contained Metal in Concentrate					
Copper	Mt	25.0	0.2	0.4	0.6
Copper	blb	55.1	0.4	0.8	1.3
Gold	Moz	26.3	0.3	0.4	0.6
Silver	Moz	190.2	1.4	2.5	4.3

Table 24.37 Unit Operating Costs by Copper Production – Resources 120 Case

Description	US\$/lb Payable Copper
Mine Site Cash Cost	1.75
By-product Credit	0.71
Cash Costs (Net of By-product Credit)	1.04

Table 24.38 Operating Costs and Revenues – Resources 120 Case

	US\$b	US\$/t Ore Milled		
	Total Resources 120 Case	5-Year Average	10-Year Average	Resources 120 Case Average
Revenue				
Gross Sales Revenue	195.90	31.14	59.61	57.81
Less: Realization Costs				
Realization Costs	22.53	4.59	6.72	6.65
Government Royalty	10.21	1.63	3.10	3.01
Total Realization Costs	32.74	6.22	9.82	9.66
Net Sales Revenue	163.16	24.92	49.79	48.15
Less: Site Operating Costs				
Mining (all sources)	16.13	4.31	5.39	4.76
Processing and Tailings	25.01	7.20	7.51	7.38
G&A and Operations Support	7.12	2.50	2.41	2.10
Infrastructure and Other	2.80	1.34	0.78	0.83
Government Fees & Charges	5.37	1.80	1.88	1.58
Management and JV Payments	6.51	2.43	2.30	1.92
Total Site Operating Costs	62.94	19.58	20.26	18.57
Operating Margin	100.22	5.34	29.53	29.57

Table 24.39 Total Project Expansion and Sustaining Capital Cost – Resources 120 Case

US\$b	Expansion	Sustaining	Total
Direct Costs			
Open Pit	–	1.19	1.19
Underground	7.36	16.24	23.60
Concentrator	5.02	0.46	5.48
Infrastructure	0.35	0.31	0.66
Tailings Storage Facility (TSF)	–	2.69	2.69
Subtotal	12.74	20.88	33.62
Construction Indirect	0.86	–	0.86
Contractor Execution – EPCM	0.31	–	0.31
Owner Execution	0.43	0.24	0.67
GOM Fees & Charges – Mongolian VAT	0.52	1.86	2.39
Total	14.86	22.99	37.85

Note: Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.

The changes in financial results for a range of copper and gold prices, calculated at a silver price of US\$19.00/oz, are shown in Table 24.40.

The changes in financial results for a range of silver prices, calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz, are shown in Table 24.41.

A cumulative cash flow for the Resources 120 Case is depicted in Figure 24.29 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and a complete cash flow is provided in Table 24.42.

Table 24.40 After Tax Metal Price Sensitivity – Resources 120 Case

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	–6.21	–5.67	–5.12	–4.58	–4.03	–3.49	–2.13
2.00	–4.08	–3.53	–2.99	–2.44	–1.90	–1.35	0.01
2.50	1.27	1.81	2.36	2.90	3.45	3.99	5.36
2.80	4.48	5.02	5.57	6.11	6.66	7.20	8.57
3.00	6.62	7.16	7.71	8.25	8.80	9.34	10.70
3.50	11.96	12.51	13.05	13.60	14.14	14.69	16.05
4.00	17.31	17.85	18.40	18.94	19.49	20.03	21.39
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	–
2.00	–	–	–	–	–	3	8
2.50	11	12	13	14	15	16	18
2.80	16	16	17	18	19	20	22
3.00	18	19	20	21	21	22	24
3.50	24	24	25	26	27	27	29
4.00	28	29	29	30	31	32	34
Project Payback After Tax (Years)							
1.80	–	–	–	–	–	13.5	11.5
2.00	–	13.5	12.7	12.1	11.3	10.7	9.8
2.50	9.7	9.5	9.3	9.1	8.9	8.7	8.3
2.80	8.8	8.7	8.5	8.4	8.3	8.1	7.8
3.00	8.4	8.3	8.2	8.1	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.16	1.12	1.07	1.03	0.98	0.94	0.83
2.00	1.17	1.13	1.08	1.04	0.99	0.95	0.84
2.50	1.20	1.15	1.11	1.06	1.02	0.97	0.86
2.80	1.21	1.17	1.12	1.08	1.03	0.99	0.88
3.00	1.22	1.18	1.13	1.09	1.04	1.00	0.89
3.50	1.25	1.20	1.16	1.11	1.07	1.02	0.91
4.00	1.27	1.23	1.18	1.14	1.10	1.05	0.94

[†] Calculated at a silver price of US\$19.00/oz

Table 24.41 After Tax Silver Price Sensitivity – Resources 120 Case

After Tax Values†	Silver (US\$/oz)						
	10.00	12.00	15.00	17.00	19.00	24.00	30.00
NPV8% (US\$b)	8.48	8.55	8.65	8.72	8.80	8.97	9.19

† Calculated at a copper price of US\$3.00/lb and a gold price of US\$1,300/oz

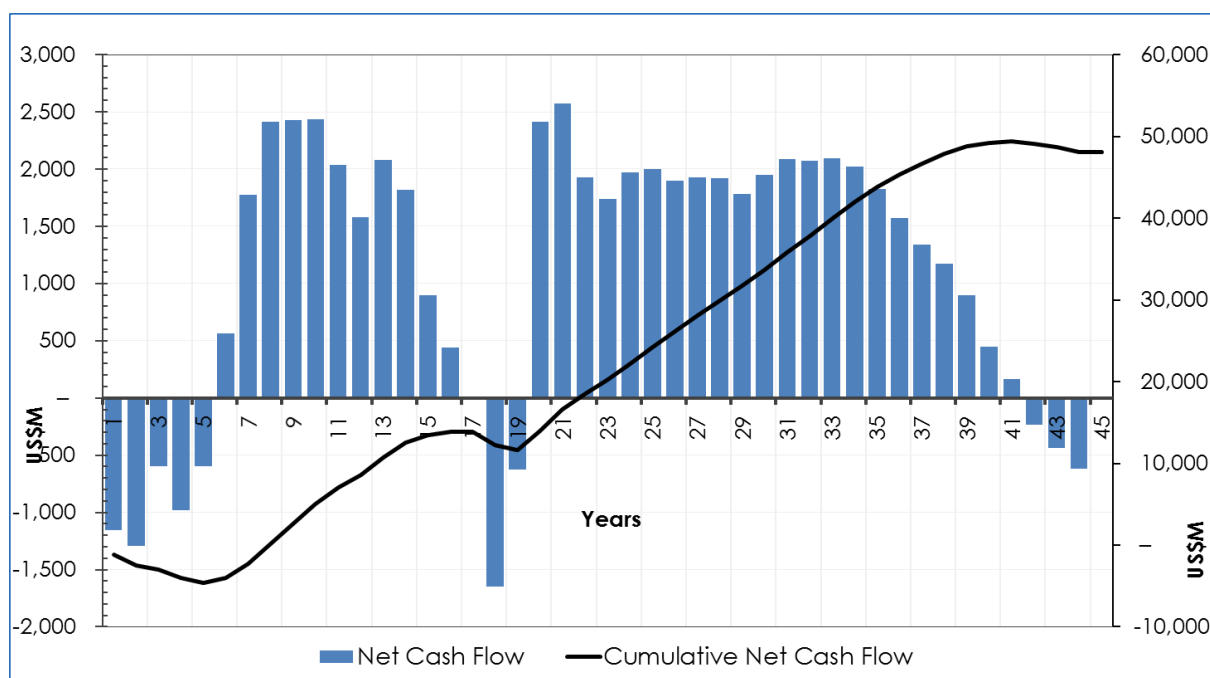
Figure 24.29 Cumulative Cash Flow – Resources 120 Case


Table 24.42 Cash Flow – Resources 120 Case

Cash Flow Statement (US\$M)	Year																		Total	
Year Number	1	2	3	4	5	6	7	8	9	10	11	21	31	41	51	61	71	81		91
Year To											20	30	40	50	60	70	80	90		100
Gross Revenue	819	1,074	1,834	1,526	1,772	2,647	4,009	4,826	4,728	4,752	43,698	65,575	53,404	5,231	–	–	–	–	–	195,895
Realization Costs	251	252	295	244	361	445	602	702	716	742	7,204	11,212	8,856	857	–	–	–	–	–	32,739
Net Sales Revenue	568	822	1,540	1,282	1,411	2,202	3,408	4,124	4,012	4,010	36,494	54,363	44,548	4,374	–	–	–	–	–	163,156
Site Operating Costs																				
Mining	141	191	199	201	239	267	300	317	329	343	4,426	4,169	4,054	949	–	–	–	–	–	16,125
Processing and Tailings	288	328	326	322	360	367	388	384	380	382	4,335	8,474	7,403	1,277	–	–	–	–	–	25,013
G&A and Operations Support	103	105	104	114	140	125	117	114	107	104	1,182	2,402	2,092	314	–	–	–	–	–	7,123
Infrastructure and Other	65	66	48	36	87	22	2	19	14	4	535	914	780	206	–	–	–	–	–	2,799
Total Site Operating Costs	598	690	677	673	827	781	807	834	831	833	10,478	15,959	14,329	2,746	–	–	–	–	–	51,061
Operating Surplus / (Deficit)	–30	132	863	610	584	1,421	2,601	3,291	3,181	3,177	26,016	38,404	30,220	1,628	–	–	–	–	–	112,095
Indirect Costs	161	189	187	200	217	202	207	212	196	191	2,431	3,353	3,549	586	–	–	–	–	–	11,880
Net Profit Before Income Tax	–191	–57	676	410	367	1,219	2,393	3,079	2,985	2,986	23,585	35,050	26,671	1,042	–	–	–	–	–	100,215
Income Tax	–	–	–	–	–	–	–	–	–	–	1,335	5,830	4,719	–	–	–	–	–	–	11,884
Net Profit After Income Tax	–191	–57	676	410	367	1,219	2,393	3,079	2,985	2,986	22,250	29,221	21,951	1,042	–	–	–	–	–	88,331
Capital Expenditure																				
Expansion Capital	874	1,072	1,080	831	387	96	37	51	123	84	9,335	364	–	–	–	–	–	–	–	14,334
Sustaining Capital	57	103	74	369	434	385	424	485	323	337	3,206	8,374	5,724	829	–	–	–	–	–	21,124
VAT & Duties	76	83	68	104	76	46	48	55	42	41	512	703	473	61	–	–	–	–	–	2,388
Subtotal	1,008	1,258	1,223	1,304	898	527	508	591	488	462	13,053	9,441	6,197	890	–	–	–	–	–	37,847
Working Capital, Capitalised Operating Costs and Closure	–48	–28	43	79	68	119	100	67	60	84	179	90	237	1,276	–	–	–	–	–	2,326
VAT & Duties (Capex)	3	3	4	5	0	7	9	7	7	8	32	–	–	–	–	–	–	–	–	85
Total Capital Expenditure	962	1,233	1,270	1,388	965	653	617	665	555	554	13,264	9,531	6,434	2,166	–	–	–	–	–	40,257
Net Cash Flow After Tax	–1,152	–1,290	–594	–979	–599	566	1,776	2,413	2,431	2,432	8,985	19,690	15,518	–1,124	–	–	–	–	–	48,074

24.1.6 Comparison – 2016 Reserves Case and Alternative Production Cases

A comparison was made of the 2016 Reserves Case (base case) with the Alternative Production Cases. Four cost sensitivity options were analyzed. Each sensitivity assumes an improvement in the costs and productivities. The improvements could be the result of optimization and efficiencies from the experience that will be gained over the years of developing and operating the plant and mines at Oyu Tolgoi. The cost assumptions are:

- Underground construction capital costs reduced by 30%.
- Operating costs reduced by 15%.
- G&A costs are assumed to reach a long-term average annual cost of US\$50 M from Year-7. This cost is based on a review of costs from studies of other copper projects.
- Rail freight available to the project after 2020 and the concentrate freight cost is reduced to US\$25/t.

The analyses of the Alternative Production Cases are effectively Preliminary Economic Assessments under NI 43-101 and therefore do not have as high a level of certainty as the 2016 Reserves Case (base case). The Alternative Production Cases are preliminary in nature and include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the cases will be realized.

The after tax NPV8% results of the comparisons are shown in Table 24.43. Option 'A' compares the results using the base case assumptions and Options 'B' through 'E' show the results of applying the sensitivities cumulatively. The results indicate that for the Option 'A' (base case assumptions) there is an improvement in after tax NPV8% for the 2016 Resources Case and the Resources 50 Case has the highest value. When each of the options is applied cumulatively there is an increase in value and the Resources 100 Case has the highest value for Options 'B' and 'C' and the Resources 120 Case has the highest value for Options 'D' and 'E'.

The after tax NPV8% results based on US\$3.50/lb copper and US\$1,400/oz gold is shown in Table 24.44. The expansion capital and variation by the Options is shown in Table 24.45. These costs include the expansion capital for each new mine and for plant and infrastructure. In each of the cases there will be opportunity to fund from the project cash flow.

Metal price sensitivities with variation in after tax NPV8%, IRR, Payback, and Cash Cost for a number of cases are shown in Table 24.46 through Table 24.70.

Table 24.43 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.00/lb Copper and US\$1,300/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	6.94	8.37	9.32	8.88	8.80
B	Underground Construction Capital Reduced by 30%	US\$b	7.85	9.64	10.57	10.59	10.51
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	8.97	10.20	11.86	12.00	11.98
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	9.14	10.43	12.20	12.50	12.57
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	9.62	11.02	13.15	13.58	13.69

Note: Based on US\$3.00/lb copper, US\$1,300/oz gold, US\$19.00/oz silver, and 8% discount rate.

Table 24.44 2016 Reserves Case and Alternative Production Cases – NPV8% After Tax Comparison based on US\$3.50/lb Copper and US\$1,400/oz Gold

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	10.80	12.95	14.36	14.59	14.69
B	Underground Construction Capital Reduced by 30%	US\$b	11.72	14.21	15.62	16.30	16.40
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	12.83	14.78	16.91	17.71	17.87
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	13.00	15.01	17.25	18.21	18.46
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	13.48	15.59	18.20	19.29	19.58

Note: Based on US\$3.50/lb copper, US\$1,400/oz gold, US\$19.00/oz silver, and 8% discount rate.

Table 24.45 2016 Reserves Case and Alternative Production Cases – Expansion Capital Costs

Option	Cost Assumptions	Unit	2016 Reserves Case	2016 Resources Case	Resources 50 Case	Resources 100 Case	Resources 120 Case
A	2016 Base Case	US\$b	4.63	9.73	9.73	13.47	14.86
B	Underground Construction Capital Reduced by 30%	US\$b	4.13	7.69	7.69	11.43	12.82
C	Underground Construction Capital Reduced by 30% and Operating Costs by 15%.	US\$b	4.13	7.69	7.69	11.43	12.82
D	Underground Construction Capital, Operating, and G&A Costs Reduced	US\$b	4.13	7.69	7.69	11.43	12.82
E	Underground Construction Capital, Operating, and G&A Costs Reduced and Rail Transport	US\$b	4.13	7.69	7.69	11.43	12.82

Notes:

1. Capital costs include only direct project costs and exclude interest expense, capitalized interest, debt repayments, tax pre-payments and forex adjustments.
2. In all cases total capital cost excludes capital costs for the year 2016. Expansion capital for 2016 excluded is US\$0.46.

Table 24.46 After Tax Metal Price Sensitivity – 2016 Reserves Case, Option A

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.89	-2.54	-2.20	-1.86	-1.51	-1.17	-0.31
2.00	-1.48	-1.14	-0.79	-0.45	-0.11	0.24	1.10
2.50	2.04	2.39	2.73	3.07	3.42	3.76	4.62
2.80	4.16	4.50	4.84	5.19	5.53	5.87	6.73
3.00	5.56	5.91	6.25	6.59	6.94	7.28	8.14
3.50	9.09	9.43	9.77	10.12	10.46	10.80	11.66
4.00	12.61	12.95	13.29	13.64	13.98	14.32	15.18
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	7
2.00	–	–	5	6	8	9	11
2.50	13	14	14	15	16	17	18
2.80	16	17	18	18	19	20	21
3.00	19	19	20	20	21	22	23
3.50	23	23	24	25	25	26	27
4.00	27	27	28	28	29	30	31
Project Payback After Tax (Years)							
1.80	14.4	13.7	13.1	12.7	12.2	11.7	10.6
2.00	12.2	11.8	11.4	11.0	10.7	10.3	9.7
2.50	9.7	9.5	9.3	9.2	9.0	8.9	8.6
2.80	9.0	8.8	8.7	8.6	8.5	8.4	8.2
3.00	8.6	8.5	8.4	8.3	8.3	8.2	8.0
3.50	8.1	8.0	7.9	7.8	7.7	7.7	7.5
4.00	7.6	7.5	7.5	7.4	7.4	7.3	7.2
Cash Costs (Net of By-product Credits) US\$/lb Payable Copper)							
1.80	1.35	1.31	1.27	1.23	1.18	1.14	1.04
2.00	1.36	1.32	1.28	1.24	1.19	1.15	1.05
2.50	1.38	1.34	1.30	1.26	1.22	1.18	1.08
2.80	1.40	1.36	1.32	1.27	1.23	1.19	1.09
3.00	1.41	1.37	1.33	1.28	1.24	1.20	1.10
3.50	1.43	1.39	1.35	1.31	1.27	1.23	1.12
4.00	1.46	1.42	1.37	1.33	1.29	1.25	1.15

[†] Calculated at a silver price of US\$19.00/oz

Table 24.47 After Tax Metal Price Sensitivity – 2016 Reserves Case, Option B

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-1.97	-1.63	-1.29	-0.94	-0.60	-0.26	0.60
2.00	-0.56	-0.22	0.12	0.47	0.81	1.15	2.01
2.50	2.96	3.30	3.64	3.99	4.33	4.67	5.53
2.80	5.07	5.41	5.76	6.10	6.44	6.79	7.65
3.00	6.48	6.82	7.17	7.51	7.85	8.20	9.05
3.50	10.00	10.34	10.69	11.03	11.37	11.72	12.58
4.00	13.52	13.87	14.21	14.55	14.90	15.24	16.10
Project After Tax IRR (%)							
1.80	-	-	-	-	3	7	11
2.00	5	7	8	10	11	12	15
2.50	15	16	17	18	19	19	21
2.80	19	20	21	21	22	23	25
3.00	21	22	23	23	24	25	27
3.50	26	26	27	28	28	29	31
4.00	29	30	31	31	32	33	34
Project Payback After Tax (Years)							
1.80	12.5	12.0	11.5	11.0	10.6	10.3	9.6
2.00	10.8	10.5	10.2	9.9	9.7	9.5	9.0
2.50	9.0	8.9	8.8	8.6	8.5	8.4	8.1
2.80	8.5	8.4	8.3	8.2	8.1	8.0	7.8
3.00	8.2	8.1	8.1	8.0	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.2
4.00	7.3	7.3	7.2	7.1	7.1	7.0	6.9
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.35	1.31	1.27	1.22	1.18	1.14	1.04
2.00	1.36	1.32	1.28	1.23	1.19	1.15	1.05
2.50	1.38	1.34	1.30	1.26	1.22	1.18	1.07
2.80	1.40	1.36	1.31	1.27	1.23	1.19	1.09
3.00	1.41	1.37	1.32	1.28	1.24	1.20	1.10
3.50	1.43	1.39	1.35	1.31	1.27	1.23	1.12
4.00	1.46	1.41	1.37	1.33	1.29	1.25	1.15

[†] Calculated at a silver price of US\$19.00/oz

Table 24.48 After Tax Metal Price Sensitivity – 2016 Reserves Case, Option C

After Tax Values†	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.86	-0.51	-0.17	0.17	0.52	0.86	1.72
2.00	0.55	0.90	1.24	1.58	1.93	2.27	3.13
2.50	4.07	4.42	4.76	5.10	5.45	5.79	6.65
2.80	6.19	6.53	6.87	7.22	7.56	7.90	8.76
3.00	7.60	7.94	8.28	8.63	8.97	9.31	10.17
3.50	11.12	11.46	11.80	12.15	12.49	12.83	13.69
4.00	14.64	14.98	15.33	15.67	16.01	16.36	17.21
Project After Tax IRR (%)							
1.80	-	5	7	9	10	12	14
2.00	10	11	12	13	14	15	18
2.50	18	19	20	21	22	22	24
2.80	22	22	23	24	25	26	28
3.00	24	24	25	26	27	28	30
3.50	28	29	30	30	31	32	34
4.00	32	33	33	34	35	36	37
Project Payback After Tax (Years)							
1.80	10.9	10.5	10.2	9.9	9.7	9.4	8.9
2.00	9.9	9.6	9.4	9.2	9.0	8.9	8.5
2.50	8.6	8.5	8.3	8.2	8.1	8.0	7.8
2.80	8.1	8.1	8.0	7.9	7.8	7.7	7.5
3.00	7.9	7.8	7.7	7.6	7.6	7.5	7.3
3.50	7.4	7.4	7.3	7.2	7.2	7.1	6.9
4.00	7.1	7.1	7.0	6.9	6.9	6.8	6.6
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.18	1.14	1.10	1.05	1.01	0.97	0.87
2.00	1.19	1.15	1.11	1.06	1.02	0.98	0.88
2.50	1.21	1.17	1.13	1.09	1.05	1.01	0.90
2.80	1.23	1.19	1.14	1.10	1.06	1.02	0.92
3.00	1.24	1.20	1.15	1.11	1.07	1.03	0.93
3.50	1.26	1.22	1.18	1.14	1.10	1.06	0.95
4.00	1.29	1.24	1.20	1.16	1.12	1.08	0.98

† Calculated at a silver price of US\$19.00/oz

Table 24.49 After Tax Metal Price Sensitivity – 2016 Reserves Case, Option D

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.69	-0.35	-0.00	0.34	0.68	1.03	1.89
2.00	0.72	1.06	1.41	1.75	2.09	2.44	3.29
2.50	4.24	4.58	4.93	5.27	5.61	5.96	6.82
2.80	6.35	6.70	7.04	7.38	7.73	8.07	8.93
3.00	7.76	8.11	8.45	8.79	9.14	9.48	10.34
3.50	11.28	11.63	11.97	12.31	12.66	13.00	13.86
4.00	14.81	15.15	15.49	15.84	16.18	16.52	17.38
Project After Tax IRR (%)							
1.80	–	6	8	10	11	12	15
2.00	11	12	13	14	15	16	18
2.50	19	19	20	21	22	23	25
2.80	22	23	23	24	25	26	28
3.00	24	25	25	26	27	28	30
3.50	28	29	30	31	31	32	34
4.00	32	33	34	34	35	36	38
Project Payback After Tax (Years)							
1.80	10.7	10.4	10.1	9.8	9.6	9.3	8.9
2.00	9.8	9.5	9.3	9.1	9.0	8.8	8.5
2.50	8.5	8.4	8.3	8.2	8.1	8.0	7.7
2.80	8.1	8.0	7.9	7.8	7.7	7.6	7.4
3.00	7.9	7.8	7.7	7.6	7.5	7.5	7.3
3.50	7.4	7.3	7.3	7.2	7.2	7.1	6.9
4.00	7.1	7.0	7.0	6.9	6.8	6.8	6.6
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.15	1.11	1.07	1.03	0.99	0.95	0.85
2.00	1.16	1.12	1.08	1.04	1.00	0.96	0.86
2.50	1.19	1.15	1.11	1.06	1.02	0.98	0.88
2.80	1.20	1.16	1.12	1.08	1.04	1.00	0.89
3.00	1.21	1.17	1.13	1.09	1.05	1.01	0.90
3.50	1.24	1.20	1.15	1.11	1.07	1.03	0.93
4.00	1.26	1.22	1.18	1.14	1.10	1.06	0.95

[†] Calculated at a silver price of US\$19.00/oz

Table 24.50 After Tax Metal Price Sensitivity – 2016 Reserves Case, Option E

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.21	0.13	0.48	0.82	1.16	1.51	2.37
2.00	1.20	1.54	1.89	2.23	2.57	2.92	3.77
2.50	4.72	5.06	5.41	5.75	6.09	6.44	7.30
2.80	6.83	7.18	7.52	7.86	8.21	8.55	9.41
3.00	8.24	8.59	8.93	9.27	9.62	9.96	10.82
3.50	11.76	12.11	12.45	12.79	13.14	13.48	14.34
4.00	15.29	15.63	15.97	16.32	16.66	17.00	17.86
Project After Tax IRR (%)							
1.80	7	9	10	11	12	14	16
2.00	12	13	14	15	16	17	19
2.50	19	20	21	22	23	23	26
2.80	23	23	24	25	26	27	29
3.00	25	25	26	27	28	28	30
3.50	29	30	30	31	32	33	35
4.00	33	33	34	35	36	36	38
Project Payback After Tax (Years)							
1.80	10.3	10.0	9.7	9.5	9.3	9.1	8.7
2.00	9.5	9.3	9.1	8.9	8.8	8.6	8.3
2.50	8.4	8.3	8.2	8.1	8.0	7.9	7.7
2.80	8.0	7.9	7.8	7.7	7.6	7.6	7.4
3.00	7.8	7.7	7.6	7.5	7.5	7.4	7.2
3.50	7.3	7.3	7.2	7.2	7.1	7.0	6.9
4.00	7.0	7.0	6.9	6.9	6.8	6.7	6.6
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.07	1.03	0.99	0.95	0.91	0.87	0.76
2.00	1.08	1.04	1.00	0.96	0.92	0.88	0.77
2.50	1.11	1.06	1.02	0.98	0.94	0.90	0.80
2.80	1.12	1.08	1.04	1.00	0.96	0.91	0.81
3.00	1.13	1.09	1.05	1.01	0.97	0.92	0.82
3.50	1.15	1.11	1.07	1.03	0.99	0.95	0.85
4.00	1.18	1.14	1.10	1.06	1.01	0.97	0.87

[†] Calculated at a silver price of US\$19.00/oz

Table 24.51 After Tax Metal Price Sensitivity – 2016 Resources Case, Option A

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-3.22	-2.84	-2.47	-2.09	-1.72	-1.34	-0.40
2.00	-1.54	-1.16	-0.79	-0.41	-0.04	0.34	1.28
2.50	2.67	3.04	3.42	3.79	4.17	4.54	5.48
2.80	5.19	5.57	5.94	6.32	6.69	7.06	8.00
3.00	6.87	7.25	7.62	8.00	8.37	8.75	9.68
3.50	11.07	11.45	11.82	12.20	12.57	12.95	13.89
4.00	15.28	15.65	16.03	16.40	16.78	17.15	18.09
Project After Tax IRR (%)							
1.80	-	-	-	-	-	-	6
2.00	-	-	-	6	8	9	11
2.50	13	14	15	15	16	17	18
2.80	17	17	18	19	19	20	22
3.00	19	19	20	21	21	22	23
3.50	23	24	24	25	25	26	28
4.00	27	27	28	28	29	30	31
Project Payback After Tax (Years)							
1.80	-	14.0	13.3	12.8	12.3	11.8	10.7
2.00	12.3	11.9	11.5	11.1	10.7	10.4	9.8
2.50	9.7	9.5	9.4	9.2	9.1	8.9	8.6
2.80	9.0	8.9	8.8	8.7	8.6	8.4	8.2
3.00	8.7	8.6	8.5	8.4	8.3	8.2	8.0
3.50	8.1	8.0	7.9	7.8	7.8	7.7	7.5
4.00	7.6	7.5	7.5	7.4	7.4	7.3	7.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.27	1.22	1.18	1.13	1.09	1.04	0.93
2.00	1.28	1.23	1.19	1.14	1.10	1.05	0.94
2.50	1.30	1.26	1.21	1.17	1.12	1.08	0.97
2.80	1.32	1.27	1.23	1.18	1.14	1.09	0.98
3.00	1.33	1.28	1.24	1.19	1.15	1.10	0.99
3.50	1.35	1.31	1.26	1.22	1.17	1.13	1.02
4.00	1.38	1.33	1.29	1.24	1.20	1.15	1.04

[†] Calculated at a silver price of US\$19.00/oz

Table 24.52 After Tax Metal Price Sensitivity – 2016 Resources Case, Option B

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-1.95	-1.58	-1.20	-0.83	-0.45	-0.08	0.86
2.00	-0.27	0.10	0.48	0.85	1.23	1.60	2.54
2.50	3.93	4.31	4.68	5.06	5.43	5.81	6.74
2.80	6.45	6.83	7.20	7.58	7.95	8.33	9.27
3.00	8.14	8.51	8.89	9.26	9.64	10.01	10.95
3.50	12.34	12.71	13.09	13.46	13.84	14.21	15.15
4.00	16.54	16.92	17.29	17.67	18.04	18.42	19.35
Project After Tax IRR (%)							
1.80	-	-	-	-	-	8	11
2.00	7	8	10	11	12	13	15
2.50	16	17	17	18	19	20	22
2.80	19	20	21	21	22	23	25
3.00	21	22	23	23	24	25	27
3.50	26	26	27	28	28	29	31
4.00	30	30	31	31	32	33	35
Project Payback After Tax (Years)							
1.80	12.5	12.0	11.5	11.0	10.7	10.3	9.6
2.00	10.8	10.5	10.2	9.9	9.7	9.5	9.1
2.50	9.1	8.9	8.8	8.7	8.5	8.4	8.2
2.80	8.5	8.4	8.3	8.2	8.1	8.0	7.8
3.00	8.2	8.2	8.1	8.0	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.2
4.00	7.3	7.3	7.2	7.1	7.1	7.0	6.9
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.27	1.23	1.18	1.14	1.09	1.05	0.94
2.00	1.28	1.24	1.19	1.15	1.10	1.06	0.95
2.50	1.31	1.26	1.22	1.17	1.13	1.08	0.97
2.80	1.32	1.28	1.23	1.19	1.14	1.10	0.99
3.00	1.33	1.29	1.24	1.20	1.15	1.11	1.00
3.50	1.36	1.31	1.27	1.22	1.18	1.13	1.02
4.00	1.38	1.34	1.29	1.25	1.20	1.16	1.05

[†] Calculated at a silver price of US\$19.00/oz

Table 24.53 After Tax Metal Price Sensitivity – 2016 Resources Case, Option C

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-1.39	-1.02	-0.64	-0.27	0.11	0.48	1.42
2.00	0.29	0.67	1.04	1.42	1.79	2.17	3.10
2.50	4.49	4.87	5.24	5.62	5.99	6.37	7.31
2.80	7.02	7.39	7.77	8.14	8.52	8.89	9.83
3.00	8.70	9.07	9.45	9.82	10.20	10.57	11.51
3.50	12.90	13.28	13.65	14.03	14.40	14.78	15.71
4.00	17.10	17.48	17.85	18.23	18.60	18.98	19.92
Project After Tax IRR (%)							
1.80	–	–	–	7	8	10	13
2.00	9	10	11	12	13	14	17
2.50	17	18	19	20	20	21	23
2.80	21	21	22	23	24	24	26
3.00	23	23	24	25	25	26	28
3.50	27	28	28	29	30	30	32
4.00	31	31	32	33	33	34	36
Project Payback After Tax (Years)							
1.80	11.7	11.2	10.8	10.5	10.1	9.9	9.3
2.00	10.3	10.0	9.8	9.6	9.4	9.2	8.8
2.50	8.8	8.7	8.6	8.5	8.3	8.2	8.0
2.80	8.3	8.2	8.1	8.0	7.9	7.8	7.6
3.00	8.1	8.0	7.9	7.8	7.7	7.6	7.4
3.50	7.6	7.5	7.4	7.3	7.3	7.2	7.1
4.00	7.2	7.1	7.1	7.0	7.0	6.9	6.7
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.20	1.16	1.11	1.07	1.02	0.98	0.87
2.00	1.21	1.17	1.12	1.08	1.03	0.99	0.88
2.50	1.24	1.19	1.15	1.10	1.06	1.01	0.90
2.80	1.25	1.21	1.16	1.12	1.07	1.03	0.92
3.00	1.26	1.22	1.17	1.13	1.08	1.04	0.93
3.50	1.29	1.24	1.20	1.15	1.11	1.06	0.95
4.00	1.31	1.27	1.22	1.18	1.13	1.09	0.98

[†] Calculated at a silver price of US\$19.00/oz

Table 24.54 After Tax Metal Price Sensitivity – 2016 Resources Case, Option D

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-1.16	-0.78	-0.41	-0.03	0.34	0.72	1.66
2.00	0.53	0.90	1.28	1.65	2.03	2.40	3.34
2.50	4.73	5.10	5.48	5.85	6.23	6.60	7.54
2.80	7.25	7.63	8.00	8.38	8.75	9.13	10.06
3.00	8.93	9.31	9.68	10.06	10.43	10.81	11.74
3.50	13.14	13.51	13.89	14.26	14.64	15.01	15.95
4.00	17.34	17.71	18.09	18.46	18.84	19.21	20.15
Project After Tax IRR (%)							
1.80	–	–	6	8	9	11	13
2.00	10	11	12	13	14	15	17
2.50	18	18	19	20	21	21	23
2.80	21	22	22	23	24	25	26
3.00	23	24	24	25	26	26	28
3.50	27	28	29	29	30	31	32
4.00	31	32	32	33	34	34	36
Project Payback After Tax (Years)							
1.80	11.4	11.0	10.6	10.3	10.0	9.7	9.2
2.00	10.2	9.9	9.7	9.5	9.3	9.1	8.7
2.50	8.8	8.7	8.5	8.4	8.3	8.2	7.9
2.80	8.3	8.2	8.1	8.0	7.9	7.8	7.6
3.00	8.1	8.0	7.9	7.8	7.7	7.6	7.4
3.50	7.5	7.5	7.4	7.3	7.3	7.2	7.0
4.00	7.2	7.1	7.1	7.0	7.0	6.9	6.7
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.17	1.12	1.08	1.03	0.99	0.94	0.83
2.00	1.18	1.13	1.09	1.04	1.00	0.95	0.84
2.50	1.20	1.16	1.11	1.07	1.02	0.98	0.87
2.80	1.22	1.17	1.13	1.08	1.04	0.99	0.88
3.00	1.23	1.18	1.14	1.09	1.05	1.00	0.89
3.50	1.25	1.21	1.16	1.12	1.07	1.03	0.92
4.00	1.28	1.23	1.19	1.14	1.10	1.05	0.94

[†] Calculated at a silver price of US\$19.00/oz

Table 24.55 After Tax Metal Price Sensitivity – 2016 Resources Case, Option E

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.57	-0.20	0.18	0.55	0.93	1.30	2.24
2.00	1.11	1.49	1.86	2.24	2.61	2.99	3.92
2.50	5.31	5.69	6.06	6.44	6.81	7.19	8.13
2.80	7.84	8.21	8.59	8.96	9.34	9.71	10.65
3.00	9.52	9.89	10.27	10.64	11.02	11.39	12.33
3.50	13.72	14.10	14.47	14.85	15.22	15.59	16.53
4.00	17.92	18.30	18.67	19.05	19.42	19.80	20.74
Project After Tax IRR (%)							
1.80	–	7	9	10	11	12	15
2.00	11	12	13	14	15	16	18
2.50	18	19	20	21	21	22	24
2.80	22	22	23	24	25	25	27
3.00	24	24	25	26	26	27	29
3.50	28	28	29	30	31	31	33
4.00	31	32	33	33	34	35	37
Project Payback After Tax (Years)							
1.80	10.8	10.5	10.2	9.9	9.7	9.5	9.0
2.00	9.9	9.7	9.5	9.3	9.1	8.9	8.6
2.50	8.6	8.5	8.4	8.3	8.2	8.1	7.8
2.80	8.2	8.1	8.0	7.9	7.8	7.7	7.5
3.00	8.0	7.9	7.8	7.7	7.6	7.5	7.3
3.50	7.5	7.4	7.3	7.3	7.2	7.1	7.0
4.00	7.1	7.1	7.0	7.0	6.9	6.8	6.7
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.08	1.03	0.99	0.94	0.90	0.85	0.74
2.00	1.09	1.04	1.00	0.95	0.91	0.87	0.75
2.50	1.11	1.07	1.02	0.98	0.94	0.89	0.78
2.80	1.13	1.08	1.04	0.99	0.95	0.91	0.79
3.00	1.14	1.09	1.05	1.01	0.96	0.92	0.80
3.50	1.16	1.12	1.08	1.03	0.99	0.94	0.83
4.00	1.19	1.15	1.10	1.06	1.01	0.97	0.85

[†] Calculated at a silver price of US\$19.00/oz

Table 24.56 After Tax Metal Price Sensitivity – Resources 50 Case, Option A

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-3.52	-3.07	-2.61	-2.16	-1.71	-1.26	-0.13
2.00	-1.68	-1.23	-0.78	-0.32	0.13	0.58	1.71
2.50	2.91	3.37	3.82	4.27	4.72	5.18	6.31
2.80	5.67	6.12	6.57	7.03	7.48	7.93	9.06
3.00	7.51	7.96	8.41	8.87	9.32	9.77	10.90
3.50	12.10	12.56	13.01	13.46	13.91	14.36	15.50
4.00	16.70	17.15	17.60	18.05	18.51	18.96	20.09
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	8
2.00	–	–	5	7	8	10	12
2.50	14	14	15	16	17	18	20
2.80	17	18	19	20	20	21	23
3.00	20	20	21	22	23	23	25
3.50	24	25	26	27	27	28	30
4.00	29	29	30	31	31	32	34
Project Payback After Tax (Years)							
1.80	–	–	–	13.9	13.1	12.5	10.8
2.00	13.2	12.6	12.1	11.3	10.8	10.4	9.6
2.50	9.5	9.3	9.1	9.0	8.8	8.6	8.3
2.80	8.7	8.6	8.5	8.3	8.2	8.1	7.8
3.00	8.4	8.2	8.1	8.0	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.4	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.20	1.15	1.11	1.06	1.02	0.97	0.86
2.00	1.21	1.16	1.12	1.07	1.03	0.98	0.87
2.50	1.23	1.19	1.14	1.10	1.05	1.01	0.90
2.80	1.25	1.20	1.16	1.11	1.07	1.02	0.91
3.00	1.26	1.21	1.17	1.12	1.08	1.03	0.92
3.50	1.28	1.24	1.19	1.15	1.10	1.06	0.95
4.00	1.31	1.26	1.22	1.17	1.13	1.08	0.97

[†] Calculated at a silver price of US\$19.00/oz

Table 24.57 After Tax Metal Price Sensitivity – Resources 50 Case, Option B

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.27	-1.81	-1.36	-0.91	-0.46	-0.01	1.13
2.00	-0.43	0.02	0.48	0.93	1.38	1.83	2.96
2.50	4.17	4.62	5.07	5.52	5.98	6.43	7.56
2.80	6.92	7.38	7.83	8.28	8.73	9.18	10.32
3.00	8.76	9.21	9.67	10.12	10.57	11.02	12.15
3.50	13.36	13.81	14.26	14.71	15.16	15.62	16.75
4.00	17.95	18.40	18.85	19.31	19.76	20.21	21.34
Project After Tax IRR (%)							
1.80	–	–	–	–	6	8	12
2.00	6	8	9	11	12	13	16
2.50	16	17	18	19	20	21	23
2.80	20	21	22	23	24	25	27
3.00	22	23	24	25	26	27	29
3.50	27	28	29	30	31	32	34
4.00	32	32	33	34	35	36	38
Project Payback After Tax (Years)							
1.80	13.8	13.0	12.3	11.6	10.8	10.4	9.5
2.00	11.1	10.6	10.2	9.9	9.6	9.4	8.8
2.50	8.9	8.7	8.5	8.4	8.2	8.1	7.8
2.80	8.2	8.1	8.0	7.9	7.7	7.6	7.4
3.00	7.9	7.8	7.7	7.6	7.5	7.4	7.2
3.50	7.4	7.3	7.2	7.1	7.1	7.0	6.8
4.00	7.0	6.9	6.8	6.8	6.7	6.6	6.5
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.20	1.16	1.11	1.07	1.02	0.98	0.87
2.00	1.21	1.17	1.12	1.08	1.03	0.99	0.88
2.50	1.24	1.19	1.15	1.10	1.06	1.01	0.90
2.80	1.25	1.21	1.16	1.12	1.07	1.03	0.92
3.00	1.26	1.22	1.17	1.13	1.08	1.04	0.93
3.50	1.29	1.24	1.20	1.16	1.11	1.07	0.95
4.00	1.32	1.27	1.23	1.18	1.14	1.09	0.98

[†] Calculated at a silver price of US\$19.00/oz

Table 24.58 After Tax Metal Price Sensitivity – Resources 50 Case, Option C

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	−0.98	−0.53	−0.07	0.38	0.83	1.28	2.42
2.00	0.86	1.31	1.77	2.22	2.67	3.12	4.25
2.50	5.46	5.91	6.36	6.81	7.26	7.72	8.85
2.80	8.21	8.66	9.12	9.57	10.02	10.47	11.60
3.00	10.05	10.50	10.95	11.41	11.86	12.31	13.44
3.50	14.64	15.10	15.55	16.00	16.45	16.91	18.04
4.00	19.24	19.69	20.14	20.60	21.05	21.50	22.63
Project After Tax IRR (%)							
1.80	–	5	8	9	11	12	16
2.00	11	12	13	14	15	17	20
2.50	19	20	21	22	23	24	27
2.80	23	24	25	26	27	28	30
3.00	25	26	27	28	29	30	33
3.50	30	31	32	33	34	35	37
4.00	35	36	36	37	38	39	42
Project Payback After Tax (Years)							
1.80	11.3	10.7	10.2	9.9	9.6	9.3	8.7
2.00	9.8	9.5	9.3	9.0	8.8	8.6	8.2
2.50	8.3	8.2	8.0	7.9	7.8	7.6	7.4
2.80	7.8	7.7	7.6	7.5	7.4	7.3	7.1
3.00	7.5	7.4	7.4	7.3	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.8	6.8	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.06	1.02	0.97	0.93	0.88	0.84	0.73
2.00	1.07	1.03	0.98	0.94	0.89	0.85	0.74
2.50	1.10	1.05	1.01	0.96	0.92	0.87	0.76
2.80	1.11	1.07	1.02	0.98	0.93	0.89	0.78
3.00	1.12	1.08	1.03	0.99	0.95	0.90	0.79
3.50	1.15	1.11	1.06	1.02	0.97	0.93	0.81
4.00	1.18	1.13	1.09	1.04	1.00	0.95	0.84

[†] Calculated at a silver price of US\$19.00/oz

Table 24.59 After Tax Metal Price Sensitivity – Resources 50 Case, Option D

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.63	-0.18	0.27	0.72	1.18	1.63	2.76
2.00	1.21	1.66	2.11	2.56	3.01	3.47	4.60
2.50	5.80	6.25	6.70	7.16	7.61	8.06	9.19
2.80	8.56	9.01	9.46	9.91	10.37	10.82	11.95
3.00	10.39	10.85	11.30	11.75	12.20	12.66	13.79
3.50	14.99	15.44	15.89	16.35	16.80	17.25	18.38
4.00	19.58	20.04	20.49	20.94	21.39	21.85	22.98
Project After Tax IRR (%)							
1.80	4	7	9	10	12	13	16
2.00	11	13	14	15	16	17	20
2.50	20	21	22	23	24	25	27
2.80	24	24	25	26	27	28	31
3.00	26	27	28	29	29	30	33
3.50	31	32	32	33	34	35	38
4.00	35	36	37	38	39	39	42
Project Payback After Tax (Years)							
1.80	10.8	10.4	10.0	9.7	9.4	9.1	8.6
2.00	9.6	9.4	9.1	8.9	8.7	8.5	8.1
2.50	8.2	8.1	8.0	7.8	7.7	7.6	7.3
2.80	7.8	7.6	7.5	7.4	7.3	7.3	7.0
3.00	7.5	7.4	7.3	7.2	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.8	6.7	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.01	0.97	0.92	0.88	0.83	0.79	0.68
2.00	1.02	0.98	0.93	0.89	0.84	0.80	0.69
2.50	1.05	1.00	0.96	0.91	0.87	0.82	0.71
2.80	1.06	1.02	0.97	0.93	0.88	0.84	0.73
3.00	1.07	1.03	0.98	0.94	0.89	0.85	0.74
3.50	1.10	1.05	1.01	0.97	0.92	0.88	0.76
4.00	1.13	1.08	1.04	0.99	0.95	0.90	0.79

[†] Calculated at a silver price of US\$19.00/oz

Table 24.60 After Tax Metal Price Sensitivity – Resources 50 Case, Option E

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	0.32	0.77	1.22	1.67	2.13	2.58	3.71
2.00	2.16	2.61	3.06	3.51	3.96	4.42	5.55
2.50	6.75	7.20	7.65	8.11	8.56	9.01	10.14
2.80	9.51	9.96	10.41	10.86	11.32	11.77	12.90
3.00	11.34	11.80	12.25	12.70	13.15	13.61	14.74
3.50	15.94	16.39	16.84	17.30	17.75	18.20	19.33
4.00	20.53	20.99	21.44	21.89	22.34	22.80	23.93
Project After Tax IRR (%)							
1.80	9	10	12	13	14	15	19
2.00	14	15	16	17	18	19	22
2.50	21	22	23	24	25	26	29
2.80	25	26	27	28	29	30	32
3.00	27	28	29	30	31	32	34
3.50	32	33	34	34	35	36	39
4.00	36	37	38	39	39	40	43
Project Payback After Tax (Years)							
1.80	10.0	9.7	9.4	9.2	8.9	8.7	8.2
2.00	9.2	9.0	8.8	8.6	8.4	8.2	7.8
2.50	8.0	7.9	7.8	7.7	7.5	7.4	7.2
2.80	7.6	7.5	7.4	7.3	7.2	7.1	6.9
3.00	7.4	7.3	7.2	7.1	7.0	7.0	6.7
3.50	7.0	6.9	6.8	6.7	6.6	6.6	6.4
4.00	6.6	6.5	6.5	6.4	6.4	6.3	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	0.89	0.84	0.80	0.75	0.71	0.67	0.55
2.00	0.90	0.85	0.81	0.77	0.72	0.68	0.56
2.50	0.93	0.88	0.84	0.79	0.75	0.70	0.59
2.80	0.94	0.90	0.85	0.81	0.76	0.72	0.60
3.00	0.95	0.91	0.86	0.82	0.77	0.73	0.61
3.50	0.98	0.93	0.89	0.84	0.80	0.75	0.64
4.00	1.00	0.96	0.91	0.87	0.82	0.78	0.66

[†] Calculated at a silver price of US\$19.00/oz

Table 24.61 After Tax Metal Price Sensitivity – Resources 100 Case, Option A

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-5.65	-5.14	-4.62	-4.10	-3.59	-3.07	-1.78
2.00	-3.58	-3.06	-2.54	-2.03	-1.51	-0.99	0.30
2.50	1.62	2.14	2.65	3.17	3.68	4.20	5.49
2.80	4.74	5.25	5.77	6.28	6.80	7.32	8.61
3.00	6.81	7.33	7.85	8.36	8.88	9.40	10.69
3.50	12.01	12.52	13.04	13.56	14.07	14.59	15.88
4.00	17.20	17.72	18.23	18.75	19.27	19.78	21.07
Project After Tax IRR (%)							
1.80	-	-	-	-	-	-	-
2.00	-	-	-	-	-2	5	9
2.50	11	12	13	14	15	16	18
2.80	16	17	18	18	19	20	22
3.00	18	19	20	21	22	22	25
3.50	24	24	25	26	27	27	29
4.00	28	29	29	30	31	32	34
Project Payback After Tax (Years)							
1.80	-	-	-	-	-	13.4	11.5
2.00	-	13.4	12.7	12.1	11.3	10.7	9.8
2.50	9.7	9.5	9.3	9.1	8.9	8.7	8.3
2.80	8.8	8.7	8.5	8.4	8.3	8.1	7.8
3.00	8.4	8.3	8.2	8.1	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.18	1.13	1.09	1.04	1.00	0.95	0.84
2.00	1.19	1.14	1.10	1.05	1.01	0.96	0.85
2.50	1.21	1.17	1.12	1.08	1.03	0.99	0.88
2.80	1.23	1.18	1.14	1.09	1.05	1.00	0.89
3.00	1.24	1.19	1.15	1.10	1.06	1.01	0.90
3.50	1.26	1.22	1.17	1.13	1.08	1.04	0.93
4.00	1.29	1.24	1.20	1.15	1.11	1.06	0.95

[†] Calculated at a silver price of US\$19.00/oz

Table 24.62 After Tax Metal Price Sensitivity – Resources 100 Case, Option B

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-3.95	-3.43	-2.91	-2.40	-1.88	-1.36	-0.07
2.00	-1.87	-1.35	-0.84	-0.32	0.20	0.71	2.00
2.50	3.33	3.84	4.36	4.87	5.39	5.91	7.20
2.80	6.44	6.96	7.48	7.99	8.51	9.02	10.31
3.00	8.52	9.04	9.55	10.07	10.59	11.10	12.39
3.50	13.71	14.23	14.75	15.26	15.78	16.30	17.59
4.00	18.91	19.43	19.94	20.46	20.97	21.49	22.78
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	8
2.00	–	–	4	7	9	10	14
2.50	15	16	17	18	19	20	22
2.80	19	20	21	22	23	24	26
3.00	22	23	23	24	25	26	28
3.50	27	28	29	29	30	31	33
4.00	31	32	33	34	35	35	38
Project Payback After Tax (Years)							
1.80	–	13.8	12.9	12.2	11.3	10.7	9.6
2.00	11.6	10.9	10.5	10.0	9.7	9.5	8.9
2.50	8.9	8.8	8.6	8.4	8.3	8.1	7.8
2.80	8.3	8.1	8.0	7.9	7.8	7.7	7.4
3.00	8.0	7.8	7.7	7.6	7.5	7.4	7.2
3.50	7.4	7.3	7.2	7.1	7.1	7.0	6.8
4.00	7.0	6.9	6.8	6.8	6.7	6.6	6.5
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.18	1.14	1.09	1.05	1.00	0.96	0.85
2.00	1.19	1.15	1.10	1.06	1.01	0.97	0.86
2.50	1.22	1.17	1.13	1.09	1.04	1.00	0.88
2.80	1.23	1.19	1.15	1.10	1.06	1.01	0.90
3.00	1.24	1.20	1.16	1.11	1.07	1.02	0.91
3.50	1.27	1.23	1.18	1.14	1.09	1.05	0.93
4.00	1.30	1.25	1.21	1.16	1.12	1.07	0.96

[†] Calculated at a silver price of US\$19.00/oz

Table 24.63 After Tax Metal Price Sensitivity – Resources 100 Case, Option C

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.54	-2.02	-1.50	-0.99	-0.47	0.05	1.34
2.00	-0.46	0.06	0.58	1.09	1.61	2.12	3.42
2.50	4.74	5.25	5.77	6.29	6.80	7.32	8.61
2.80	7.85	8.37	8.89	9.40	9.92	10.44	11.73
3.00	9.93	10.45	10.96	11.48	12.00	12.51	13.80
3.50	15.13	15.64	16.16	16.67	17.19	17.71	19.00
4.00	20.32	20.84	21.35	21.87	22.39	22.90	24.19
Project After Tax IRR (%)							
1.80	–	–	–	–	6	8	13
2.00	6	8	10	12	13	14	18
2.50	18	19	20	21	22	23	26
2.80	22	23	24	25	26	27	30
3.00	25	26	27	28	29	30	32
3.50	30	31	32	33	34	35	37
4.00	34	35	36	37	38	39	41
Project Payback After Tax (Years)							
1.80	11.9	11.0	10.5	10.0	9.7	9.4	8.8
2.00	9.9	9.7	9.4	9.1	8.9	8.7	8.2
2.50	8.4	8.2	8.1	7.9	7.8	7.7	7.4
2.80	7.8	7.7	7.6	7.5	7.4	7.3	7.1
3.00	7.6	7.5	7.4	7.3	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.9	6.8	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.05	1.00	0.96	0.91	0.87	0.82	0.71
2.00	1.06	1.01	0.97	0.92	0.88	0.83	0.72
2.50	1.08	1.04	0.99	0.95	0.90	0.86	0.74
2.80	1.10	1.05	1.01	0.96	0.92	0.87	0.76
3.00	1.11	1.06	1.02	0.97	0.93	0.88	0.77
3.50	1.13	1.09	1.04	1.00	0.95	0.91	0.80
4.00	1.16	1.11	1.07	1.02	0.98	0.93	0.82

[†] Calculated at a silver price of US\$19.00/oz

Table 24.64 After Tax Metal Price Sensitivity – Resources 100 Case, Option D

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.03	-1.52	-1.00	-0.48	0.03	0.55	1.84
2.00	0.05	0.56	1.08	1.60	2.11	2.63	3.92
2.50	5.24	5.76	6.27	6.79	7.31	7.82	9.11
2.80	8.36	8.87	9.39	9.91	10.42	10.94	12.23
3.00	10.44	10.95	11.47	11.98	12.50	13.02	14.31
3.50	15.63	16.15	16.66	17.18	17.69	18.21	19.50
4.00	20.82	21.34	21.86	22.37	22.89	23.41	24.70
Project After Tax IRR (%)							
1.80	–	–	0	6	8	10	14
2.00	8	10	11	13	14	15	19
2.50	19	20	21	22	23	24	26
2.80	23	24	25	26	27	28	30
3.00	25	26	27	28	29	30	32
3.50	30	31	32	33	34	35	37
4.00	35	36	36	37	38	39	42
Project Payback After Tax (Years)							
1.80	11.2	10.6	10.2	9.8	9.5	9.2	8.6
2.00	9.8	9.5	9.2	9.0	8.8	8.6	8.1
2.50	8.3	8.1	8.0	7.9	7.7	7.6	7.4
2.80	7.8	7.7	7.6	7.5	7.4	7.3	7.1
3.00	7.5	7.4	7.3	7.3	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.8	6.8	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	0.97	0.93	0.88	0.84	0.79	0.75	0.63
2.00	0.98	0.94	0.89	0.85	0.80	0.76	0.64
2.50	1.01	0.96	0.92	0.87	0.83	0.78	0.67
2.80	1.02	0.98	0.93	0.89	0.84	0.80	0.68
3.00	1.03	0.99	0.94	0.90	0.85	0.81	0.69
3.50	1.06	1.01	0.97	0.92	0.88	0.83	0.72
4.00	1.08	1.04	0.99	0.95	0.90	0.86	0.74

[†] Calculated at a silver price of US\$19.00/oz

Table 24.65 After Tax Metal Price Sensitivity – Resources 100 Case, Option E

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-0.95	-0.43	0.08	0.60	1.11	1.63	2.92
2.00	1.13	1.64	2.16	2.68	3.19	3.71	5.00
2.50	6.32	6.84	7.35	7.87	8.39	8.90	10.19
2.80	9.44	9.95	10.47	10.99	11.50	12.02	13.31
3.00	11.52	12.03	12.55	13.06	13.58	14.10	15.39
3.50	16.71	17.23	17.74	18.26	18.78	19.29	20.58
4.00	21.90	22.42	22.94	23.45	23.97	24.49	25.78
Project After Tax IRR (%)							
1.80	3	6	8	10	12	13	17
2.00	11	13	14	15	16	18	21
2.50	20	21	22	23	24	25	28
2.80	24	25	26	27	28	29	32
3.00	26	27	28	29	30	31	34
3.50	31	32	33	34	35	36	38
4.00	36	37	37	38	39	40	43
Project Payback After Tax (Years)							
1.80	10.2	9.8	9.5	9.3	9.0	8.8	8.3
2.00	9.3	9.1	8.8	8.6	8.5	8.3	7.9
2.50	8.1	7.9	7.8	7.7	7.6	7.5	7.2
2.80	7.6	7.5	7.4	7.3	7.2	7.1	6.9
3.00	7.4	7.3	7.2	7.1	7.1	7.0	6.8
3.50	7.0	6.9	6.8	6.7	6.7	6.6	6.4
4.00	6.6	6.5	6.5	6.4	6.4	6.3	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	0.85	0.80	0.76	0.71	0.67	0.62	0.51
2.00	0.86	0.81	0.77	0.72	0.68	0.63	0.52
2.50	0.88	0.84	0.79	0.75	0.70	0.66	0.55
2.80	0.90	0.85	0.81	0.76	0.72	0.67	0.56
3.00	0.91	0.86	0.82	0.77	0.73	0.68	0.57
3.50	0.93	0.89	0.84	0.80	0.75	0.71	0.60
4.00	0.96	0.91	0.87	0.82	0.78	0.73	0.62

[†] Calculated at a silver price of US\$19.00/oz

Table 24.66 After Tax Metal Price Sensitivity – Resources 120 Case, Option A

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	–6.21	–5.67	–5.12	–4.58	–4.03	–3.49	–2.13
2.00	–4.08	–3.53	–2.99	–2.44	–1.90	–1.35	0.01
2.50	1.27	1.81	2.36	2.90	3.45	3.99	5.36
2.80	4.48	5.02	5.57	6.11	6.66	7.20	8.57
3.00	6.62	7.16	7.71	8.25	8.80	9.34	10.70
3.50	11.96	12.51	13.05	13.60	14.14	14.69	16.05
4.00	17.31	17.85	18.40	18.94	19.49	20.03	21.39
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	–
2.00	–	–	–	–	–	3	8
2.50	11	12	13	14	15	16	18
2.80	16	16	17	18	19	20	22
3.00	18	19	20	21	21	22	24
3.50	24	24	25	26	27	27	29
4.00	28	29	29	30	31	32	34
Project Payback After Tax (Years)							
1.80	–	–	–	–	–	13.5	11.5
2.00	–	13.5	12.7	12.1	11.3	10.7	9.8
2.50	9.7	9.5	9.3	9.1	8.9	8.7	8.3
2.80	8.8	8.7	8.5	8.4	8.3	8.1	7.8
3.00	8.4	8.3	8.2	8.1	7.9	7.8	7.6
3.50	7.7	7.6	7.5	7.5	7.4	7.3	7.1
4.00	7.3	7.2	7.1	7.1	7.0	6.9	6.8
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.16	1.12	1.07	1.03	0.98	0.94	0.83
2.00	1.17	1.13	1.08	1.04	0.99	0.95	0.84
2.50	1.20	1.15	1.11	1.06	1.02	0.97	0.86
2.80	1.21	1.17	1.12	1.08	1.03	0.99	0.88
3.00	1.22	1.18	1.13	1.09	1.04	1.00	0.89
3.50	1.25	1.20	1.16	1.11	1.07	1.02	0.91
4.00	1.27	1.23	1.18	1.14	1.10	1.05	0.94

[†] Calculated at a silver price of US\$19.00/oz

Table 24.67 After Tax Metal Price Sensitivity – Resources 120 Case, Option B

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	–4.50	–3.95	–3.41	–2.86	–2.32	–1.77	–0.41
2.00	–2.36	–1.81	–1.27	–0.72	–0.18	0.37	1.73
2.50	2.99	3.53	4.08	4.62	5.17	5.71	7.08
2.80	6.19	6.74	7.29	7.83	8.38	8.92	10.28
3.00	8.33	8.88	9.42	9.97	10.51	11.06	12.42
3.50	13.68	14.22	14.77	15.31	15.86	16.40	17.77
4.00	19.02	19.57	20.11	20.66	21.21	21.75	23.11
Project After Tax IRR (%)							
1.80	–	–	–	–	–	–	6
2.00	–	–	–	5	7	9	13
2.50	14	15	17	18	19	20	22
2.80	19	20	21	22	23	24	26
3.00	22	22	23	24	25	26	28
3.50	27	28	29	29	30	31	33
4.00	31	32	33	34	35	35	38
Project Payback After Tax (Years)							
1.80	–	14.0	12.9	12.2	11.3	10.7	9.6
2.00	11.6	10.9	10.5	10.0	9.7	9.5	8.9
2.50	8.9	8.8	8.6	8.4	8.3	8.1	7.8
2.80	8.3	8.1	8.0	7.9	7.8	7.7	7.4
3.00	8.0	7.8	7.7	7.6	7.5	7.4	7.2
3.50	7.4	7.3	7.2	7.1	7.1	7.0	6.8
4.00	7.0	6.9	6.8	6.8	6.7	6.6	6.5
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.17	1.13	1.08	1.04	0.99	0.95	0.83
2.00	1.18	1.14	1.09	1.05	1.00	0.96	0.84
2.50	1.21	1.16	1.12	1.07	1.03	0.98	0.87
2.80	1.22	1.18	1.13	1.09	1.04	1.00	0.88
3.00	1.23	1.19	1.14	1.10	1.05	1.01	0.89
3.50	1.26	1.21	1.17	1.12	1.08	1.03	0.92
4.00	1.28	1.24	1.19	1.15	1.10	1.06	0.95

[†] Calculated at a silver price of US\$19.00/oz

Table 24.68 After Tax Metal Price Sensitivity – Resources 120 Case, Option C

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	–3.03	–2.48	–1.94	–1.39	–0.85	–0.30	1.06
2.00	–0.89	–0.35	0.20	0.75	1.29	1.84	3.20
2.50	4.46	5.00	5.55	6.09	6.64	7.18	8.54
2.80	7.66	8.21	8.75	9.30	9.84	10.39	11.75
3.00	9.80	10.35	10.89	11.44	11.98	12.53	13.89
3.50	15.15	15.69	16.24	16.78	17.33	17.87	19.24
4.00	20.49	21.04	21.58	22.13	22.67	23.22	24.58
Project After Tax IRR (%)							
1.80	–	–	–	–	2	7	12
2.00	3	7	9	11	12	14	17
2.50	18	19	20	21	22	23	26
2.80	22	23	24	25	26	27	30
3.00	25	26	26	27	28	29	32
3.50	30	31	32	33	34	35	37
4.00	34	35	36	37	38	39	41
Project Payback After Tax (Years)							
1.80	11.9	11.0	10.5	10.0	9.7	9.4	8.8
2.00	9.9	9.7	9.4	9.1	8.9	8.7	8.2
2.50	8.4	8.2	8.1	7.9	7.8	7.7	7.4
2.80	7.8	7.7	7.6	7.5	7.4	7.3	7.1
3.00	7.6	7.5	7.4	7.3	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.9	6.8	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	1.03	0.99	0.94	0.90	0.85	0.81	0.70
2.00	1.04	1.00	0.95	0.91	0.86	0.82	0.71
2.50	1.07	1.02	0.98	0.93	0.89	0.85	0.73
2.80	1.08	1.04	0.99	0.95	0.90	0.86	0.75
3.00	1.09	1.05	1.00	0.96	0.92	0.87	0.76
3.50	1.12	1.07	1.03	0.99	0.94	0.90	0.78
4.00	1.14	1.10	1.06	1.01	0.97	0.92	0.81

[†] Calculated at a silver price of US\$19.00/oz

Table 24.69 After Tax Metal Price Sensitivity – Resources 120 Case, Option D

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-2.44	-1.89	-1.35	-0.80	-0.26	0.29	1.65
2.00	-0.30	0.24	0.79	1.33	1.88	2.42	3.79
2.50	5.04	5.59	6.13	6.68	7.22	7.77	9.13
2.80	8.25	8.80	9.34	9.89	10.43	10.98	12.34
3.00	10.39	10.94	11.48	12.03	12.57	13.12	14.48
3.50	15.74	16.28	16.83	17.37	17.92	18.46	19.82
4.00	21.08	21.63	22.17	22.72	23.26	23.81	25.17
Project After Tax IRR (%)							
1.80	–	–	–	4	7	9	14
2.00	7	9	11	12	13	15	18
2.50	18	19	20	22	23	24	26
2.80	23	24	25	26	26	28	30
3.00	25	26	27	28	29	30	32
3.50	30	31	32	33	34	35	37
4.00	35	35	36	37	38	39	42
Project Payback After Tax (Years)							
1.80	11.2	10.6	10.2	9.8	9.5	9.2	8.6
2.00	9.8	9.5	9.2	9.0	8.8	8.6	8.1
2.50	8.3	8.1	8.0	7.9	7.7	7.6	7.4
2.80	7.8	7.7	7.6	7.5	7.4	7.3	7.1
3.00	7.5	7.4	7.3	7.3	7.2	7.1	6.9
3.50	7.1	7.0	6.9	6.8	6.8	6.7	6.5
4.00	6.7	6.6	6.6	6.5	6.4	6.4	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	0.95	0.90	0.86	0.81	0.77	0.72	0.61
2.00	0.96	0.91	0.87	0.82	0.78	0.73	0.62
2.50	0.98	0.94	0.89	0.85	0.80	0.76	0.65
2.80	1.00	0.95	0.91	0.86	0.82	0.77	0.66
3.00	1.01	0.96	0.92	0.87	0.83	0.78	0.67
3.50	1.03	0.99	0.94	0.90	0.85	0.81	0.70
4.00	1.06	1.01	0.97	0.92	0.88	0.83	0.72

[†] Calculated at a silver price of US\$19.00/oz

Table 24.70 After Tax Metal Price Sensitivity – Resources 120 Case, Option E

After Tax Values [†]	Gold (US\$/oz)						
Copper (US\$/lb)	900	1,000	1,100	1,200	1,300	1,400	1,650
Project After Tax NPV8% (US\$b)							
1.80	-1.32	-0.78	-0.23	0.31	0.86	1.40	2.77
2.00	0.82	1.36	1.91	2.45	3.00	3.54	4.90
2.50	6.16	6.71	7.25	7.80	8.34	8.89	10.25
2.80	9.37	9.91	10.46	11.00	11.55	12.09	13.46
3.00	11.51	12.05	12.60	13.14	13.69	14.23	15.59
3.50	16.85	17.40	17.94	18.49	19.03	19.58	20.94
4.00	22.20	22.74	23.29	23.83	24.38	24.92	26.29
Project After Tax IRR (%)							
1.80	–	4	7	9	11	13	16
2.00	10	12	13	15	16	17	20
2.50	20	21	22	23	24	25	28
2.80	24	25	26	27	28	29	31
3.00	26	27	28	29	30	31	34
3.50	31	32	33	34	35	36	38
4.00	36	36	37	38	39	40	43
Project Payback After Tax (Years)							
1.80	10.2	9.8	9.5	9.3	9.0	8.8	8.3
2.00	9.3	9.1	8.8	8.6	8.5	8.3	7.9
2.50	8.1	7.9	7.8	7.7	7.6	7.5	7.2
2.80	7.6	7.5	7.4	7.3	7.2	7.1	6.9
3.00	7.4	7.3	7.2	7.1	7.1	7.0	6.8
3.50	7.0	6.9	6.8	6.7	6.7	6.6	6.4
4.00	6.6	6.5	6.5	6.4	6.4	6.3	6.2
Cash Costs (Net of By-product Credits) (US\$/lb Payable Copper)							
1.80	0.82	0.78	0.73	0.69	0.64	0.60	0.49
2.00	0.83	0.79	0.74	0.70	0.65	0.61	0.50
2.50	0.86	0.81	0.77	0.72	0.68	0.63	0.52
2.80	0.87	0.83	0.78	0.74	0.69	0.65	0.54
3.00	0.88	0.84	0.79	0.75	0.70	0.66	0.55
3.50	0.91	0.86	0.82	0.77	0.73	0.68	0.57
4.00	0.93	0.89	0.84	0.80	0.75	0.71	0.60

[†] Calculated at a silver price of US\$19.00/oz

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-sulphidation types of deposits. The deposits are grouped into three areas, from south to north: Heruga, Oyut, and Hugo Dummett.

The exploration programme relies strongly on geophysical survey data (IP and magnetics), and other target anomalies still remain within the project land holdings.

25.1.1 Oyut Zones

Four discrete zones have been identified in the Oyut system, each of which are potentially mineable by open pit methods. In the current Technical Report, the resource estimate has been re-estimated in 2012. The base case assumes a CuEq cut-off of 0.22% but includes the results of alternative scenarios down to 0.2% CuEq 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 zone consists primarily of pyrite–chalcopyrite mineralization related to biotite–magnetite alteration, overprinted by chlorite–sericite alteration. Mineralization is characterized by high gold contents with Au : Cu ratios (g/t Au : % 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 zone is closely associated with chalcopyrite and occurs intergrown with chalcopyrite, as inclusions and fracture infills within pyrite, or on grain boundaries of pyrite. The Southwest zone is essentially hosted in augite basalts.

South zone mineralization is hosted in quartz monzodiorite in the south-west and basalt throughout the central portion of the zone. Chalcopyrite is the principal copper sulphide, 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 zone is characterized by an upward-flaring high-sulphidation zone that overprints and overlies porphyry-style chalcopyrite–gold mineralization. A secondary-enriched supergene chalcocite blanket tens of metres in thickness overlies the high-sulphidation covellite–pyrite zone. The high-sulphidation portion of the Central zone 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 sulphides 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 zone within either basalt or quartz monzodiorite adjacent to intrusive contacts with basalt.

The high-sulphidation part of the Central zone lacks significant gold. Alteration in the Central zone 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-sulphidation mineralization within quartz monzodiorite, and ignimbrite.

The Wedge deposit contains a zone of high sulphidation mineralization hosted principally in dacite tuff, grading downward and southward into chalcopryite mineralization in basalt and quartz monzodiorite host rocks. High-sulphidation mineralization consists of pyrite, chalcopryite, bornite, enargite, covellite, and primary chalcocite in advanced argillically-altered host rocks. The high-sulphidation mineralization grades downward into chalcopryite, with lesser bornite within basalt host rocks, and pyrite + chalcopryite 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%–5% Cu), and is zoned outward to chalcopryite (2%). At grades of <1% Cu, pyrite–chalcopryite ± 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 chalcopryite, although in hand specimen the chalcopryite is usually not visible. The sulphides are disseminated throughout the rock in the matrix as well as in quartz veins. The fine-grained sulphide 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–zunnyite–topaz–dickite (advanced argillic), and sericite–muscovite zones.

25.1.4 Current Resource Estimation

The current Hugo North and Hugo South resource models were developed using industry-accepted methods and are considered acceptable for incorporation into resource estimates. The updated Oyut model has been developed using a 2D level-plan projection method. It is recommended that a wireframe solid grade shell approach be used for further updates. If OT LLC continue to use the 2D level-plan projection method then a separate wireframe solid grade shell approach should also be prepared to allow a comparison of the resource estimate and provide a confirmation of the results. OreWin concurs with OT LLC's conclusion that the estimates for Ag, As, and Mo may require review and revision. Revision of the Ag estimate may even result in an increase in grade relative to the 2005 model. OT LLC has reported the grades for Ag from the 2012 resource model for Mineral Resources but has used the Ag grade from the 2005 model for Mineral Reserves.

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 license, contains a large zone of porphyry copper-gold-molybdenum mineralization that has been subject to ongoing systematic drilling by OT LLC.

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 southern 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 sulphides occur at Heruga in both disseminations and veins/fractures. Mineralized veins have a much lower density at Heruga than in the more northerly Oyut 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 under the supervision of Scott Jackson of Quantitative Group (QG). A close-off date of 21 June 2009 applied for all survey (collar and downhole) and assay data incorporated into the database.

Ivanhoe created 3D shapes (wireframes) of the major geological features of the Heruga deposit. To assist in the estimation of grades in the model, Ivanhoe also manually created 3D 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 sub-vertical 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 drillhole data.

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 downhole 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 3D wireframe was constructed, inside of which the nominal drill spacing was less than 150 m. This wireframe was constructed with the aim of removing 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 ultimately included because it was considered that geological and grade continuity could be reasonably assumed 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 are based on feasibility study level work carried out by OT LLC. The study work has been used to support project financing and meets requirements for Mineral Reserve reporting under the NI 43-101 Standards of Disclosure for Mineral Projects and those of US Industry Guide 7. This result meets the main objective of the 2016 OTTR. OT LLC plans to continue to study and optimize the operation in order to maximize the value of the Mineral Reserve.

OT LLC identified a problem with the Ag grade in the Oyu resource model and reverted to the Ag grade from the 2005 resource model for mine planning work. As discussed in Section 14, the Ag grades in the current Oyu resource model are lower than the 2005 model Ag grades. Silver is a by-product and this difference has been calculated to have an NSR value of approximately US\$0.10/t. OreWin considers that this is within the accuracy of the study work and is not a material issue. OreWin has therefore concluded that the Mineral Reserve is valid. However, it is recommended that OT LLC review the resource estimates for silver and that the updated mine planning work take account of any changes to silver and other elements.

25.3 Alternative Production Cases

Oyu Tolgoi is a very large project that includes five separate deposits. The long-term development of Oyu Tolgoi would involve the development of the resources on all deposits. Alternative Production Cases have been developed to provide early-stage analysis of the development flexibility that exists with respect to later phases of the Oyu Tolgoi deposits (Heruga, Hugo South, and the second lift of Hugo North). Development of these deposits will require separate development decisions in the future based on then prevailing conditions and the development experience obtained from developing and operating the initial phases of Oyu Tolgoi.

The extensive resources at Oyu Tolgoi provide the company with the opportunity to continually evaluate investment decisions as the mine progresses, deposit knowledge increases and economic conditions change.

25.4 Concentrator

OTFS16 is based on sufficient metallurgical testwork, and operating experience in the concentrator, to enable reasonable predictions of orebody performance when treated through the existing, as constructed, concentrator and an expanded concentrator.

Ongoing metallurgical testwork supported by operating experience is expected to improve forecast modelling, grade / recovery / throughput, parameters for future ore types and blends.

25.5 Infrastructure

There are limited changes to site infrastructure beyond that already established. The changes proposed are reasonable and in most cases are expansions of existing systems.

25.6 Water

Ongoing hydrogeological aquifer modelling and monitoring of actual aquifer performance with operations is indicating that the aquifer has a greater capacity than currently licensed. However, the expansion does not require any additional water than that licensed, 918 L/s.

26 RECOMMENDATIONS

Recommendations have been stated throughout the text of the report. OT LLC has been in production from the open pit and the concentrator for three years, and the underground mine at Hugo North is being developed. OT LLC intends to use this experience and apply it to the continued study, operation and development of Oyu Tolgoi. OreWin concurs with this plan. The key areas of further work are as follows.

26.1 Geology and Resources

The present geological interpretations are acceptable for the geological and resource models at the levels of confidence stated. The geological interpretations and models are likely to change as the project proceeds to detailed mine design and construction.

From 2010–2014, OT LLC reviewed and re-evaluated much of the data collected in support of mine design and construction. These reviews included consideration of the additional drilling completed in the project area, incorporation of revised, and more detailed, structural and lithological interpretations, consideration of changes in interpretation of the evolution and genesis of Oyu Tolgoi deposits, and the results of geotechnical reviews. A number of areas were identified that could benefit from targeted work programmes, particularly in the areas of structure and rock mechanics.

The result was development of proposed work for 2014 and beyond that addressed geological issues that could directly affect the mine design and construction. The OT LLC work programme is likely to include the following:

- Continuing to incorporate reconciliation data to assess performance of resource models, especially for the Oyu deposit.
- Continuing with underground and open pit face mapping to improve geological and structural understanding and validate the structural and geological models.
- Undertaking drill testing of structural discontinuities identified in the structural and geotechnical reviews. The results of such testwork would be incorporated into the block model.
- Reviewing the updated block models to pinpoint areas where there are gaps in knowledge concerning lithology, alteration, structure, and mineralization that require targeted drill testing. Such gaps are considered to be more likely on the orebody margins or in areas where high-grade mineralization is in direct contact with areas interpolated as waste.
- Building a 3D district geological and structural model that will assist in further exploration and in the definition of additional drill targets in the near-mine environment.
- Evaluating future work in the Hugo South and Hugo North Lift 2 areas to further enhance the understanding of the overall resource development strategy.
- Completing the quantification of the mineralized inventory within Heruga North.
- Continuing with exploration programmes scaled to business needs to investigate opportunities to expand the project resource base and improve the mine development sequence.

26.2 Open Pit Mining

The Oyut open pit design consists of ten mining phases. Phases 1 and 2 are complete. The open pit reserves are mined over approximately 40 years, continuing while underground mining commences with the Hugo North Lift block cave. Various alternative operating scenarios for the open pit are currently being evaluated around pit phase design and sequencing. This evaluation shows strong potential for improving the value of the project, but the associated work streams for these improvements exceeded the time-frames available to finalise OTFS16. The areas of future work in the open pit are:

- Develop a scope of work to implement mining bottom benches.
- Develop a reconciliation of planned versus actual production according to conditions.
- Review the operator efficiency factors with actual performance and equipment productivity and cost rates.
- Review the equipment utilization factors and revise the effective hours and hours used for costing.
- Reconcile the current catch bench widths achieved versus design widths.
- Develop a contingency plan to manage risks due to potential geotechnical instability of slopes for open pit phases.
- Provide better communication of short-range and medium-range planning results and planning cycles.
- Modify and evaluate the current configuration of trim blasts as a function of achievable mining of bench toes.
- Focus the mine planning process to exploit production on the gold core of the Southwest zone.

26.3 Underground Mining

Underground mine design will proceed into detail design work, with the first priority being to support the restart plan. Detailed mine design will continue to be refined, focusing on final access, mass excavations and infrastructure excavation layouts, Panel 0 layout, and related boundaries associated with both geotechnical structures as well as panel interfaces. The following have been identified as key focus areas:

- Reviewing and managing geotechnical risks for the cave; benchmarking drawpoint loss and drawpoint availability, and further work on potential impacts for Oyu Tolgoi.
- Maximizing extraction drive and pillar stability to manage the risk of ground collapse and impact on production ramp-up and reserves. This will include a review of smaller-profile extraction drives and operating equipment and a detailed design review of ground support for on-footprint poor ground areas
- Improving the understanding and incorporating any impacts of draw and dilution on production schedule. This will include a review of LHD draw strategies for pillar load shedding.
- Analyzing designs and schedules around crossing cave boundaries in detail. This will include a trade-off in parallel of alternate production sequencing to leaving a minable pillar at either end of Panel 0 and a review of Hazmap and identified poor ground areas against design and incorporate improvements.
- Review sequencing of footprint orepass development within cave stress shadow.
- Identify opportunities to de-risk and seek improvements in ramp-up and full production tonnage through increased drawbell construction rate and improved draw management.
- Continue to explore ways of exceeding development rates in critical path areas including access to and initial footprint panel development.
- Review and improve planning tools, systems, processes, and integration. Further work on simulation integration with development and production planning to progress detailed designs, sequencing, and scheduling. Focus on interaction and congestion management to seek improvement opportunities for preproduction and project ramp-up phases.

26.4 Metallurgical Process and Plant

Additional metallurgical work is required to advance designs for evolution of the process plant from Phase 1 to Phase 2. These studies will inform both short and long-term project planning. This will include:

- Additional metallurgical work: variability, comminution, tailings, concentrate, flotation, heap leach potential and gravity gold.
- Debottlenecking and incremental capacity studies.
- Concentrator conversion detailed design.
- Coarse flotation studies.

26.5 Infrastructure and Logistics

Further studies will continue to create a multi-channel and multi-transport mode solution. In particular, direct rail transport is considered a long-term transportation solution after this initial development period when other non-Oyu Tolgoi development projects are initiated in the region. OT LLC has advised that it expects revisions to the project infrastructure 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 further studies.

26.6 Tailings Storage Facility

Ongoing independent quality control and quality assurance will ensure that all earthworks, rockfill placement, and management and monitoring activities are undertaken in accordance with the design and related management and monitoring plans. OT LLC has an ongoing programme of studies to optimize the TSF design. This optimization of the TSF will potentially provide cost savings over the assumptions in OTFS16.

26.7 Power Supply Determination

The supply of power has been recognized as being critical to the execution of Oyu Tolgoi in the IA. In terms of power, the IA includes an overarching commitment from the GOM and OT LLC to work together to determine the most optimal and reliable solutions for power supply. OT LLC will continue to work with the GOM and other stakeholders to evaluate and develop the power requirements for Oyu Tolgoi.

26.8 Health and Safety

Oyu Tolgoi has developed a comprehensive Health, Safety, Environment and Communities Management System (HSEC MS) that meets the requirements of the Rio Tinto HSEC Management System Standard and Health, Environment, Safety and Community Performance Standards. The management system is designed on the principles of continual improvement and adopts the methodology of Plan, Do, Check and Review, which comprises 17 discrete elements for implementation. The Oyu Tolgoi HSEC MS has been audited and is certified to ISO14001 and OHSAS18001. Continued focus on Health Safety and Environment

(HSE) is an important activity that will be required to minimize exposure of personnel and the project to risks while maximizing the overall project value.

26.9 Water Management

The GOM awarded a water utilization contract to OT LLC until 2040, which may in turn be extended for 20-year periods beyond 2040, in accordance with the Law on Water. OT LLC is currently entitled to utilize water at a rate of 918 L/s. Updated hydrogeological modelling, completed in 2013, and based on all three hydrogeological investigation programmes, demonstrates that the Gunii Hooloi aquifer is capable of providing 1,475 L/s, based on the same time and drawdown conditions.

Studies continue per defined ongoing monitoring and socio-economic programmes developed by OT LLC, specifically with regard to water resource management. 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 if any on 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.

26.10 Innovation and Technology Opportunities

OT LLC plans to investigate and implement projects in these areas: project monitoring, process technology, and underground technology. The innovations and possible applications outlined below are not exhaustive, nor definitive, but rather are currently viewed as having the most significant potential impact for Oyu Tolgoi.

OT LLC plans a longer-term view to developing its operations management capability to maximize performance through evaluation and implementation of advanced technologies. This approach would involve the strategic evaluation and collaboration with technology partners before further developing and implementing system capabilities in a phased and prioritized manner. Experience from a variety of industries has shown this approach to be crucial in achieving an integrated system that maximizes the potential of the various technologies and the benefit to Oyu Tolgoi. The innovation and technology opportunities that Rio Tinto is currently examining are:

- Project Monitoring and Optimization
- Process Technology
- Underground Technology
- Operating Systems and Technologies
- Data Management
- Geotechnical Research
- Extraction Level Construction
- Cave Production
- Cave Monitoring

26.11 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.

27 REFERENCES

References to other reports and information have been identified throughout the report in the sections to which the references relate.

27.1 References

OT LLC, 2016: Oyu Tolgoi Feasibility Study 2016 prepared by OT LLC April 2016.

Section 01	Executive Summary
Section 02	Country and Regional settings
Section 03	Ownership and Legal
Section 04	Government and Community Relations
Section 05	Human Resources and Capability Development
Section 06	Occupational Health, Hygiene, and Safety
Section 07	Environment
Section 08	Water Management
Section 09	Geology and Mineral Resource Estimates
Section 10	Underground Mining
Section 11	Open Pit Mining
Section 12	Metallurgical Process and Process Plant
Section 13	Tailings Storage Facility
Section 14	Infrastructure
Section 15	Project Execution Plan
Section 16	Operational Readiness
Section 17	Marketing
Section 18	Capital Costs
Section 19	Operating Costs
Section 20	Summary of Financial Estimates
Section 21	Risk Assessment
Section 22	Next Study Stage
Section 23	Life of Mine Case

OreWin Pty Ltd, 2016: Oyu Tolgoi Alternative Production Analyses prepared for TRQ, September 2016.

CIM Definition Standards on Mineral Resources and Reserves: Canadian Institute of Mining, Metallurgy and Petroleum, May 2014

27.2 Glossary of Symbols and Units

Table 27.1 Table of Symbols and Units

Symbol / Unit	Meaning
'	Minute (plane angle)
"	Second (plane angle)
%	Percent
<	Less than
>	Greater than
°C	Degrees Celsius
µm	Micrometre (micron)
a	Annum (year)
b	Billion
blb	Billion pounds
bt	Billion tonnes
cm	Centimetre
cm ²	Square centimetre
cm ³	Cubic centimetre
d	Day
d/wk	Days per week
dmt	Dry metric tonne
g	Gram
g/t	Grams per tonne
h	Hour (not hr)
ha	Hectare (10,000 m ²)
kg	Kilogram
kg/m ³	Kilograms per cubic metre
kg/t	Kilograms per tonne
kh	Thousand hours
km	Kilometre
km/h	Kilometre per hour
km ²	Square kilometre
koz	Thousand Troy ounces
kPa	Kilopascal
kt	Thousand tonnes
kt/d	Thousand tonnes per day
kt/h	Thousand tonnes per hour
L	Litre
lb	Pound
m	Metre

Symbol / Unit	Meaning
M	Million
m ²	Square metre
m ³	Cubic metre
m/s	Metres per second
Ma	Million years
masl	Metres above (mean) sea level
mg	Milligram
Mlb	Million pounds
mm	Millimetre
mm/a	Millimetres per annum
mm/h	Millimetres per hour
Moz	Million ounces
MPa	Megapascal
Mt	Million tonnes
mV	Millivolts
oz	Ounce
ppb	Parts per billion
ppm	Parts per million
t	Metric tonne (1,000 kg)
t/a	Tonnes per annum
t/d	Tonnes per day
t/h	Tonnes per hour
t/m ³	Tonnes per cubic metre
W/m ²	Watts per m ²
wk	Week (seven days)
wmt	Wet metric tonnes

27.3 Glossary of Abbreviations and Terms

Table 27.2 Table of Abbreviations and Terms

Abbreviation / Term	Description
AAS	Atomic Absorption Spectroscopy
AB	Multiple Current Electrode IP
ABA	Acid Base Accounting
ACS	Access control system
AC-Tek	Advanced Conveyor Technologies
AFS	Aminpro-Flot Simplex
AIF	Annual Information Form
Aminpro	Amelunxen Mineral Processing Ltd.
ANFO	Ammonium nitrate fuel oil
ARD	Acid rock drainage
ATV	Acoustic televiewer
BCF	Proprietary software
BiGd	Biotite-granodiorite
BWI	Ball Mill Work Index
CATV	Cable television
CCTV	Closed-circuit television
CEET	Comminution Economic Evaluation Tool
CHR	Critical hydraulic radius
CIM	Canadian Institute of Mining
CLY	Cumulative liberation yield
CuEq	Copper-equivalent
CV	Coefficient of variation
DAP	Delivered in place
DCS	Distributed Control System
DDH	Diamond drillhole
DEIA	Detailed Environmental Impact Assessments
DEM	Discrete element modelling
DFN	Discrete fracture networks
DIDOP	Detailed Integrated Development Operations Plan (2012)
DTR	Digital trunk radio system
DWT	Deadweight tonnage (shipping)
EBRD	European Bank for Reconstruction and Development
EFNARC	European Federation of National Associations Representing producers and applicators of specialist building products for Concrete
EIA	Environmental Impact Assessment
EJV	Entrée-OT LLC Joint Venture

Abbreviation / Term	Description
EMS	Environmental Management System
EMSys	Electrical monitoring system
Entrée LLC	Subsidiary of Entrée Gold Inc.
EP	Equivalent people
EPCM	Engineering, Procurement, and Construction Management
ESIA	Environmental and Social Impact Assessment
FAS	Fire alarm system
FEL	Front end loader
FIFO	Fly-in-fly-out
FLEET	Flotation Economic Evaluation Tool
FRS	Fibre-reinforced shotcrete
FS	Feasibility study
FoS	Factor of Safety
G&A	General and administration
Gbit	Gigabit (10 ⁹ bits)
GIS	Geographic information system
GOM	Government of Mongolia
GSI	Geological Strength Index
H&S MS	Health & Safety Management System
HEPA	High-efficiency particulate air
HI	Hollow inclusion
HME	Heavy mobile equipment
HSE MS	Health, Safety, and Environment Management System
IA	Investment Agreement with the GOM, effective October 2009
ICP-OES/MS	Inductively-Coupled Plasma Optical Emission Spectroscopy/Mass Spectrometry
ICT	Information and Communications Technology
IDOP	Integrated Development Operations Plan (2011)
IDP05	Integrated Development Plan 2005
IDP07	Integrated Development Plan 2007
IDP10	Integrated Development Plan 2010
IDZ	Isolated Draw Zone
IFC	International Finance Corporation
IFC	Issued for Construction
IMMI	Ivanhoe Mines Mongolia Incorporated
IMMI EIA	IMMI Environmental Impact Assessment
IMPIC	Inner Mongolia Power International Cooperation Company Ltd.
IP	Induced Polarization
IRR	Internal Rate of Return

Abbreviation / Term	Description
ISBL	Outside battery limits
JV	Joint venture
LAN	Local Area Network
LCT	Locked cycle testwork
LHD	Load-haul-dump
LIBOR	London Interbank Offered Rate
LOM	Life-of-mine
MagTell	Magnetotellurics
MCE	Maximum Credible Earthquake
MEL	Mineral exploration license
MFT	Modified Flotation Test
MMI	Mobile Metal Ion
MNT	Mongolian Tugrik
MoA	Memorandum of Agreement
MoU	Memorandum of Understanding
MRMR	Laubscher Modified Rock Mass Rating
MRS	Mesh-reinforced shotcrete
MS	Mass Spectrometry
MSP	Management Services Payment
MTO	Material take-off quantity (m ³)
NAF	Non-acid forming
NGI	Norwegian Geotechnical Institute
NI 43-101	Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects
NN	Nearest neighbour estimation method
NPTG	Mongolian National Power Transmission Grid
NPV	Net Present Value
NPV8%	Net Present Value at 8% Discount Rate
NSR	Net Smelter Return
NUM	National University of Mongolia
OSBL	Outside battery limits
OCDB	Oracle Content Database
OEL	Occupational exposure limits
OK	Ordinary kriging estimation method
OT BOO	Oyu Tolgoi Build Own Operate
OT LLC	Oyu Tolgoi LLC
PAF	Potentially acid forming
PCBC	Block cave modelling proprietary software
PEP	Project Execution Plan

Abbreviation / Term	Description
PFS	Prefeasibility study
PIMA	Portable Infrared Mineral Analyzer
PLC	Programmable Logic Controller
PMF	Probable maximum flood
PPA	Power Purchase Agreement
PSMP	Pit Slope Management Programme
QA/QC	Quality assurance and quality control
Qmd	Porphyritic quartz monzodiorite
QP	Qualified Person
RC	Reverse circulation
RCAG	Research Centre of Astronomy and Geophysics
RDP	Round Determinate Panel
RMB	Chinese Renminbi (also called Yuan)
RMR	Rock Mass Rating
ROM	Run of mine
RQD	Rock quality designation
RSP	Review and Strategic Plan
RTCP	Rio Tinto Copper Projects
RTOTM	Rio Tinto OT Management Limited
RTTI	Rio Tinto Technology and Innovation
SAG	Semi-autogenous grinding
SBR	Sequencing Batch Reactor
SFH	Soft, Firm, and Hard (as pertains to treatment of boundaries in grade estimation)
SGSPA	Small Gobi Strictly Protected Area
SIA	Social Impact Assessment
SLC	Sub-level cave
SMU	Selective mining unit
SOM	Stockpiled in an oxide material
SPI	SAG Performance Index
SPI	SAG Power Index
SRG	North seeking Gyro
SRM	Standard Reference Material
SWIR	Short-wave infrared
TDR	Time Domain Reflectometers
TDS	Total Dissolved Solids
TEC	Trace elements composites
TEG	Technical Evaluation Group
TEM	Telluric electromagnetic

Abbreviation / Term	Description
TR	Technical Report
TRQ	Turquoise Hill Resources Ltd.
TSF	Tailings storage facility
TTPP	Tavan Tolgoi Power Plant Project
UCS	Unconfined Compressive Strength
UDP	Underground Mining Development and Financing Plan
UG FS	Underground feasibility study
US CPI	United States Consumer Price Index
US SEC	United States Securities and Exchange Commission
UTS	Unconfined Tensile Strength
VAT	Value Added Tax
VFR	Visual flight rules
VoIP	Voice over Internet Protocol
VWP	Vibrating wire piezometer
WPT	Windfall Profits Tax