NI 43-101 TECHNICAL REPORT FOR THE YOUNG-DAVIDSON MINE, MATACHEWAN, ONTARIO

ALAMOS GOLD INC

181 Bay Street, Suite 3910

Toronto, ON M5J 2T3

Effective Date: **December 31, 2016**

Report Date: **January 25, 2017**

Prepared by:

Jeffrey Volk, CPG, FAusIMM

Chris Bostwick, FAusIMM

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1 SUMMARY

EXECUTIVE SUMMARY

Alamos Gold Inc. (Alamos) has prepared this technical report (the Report) on the Young-Davison underground gold mine (the Project) located near the town of Matachewan, Ontario, Canada.

The purpose of this report is to provide an update to the 2009 AMEC report entitled “NI 43-101 Technical Report and Preliminary Feasibility Study on the Young-Davidson Property, Matachewan, Ontario”. Alamos has updated this report in order to reflect updated Mineral Reserves and Mineral Resources, operating history since initial production, and other relevant technical changes since the 2009 report.

1.1 Property Setting

The Project is located 3 kilometres (km) west of the community of Matachewan in northern Ontario (Figure 1-1). Kirkland Lake is 60 km east of the Project site and Timmins, which is the regional centre, is located 170 km by road northwest of the site. There are a number of other communities also within reasonable daily commuting distance to the Project. The Matachewan First Nation Reserve, located approximately 12 km north of the Project, is the closest First Nation community to the site.

The site is connected by means of paved Highway 566 to the northern Ontario highway network.

Figure 1-1: Matachewan Location

1.2 Tenure and Agreements

The Project is situated on the Northgate claim block which occupies an area of approximately 5,685 hectares (ha). The claim block includes mineral claims and dispositions.

The claim block is located in the Powell, Yarrow and Cairo townships, within the Larder Lake Mining district. A majority of the claim block is found between Mistinikon Lake to the west and the West Montreal River to the east.

1.3 Geology and Mineralization

The Project is situated within the south-western section of the Abitibi Greenstone Belt in north-east Ontario. The Abitibi Greenstone Belt consists of a complex and diverse array of volcanic, sedimentary, and plutonic rocks typically metamorphosed to greenschist facies grade, but locally attaining amphibolite facies grade. Volcanic rocks range in composition from rhyolitic to komatiitic and commonly occur as mafic to felsic volcanic cycles. Sedimentary rocks consist of both chemical and clastic varieties and occur as both intravolcanic sequences and as unconformably overlying sequences. A wide spectrum of mafic to felsic, pre-tectonic, syn-tectonic and post-tectonic intrusive rocks are present. All lithologies are cut by late, generally northeast-trending Proterozoic diabase dikes.

The Abitibi Greenstone Belt rocks have undergone a complex sequence of deformation events ranging from early fabricless folding and faulting through later upright folding, faulting and ductile shearing resulting in the development of large, dominantly east-west trending, crustal-scale structures (“breaks”) that form a lozenge- like pattern. The regional Larder Lake-Cadillac Fault Zone (LLCFZ) cuts across the Project. The LLCFZ has a subvertical dip and generally strikes east-west. The LLCFZ is characterized by chlorite-talc-carbonate schist and the deformation zone can be followed for over 120 miles from west of Kirkland Lake to Val d’Or.

There are three important groups of Archean sedimentary rocks in the district. The oldest are Pontiac Group quartz greywacke and argillite, which occur as thick assemblages in Québec, while interbedded within the Larder Lake Group volcanic rocks are turbiditic siltstones and greywackes of the Porcupine Group. Unconformably overlying is Timiskiming Group conglomerate, turbidite and iron formation with minor interbedded alkalic volcaniclastic units.

Archean intrusive rocks are numerous in the district but are largely manifested as small stocks, dikes and plugs of augite syenite, syenite and feldspar porphyry occurring in close temporal and spatial association with the distribution of Timiskiming Group sediments. The main syenite mass, which hosts most of the gold mineralization on the property, measures almost 900 m east-west by 300 m north-south.

Huronian Proterozoic sedimentary rocks onlap and define the southern limit of the Abitibi in Ontario. In the project area these rocks are correlative to the Gowganda Formation tillite. Post-Archean dike rocks include Matachewan Diabase and younger Nipissing Diabase, which respectively bracket the Huronian unconformity in the project area.

Essentially all of the historical production at the Young-Davidson Mine and approximately 60% of the production from the MCM Mine is from syenite-hosted gold mineralization (Lovell, 1967). Most of the current open pit and underground Mineral Resources are also related to syenite-hosted gold. The syenite-hosted gold mineralization consists of a stockwork of quartz veinlets and narrow quartz veins, rarely greater than a few inches in thickness, situated within a broader halo of disseminated pyrite and potassic alteration. Visible gold is common in the narrower, glassy-textured quartz veinlets. In general, gold grades increase with quartz veinlet abundance, pyrite abundance, and alteration intensity. Mineralized areas are visually distinctive and are characterized by brick red to pink K-feldspar-rich syenite containing two to three percent disseminated pyrite and several orientations of quartz extension veinlets and veins. The quartz veins and veinlets commonly contain accessory carbonate, pyrite, and feldspar.

1.4 Mineral Resource Estimate

The underground Mineral Resource estimate is based on data available from 2,695 drill holes drilled from both surface and underground, comprising 328,202 metres of non-zero assayed gold intervals.

Mining from the open pit ceased in June 2013. The open pit Mineral Resource is based on a Mineral Resource estimate conducted by AuRico in 2012 (Edmunds 2012).

Separate block models were constructed for both the open pit and underground deposits and are constrained by mineralized shapes, based on cutoff grade, and barren diabase dykes modeled as solids.

Table 1-1: Young-Davidson Underground Mineral Resources as at December 31, 2015

Category	Tonnes (000's)	Au Grade (g/t)	Au Ounces (000's)
Measured	4,248	3.47	474
Indicated	3,707	3.43	408
Total Measured and Inferred	**7,955**	**3.45**	**883**
Inferred	3,523	2.76	312

Notes to Mineral Resources:

- The effective date is December 31, 2015
- The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
- Underground Mineral Resources are stated as contained within potentially economically minable gold grade shapes above a 1.50 g/t Au cut-off and as such includes internal dilution within the 1.50 g/t Au solid.
- The gold price used for Mineral Resources was USD $1400 per ounce.

- Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
- Contained Au ounces are in-situ and do not include metallurgical recovery losses.
- Mineral Resources are exclusive of Mineral Reserves.

1.5 Mineral Reserves

The Mineral Reserves are provided in Table 1-2 and have an effective date of December 31, 2015. Mineral Reserves are calculated using an assumed gold price of USD $1250 per ounce and a metallurgical recovery of 91%.

Table 1-2: Young-Davidson Mineral Reserves as at December 31, 2015

Category	Tonnes (x 1,000)	Au Grade (g/t)	Au Ounces (x 1,000)
Underground – Proven	14,282	2.73	1,255
Underground – Probable	30,008	2.68	2,582
Underground - Proven and Probable	**44,290**	**2.69**	**3,837**
Stockpiles – Proven	1,396	0.82	37
Total Proven and Probable	**45,686**	**2.64**	**3,874**

Notes to Mineral Reserves:

- The estimate has an effective date of December 31st, 2015.
- Chris Bostwick, FAusIMM, Vice President Technical Service, Alamos Gold is the Qualified Person for the estimate.
- The Mineral Reserves are classified as Proven and Probable Mineral Reserves, and are based on the 2014 CIM Definition Standards.
- Mineral Reserve estimates are based on a gold price of USD$1,250 per ounce, and a long-term exchange rate of CAD$1= USD$0.85.
- For underground estimates, a gold cutoff grade of 1.90 grams per tonne is used as an economic indicator only.
- For the surface stockpiles, a gold cutoff grade of 0.50 grams per tonne is used.
- The estimated gold metallurgical recovery rate is 91%.
- Underground mining dilution ranges from 8.4% to 10.9% depending on mining method.
- Underground Mineral Reserves take into account mining recoveries of between 90% and 95% depending on mining method.
- Numbers may not sum due to rounding.

The surface Mineral Reserves are based on stockpile survey measurements as at December 31, 2015.

The underground Mineral Reserves have been estimated within a detailed underground mine design based on a life of mine (LOM) plan.

1.6 Mining Methods

The Young-Davidson underground mine is accessed via the MCM #3 (MCM) shaft, and a ramp system from surface. The Northgate shaft is used for hoisting ore and waste from the mid-shaft location. The MCM shaft is used for transporting personnel and material, and for hoisting ore and waste.

Longhole stoping methods are employed with transverse stoping generally utilized where the orebody width is greater than 12 metres and longitudinal stoping used where the orebody width is less than 12 metres. Paste fill is employed to backfill mined out stopes. Sublevels are in place every 30 metres and are accessed from the main ramp. Below the 9400 level sublevel intervals will be increased to 35 metres.

Work is currently being undertaken to bring the Lower Mine into production. The Northgate shaft is being deepened via raise boring to the 8900 level, and material handling infrastructure will be installed to allow hoisting from that elevation. The MCM shaft has been sunk to full depth at the 8900 and is being used as the means of access to the Lower Mine until such time as the main ramp has been extended to the bottom.

It is anticipated that the Lower Mine facility will be in production in late 2019.

1.7 Metallurgy and Processing

During the pre-feasibility and feasibility stage extensive metallurgical test work was undertaken on both drill composites and a bulk sample to determine the most appropriate flow sheet for the treatment of Young-Davidson ore.

The Young-Davidson mill can be considered a standard flotation/CIL gold mill. The mill was commissioned during Q1 of 2012 and the first gold pour occurred on April 30th, 2012. Ore is currently sourced from two sources, the underground mine, currently ramping up to 8,000 tpd and from low grade surface stockpiles, mined from the open pit between 2011 and 2014.

Open pit stockpiled ore is crushed in a jaw crusher to a P_{80} of 150 mm. Crushed ore is then conveyed to the coarse ore bins feeding the processing plant grinding circuit.

The underground mine ore is fed from an ore pass onto a grizzly feeder into a jaw crusher located underground on the 9530 meter level. The crusher product is conveyed to the skip loading system from an underground bin. The underground ore skipped to surface is fed into a bin located in the headframe and a series of feeders and conveyors feed the coarse ore bins located at the process plant.

Feed from the coarse ore bin discharges onto the SAG mill feed conveyor. The grinding circuit is operated in a closed circuit with cyclones and features a single-stage SAG mill. The SAG mill discharge reports to the cyclone feed pumpbox. The cyclone underflow returns to the SAG mill and the cyclone overflow flows by gravity to the flotation circuit.

The cyclone overflow feeds a trash screen prior to feeding the flotation conditioner. The flotation conditioner overflows into the first flotation cell. The flotation circuit is comprised of four tank cells.

The flotation concentrate is pumped to the concentrate thickener. Thickener underflow is pumped to the regrinding circuit. The thickened concentrate is pumped to the regrinding circuit, comprising a vertical tower mill, where the solids are reduced to a size distribution with a nominal P_{80} target of 15 µm, operated in closed circuit with a cyclone cluster. The cyclone underflow returns to the tower mill box while the cyclone overflow flows directly to the concentrate CIL circuit.

The flotation tailings are pumped to the flotation tailings thickener. The flotation tailings thickener underflow is pumped to the combined CIL circuit. Thickener overflows from both thickeners report to the mill process water tank for reuse.

The flotation concentrate from the regrind circuit reports to a pre-leach tank. The slurry overflows from the pre-leach tank into a series of four CIL tanks providing a total of 48 hours of retention time.

The loaded carbon extracted from the first tank of the concentrate CIL tank is pumped to the carbon stripping circuit for carbon elution.

The flotation tailings, along with the previously leached flotation concentrate are fed to the combined CIL circuit consisting of five leach tanks providing an overall retention time of 24 hours. The combined leach circuit tailings report to the cyanide destruction circuit.

The carbon elution circuit has been sized for processing 4 t/d of carbon. The loaded carbon is pumped from the first flotation concentrate leach tank across a loaded carbon screen. The loaded carbon flows by gravity into the acid wash vessel where the loaded carbon is rinsed by a solution of hydrochloric acid followed by neutralization. The loaded carbon is transferred by a recessed impeller pump to the elution vessel. The stripped carbon is reactivated in an electric kiln and reused in the CIL circuit.

The carbon stripping circuit elutes the precious metals into the pregnant solution. The pregnant solution feeds the electrowinning circuit. Pregnant solution is pumped through the electrowinning circuit to produce sludge on the cathodes containing the precious metals. The cathodes are washed, dewatered and dried. The dried cathode sludge is melted in an electric induction furnace and poured into doré bars.

The combined leach circuit tailings report to the cyanide destruction circuit using the SO_2/air process and copper sulphate solution. The treated tailings slurry is pumped to the tailings impoundment area for settling or to the paste fill plant and a portion of the reclaim water is sent back to the process plant as process water.

The target gold recovery for the Young-Davidson mill in 91%. Young-Davidson is in the process of adding a pebble crusher to the primary grinding circuit and is projecting a fourth quarter 2017 commissioning.

1.8 Environmental, Permitting and Social Considerations

The Young Davidson Mine has been designed to maximize utilization of lands previously altered by historic mining-related activities and, where practical, reclaim brownfield areas progressively during operations; and, minimize overall Project footprint and impacts.

Environmental permits were required by various Federal, Provincial, and municipal agencies for the construction and operation of the Young-Davidson Mine, and are in place. A legal registry is used to maintain a list of active environmental permits covering operation of the Young-Davidson Mine.

Permitting is underway for the redevelopment of the historical Young-Davidson Tailings Area (TIA 1) for additional storage for the life of mine.

Alamos Gold is committed to the ongoing development and operation of the Young-Davidson Mine while engaging in an open and respectful dialogue with Aboriginal people, having an interest in the Mine. Alamos has committed to fostering positive working relationships with the First Nation (FN) groups in an effort to better understand each other's perspectives and shared interests in the environment. Using varied and culturally appropriate engagement activities Alamos will continue to engage and share information in an open, honest and transparent manner.

Impact Benefit Agreements (IBA) have been developed with both the Matachewan First Nation and Temagami First Nation to ensure that the communities derive positive benefit from Young-Davidson Mine operations as it is in part of their traditional territories. Several committees have been implemented, as part of these agreements, to share and gather input on all aspect of the mining operations.

1.9 Capital and Operating Costs

Anticipated capital costs for Young-Davidson are as follows:

Table 1-3: 2017 Budget and LOM Capital Costs

$CDN	2017 Budget	2017-2031 LOM
Surface		
Pebble Crusher	$5,200,000	$5,200,000
Tailings	$6,954,000	$30,193,000
Environmental & Reclamation	$600,000	$10,427,000
Underground		
Capital Development	$42,778,000	$336,283,000
Underground Infrastructure	$8,045,000	$40,518,000
Lower Mine Infrastructure	$13,658,000	$52,753,000
New Equipment	$8,861,000	$13,194,000
Replacement Equipment	$2,383,000	$35,311,000
Equipment Rebuilds & Misc. Maint.	$7,404,000	$64,281,000
Diamond Drilling	$1,554,000	$11,630,000
Total Young-Davidson	**$97,437,000**	**$599,790,000**

Anticipated operating costs for Young-Davidson are as follows:

Table 1-4: 2017 Budget and Life of Mine Operating Costs

	2017 Budget	2017-2031 LOM
Mining ($/t mined)	44.84	36.80
Milling ($/t milled)	14.95	14.34
Admin ($/t milled)	4.30	3.94
Refining ($/ounce)	1.30	1.50

1.10 Economic Analysis

Under NI 43-101, producing issuers may exclude the information required in this section on properties currently in production, unless the Technical Report includes a material expansion of current production. Alamos is a producing issuer, the Young-Davidson Mine is currently in production, and a material expansion is not being planned. Alamos has performed an economic analysis of the Young-Davidson Mine using the estimates presented in this report and confirms that the outcome is a positive after tax cash flow and net present value at a 5% discount rate at USD $1,250 per ounce of gold price that supports the statement of Mineral Reserves.

1.11 Conclusions and Recommendations

1.11.1 Geology and Mineral Resources

The Young-Davidson deposit is a syenite hosted gold deposit hosted by granodioritic to syenitic stocks and dykes. The sampling, sample preparation, analyses, security, and data verification meet or exceed industry standards and are appropriate for Mineral Resource estimation. The parameters, assumptions, and methodology used for Mineral Resource estimation are appropriate for the style of mineralization.

The mine should develop a plan and budget to drill below the 8900 level for additional Mineral Resources when access and resources are available to undertake such a program.

1.11.2 Mining and Mineral Reserves

Mineral Reserves at Young-Davidson are estimated in a sound, consistent and well documented manner. Mineral Reserves are backed up by a mine plan and cash flow model and demonstrate positive economics. Economic inputs including metal prices, operating costs and capital costs are re-evaluated on an annual basis and are used in the generation of Mineral Reserve estimates.

The mine should evaluate the impact of lowering Mineral Reserve cut-off grade to below 1.9 g/t, this could have the impact of increasing Mineral Reserves and increasing the life-of-mine cash flow, metal production and mine life.

1.11.3 Processing

The process plant is currently operating at design rates of 8,000 tpd and with recoveries of 91%. No anticipated long term changes in ore characteristics are anticipated that might cause the process plant to deviate from these operating levels.

1.11.4 Environmental

The mine has all necessary permits in place to continue operations. The mine is the progress of permitting the proposed TIA 1 tailing facility, and no delays are anticipated. The mine has regularly engaged, and expects to continue to engage, the local communities in operating and present and future permitting issues.

2 INTRODUCTION

2.1 Introduction

Alamos Gold Inc. (Alamos) has elected to provide an update on its Young-Davison Mine located near Matachewan, Ontario. The purpose of this update is to provide new information related to Mineral Reserves and Mineral Resources, operational information and general project information since the issuance of the most recent 2009 Technical Report. This previous report was issued prior to a production decision and the start-up of the current operation.

Alamos is a Canadian publicly traded company with a portfolio of operating mines and development projects in Canada, Mexico, US and Turkey. The Young-Davidson mine is located within the Larder Lake Mining District in the Townships of Powell, Yarrow and Cairo, and is approximately 3 kilometres west of the community of Matachewan (Figure 2-1).

The mine is 100% owned and operated by Alamos. The mine is subject to a 1.5% Net Smelter Return (NSR) royalty paid to AuRico Metals Inc.

2.2 Terms of Reference

This Report supports the disclosure of updated Mineral Resources and Mineral Reserves for the Project. Alamos will be using the Report in support of its 2016 Annual Information Form (AIF) filing and other Corporate purposes.

The present day Young-Davidson mine has been built and operated by a number of successive companies. Throughout this report works undertaken by Northgate Minerals Inc., AuRico Gold Inc., and Alamos Gold Inc. may simply be referred to as the Company.

All measurement units used in this report are metric and currency is expressed in Canadian dollars (CDN$), unless stated otherwise.

2.3 Qualified Persons

This Report has been prepared by the following QPs:

- Mr. Jeffrey Volk, CPG, FAusIMM, Director, Reserves and Resources, Alamos Gold Inc.
- Mr. Chris Bostwick, FAusIMM, Vice President, Technical Services, Alamos Gold Inc.

2.4 Site Visits and Technical Report Sections of Responsibilities

Mr. Volk has been involved with the Young-Davidson mine since August, 2012 and has been responsible for the annual and mid-year Mineral Resource model updates since that time. Mr. Bostwick has been involved with the Young-Davidson mine since October 2011, and regularly visits the site in a Corporate review and oversight capacity. Table 2-1 outlines the dates of recent site visits by the QPs and their sections of responsibility in this Technical Report.

Table 2-1: Site Visits Completed by QPs in Support of the Technical Report

Qualified Person	Date(s) of Site Visit		Sections of Responsibility (or Shared Responsibility)
	From	to	
Jeffrey Volk	Dec 3, 2015 Nov 20, 2016	Dec 4, 2015 Nov 30, 2016	1.4,1.5, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 26.1
Chris Bostwick	Jan 18, 2016 Apr 18, 2016 Jun 7, 2016 Jul 25, 2016 Sep 27, 2016 Oct 12, 2016 Oct 26, 2016 Dec 8, 2015	Jan 20, 2016 Apr 19, 2016 Jun 8, 2016 Jul 26, 2016 Sep 28, 2016 Oct 14, 2016 Oct 27, 2016 Dec 9, 2016	1.1-1.3, 1.7-1.12, 2, 3, 4, 5, 6, 13, 15, 16, 17, 18, 19, 20, 21, 22, 24, 25.1, 25.3-25.7, 26.2, 27

2.5 Effective Dates

The effective date of this report is December 31, 2016. The effective date of the Mineral Resources and Mineral Reserves contained within this report, is December 31, 2015.

2.6 Information Sources and References

This Report is based in part on internal company reports, maps, published government reports, and public information, as listed in Section 27 of this Report. Specialist input from Alamos employees in other disciplines, including legal, process, geology, geotechnical, hydrological and financial, was sought to support the preparation of the Report. Information used to support this Report is also derived from previous technical reports on the property. In addition to the Qualified Persons listed in this section, a number of Young-Davidson site personnel have contributed to the content of this report, including:

- Jim Janzen – Regional Exploration Manager
- Gordon Skrecky – Technical Services Superintendent

- Nancy Duquet-Harvey – Environmental Superintendent
- Shawn Deforge – Maintenance Superintendent
- Tyler Clark – Chief Mining Engineer

2.6.1 Previous Technical Reports

Alamos and its predecessors have previously filed the following technical reports on the Project:

- AMEC, NI 43-101, Technical Report and Preliminary Feasibility Study on the Young-Davidson Property, Matachewan Ontario, Effective Date: August 27, 2009
- Carl Edmunds, Northgate Minerals Inc., Technical Report on Underground and Open Pit Mineral Resource Estimates Young-Davidson Property, Matachewan, Ontario, Effective date: January 23, 2009
- AMEC, NI 43-101 Technical Report and Preliminary Assessment on the Young-Davidson Property, Matachewan, Canada, Effective Date, April 01, 2008
- Carl Edmunds, Northgate Minerals Inc., Technical Report on Underground and Open Pit Mineral Resource Estimates Young-Davidson Property, Matachewan, Ontario, Effective date: May 09, 2008
- The Lower boundary Zone, Luckv Zone, and Lower YD Zone Mineral Resource Estimates, Young-Davidson Property, Matachewan, Ontario. Effective Date: January 12, 2007

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Alamos Gold Inc. (Alamos). The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Alamos at the time of preparation of this report;
- Assumptions, conditions, and qualifications as set forth in this report; and,
- Data, reports, and other information supplied to Alamos by third party sources.
- Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

The QPs opinions contained herein are based on information provided by Alamos and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

4 PROPERTY DESCRIPTION AND LOCATION

The Young-Davidson Mine is located 3 kilometres west of the community of Matachewan in northern Ontario at UTM 5310205 N 522230 E, or latitude 47°56’48” N and longitude 80°40’28” W (Figure 4-1). The mine is situated on a claim block which occupies an area of approximately 5,685 ha. The claim block includes patent leases and mineral claims.

4.1 Project Ownership

In November 2005 Young-Davidson Mines Limited became a wholly-owned subsidiary of Northgate Minerals Inc. (Northgate). Northgate owned 100% of mineral rights to all of the Young-Davidson Mine and Matachewan Consolidated Mine claims. On October 26, 2011, AuRico Gold Inc. (AuRico) acquired Northgate. On July 2, 2015 AuRico and Alamos Gold Inc. (Alamos) merged to become Alamos. The Young-Davidson mine is 100% owned by Alamos.

4.2 Mineral Tenure

The claim block on which the Project is situated is located in the Powell, Yarrow and Cairo townships, within the Larder Lake Mining district. A majority of the claim block is found between Mistinikon Lake to the west and the West Montreal River to the east (Figure 4.2) The Company owns 100% of the mineral rights to all of the Mineral Resource related claims at the former Young-Davidson mine and the adjoining Matachewan Consolidated Mines Limited Mine (the “MCM Mine”), which together comprise the modern day Young-Davidson Mine. The Company also holds the mineral rights to 200 tenures from mining leases to exploration claims covering 4,734 hectares surrounding and including the Young-Davidson Mine. The contiguous claim block that covers the Young-Davidson Mine is hereinafter referred to as “Young-Davidson”. These tenures were acquired either through staking, application, or option agreements.

Claim details are included in Appendix A. All claims are in good standing.

Torys LLP has issued a Title Opinion on the Mineral Tenure for the Young-Davidson Project and the QP relies on this Title Opinion.

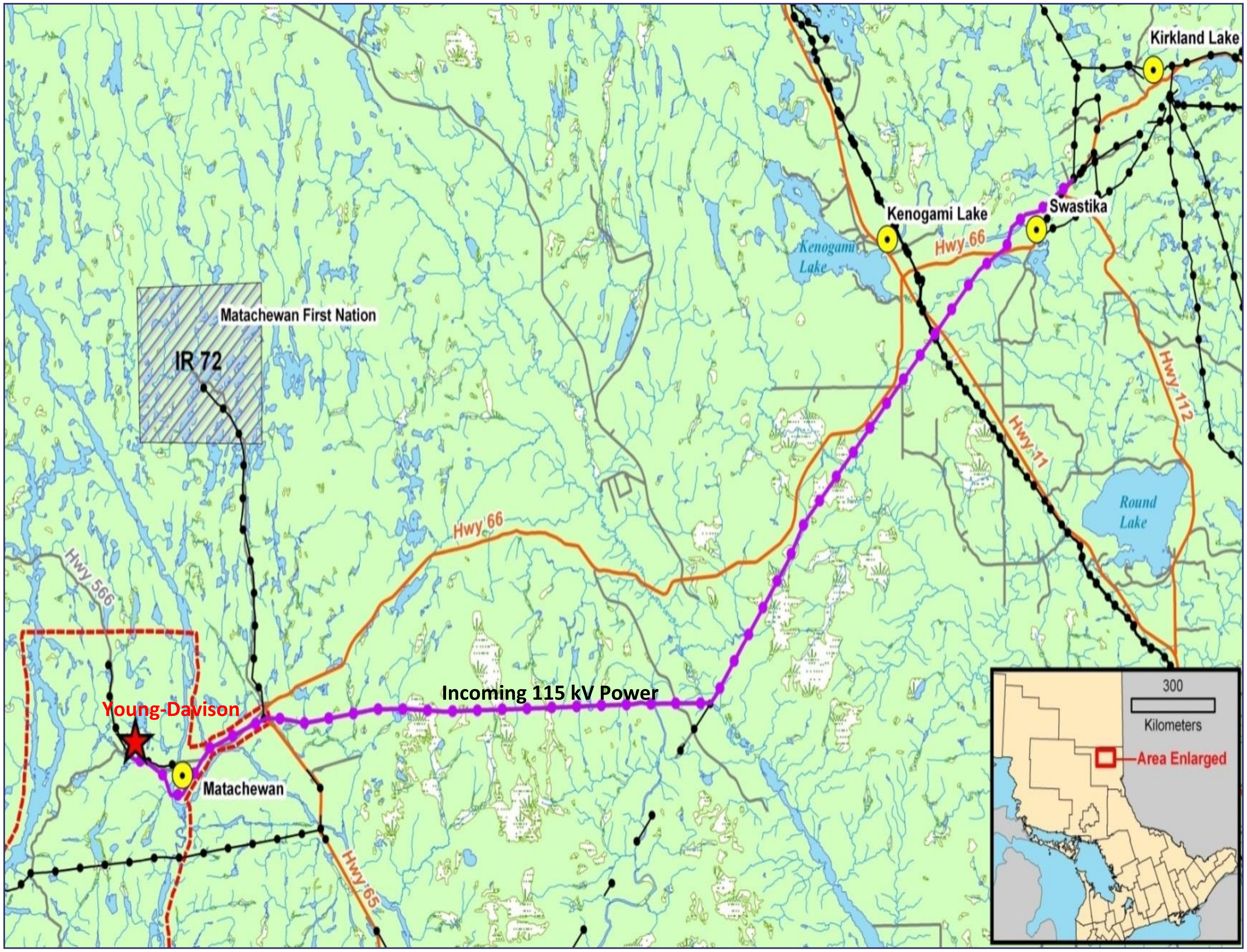


Figure 4-1: Young Davidson Location

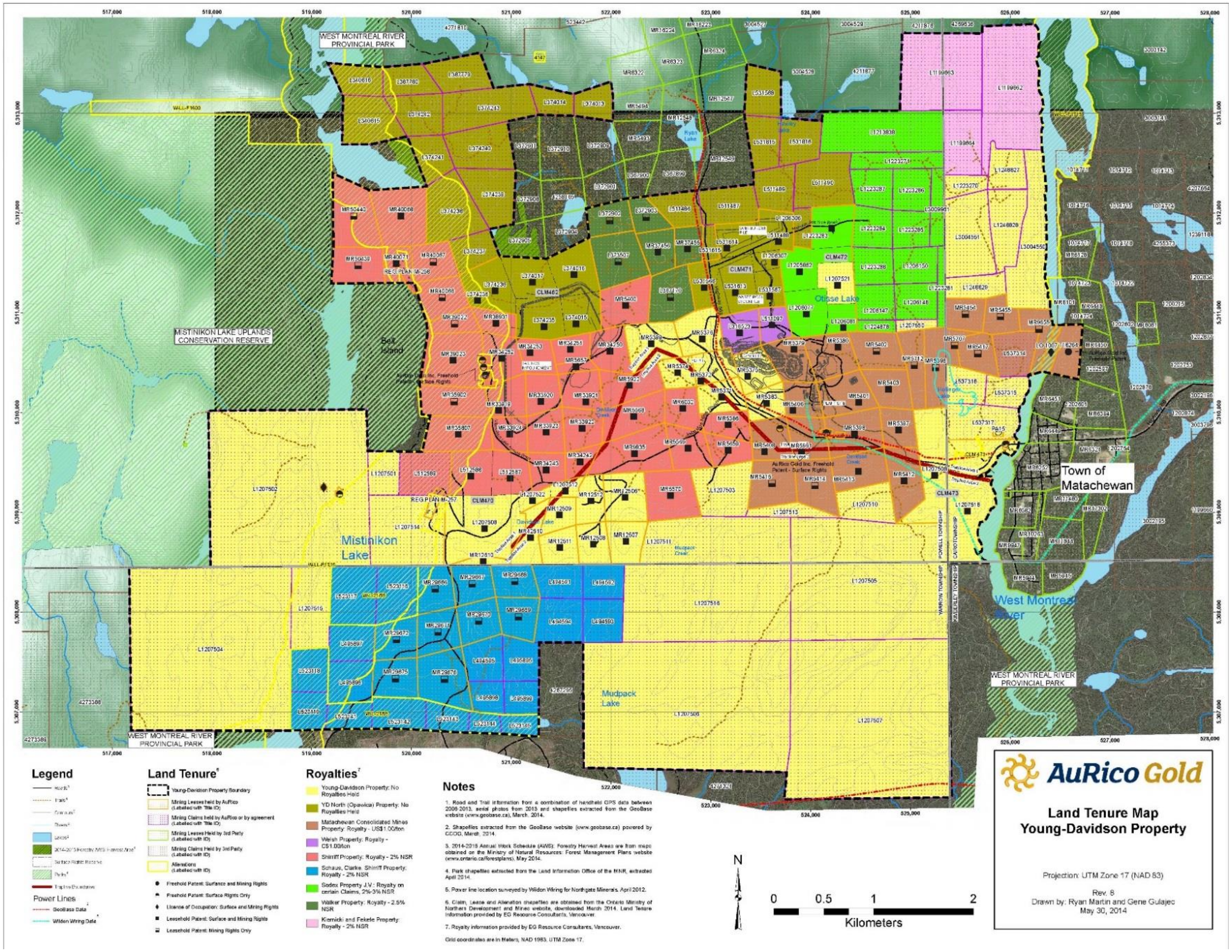


Figure 4-2: Land Tenure Map

4.3 Surface Rights

The Company controls sufficient surface rights to cover the sites required for all project buildings and fixed installations for the life of mine. The Company believes it has all of the necessary surface rights to dispose of waste rock and tailings on additional areas of the property. Alamos' land ownership and mineral tenures are registered with the Government of Ontario. All permits required to operate the mine are currently in place.

4.4 Royalties and Encumbrances

Collectively, Young-Davidson is subject to nine separate agreements with different obligations and royalties for each agreement. Based on the currently defined Mineral Reserves and Mineral Resources, the only royalties to apply are:

- a sliding scale royalty held by Matachewan Consolidated Mines Limited that relates to the eastern portion of the open pit and a small portion of the underground Mineral Resource, which together total approximately 1,000,000 tonnes;
- a per ton royalty held by the Welsh Estate that affects almost 424,000 tonnes; and
- a 1.5% net smelter return due to AuRico Metals Inc., applicable since July 2015.

4.5 Property Agreements

The Company has worked to establish and maintain cooperative relationships with the Matachewan First Nation (MFN), and the Temagami First Nation (TFN). Impact Benefit Agreements (IBA) have been developed with both the MFN and TFN to ensure that the communities derive positive benefit from Young-Davidson Mine operations as it is in part of their traditional territories.

The responsible QP has reviewed the agreements and there are no terms in the agreement that would have a negative impact on the mine. The current and future payments are incorporated into the mine's financial model.

4.6 Permits, Environment and Social Licence

The current status of the environment permitting, community consultation and the social license to operate is discussed in Section 20.

Other than as described above, the responsible QP is not aware of any rights, agreements or encumbrances to which Young-Davidson is subject, which would adversely affect the value of the property or Alamos' ownership.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The mine site is 3 km west of the community of Matachewan in northern Ontario. Kirkland Lake is 60 km east of the project site and Timmins, which is the regional centre, is located 170 km by road northwest of the site (Figure 4-1). There are a number of other communities also within daily commute distance to the Young-Davidson Project. The MFN Reserve, located approximately 12 km north of the site, is the closest First Nation community to the site.

The site is connected by means of paved Highway 566 to the northern Ontario highway network. Relocation of a section of Highway 566 was completed to support mine development as a portion of the existing highway intersected the proposed open pit.

Commercial air services are offered in Timmins, and charter aircraft can be flown into the Kirkland Lake Municipal Airport (1,373 metre paved airstrip).

5.2 Climate

The Matachewan area experiences cold winters and warm summers, typical of most of northern Ontario. The nearest climatic station to the site is Kirkland Lake, located approximately 55 km to the northeast, and serves as the reference station for the mine site. The climate in Kirkland Lake can be summarized as follows, based on climate normal data (1950 to 1995):

- Average annual daily temperature is 1.7°C. Monthly average daily temperatures range from -16.9°C in January to 17.6°C in July.
- Precipitation averages 790 mm/yr. The highest average monthly precipitation generally occurs in September with 95.8 mm. The lowest average precipitation falls in February with 47.1 mm.
- The maximum average monthly snowfall is 123.2 cm falling during the month of January (Environment Canada 2007).

Information on wind speed and direction is not available for the Kirkland Lake station. The next closest weather station to the project site is in Timmins approximately 70 km to the northwest. Winds in Timmins are gentle to moderate, averaging 11.9 km/h annually. Monthly averages range from 9.8 km/h to 13.5 km/h, with calmer conditions prevailing during the summer months.

Mining and exploration activities are conducted year-round.

5.3 Local Resources and Infrastructure

Power supply for the Young-Davidson site is 115 kV from a short tap connecting to an existing power transmission line from Hydro One’s Kirkland Lake transformer station approximately 54 km from the site. Sufficient capacity exists on the line for current and future site requirements.

Matachewan is located near Kirkland Lake and Timmins, two mining towns with a significant pool of trained personnel that allow the Company to easily recruit and retain employees.

5.4 Physiography and Vegetation

The Project site is located on a raised plateau within an area of rugged hummocky terrain and significant topographic relief. Elevation changes in the range of 20 m to 50 m are common. Elevation at the project site is approximately 350 metres above sea level (masl) while the West Montreal River is 300 masl.

The local area has been defined as bedrock knobs with subdominant areas of ground moraine or till. Shallow soils are present, within limited organic topsoil. Sandy loam soil characterizes the area surrounding the project area. The low areas between the bedrock knobs often contain water bodies or marsh areas.

Undisturbed portions of the site are covered by a conifer-dominated boreal forest. Dominant species within the local forest stands include: balsam fir, black spruce, white birch with some white spruce and trembling aspen. This vegetation is typical of the region and representative of the local climate and subsurface conditions.

6 HISTORY

Historical records show that the local area was first staked by two prospectors in 1910: Jake Davidson identified the Young-Davidson prospect and Sam Otisse the Matachewan Consolidated prospect. This eventually led to gold production between 1934 and 1956 from two significant gold mines located in close proximity: the Young-Davidson Mine and the Matachewan Consolidated Mine. The mines together produced approximately 970,000 oz of gold from 9.8 Mt of ore (SWRPA 2007).

There was no mining on the property from 1956 until Pamour Porcupine Mines Limited operated a short-lived, small, open pit mine to extract near surface mineralization between 1981 and 1982. All ore was trucked offsite and there was no processing on site or tailings resulting from this operation. Since that time there were a series of exploration programs conducted by Pamour Porcupine Mines Limited, and its successors Pamour Inc. and Royal Oak Mines between 1982 and the mid-1990's but no further mining on the property.

Royal Oak Mines Inc. (RYO) continued its involvement in the property and completed a number of engineering studies and environmental baseline investigations between 1995 and 1997. It is Alamos understanding that RYO withdrew from the environmental approvals process for corporate reasons prior to its completion. The property remained inactive for a number of years after this withdrawal.

Northgate became interested in the Matachewan area for its exploration potential and in November 2005 Young-Davidson Mines Limited became a wholly-owned subsidiary of Northgate. Northgate owned 100% of mineral rights to all of the Young-Davidson Mine and Matachewan Consolidated Mine claims.

In January, 2007, Northgate, via contractors, established a portal and proceeded to develop an exploration ramp down to the orebody, while simultaneously dewatering the existing underground workings.

On the basis of a positive feasibility study Northgate began construction of the project in August, 2010.

On October 26, 2011, AuRico Gold Inc. acquired all of the issued and outstanding common shares of Northgate.

Mining in the open pit commenced in November, 2011, with the use of a mining contractor.

Commissioning of the mill commenced in the first quarter of 2012, with the first gold pour occurring on April 30th, 2012. Commercial production for the open pit and mill was declared as of September 1, 2012.

Production of underground stoping ore began in October 2012, and commercial production for the underground was declared on October 31, 2013, with the commissioning of the Northgate Production Shaft.

On July 2, 2015, Alamos Gold Inc. and AuRico amalgamated to form Alamos Gold Inc. (Alamos).

6.1 Historical Production and Exploration

Almost one million ounces of gold has been produced historically (pre 2012) from the Young-Davidson property (Table 6-1).

Table 6-1: Historical Production at the Young-Davidson Property

Period	Mine	Tons	Recovered Grade (oz/ton)	Produced Ounces Au
1934 to 1957	YD	6,218,272	0.094	585,690
1934 to 1954	MCM	3,525,200	0.107	378,101
1981 to 1982	MCM	106,000	0.069	7,314
Total		**9,849,472**	**0.099**	**971,105**

A chronological listing of the historical exploration and development work undertaken on the main claim groups, prior to the Company's involvement, is summarized below.

6.1.1 Young-Davidson Property

A significant amount of work has been done on the Young-Davidson property over the years, beginning with the initial gold discovery on the Davidson Claims by Mr. Jake Davidson in 1916.

Since then, much exploration and development has been done on the property, including the excavation of two shafts, six production levels, and one exploration level. Production of gold from this property took place mainly from 1934 to 1957. A brief chronological summary of work done is detailed below:

- 1916-1933: Young-Davidson Mines Ltd.: surface prospecting, hand trenching and stripping. Pre-production activities.
- 1934-1957: Hollinger Corporation: production of gold under contract from Young- Davidson Mines. According to records maintained by the Ontario Geological Survey, a total of 6,218,272 tons were produced at an average recovered grade of 0.094 oz/ton (3.22 g/tonne) gold, producing 585,690 ounces of gold.
- 1979: Pamour: concluded Option Agreement.
- 1980: Pamour: Diamond drilling, 44 holes totaling 12,841 feet (ft).
- 1986: Pamour: additional diamond drilling in Boundary Pit Area, 84 holes totaling 32,130 ft.
- 1988: Pamour: diamond drilling, 10 holes totaling 3,882 ft, testing shallow targets and upper portions of the Boundary Zone.

- 1989: Pamour: diamond drilling, 10 holes totaling 13,052 ft, testing Boundary Zone (underground target).
- 1990: Pamour: diamond drilling, 31 holes, totaling 68,391 ft., testing several targets in syenite.
- 1995: Royal Oak Mines Inc.: diamond drilling, 68 holes totaling 71,102 ft that tested selected underground targets, and testing for the western and depth extensions of the open pit mineralization
- 1996: Royal Oak Mines Inc.: diamond drilling, 23 holes totaling 11,832 ft, confirming the western and depth extensions of the open pit mineralization, and to confirm the assay values indicated by the 1980 vintage drill holes.
- 1997: Royal Oak Mines Inc.: diamond drilling, 9 holes totaling 3,383 ft.
- 2003: Young-Davidson Mines, Limited: diamond drilling, 10 holes totaling 5,407 ft.

6.1.2 Matachewan Consolidated Mines Property

The initial gold discovery on the adjoining Davidson Claims in 1916 sparked a staking rush which resulted in the discovery of gold on the MCM property later that year. Since then, much exploration and development has been done on the property, including excavation of three shafts and eleven production levels. Production of gold from this property took place mainly from 1934 to 1954. A brief chronological summary is detailed below:

- 1916: Discovery of gold on the mine property by Samuel Otisse.
- 1916-1933: Ventures Limited: surface prospecting, hand trenching and stripping. Pre-production activities.
- 1934-1954: Ventures Limited: production period. According to records maintained by the Ontario Geological Survey, a total of 3,525,200 tons were produced at an average recovered grade of 0.107 oz/ton (3.67 g/tonne) gold, producing 378,101 ounces of gold.
- 1980: Pamour: concluded Option Agreement.
- 1981-1982: Pamour: gold production from five small open pits, whole ore trucked back to Pamour mill in Timmins for processing. Internal company documents indicate that 106,000 tons at an average grade of 0.069 oz/ton Au were produced (Skeeles, 1989).
- 1995: Royal Oak Mines Inc.: diamond drilling, 34 holes totaling 22,984 ft testing for the eastern extensions of the open pit mineralization.
- 1996: Royal Oak Mines Inc.: diamond drilling, 6 holes totaling 2,717 ft testing for the eastern extensions of the open pit mineralization. Commencement of an underground exploration program in the fall of 1996 and proceeding through to the fall of 1997. Work completed included dewatering to the 9th Level loading pocket, shaft rehabilitation to the 8th Level station, and geological mapping and sampling to the 5th Level. Preparation of necessary documentation to produce a Comprehensive Study Report and Environmental Impact Statement in fulfillment of the requirements outlined in the Canadian Environmental Assessment Act.

- 1997: Royal Oak Mines Inc.: trenching program, including geological mapping and channel sampling. Reconnaissance level Induced Polarization (IP) survey.
- 2000: 1519864 Ontario, Limited: diamond drilling, 11 holes totaling 983 ft to test for the presence of gold mineralization in the crown pillars of selected stopes. Limited stripping and trenching.
- 2002: Young-Davidson Mines, Limited: Line cutting, IP and magnetometer surveys, soil sampling, trenching, and geological mapping of the area located from the MCM #3 shaft eastwards to the Montreal River. Diamond drilling, 21 holes totaling 12,793 ft to test selected IP anomalies or geological targets for the potential of hosting significant gold values.
- 2003: Young-Davidson Mines, Limited: Diamond drilling, 37 holes totaling 17,759 ft.

6.1.3 Welsh Claims

- 1916: Discovery of gold on adjoining claims.
- 1979: Pamour: concluded Option Agreement.
- 1986: Pamour: diamond drilling to test for the western extension of the Boundary Pit mineralization
- 1995: Royal Oak Mines Inc.: diamond drilling, total 4,982 ft in 18 holes.
- 1996: Royal Oak Mines Inc.: diamond drilling, total 1,375 ft in 7 holes.
- 1997: Royal Oak Mines Inc.: diamond drilling, total 2,812 ft in 6 holes.

6.1.4 Shirriff Claims

- 1936: Matachewan Consolidated Mines: 3 drill holes completed on the Shirriff claims just south of the Young-Davidson Mine.
- 1949: British Matachewan Gold Mines Ltd.: Diamond drilling, 1 hole (1,001 ft).
- 1960-1966: British Matachewan Gold Mines Ltd.: Diamond drilling, 7 holes totaling 3,858 ft.
- 1971: British Matachewan Gold Mines Ltd.: Magnetic and electro-magnetic (EM) surveys in the Mistinikon Lake area. Diamond drilling, 2 holes.
- 1973: British Matachewan Gold Mines Ltd.: IP survey.
- 1990: Pamour: Line cutting, magnetic and CSAMT surveys, geological mapping, diamond drilling (4,335 ft in one complete hole, 1 partial hole, crossing claim boundary (YD90-21)).
- 1994: Royal Oak Mines Inc.: Minor line cutting, diamond drilling (9,568 ft in 3 holes).
- 1995: Royal Oak Mines Inc.: Minor line cutting, diamond drilling (4,801 ft in 3 holes).

- 2003: Young-Davidson Mines, Limited: Diamond drilling, one hole totaling 2,579.7 ft to search for the western depth projection of the Young-Davidson deposit.

6.1.5 Schaus-Clarke-Shirriff Claims

The Ministry of Northern Development and Mines assessment files have no records of exploration activities conducted on the Schaus-Clarke-Shirriff claims.

6.2 Previous Mineral Resource Estimates

Details from a previous Micon Mineral Resource estimate are disclosed in a NI 43-101 technical report dated August 2004.

In December 2006, Scott Wilson, Roscoe, Postle and Associates (SWRPA) estimated the underground Mineral Resources for the Lower Boundary Zone, the Lucky Zone, and the Lower YD Zone.

In December 2008, Northgate Minerals Corporation estimated the open pit and underground Mineral Resources as disclosed in a NI 43-101 technical report dated January 23, 2009.

Since 2011 AuRico, and then later Alamos, have continuously disclosed updated Mineral Reserves and Mineral Resources in their Annual Information Forms.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Young-Davidson Property is located within the southwestern part of the Abitibi Greenstone Belt, which is the largest preserved Archean greenstone belt in the world and one of the most continuous units of the Superior Geologic Province. The Abitibi Greenstone Belt extends for 750 km from the Grenville Province in the east to the Kapuskasing Gneiss Belt in the west, and for over 170 km from the Opatica Gneissic belt in the north to the Proterozoic Huronian sediments in the south (Figure 7-1).

The Abitibi Greenstone Belt consists of a complex and diverse array of volcanic, sedimentary, and plutonic rocks typically metamorphosed to greenschist facies grade, but locally attaining amphibolite facies grade adjacent to large plutons and along the Grenville Front.

Volcanic rocks range in composition from rhyolitic to komatiitic, and commonly occur as mafic to felsic volcanic cycles. Sedimentary rocks consist of both chemical and clastic varieties, and occur as both intravolcanic sequences and as uncomformably overlying sequences. A wide spectrum of mafic to felsic, pre-tectonic, syn-tectonic and post-tectonic intrusive rocks is present. All lithologies are cut by late, generally northeast-trending Proterozoic diabase dykes.

The Abitibi Greenstone Belt rocks have undergone a complex sequence of deformation events ranging from early folding and faulting through later upright folding, faulting and ductile shearing, resulting in the development of large, dominantly east- west trending, crustal-scale structures (“breaks”) that form a lozenge-like pattern.

The regional Larder Lake-Cadillac Fault Zone (LLCFZ) cuts across the Young-Davidson Property. The LLCFZ has a sub-vertical dip, and generally strikes east-west. The LLCFZ is characterized by chlorite-talc-carbonate schist, and the deformation zone can be followed for over 200 km from west of Kirkland Lake to Val d’Or.

7.2 Local and Property Geology

In general, the Archean volcanic rocks comprise two similar volcanic sequences characterized by basal komatiitic units, and overlain by magnesium and iron tholeiitic units, which finally grade into calc-alkaline volcanic rocks. The oldest rocks of these sequences are 2,747 million years (Ma), while the youngest are 2,700 Ma (Corfu et al., 1989); at Matachewan, they are likely correlative to the Larder Lake Group.

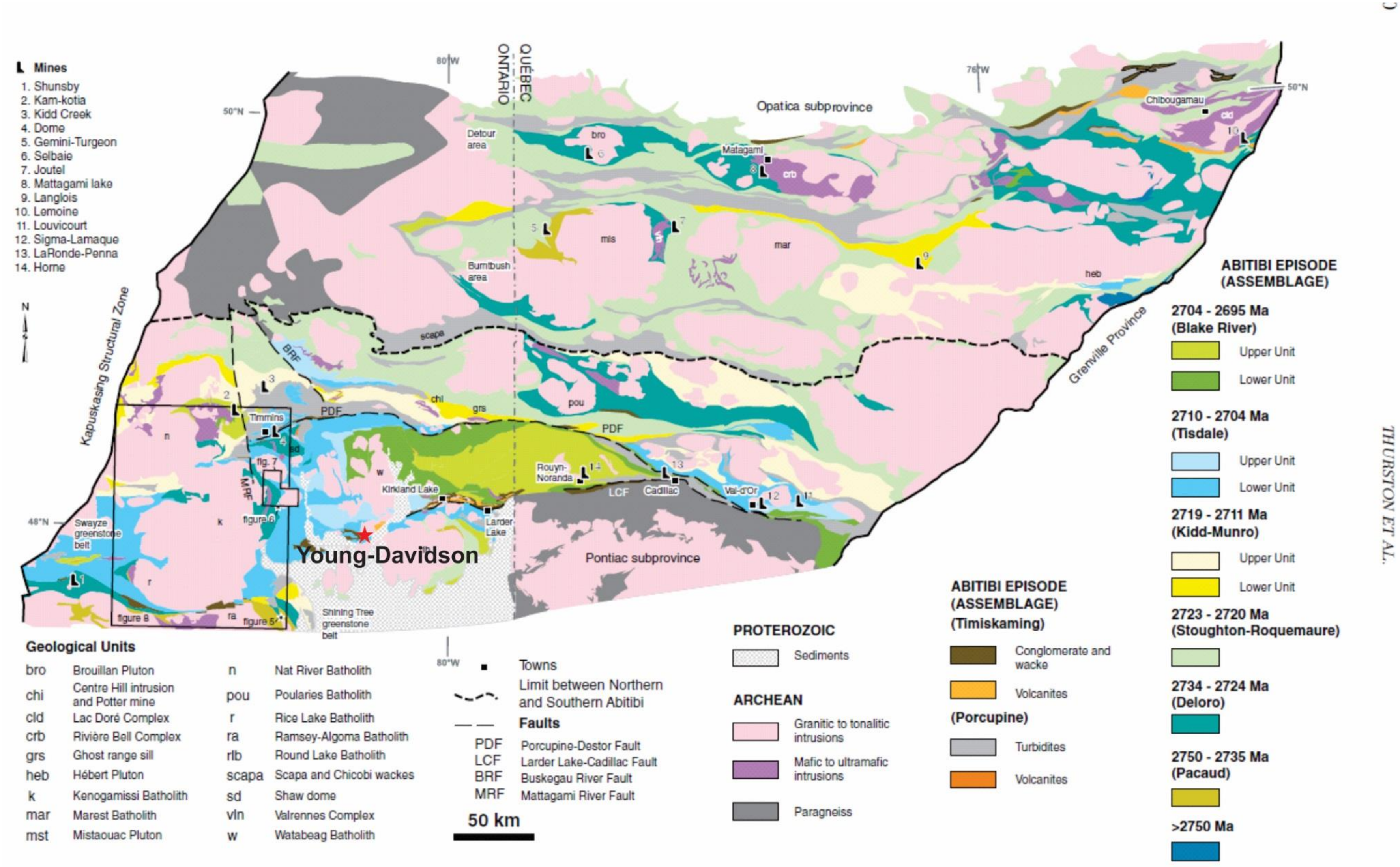


Figure 7-1: Regional Geology of the Young-Davidson Property

There are three important groups of Archean sedimentary rocks in the district. The oldest are the Pontiac Group quartz greywackes and argillites, which occur as thick assemblages in Québec, while interbedded within the Larder Lake Group volcanic rocks are turbiditic siltstones and greywackes of the Porcupine Group. Uncomformably overlying the former is the Timiskiming Group conglomerates, turbidites and iron formations, with minor interbedded alkalic volcaniclastic units.

Archean intrusive rocks are numerous in the district, but are largely manifested as small stocks, dykes and plugs of augite syenite, syenite and feldspar porphyry occurring in close temporal and spatial association with the distribution of Timiskiming Group sediments.

Huronian Proterozoic sedimentary rocks onlap and define the southern limit of the Abitibi in Ontario. In the project area, these rocks are correlative to the Gowganda Formation, which is a tillite yielding an age date of 2,288 ±87 Ma (Fairbairn et al., 1969).

Post-Archean dyke rocks include Matachewan Diabase (2,454 ±2 Ma; Heaman, 1988) and Nipissing Diabase (2,219 ±4 Ma) (Corfu and Andrews, 1986), which respectively bracket the Huronian unconformity in the project area.

The local and property geology and stratigraphy are shown in Figures 7-2 and 7-3, respectively.

7.3 Lithology

The lithologic assemblage exposed on the Young-Davidson Property is described below in order of super-position or from the oldest to youngest (Edmunds, 2007). The descriptions and interpretations are based on previous studies (Micon, 2004; Panterra Geoservices (PGI), 2003; Powell et al., 1991) complemented by direct core observations.

7.3.1 Larder Lake Group (Archean)

The Larder Lake Group is intermittently exposed in the central eastern portion of the property. It is composed of interbedded mafic to ultramafic flows of tholeiitic to komatiitic composition and interflow volcanic-derived sediments. The ultramafic rocks occur as dark grey-green, talc-chlorite+carbonate schists with all primary textures destroyed by metamorphism and deformation. In core, these units range from 15 m to 100 m in intersected thickness.

Tholeiitic basalts occur as finer grained massive featureless chlorite sericite schists, often grey-green in colour. Areas exposed at surface northeast of the #2 Shaft show well developed pillow textures indicating younging directions to the south. Some core holes have intersected narrow sections of mafic lapilli tuffs and related fragmental breccias. The proportion of ultramafics in the Larder Lake rocks varies across the property, and appears to increase towards the west.

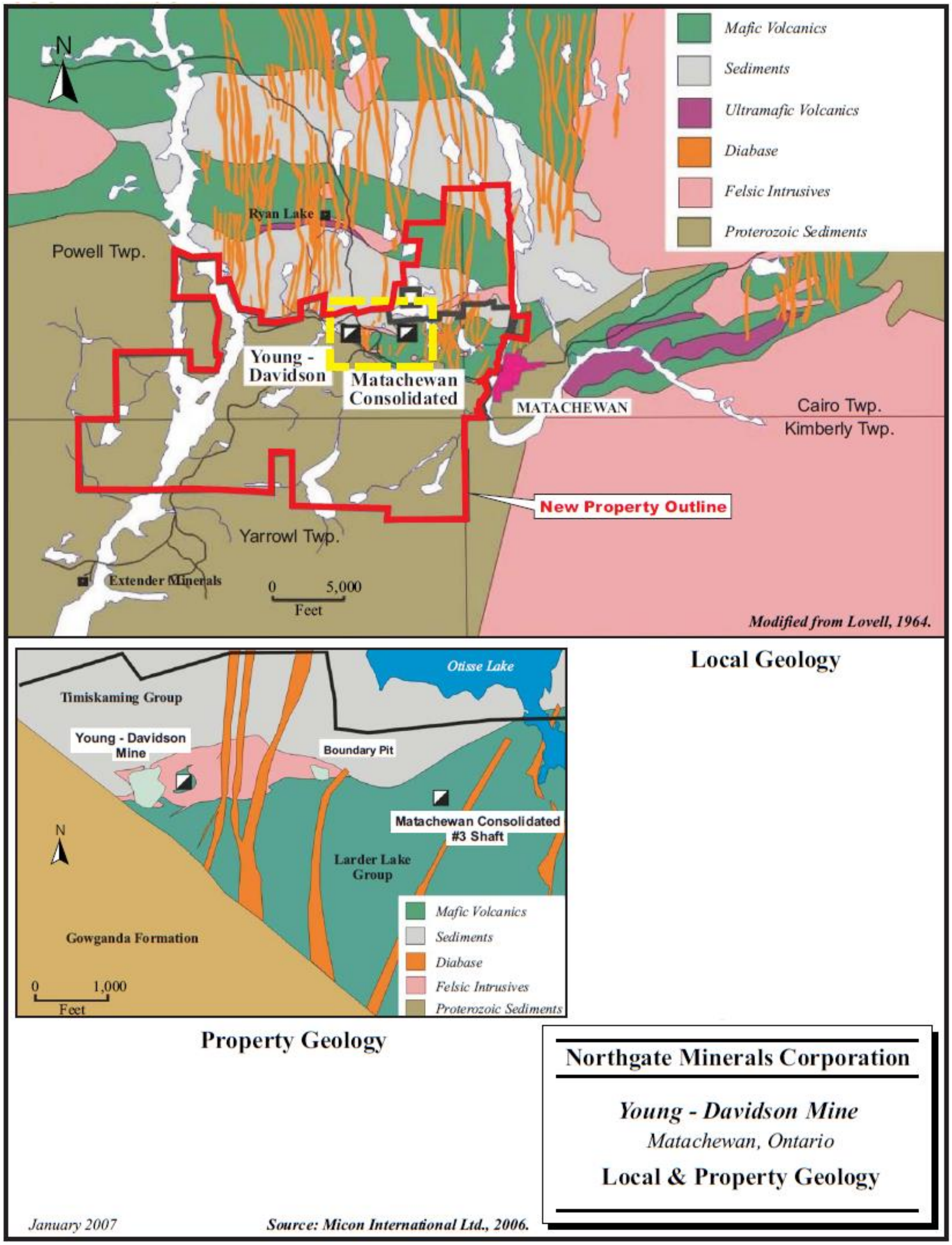


Figure 7-2: Local and Property Geology of the Young-Davidson Property

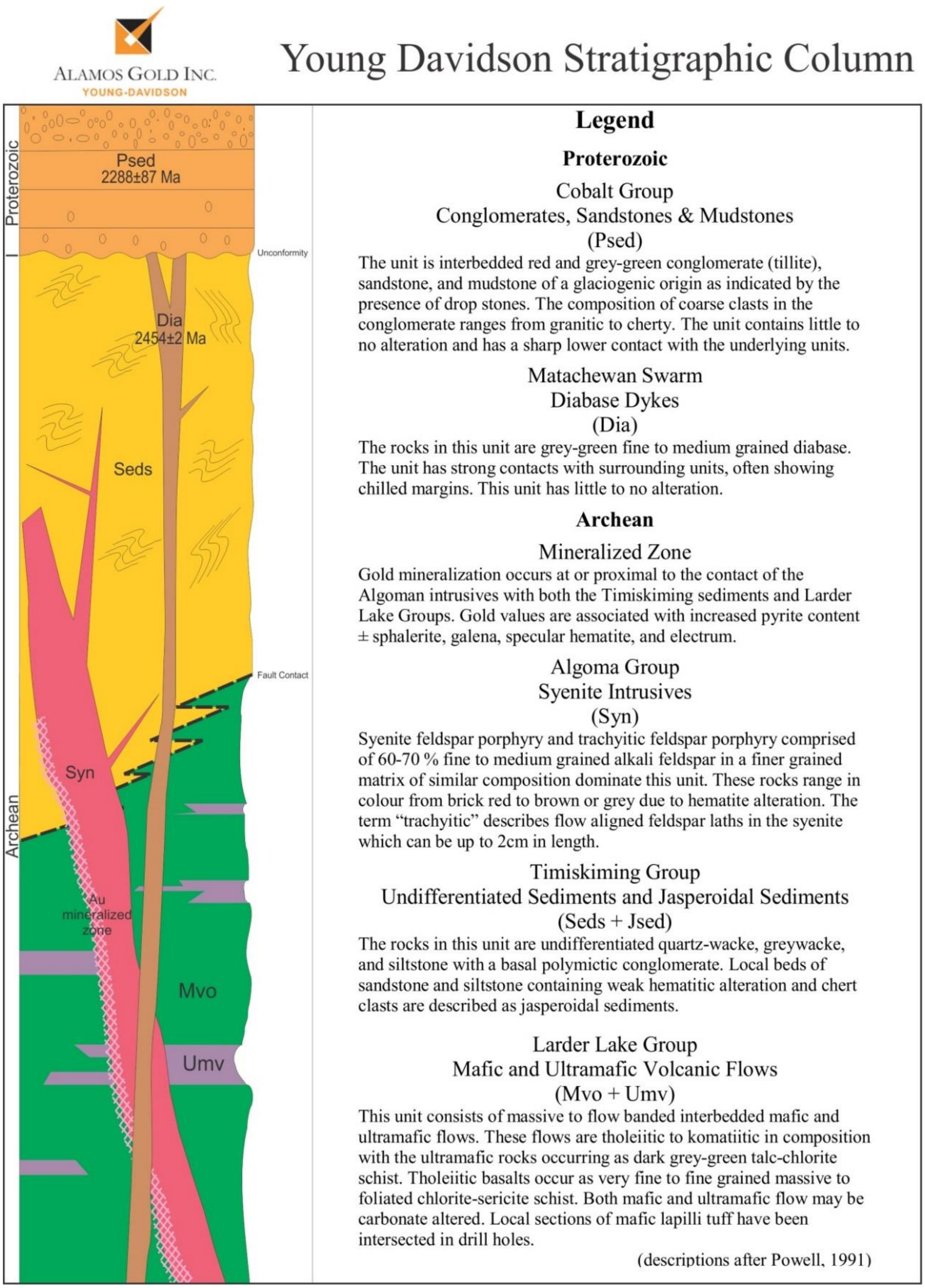


Figure 7-3: Local and Property Stratigraphy of the Young-Davidson Property

7.3.2 Timiskiming Group (Archean)

The Timiskiming Group dominates the northern reaches of the property and, at surface, is presented as a monotonous sequence of quartz-wackes, greywackes, and siltstones. There are rare surface exposures of the contact between Larder Lake Group and Timiskiming sediments, and these often show a basal polymictic conglomerate composed of clasts ranging from 1 centimetre (cm) to 3 cm diameter. This observation implies an opposing or northerly younging direction for the Timiskiming assemblage. Core logging of deeper drill holes has identified polymictic fragmental units within the Timiskiming, which have been described as jasperoidal sediments due to the presence of chert clasts. These fragmental units may correspond to the basal conglomerate.

7.3.3 Gowganda Formation (Proterozoic)

Gowganda Formation rocks are dominated by red to brown-grey interbedded conglomerates (tillites), mudstones, and sandstones, which can be massive to well- bedded in sections. The glaciogenic origin for these sediments is suggested by the presence of drop stones in otherwise massive graded units. The contact between the base of the Gowganda and the underlying rocks is always very sharp, but has only been observed in drill core. The Gowganda Formation covers the southern two thirds of the property.

7.3.4 Intrusive Rocks

The most abundant Algoman intrusive rocks are dominated by syenite feldspar porphyry and trachytic syenite porphyry. These units are often composed of 30% to 70% fine to coarse-grained alkali feldspar phenocrysts, set in a finer-grained compositionally similar matrix. Alteration affects the colour of the syenite units, and ranges from economically significant brick red to brown or grey. The “trachytic” term is used to describe flow-aligned feldspar laths in certain sections of syenite, which can range up to 3 cm in length.

The Matachewan area is well known for diabase dykes, and those that affect the area covered by the exploration work are Matachewan diabases, which occur as dark grey- green dykes with fine to medium grain size, often showing chill-bake margins and feldspar textures that vary from glomerophyric to massive.

7.4 Structural Geology

Recent structural studies completed by PGI (2003) have documented the main structural events on the property and are summarized below.

On a regional scale, some important large-scale features transect or are in close proximity to the property. The regionally extensive and economically significant LLCFZ has been interpreted to either pass through the property (MERQ, 1983), or breaks into three splays (Powell et al., 1991). The southernmost of these splays is interpreted by Powell el al. (1991) to cross the property on the contact between the Timiskiming and Larder Lake Groups. The presence of a polymictic basal conglomerate in the Timiskiming suggests that the extension of the break may be south of this unconformity.

The Larder Lake units trend east-west, dip steeply south at 70°, and are upright such that they young to the south. Timiskiming units trend in a similar manner; however, they are overturned and, although younger, form the structural footwall to the Larder Lake Group volcanic rocks. The unconformity trends east-west for much of the property; however, it swings to the north along steeply plunging fold axes east of the main mass of syenite.

The syenite occurs near the unconformity, and appears to transgress the Archean contact, such that it is much closer to the Larder Lake Group in the east as compared to the west, where it is fully enveloped by Timiskiming sediments near surface and at depth. The main mass measures almost 1,000 m east-west by 300 m north-south, and its western limits are poorly defined. Minor apophyses of syenite occur sporadically within 500 m of the unconformity, and some portions of the deformational history have affected it, as the dykes are folded and faulted.

The Gowganda Formation forms a progressively thickening southward dipping wedge that rapidly increases in thickness towards the southwest, suggesting significant pre- Gowganda topography.

PGI (2003) confirmed the previous interpreted structural history for the property. There are at least three Archean tectonic events, followed by Proterozoic and Palaeozoic deformation. The first Archean event involved isoclinal folding, uplift, and erosion of the Larder Lake Group volcanic rocks, such that a pre-Timiskiming downward-facing limb of an isocline underlies part of the property. No fabric has been associated with this event (D1).

The second Archean event is major in terms of establishing rock fabrics, and affects the previously deformed Larder Lake Group, Timiskiming Group, and Algoman intrusive rocks, as well as gold-bearing quartz vein structures and associated alteration (D2). Schistosity forms the main D2 fabric defined by the preferred orientation of phyllosilicates, and flattening of clasts in conglomerate units (Powell, 1991).

The third Archean structural event ranges from a weakly developed northeast to southwest crenulation cleavage to a complete transposition of earlier fabric elements (D3). Mineral and elongation lineation is a feature of the D3 event, and workers such as Derry (1948) and Powell et al. (1991) have noted that this direction parallels the trend of mineralization at the Young-Davidson Property. Both Powell et al. (1991) and PGI (2003) consider a fourth Archean deformation event that produced steeply plunging open folds associated with north-northeast to north-northwest striking axial planar cleavage. This event may be a macroscopic feature related to the later stages of the D3 crenulation event.

7.5 Metamorphism

The rocks in the townships surrounding Matachewan have undergone regional metamorphism ranging from prehnite-epidote facies in Bannockburn to greenschist facies (actinolite-epidote-chlorite for metabasalts) in Powell Township, closest to the Young-Davidson Project area (Powell, 1991). These assemblages correspond to a regional low- pressure-temperature event with temperatures up to 250°C, at burial depths in the order of 6.5 km. This event appears to be synchronous with the D2 structural event (Powell, 1991).

7.6 Alteration

The most obvious form of alteration in the volcanic units of the Larder Lake Group is extensive carbonate alteration, manifested by distal calcite and proximal iron carbonate adjacent to mineralization. Timiskiming sediments show a comparatively minor amount of carbonate alteration; however, they can become hematitic adjacent to mineralized portions of the syenite masses.

The predominant alteration empirically associated with syenite-hosted gold mineralization is quartz veining, microcline development, and sulphidation in the form of increased pyrite content with accessory chalcopyrite, sphalerite and galena. Semi- quantitative ICP analyses show that, through mineralized sections of syenite, iron contents remain relatively constant, while sulphur, pyrite and gold contents vary antithetically with magnetic susceptibility and reported ICP barium. This suggests that iron oxide as magnetite is being reduced during the mineralizing process, and the silicates are metasomatized capturing barium in an insoluble silicate such as feldspar. Thin section petrology may help understand some of these observations from the ICP analyses.

7.7 Mineralization

At least five styles of gold mineralization are recognized at the Young-Davidson Project. They are described in detail in Micon (2004) and PGI (2003).

1. Syenite-hosted gold mineralization
2. Mafic volcanic-hosted gold mineralization (MCM Mine)
3. Timiskaming sediment-hosted gold mineralization
4. Ultramafic-hosted gold mineralization
5. Hanging wall contact gold mineralization

Essentially all of the historical production at the Young-Davidson Mine and approximately 60% of the production from the MCM Mine is from syenite-hosted gold mineralization (Lovell, 1967). Most of the current underground Mineral Resources are also related to syenite-hosted gold.

The syenite-hosted gold mineralization consists of a stockwork of quartz veinlets and narrow quartz veins, rarely greater than a few inches in thickness, situated within a broader halo of disseminated pyrite and potassic alteration. Visible gold is common in the narrower, glassy-textured quartz veinlets. In general, gold grades increase with quartz veinlet abundance, pyrite abundance, and alteration intensity. Mineralized areas are visually distinctive and are characterized by brick red to pink K-feldspar-rich syenite containing two to three percent disseminated pyrite and several orientations of quartz extension veinlets and veins. The quartz veins and veinlets commonly contain accessory carbonate, pyrite, and feldspar.

At least two orientations for the quartz veins and veinlets are recognized. Most dip gently to the north and are ladder-type flat veins and some dip steeply to the north. The flat veinlets are common in the large outcrop exposures and small pits on surface. Previous workers noted that the syenite-hosted gold mineralization is generally more extensive and lower grade near surface and appears to be more channeled and concentrated into higher grade corridors at depth (SWRPA, 2007).

In 2008, as part of a Northgate – NSERC funded research study, the ore cross cut driven at 23,600mE on the 10,120 elevation in the UBZ was mapped and sampled at a very detailed scale. The mapping showed that there are two stages of mineralization: an early stage manifest as boudinaged quartz-albite veins, folded quartz-pyrite veins with accessory disseminated sulphide, and a later stage comprised of planar gently north-dipping quartz-carbonate veins with sulphides that are much less deformed.

Ore shoots within the syenite may plunge moderately to the southwest parallel to the L3 lineation (PGI, 2003), however, a strong vertical down dip attenuation and plunge direction is indicated by the historical open pits and underground stopes. The current mineralization wireframes suggest that the mineralization may plunge steeply to the south-southwest or rake subvertically moderately to the west. Actual plunge direction(s) for the underground syenite-hosted gold mineralization will become more evident as new data become available.

8 DEPOSIT TYPES

The syenite-hosted gold deposits, commonly associated with quartz-monzonite to syenite stocks and dikes, has been individualized as a distinct group of gold deposits, well represented in the Abitibi Greenstone belt and particularly at the Porcupine and Kirkland Lake districts, Northern Ontario.

According to Robert (2004), the syenite-hosted gold deposits occur mainly along major fault zones, in association with preserved alluvial-fluvial, Timiskaming-type, sedimentary rocks. Robert (2004) describes the gold mineralization in these deposits as being represented by disseminated sulphide replacement zones, with variably developed stockworks of quartz-carbonate-K-feldspar veinlets within zones of carbonate, albite, K-feldspar, and sericite alteration. Syenitic intrusions are broadly contemporaneous with deposition of Timiskaming sedimentary rocks and, together with disseminated gold mineralization; they have been overprinted by subsequent regional folding and related penetrative cleavage.

The disseminated gold mineralization occurs within composite syenitic stocks or along their margins, along satellite dikes and sills, and along faults and lithologic contacts away from intrusions. Robert (2004) has interpreted the mineralized bodies as proximal to distal components of large magmatic-hydrothermal systems centered on, and possibly genetically related to, the composite syenitic stocks.

The Young-Davidson deposit, also located in the Abitibi greenstone belt, can be classified as an Archean, syenite-hosted gold deposit. The gold mineralization is primarily related to quartz veinlet stockworks and disseminated pyrite mineralization, mostly enclosed within the syenite intrusion boundaries, or very close to the contacts with the enclosing rocks, and is frequently associated with broader zones of potassic alteration.

Recent findings in similar environments of Northern Ontario and Québec have been reported at Upper Beaver, Abitibi West, Hislop, and Cadillac Break, etc., in the Kirkland Lake-Timmins area.

This type of mineralization is also present in the Yilgarn block (Kalgoorlie, Western Australia), where the Jupiter deposit shares some of its characteristics. Similarly, the Wallabi gold deposit bears similarities with the Young-Davidson deposit, in particular the Archean age, and the association of gold mineralization with hematite alteration in the proximities of a syenitic intrusion. However, in the Yilgarn block the gold mineralization is related to a broader range of granitoid host rocks.

9 EXPLORATION

9.1 Field Mapping

In 2007, a field mapping program was undertaken on the Young-Davidson property. Mapping was completed at a 1:10,000 scale geologic base map covering all Northgate owned ground in the Matachewan area (35 square kilometers). The goal of this program was to map geology, structure, veining and mineral occurrences for the entire property. Field mapping was done with the use of ATVs, trucks and on foot.

The geological map of the Young-Davidson property was updated in 2013. The newly acquired Opawika Claims to the north of the property had not been mapped prior to this and the mapping was carried out over part of the summer 2013 field season. Much of the area was covered with thick bush with very few outcrop showings. The mapping was carried out by following existing roads and trails.

Figure 9-1 shows the 2015 version of the geological map of the Young-Davidson property mapped at 1:10,000.

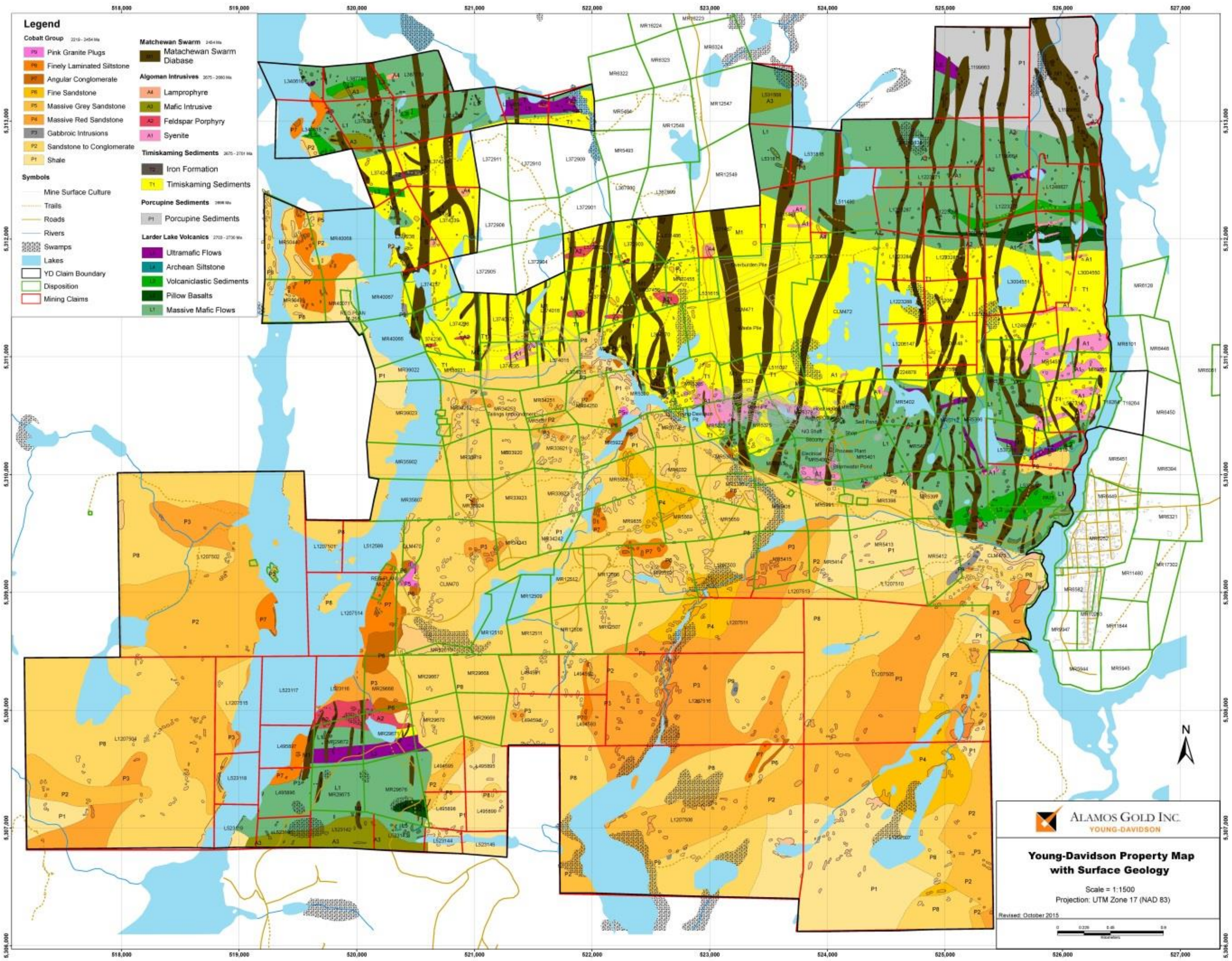


Figure 9-1: 2015 Young-Davidson Geology Base Map

9.2 Surface Sampling

A total of 41 surface samples were collected during the 2007 mapping program. Eight of these samples were chip samples and 33 were grab samples. Samples were tagged in the field at the collection location and subsequently catalogued. Samples were submitted to ALS Chemex's Timmins, Ontario laboratory for analytical work.

A total of 72 surface samples were taken during the 2012-2013 mapping programs. These samples include those taken in the southwest portion of the Young-Davidson property (Schaus, Clarke and Shirriff Claims) and the North Opawika Claims. Surface Sampling was done by foot, trucks and ATVs.

For the 2014 summer field mapping program, a total of 19 surface samples were sent for assaying, however, numerous samples were taken during the program. The samples were taken from the North Opawika Claims. Surface Sampling was carried out by foot, trucks and ATVs. The samples did not return any gold assays of any significance. The highest assay was 0.087 Au g/t.

Surface samples taken from the 2012 to 2014 mapping programs did not target gold showings but instead, various alteration types, structural features such as foliations and deformations in a variety of rock types as well as gold associated minerals and indicator minerals, such as pyrite and major and trace elements. Au g/t values in the assays returned were, therefore, not high, as expected.

9.3 Linecutting

Linecutting on the Young-Davidson Property has been carried out from 2007-2013 in preparation for various geophysical surveys completed. Lines were cut using axes and chainsaws and marked at 25 m intervals with pickets and metal tags.

In 2007, linecutting was undertaken by Northgate employees for a Quantec Titan-24 Survey. A total of ~13.5 kms were cut.

In 2008, linecutting was done by Katrine Exploration and Development Inc. in preparation for an IP Survey and Gradient Array by Larder Geophysics. Approximately 130 km of lines were cut. Line spacing was 100 m and pickets were located every 25 m. Metal tags were used every 50 m on lines and every 25 m on base and tie lines.

In 2010, line cutting was undertaken for a Gravity Survey by Allan Spector and Associates and Gradient array and IP surveys by Insight Geophysics. Linecutting was done by Bob Craig of Rouyn, Quebec. Approximately 27 kilometers of new lines were cut in preparation for the Gravity Survey.

In 2013, line cutting was carried out by All Terrain Exploration. 81.55 kilometers of lines were cut from January to May for an Insight I.P. Geophysical Survey. Cut lines were 3 feet wide and pickets were placed every 25 meters. Pickets were a minimum of 4 feet in height and spray painted 4 inches from the top with orange spray paint.

9.4 Geophysics

9.4.1 Aerial Photography Survey

In October 2006 and October 2012 Eagle Mapping was contracted to provide 1:1,000 scale photos over the mine site area and 1:5,000 scale photos over a larger area outside the 1:1,000 scale area. There was also a Satellite image created covering the Matachewan First Nations Reserve.

The first aerial survey was flown in October 2006 using colour film. It was flown by Champlain Aerial Surveys out of Pembroke, Ontario. There were 24 frames of 1:10,000 scale photo and 26 frames of 1:20,000 scale photo. The survey control was provided by staff at Young Davidson Mines Ltd. The Map Projection on this project was UTM NAD 83, zone 17.

The film was scanned at 16 microns on a Werhli scanner. It was then Aerial Triangulated using Z/I Image Station Aerial Triangulation (ISAT) software for mensuration/measuring, followed by Pat B Adjustment software. The compilation was performed on Leica and Socet Set softcopy workstations.

The “Mapfinishing/Editing” was done using Terra Model software for the contour generation, and Autocad software and in-house programs along with many hours of manual edit work.

The Orthophoto was generated at a pixel size of 0.2 meters from the 1:10000 photo, making up a total of 30 sheets at 1:1000. Another Orthophoto was generated at a pixel size of 1 meter from the 1:20,000 photo, making up a total of 6 sheets at 1:5000.

The final survey photograph is provided in Figure 9-2.

Little information is available for the survey done in October 2012 other than a 4 Band Ortho Rectified Satellite Image was taken over the Young-Davidson Property. The final survey photograph is shown in Figure 9-3.

Figure 9-2: 2007 Aerial Survey Photograph

Figure 9-3: 2012 Aerial Survey Photograph

9.4.2 Quantec Titan-24 Survey

Quantec's Titan Survey (Titan-24 Distributed Acquisition System Tensor MT and DCIP surveys) was completed over the Young-Davidson property in 2007. The Titan surveyed lines on the Young-Davidson property are provided in Figure 9-4. The survey objectives were: 1) Blind test to define DC, IP and MT signatures relating to the known Au-bearing, pyritic Syenitic Intrusive associated with the Young-Davidson Mine and Kirkland Lake – Larder Lake Belt Horizon and 2) To extend this knowledge to detect similar targets at greater depth and elsewhere at Young-Davidson.

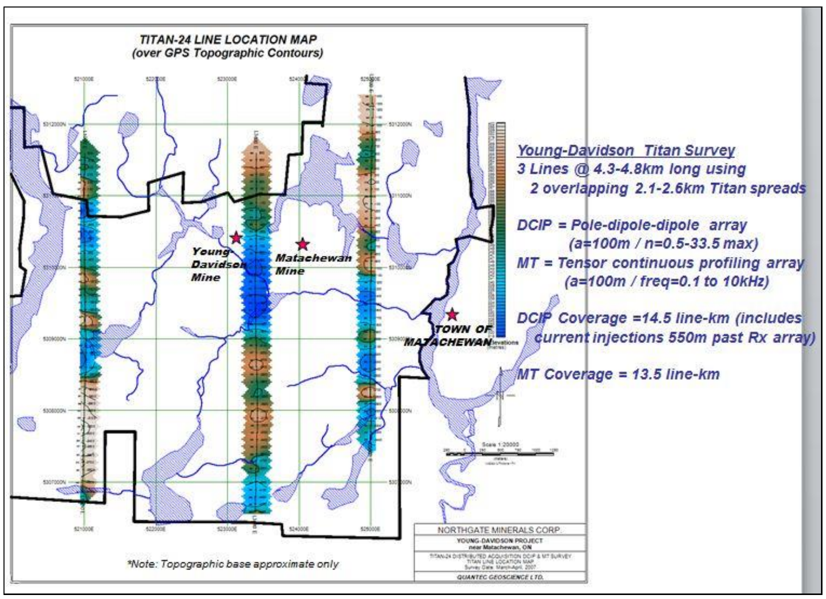


Figure 9-4: Titan-24 Line Location Map over the Young-Davidson Property

An example Quantec Geoscience pseudo section is shown in Figure 9-5.

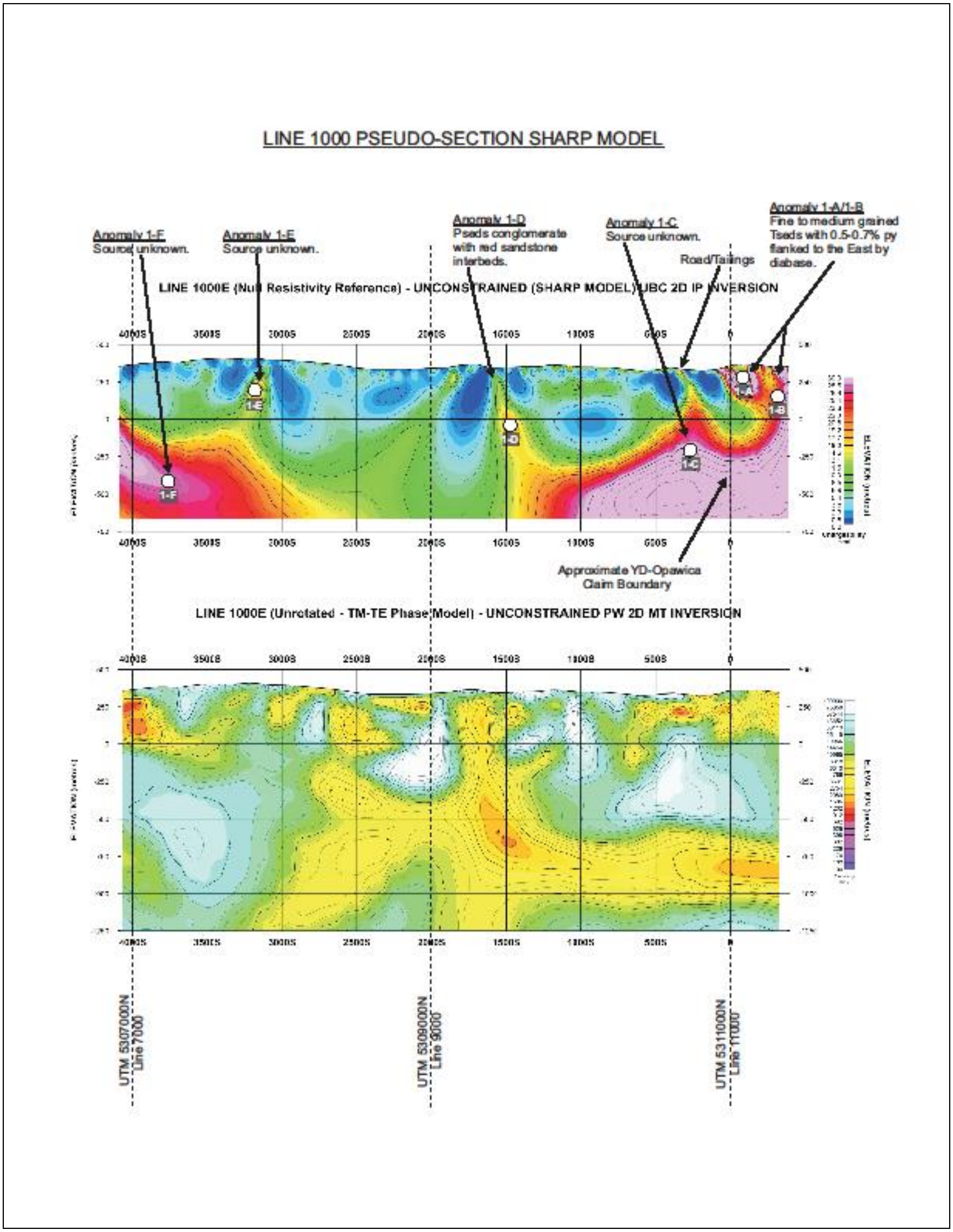


Figure 9-5: Line 1000 Pseudo-Section

9.4.3 IP Surveys

Three Induced Polarization (I.P.) surveys were completed on the Young-Davidson Property in 2008, 2010, and, 2013. In 2008, Larder Geophysics conducted an IP Survey. In 2010, Insight Geophysics completed a Tuned Gradient/ DCIP Survey and in 2013, Insight Geophysics, again, performed an IP Survey in the South West (SW) portion of the Young Davidson property.

In 2008, an IP Survey was done by Larder Geophysics Ltd. A 10 channel Elrec Pro receiver was employed for this survey. The transmitter consisted of a VIP 3000 (3kW) with a Honda 5000 as a power plant. A total of 57.65 line kilometers of gradient IP over 5 AB blocks was performed.

The grid consisted of approximately 130 line kilometers of cut grid lines. The grid lines were spaced at 100 meter intervals with the stations picketed at 25 m intervals with the northern grid baseline running at 0°N for a distance of 2.2 km.

9.4.4 Airborne and Aeromagnetic Survey

In 2010, Terraquest Ltd. conducted an airborne survey at the request of Allan Spector and Associates, in conjunction with requests from Jim Janzen. The airborne surveyed area is 36 square km in size.

On October 5, 2010 a 435km airborne high sensitivity, horizontal gradient magnetic and XDS VLF-EM survey was performed over the Young-Davidson property, with a 74 metre mean terrain clearance, 100 metre line intervals, 1000 metre tie line interval, and with data sample points at approximately 6 metres along the flight lines. The base of operations was at Kirkland Lake airport. A high sensitivity magnetic and a GPS base station located at the airport recorded the diurnal magnetic activity and reference GPS time during the survey for adherence to survey tolerances.

The survey consisted of 60 E-W lines at 100 m spacing and N-S lines at 1000 m spacing. Flight altitude was nominally 50 m. Magnetic intensity data was collected by 3 Cesium vapour magnetometers in the tail and wing tips of a Cessna 206 aircraft. Electromagnetic data was also collected using Terraquest's XDS system. The EM data was found to be of minimal geological value because of the response to variations in conductivity of Huronian sediments and the minimal depth of exploration of this high frequency EM data.

1:5,000 profiles of the magnetic intensity, vertical gradient and altimeter data were plotted and analyzed by exploration staff. The analysis results were compiled using 1:5000 scale contour maps of the magnetic data. The results were correlated with features displayed in a geological map of the property compiled by Katie Lucas in 2007 and enlarged to 1:5000 scale.

The results of the analysis is a 1:5000 scale map showing the location of magnetized zones, magnetic bedding depth and estimates of depth to magnetized rocks below ground. A summary showing number of lines and total survey lengths is provided in Table 1.

Table 9-1: Survey Kilometerage for the Aeromagnetic Survey

Survey Kilometers:	
60 Lines	390.5 km
7 Tie	44.8 km
Total	435.3 km

The final magnetic processing was achieved by base station diurnal correction followed by standard tie-line intersection leveling techniques. The intersections of traverse and control lines were calculated and the differences in observed magnetic values were attributed to residual diurnal variation and heading differences. In some active areas, with steep magnetic gradients, the difference also reflects errors due to small inaccuracies in both horizontal and vertical position at the line intersection. The corrections at individual intersections were adjusted as needed. The correction applied was a linear sloping datum between control line intersections. The final processed total magnetic field data from the tail sensor was lag corrected, microlevelled with a Butterworth procedure (400 m filter order 8, high pass, directional cosine, limited error values no greater than 5nT {92% of data}) and gridded with a bidirectional Akima spline procedure with a 25 metre grid cell size. The vertical derivative is calculated from this final data set.

The transverse magnetic gradients for each flight direction were calculated by subtracting the left wing sensor reading from the right wing sensor reading and dividing the resulting value by the tip-to-tip separation (13.5 metres), yielding the measurement expressed as nT/m. The longitudinal gradient was determined as the along-line gradient by subtracting successive tail sensor data and dividing by the specific distance travelled between each sample point (approximately 6 metres). The Reconstructed Total Field was calculated using the Nelson method and included in the database.

The Terraquest XDS VLF-EM system produced good resolution and consistent results. The x, y and z components of the XDS VLF-EM data in the half-power range of 22.0 to 26.0 kHz (which include Cutler, North Dakota and Seattle transmitter signals plus any other natural or artificial signals), were rescaled (where required), low pass filtered, DC shift corrected and levelled.

9.4.5 Gravity Survey

In 2010, Gravity Surveys were completed by Allan Spector and Associates. A Gravity Survey over the entire Young-Davidson project and a North Mudpack Gravity Survey were done. The following deliverables were provided:

- A digital recording of Principal Facts (station, line number, X-Y in UTM coordinates, elevation, Bouguer gravity, Residual Gravity)
- 1:10,000 scale profiles of Bouguer gravity, Residual Gravity and elevation for each survey line,

- 1:10,000 contoured Bouguer and Residual Gravity maps
- 1:10,000 Geological Interpretation Map
- 1:10,000 Geological Sections based on modeling of the gravity data and known geology
- Final Survey Report including Field Operations report, description of data processing methodology and description of interpretative findings.

The work consisted of 38 km of surveying on 48 roughly north-south cut lines. 1515 gravity measurements were taken at 25 m spacing. Line spacing was at 100 m.

The study area is included in the 1989 Shining Tree airborne EM and magnetic survey. Although no prospective EM conductors were observed in the vicinity of the Y-D property, the mineralization is associated with a vigorous pattern of magnetic relief.

The lithology density relations were provided to Allan Spector and are shown in Table 9-2.

Table 9-2: Lithology Density Relations

Lithology	Mean Density (g/cm^3)	Contrast (g/cm^3)
Timiskaming Group	2.75	+0.06
Lake Volcanics	2.83	+0.14
Mineralized Syenite	2.69	0
Matachewan Diabase	2.95	+0.26
Cobalt Group	2.68	-0.01
Algoman Intrusives	2.71	+0.02

High Density Units

The following high density units are distinguished in plotted profiles of the gravity data;

Unit M1 extends westward 3000 m from the east limit of the survey. Gravity decreases from over 3 m gal to the west to less than one milligal to the west. According to the published geology this 800 m wide, higher density zone is associated with a facies of the Timiskaming Group of sedimentary rocks that, according to Northgate measurements, have a mean density of 2.75 gm /cc. In places, the asymmetry of the gravity anomaly shape is indicative of a southerly dipping bedding attitude. The unit appears to be terminated by an inferred NE-SW trending fault; **F2**. This cross fault is also evident in the published Shining Tree aeromagnetic data.

A large proportion of the rocks comprising M1 are strongly magnetic. This is evident in ground magnetic survey data measured by site personnel along the 10,000N base line, where magnetic relief of 2000 to 3000nT is observed. The magnetic contact C2 marks the west contact of the most magnetic facies of M1. C2 is also associated with a dyke discerned from the gravity data near line 33.

An interpretive gravity map is provided in Figure 9-6.

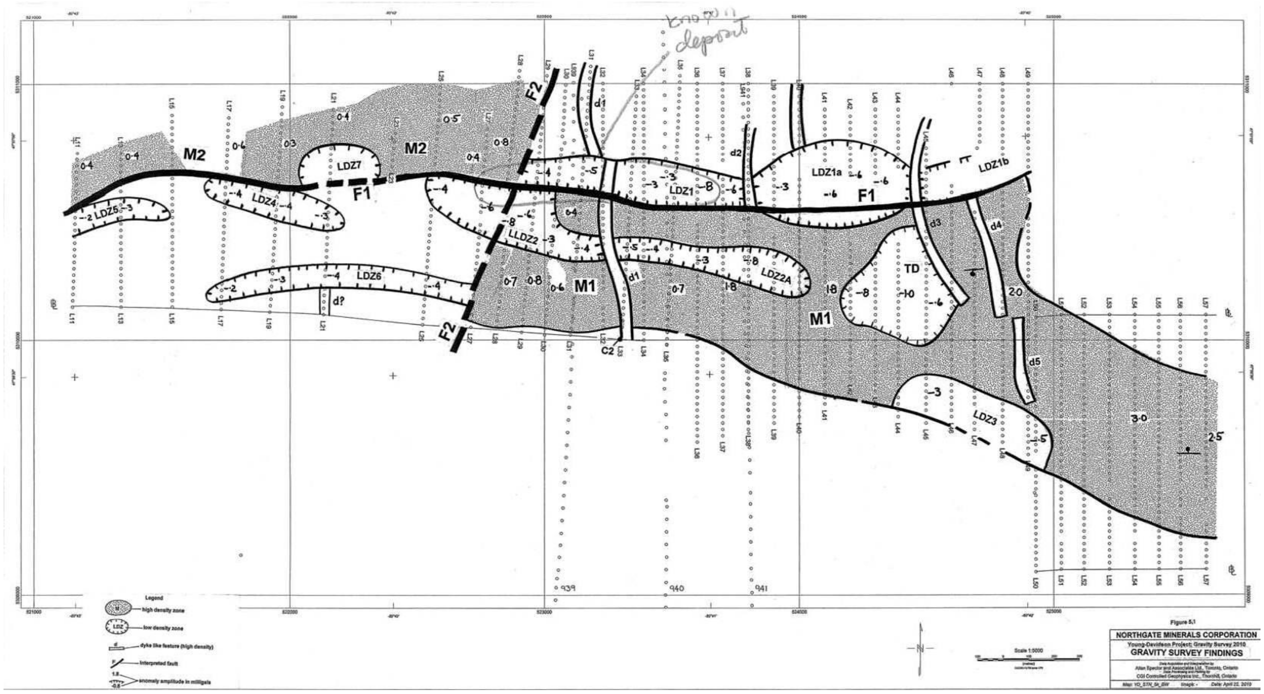


Figure 9-6: Allan Spector and Associates Interpretation Map –Main YD Deposit

The linearity of the north contact of M1 (in contrast to its southern contact) is taken as evidence of a major longitudinal fault; **F1**.

Unit M2 discerned in the west third of the survey, may be a dislocated extension of Unit M1. Gravity increase is less than one milligal. This relatively reduced amplitude could be attributed to the fact that the zone is buried by low density Huronian rocks. The south contact of Unit M2 is taken to be coincidental with fault **F1**.

Unit d is a narrow, dyke like gravity feature (high density) and northerly trending. At least 5 of these features are discerned (d1 to d5). The sub-parallel strike of these anomalies and the gravity survey likes makes it difficult to trace these features.

Low Density Units

Rocks that exhibit a reduced density compared to neighbouring units, are outlined in low density units or zones; **LDZ**. These units attract immediate attention because the Young-Davidson mineralized syenite is coincident with an LDZ.

Unit **LDZ1** parallels the north contact of Unit M1 and appears associated with the major fault **F1**. The 100 to 150 m wide zone can be continuously traced for over 1800 m and is characterized by a –0.3 to –0.6 mgal gravity decrease. Again, the mineralized syenite intrusion which has a strike length of 1000 m, is located in the west part of LDZ1. The zone appears to be dislocated at the intersection of F1 and F2. The mineralized syenite has a mean density of 2.69 gm/cc, -0.06 gm/cc in contrast to the adjacent Timiskaming units.

Unit **LDZ2** parallels the south contact of Unit M2 (fault F1) and appears to be an extension of LDZ1. The 100 to 200 m wide zone can traced for 1500 m.

Unit **LDZ3** is situated over the southern contact of unit M1. The reduced gravity is likely due to Huronian cover.

Units **LDZ4, LDZ5, LDZ6** and LDZ7are located within Huronian cover and most probably reflect facies changes within this younger sedimentary sequence.

Unit **TD** is associated with low density sediments in the tailings dump, south of the mine site. A Gravity contour map is provided in Figure 9-7.

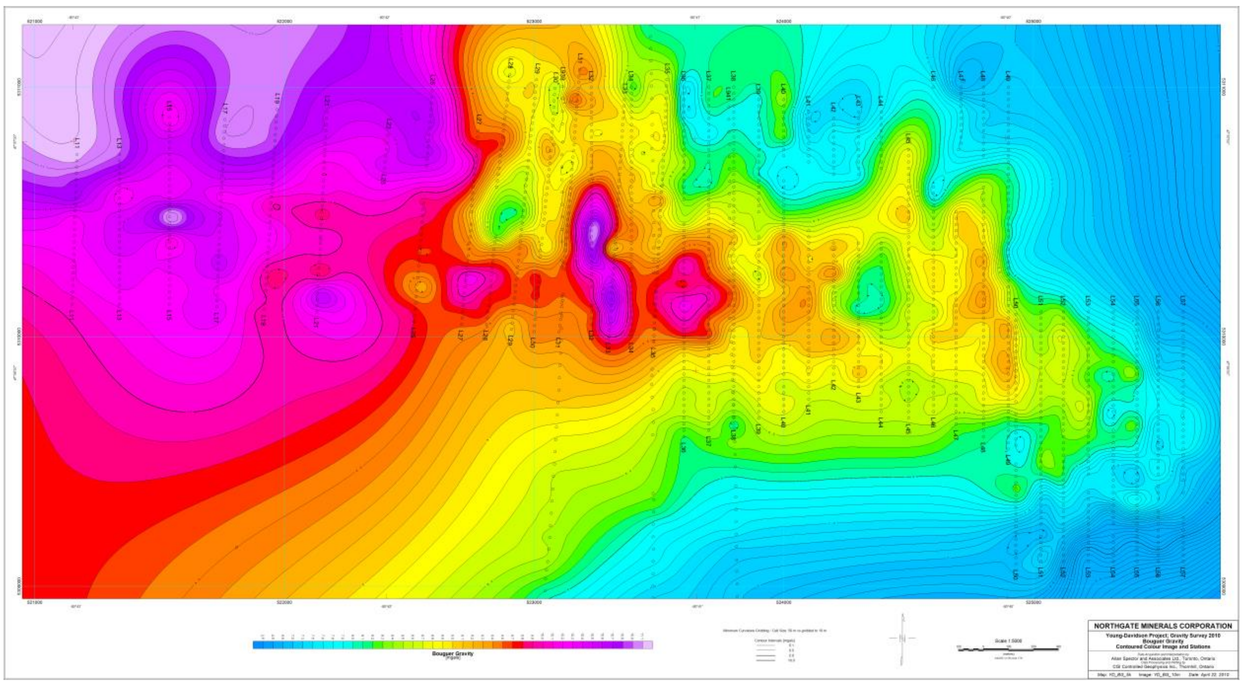


Figure 9-7: Gravity Contour Map

9.4.6 North Mudpack Gravity Survey

This work was initiated in response to a request to map density zones and structure south of Davidson Creek or Highway 566 for purposes of gold exploration. This gravity survey is intended to fill the gap between surveying north of Davidson Creek and various surveys to the south; Mudpack I and II.

The work on the North Mudpack area consisted of 10 km of surveying on 16 north-south cut lines at 100 m spacing. 378 gravity measurements were taken at 25 m spacing. Survey base was in the village of Matachewan. From this base, the survey area is accessed using Provincial Highway 566.

The survey area is entirely underlain by Huronian (Cobalt Group; Gowganda Fm.) sedimentary rocks; mostly wackes and diamicite that dip gently to the southeast and are all of glacial origin. The Gowganda Fm is estimated to be up to 200 m thick. Junnila states that lithologic types and structures favourable for gold mineralization may occur in this area, ie Witswatersrand-type paleo-placer gold deposition.

9.4.7 Gravity and Aeromagnetic Survey Findings

Findings of the gravity data analysis (using all data collected to date) are integrated with the interpretation of the recent aeromagnetic survey of the Northgate Property.

Because the gravity surveying was done on north-south lines and the aeromagnetic survey consisted of east-west lines, an integration of the two sets of findings is very constructive. For example the identification of magnetized Matachewan dykes from aeromagnetic survey serves as a basis for resolving their fairly large effects in the gravity data.

9.4.8 Induced Polarization (IP) Survey

In 2013, a Gradient and Insight Section Array Resistivity Survey was performed by Insight Geophysics Inc within the Southwest Area. A total of approximately 35.5 km of gradient array surveying was completed. A total of 7 sections were read.

9.5 Trenching

In 2012 and 2013, trenching programs were undertaken to further explore zones of interest in the Southwest Window of the Young-Davidson property. Gradient and "pseudosection" geophysical arrays were performed by Insight Geophysics, and identified zones of high chargeability at surface and at depth in the southwest window. A number of goals were accomplished over the two field seasons: the geological base map for Young-Davidson was updated in the SW window area and the Opawica area; three trenches, totalling 4662 m^2, were mechanically stripped, washed, geologically mapped and sampled; two diamond drill

holes totalling ~1000 m in length were also drilled and sampled. The trenches were placed throughout the SW window and centered on areas of high chargeability identified by the geophysical surveys.

Part of the 2012-2013 field mapping program focused on the area referred to as the "Southwest Window", located in the south-west corner of the Young-Davidson property. Prior to the 2012-2013 programs, exploration in the SW window was limited, mainly carried out in 2006-2007, with some additional attention from Iain McIlwraith in 2009. In 2006-2007, completed a 1:10,000 scale geologic base map covering all Company owned ground in the Matachewan area. The goal of this program was to map geology, structure, veining and mineral occurrences for the entire property. In 2009, a single diamond drill hole

(YD09-104 at 520,995mE, 5307740mN) was planned to test the eastern extent of the geophysical anomaly underneath the Proterozoic cover. With no encouraging results from this hole, this area was not revisited. In the years since, the Ministry of Natural Resources zoned land packages encompassed by the southwest window for logging and the resulting clear cutting of the area has opened new roads that provide better access. Thus in 2012, with easier access to potentially unmapped ground, the Company began a detailed investigation into the lithology, alteration and mineralization of the Southwest Window.

The trenching program showed that a package of deformed, strongly altered and weakly mineralized ultramafic to mafic volcanic rocks is generally the source of the chargeability highs observed in the gradient and pseudo-section I.P. surveys. Drilling near the trenches showed that the structural relationships between different rock packages is complex, and that late brittle faulting has superimposed different rock types onto one another.

9.6 *Conclusions*

The trenching and drilling program in the SW window aided in determining the distribution of Archean rock units in the SW window, as well as provided new questions for future exploration in the area. The structural story is of key interest here because at least 2 late faults have been uncovered and provide further questions about the structural history of the area. The fault located at the north end of Trench 3, between Archean and Proterozoic rocks, suggests late brittle faulting has placed the Archean rocks into tectonic contact with the Proterozoic rocks. Similarly, the fault uncovered in Trench 2 has placed diabase and altered ultramafic volcanic rocks into tectonic contact.

The conglomerate unit observed at the north end of Trench 2 stands out as unique among the rocks observed in the Matachewan area. These conglomerates contain strongly stretched clasts of feldspar porphyry hosted in a matrix also containing feldspar phenocrysts. This rock type is unusual because of its clast and matrix compositions, and also because of the amount of strain recorded in the clast stretching. These conglomerates, along with the other geological features observed in the 3 trenches from this program, are probably of interest to geologists of the OGS, as the current map indicates that this area contains only felsic and mafic volcanic rocks.

A single zone of weakly mineralized (< 0.5 g/t) mafic to ultramafic volcanic rocks was intersected in Trench 2. A mineralized zone was also intersected by YD13-282, however, the drill hole also highlighted the structural complexity of the area. The gradient I.P. survey conducted by Insight Geophysics suggests that the package of altered ultramafic rocks is continuous, on a rough 090° strike orientation, between Trench 1 and Trench 2. This zone may also extend to the west underneath the Proterozoic cover. Prior to any additional work in the area, it is recommended that Trench 1 be channel sampled to search for discreet zones of mineralization, if any. Considering the low grade of mineralization intersected at Trench 2 (< 0.5 g/t Au) and the likely problems with continuity, additional trenching between Trench 1 and Trench 2, and perhaps one east of Trench 2, may be beneficial prior to additional drilling.

10 DRILLING

Drilling on the property has been conducted in a number of campaigns by prior operators and by the Company. A summary of these campaigns by year is provided in Table 10-1. A plan map of the deposit showing all drilling through 2015 is provided in Figure 10-1.

Table 10-1: Drilling Summary – Drilling Campaigns by Year

Year	Hole Series	#holes	Metres	Location
unknown	BM	14	2,562	Surface
unknown	MR	7	660	UG
unknown	MW	12	167	Surface
unknown	MZ	30	300	Pit surface
unknown	R	7	1,881	UG
unknown	V	8	1,841	Surface
unknown	W	2	95	Surface
unknown	YD	32	2,210	Surface
unknown	YDFS	19	5,237	Surface
1934-1950	MCM	657	54,007	Surface/UG
1980	YD	62	5,437	Surface
1986	YD	84	9,812	Surface
1988	YD	10	1,183	Surface
1989	YD	10	3,978	Surface
1990	YD	31	20,846	Surface
1994	SH/SO	5	4,714	Surface
1995	MCM	29	6,985	Surface
1995	YD	74	25,347	Surface
1995	SO	7	1,641	Surface
1996	MCM	8	1,227	Surface
1996	YD	24	4,971	Surface
1996	SH/SO	4	719	Surface
1997	WL	6	857	Surface
2000	M00	11	300	Surface
2002	M02	21	3,899	Surface
2003	M03	48	7,847	Surface
2006	M6	6	486	UG
2006	MW	4	35	Surface
2006	YD	53	50,226	Surface
2007	NO/OP	67	9,272	Surface Pit
2007	YD	48	46,474	Surface
2007	YR	45	6,098	UG

Year	Hole Series	#holes	Metres	Location
2008	OW	2	300	Surface
2008	YD	50	42,389	Surface
2009	YD	60	16,456	Surface
2010	YM	29	3,473	UG
2010	YD	98	23,468	Surface
2011	YM	214	36,674	UG
2011	YD	26	23,099	Surface
2012	YM	295	44,686	UG
2012	YD	39	41,754	Surface
2013	YM	379	51,364	UG
2013	YD	21	19,393	Surface
2014	YM	263	32,968	UG
2014	YD	2	2,102	Surface
2015	YM	182	21,855	UG
2012-2015	YM UG Geotech	41	211	UG
	Total drilled	**3,146**	**641,505**	

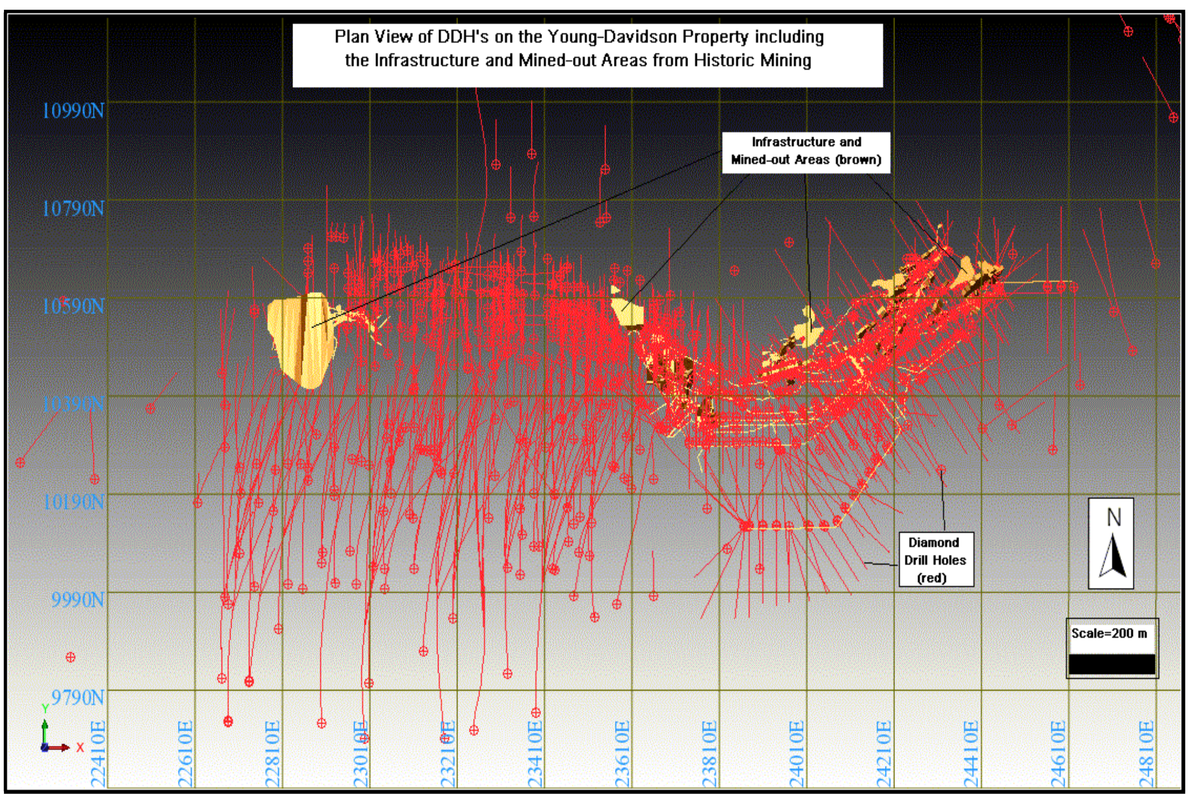


Figure 10-1: Plan View of Drill holes and UG Infrastructure

10.1 Pre-2006 Drilling Campaigns

Prior to the Company's acquisition of the Property in November 2005, 684 drill holes totalling 32,830 m had been drilled at the Property. However, detailed information regarding the drilling procedures during the pre-2002 drilling campaigns have not been available to the Company. However, historical holes from 1980 were down-hole surveyed using Pajari instruments or by acid tests.

Between late October 2002 and mid-May 2003, 1519865 Ontario Limited conducted a drilling campaign at the Property to test selected geophysical, geological or geochemical features identified during the exploration programs.

The drill holes were completed by Heath and Sherwood (1986) Inc.), an independent diamond drilling contractor from Kirkland Lake, using either a Boyles B20 or B25 diamond drill, or a custom-made diamond drill. During this campaign, 72 diamond drill holes totalling 11,747 m in length were completed. Core size was of BQ (36.5 mm) diameter, except for one drill hole, which was completed using NQ (47.6 mm) wire-line equipment.

The drill core was delivered to a secured core-logging facility twice per day. The core was re-aligned by the geologist to a consistent orientation, and was measured to confirm the accuracy of the depth markers placed in the core boxes by the diamond drilling crews. Logging included descriptions of the lithologies, alteration styles and intensities, structural features, occurrences and orientations of quartz veins, occurrences of visible gold, and the style, amount and distribution of sulphide minerals. Magnetic susceptibility was measured using a hand-held instrument.

AMEC (2009) did not identify any drilling or recovery factors that would materially impact the accuracy and reliability of the exploration data. The drill core provided samples of high quality, which were representative of alteration, veining, or sulphide accumulations intersected by drill holes.

10.2 Post 2006 Drilling Campaigns

The Company began a deep surface drilling program in early 2006, designed to confirm and expand the underground Mineral Resources on the Young-Davidson Property.

Orbit-Garant Drilling (OGD), from Val d'Or, Québec, was used as the drilling contractor. During this period, core drilling has been conducted with up to six custom-made SH-1 rigs. Core diameter on surface drilling has been NQ (47.6 mm), but underground drilling has used BQ (36.5 mm) diameter. Drilling commenced with hole YD06-01, and has continued to date. Most surface holes have been drilled with 0º ±10º azimuth, and at a range of dips from -45º to -80º; however, some holes were also drilled with approximately 90º and 180º azimuths. Depth ranged from 15.5 m to 1592.2 m, averaging 543.7 m.

Drilling has been often conducted in directional fans. IDS has been the contractor in charge of directing the holes, using deviation equipment rented by the Company from Devico Systems, and sometimes from Navy-Drill.

Surface drill holes are marked in the field with solid cement monuments and 1 m long pieces of steel bar, included a steel triangle showing the drill hole identification.

After extraction from the core barrel, the core was placed in wood boxes. All core boxes were properly identified. Small wood pieces were used to mark the drill runs. All core boxes were transported every day by truck to a secure core logging facility at Matachewan, where core was photographed, logged and sampled. All core was later transported to secure storage facilities within the project premises.

After arrival to the logging site, core boxes were placed on wood logging stands. Company personnel on site calculated the core drilling length recovery, and measured RQD. Core length recovery was in general good, usually exceeding 95%. Additionally, magnetic susceptibility was measured with a portable instrument at 3 m intervals.

10.3 Drill Logs

A complete set of drill logs are available for the drilling by Pamour Gold Mines Ltd (Pamour) and Pamorex Minerals from 1980 to 1995, the drilling by Royal Oak Mines Ltd (Royal Oak) from 1996 to 1997, by Young-Davidson Mines Ltd through 2003, and by Alamos and predecessor companies during the 2006 to 2015 drilling campaigns. The drill logs related to mostly underground drilling at both mines from the1930's to 1950's no longer exist, however, the drill hole traces and assays have been digitized from level plans and sections. All of the 2006-2015 core was photographed prior to splitting.

Logging of surface hole was done directly into a computer laptop using MS-Access©-based log forms. Logs recorded lithology, alteration and mineralization types, details about the mineralization style (approximate percentage of pyrite and chalcopyrite), and information on texture, color, grain size, presence and type of structures, and angle with core axis.

Underground core is currently logged into Dhlogger, a computerized logging system, and stored immediately on the Mine central computer system using laptop computers in the core shack on surface. Sampling as well as geological data is recorded in this manner. Holes are photographed prior to sampling. Underground definition drilling, YM series holes, are all whole core sampled. Un-sampled core (waste intervals) is retained until results are received and then discarded. YD series holes from exploration, are similarly logged and photographed but core is sampled by splitting and the hole is retained and stored on site.

10.4 Drill Collar and Down Hole Surveys

Collar survey records by professional land surveyors or trained technicians are available for most of the 1988 to 2003 drill holes and all of the 2006 through 2015 drill holes. It is assumed that most of the pre-1988 surface drill hole collars were not surveyed and were chained from the exploration grid lines. The collar coordinate uncertainty in these older holes could be in the order of plus or minus ten metres, which is insignificant for drill hole intersections that are spaced at 30 to 150 metres apart. It is believed that the underground drill holes are accurately located with respect to the surface drill holes.

In 2007 the project was converted over to metric NAD 83 UTM survey base and the details for this translation are documented in Company files. The conversion required a scale change and an Easting, Northing and Elevation translation from the origin of the imperial survey base.

The Company regularly used a FLEXIT and a gyroscopic instrument to measure hole deviation in order to locate the mineralized zones as accurately as possible. Historical downhole survey tests were taken every few hundred feet or so by using a tropari instrument made by Pajari Instruments Ltd., or a single shot camera and compass instrument made by Sperry Sun, or by using acid tests. All of the holes since 1980 have downhole survey data.

Mineralization at Young-Davidson generally strikes east-west and dips 75^{o} to the south, however there are localized areas where the orientations can deviate by 40^{o} in strike and 25^{o} in dip. Most drilling from surface has been conducted from south to north, while most underground holes are oriented to transect mineralization at a high angle. Most Young-Davidson Project core length intercepts do not represent true orthogonal width of mineralization, but rather a variable fraction of the core length intercept, depending on relative orientation of each hole and mineralization.

The Company considers that the locations of all of the surface and underground drill holes in the Young-Davidson Mineral Resource Database, with the exception of the MCM drilling, are accurate and should be included for Mineral Resource estimation work. MCM drilling has not been used in the Mineral Resource estimate.

10.5 Core Size

The 2006-2015 surface drill holes and one of the 2003 surface drill holes were drilled with NQ equipment (4.7 cm core diameter). The Company understands that all other surface holes have BQ core (3.7 cm core diameter) and that the underground drill holes from the 1930's to 1950's were probably drilled with AQ equipment (2.7 cm diameter). Post 2006 underground core has been drilled using BQ equipment until April 2014. Underground core was transitioned over to NQ in May of 2014 and has remained NQ since then.

10.6 Core Recovery

Core recovery and rock quality designation (RQD) information is available in the 2006-2015 drill logs and is generally absent in the historical drill logs. Over much of the Young-Davidson's exploration history, core recovery is excellent and the mineralization and its wall rock are very competent. RQD data is routinely collected for all underground drilling.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The Young-Davidson mine submits drill core and muck samples from underground mining operations to the Young-Davidson on-site mine laboratory. Blanks and certified reference materials (CRMs) are inserted by Geology in the sample stream and are "blind" to the laboratory. The laboratory maintains its own QC program which includes the insertion of RMs, analytical blanks and pulp duplicates.

11.1 Sample Lengths

During the exploration phase samples are generally taken on 1.5 m intervals in potentially mineralized zones, with some samples less than 1.5 m to preserve lithologic intervals. Longer samples are generally taken in barren post mineral dyke intervals. For orebody definition drilling, most sampling is on 1 metre intervals broken around lithological changes as required

11.2 Sample Preparation

Information on the historical methods used for sample preparation and gold assaying are not readily available. It is assumed that conventional crushing, pulverizing, and fire assaying techniques were carried out at the Young-Davidson and MCM Mine laboratories to analyze the 1930s to 1950s core samples. Most of the core samples from the 1980s and 1990s were likely processed at the Pamour laboratory in Timmins.

The sample preparation and assay methods are summarized in Table 11.1.

Since 2012, an on-site mine laboratory provides all sample preparation and assaying requirements for the operation. Currently, underground drill core samples weighing 3 to 5 kg are prepared and assayed at the Young-Davidson laboratory. The mine laboratory is also responsible for analysis of muck, chip and mill samples. Approximately 30,000 samples per year are assayed at the Young-Davidson mine laboratory.

Table 11-1: Summary of Preparation and Assay Methods

Period	Historical	1980s-1990s	2003	2006-2011	2012-2013	2014-2015	2012-2016
No of Samples	-	-	-	51,193	6,756	2,169	ug core and mucks
Laboratory	-	Pamour	Swastika	ALS and Swastika	ALS	ALS and YD	YD
Preparation: Crush	-	-	Crush to ~2mm		Crush to 70% passing 2mm		Crush to 80% passing 2mm
Preparation: Pulverize			Pulverize 350 g to 90% passing 150 µ		Pulverize 1000 g to 85% passing 75 µ		Pulverize 250 g to 85% passing 100 µ
Fire Assay Gold	-	-	30 g aliquot		30 g aliquot		30 g aliquot
Routine QC	-	-	-	1 in 75 blank, RM and reject duplicate	1 in 60 blank, RM and reject duplicate	2 in 60 blank, RM and reject duplicate	1 in 65 blanks,1 in 40 RMs

11.3 Analysis

Drill core samples used for Mineral Resource estimation have been assayed for gold by standard fire assay methods. A 50 gm aliquot was used at ALS and a 30 gm aliquot was used at Swastika.

The Young-Davidson mine laboratory assays sample pulps for gold by standard fire assay methods. A 30 gm aliquot is submitted for standard fire assay determination of gold with a detection limit of 0.01 g/t Au. Gold is determined by atomic absorption spectroscopy on a digest of the gold-silver bead and assays are repeated with a gravimetric finish when initial gold results are greater than 5 g/t.

11.4 Security

Exploration samples are transported by Manitoulin Transports to ALS in Sudbury using secured sealed bags. A chain of custody procedure was strictly followed during transportation. All of the underground samples are transported to the site assay lab for analysis. Some underground samples are currently being submitted to Swastika Laboratories in Kirkland Lake when the site lab gets behind in processing them. Sample security for these samples includes a similar protocol as described above.

11.5 Quality Control and Quality Assurance

No information has been compiled that describes the quality control (QC) and quality assurance (QA) procedures and results for the pre-2003 drilling programs. The main form of QC/QA in the past would have been periodic re-assaying of anomalous samples. Half-core duplicates for 632 sample intervals from 33 historic drill holes were used to assess historic assays for bias and no significant issues were identified.

Since 2006, project operators (Northgate, AuRico, and Alamos) have maintained a QC program for surface drill core that includes:

- insertion of blanks and RMs with each batch of submitted samples,
- creation of preparation duplicates, and
- submission of pulps to a secondary laboratory for check assays.

Half-core duplicates were used in 2006-2008 to document core sampling precision. The 252 half-core duplicates from 66 drill holes showed reproducibility consistent with the deposit type.

The reference materials inserted with samples are purchased from a third-party supplier Rocklabs. Quality control samples were inserted at a rate of approximately 4% which is consistent with industry standards. QC and QA results were compiled in detailed reports by R. Konst (Konst 2006-2009). Any outliers were investigated and re-assays were requested as required. Konst concluded that the sample preparation, security, and analytical procedures employed by ALS and Swastika were adequate and produced analytical results for gold and silver for the drilling programs which are accurate, precise and thus suitable for supporting Mineral Resource and Mineral Reserve estimation work.

No aspect of the sample preparation process was conducted by an employee, officer, director or associate of the Cmpany. ALS in Vancouver and Sudbury are ISO 9001:2000 and ISO 17025 accredited lab, while Swastika also holds an ISO 9001:2000 accreditation from CCRMP and the Standards Council of Canada.

Underground drill core, chips and mucks are assayed at the Young-Davidson mine laboratory. The laboratory has been operating since 2012 and has been audited by Analytical Solutions Ltd. several times. The Young-Davidson operation maintains rigorous assay quality control. Blanks and reference materials are inserted with underground drill core samples on a routine basis. Results are reviewed when received and any issues are discussed with the on-site laboratory. In addition, sample pulps are routinely submitted for check assays to an accredited commercial laboratory.

The mine laboratory also maintains a quality control program that includes reagent blanks, insertion of reference materials and pulp duplicates.

The mine laboratory has participated in the Geostats semi-annual international round robin since 2013 and performs well.

Both laboratory and the operations QC results have been reviewed regularly by Analytical Solutions Ltd. Analytical Solutions Ltd. concluded that there is no evidence of systematic gold contamination on the basis of the insertion of blanks. The QC data has occasionally identified a low or high bias of 2 to 3% based on RMs and check assays. Any trends are investigated thoroughly and minor modifications to the assay procedures have corrected these trends.

Both Geology and the Young-Davison laboratory use reference materials prepared by Rocklabs. The materials are synthetic materials consisting of mostly quartz, albite and basalt.

Young-Davidson geologic staff are considering the use of a reliable matrix-matched RM for the project.

The QC data is audited annually by Lynda Bloom of Analytical Solutions LTD (ASL). A summary of the 2015 audit results is provided below.

11.5.1 Accuracy as Determined by Blanks and Reference Materials

Coarse blanks and reference materials were submitted with samples to the mine laboratory by the Geology group. In addition, the YD laboratory has an internal quality control program of RMs, blanks and duplicates.

11.5.2 Laboratory Performance for Blank Samples submitted by Geology

Barren coarse material ("a blank") is submitted with samples for crushing and pulverizing to determine if there has been contamination or sample cross-contamination in preparation. Elevated values for blanks may also indicate sources of contamination in the fire assay procedure (contaminated reagents or crucibles) or sample solution carry-over.

A total of 433 blanks were assayed at the YD laboratory. Thirty four of these had incorrectly entered dates and were excluded from further analysis. The data are plotted in Figure 11-1.

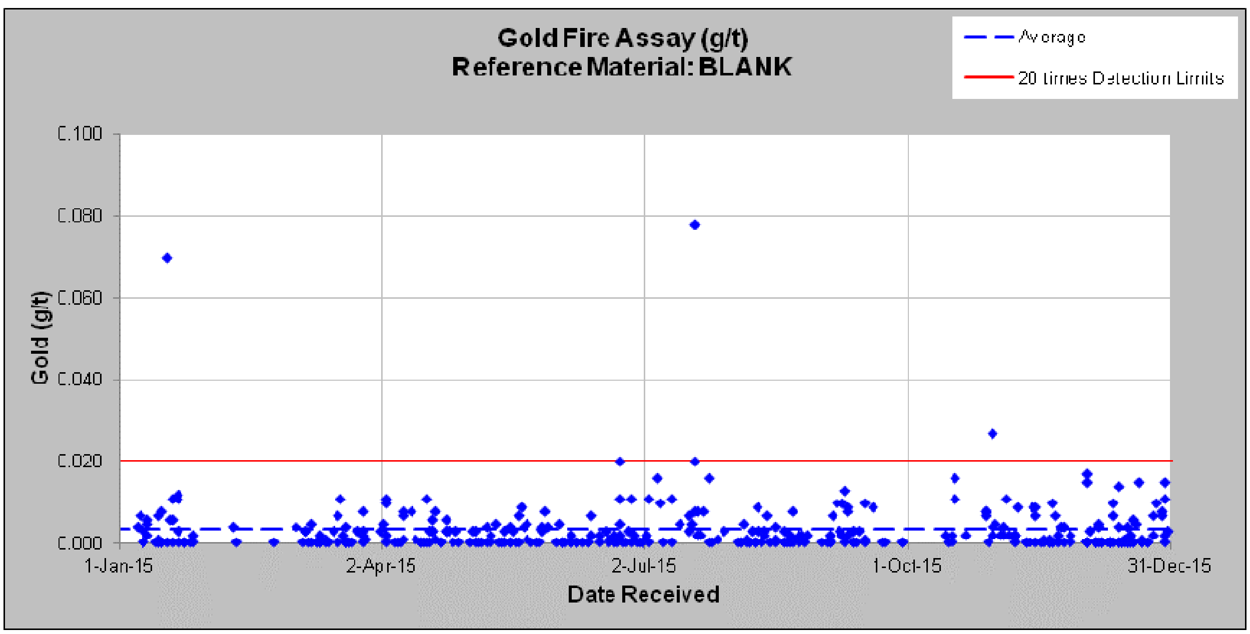


Figure 11-1: Gold Assays for Blanks at YD

There is no evidence of systematic gold contamination based on the blanks that were inserted with the samples.

11.5.3 Laboratory Performance Based on Reference Materials and Control Samples

Reference Materials ("RM") are submitted with samples for assay to identify:

- if there were assay problems with specific sample batches; and
- possible long-term biases in the overall dataset.

The definition of a quality control failure is when:

- assays for a RM are outside ±three standard deviations or 10%; and
- assays for two consecutive RMs are outside ± two standard deviations.

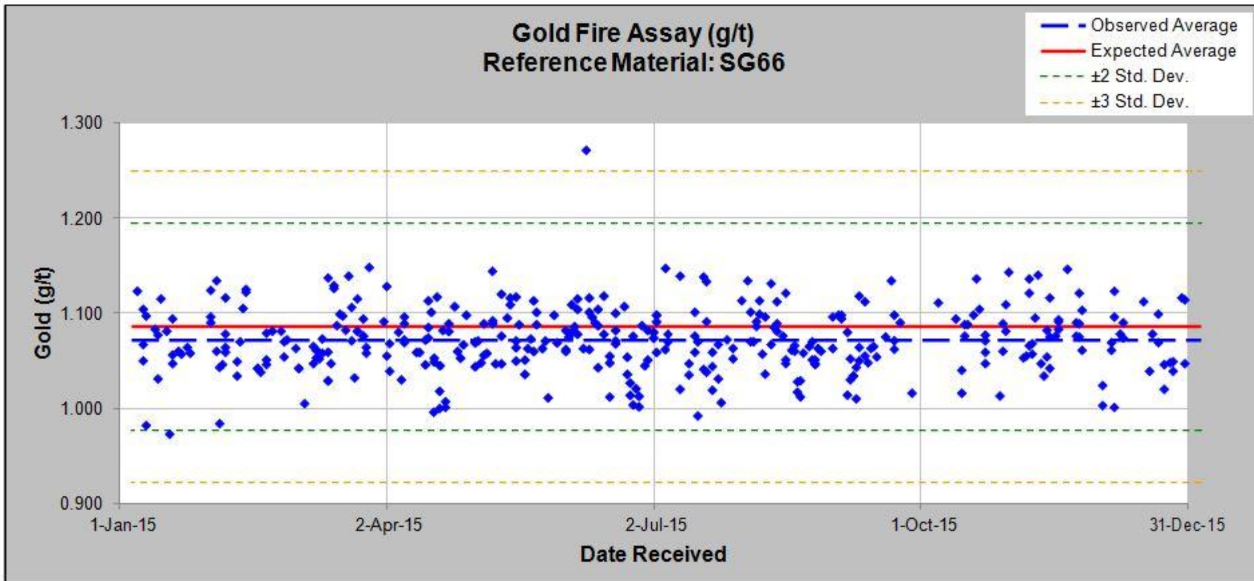
Two different RMs were submitted with samples. The reference materials are purchased from Rocklabs, a commercial supplier in New Zealand. Materials are feldspar, pyrite and basalt with some blended gold ore. Rocklabs does not provide a standard deviation for the expected values so a tolerance of ±5% of the accepted value (1 standard deviation) was used for all Rocklabs RMs in this report.

11.5.4 Quality Control Discussion Based on RMs

A total of 684 RMs (two different RMs from RockLabs) were submitted blind with samples sent for analysis to YD by the Geology department. The results are summarized in Table 11-4. Charts showing the performance for the two RM's listed below are provided in Figures 11-2 and 11-3.

Table 11-2: Summary of Reference Materials at YD

RM	**N**	**Expected Au (g/t)**		**Observed Au (g/t)**		**Percent of Expected**
		Average	**Std. Dev.**	**Average**	**Std. Dev.**	
SG66	373	1.086	0.054	1.072	0.034	98.7%
SL76	309	5.960	0.298	5.940	0.193	99.7%
Total	**682**				**Weighted Average**	**99.1%**

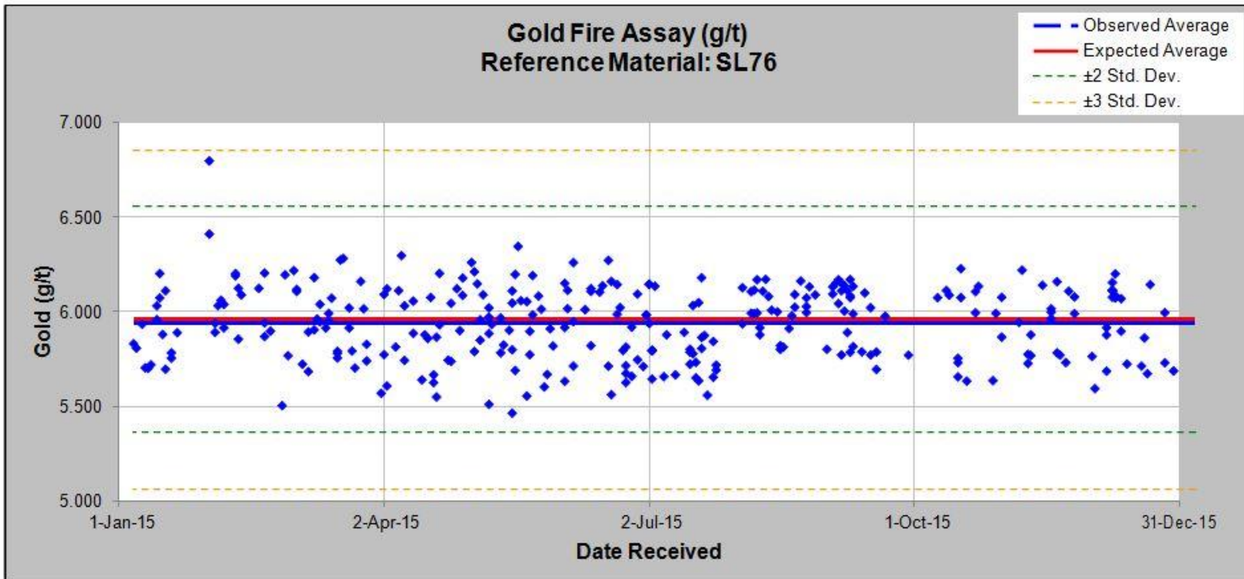


Summary Statistics

Expected Values	
Mean	1.0860
Standard Deviation	0.0543
Relative 95% confidence	9.80%

Observed Values	
Number of Samples	373
Mean	1.0716
Standard Deviation	0.0345
Relative 95% Confidence	6.31%
Falls Within 3 SD of Certified Mean	99.73%
Falls Within 2 SD of Certified Mean	99.46%
Falls Within 1 SD of Certified Mean	87.67%

Figure 11-2: 2015 Performance Chart for RM SG66



Summary Statistics

Expected Values	
Mean	5.9600
Standard Deviation	0.2980
Relative 95% confidence	9.80%

Observed Values	
Number of Samples	309
Mean	5.9405
Standard Deviation	0.1925
Relative 95% Confidence	6.35%
Falls Within 3 SD of Certified Mean	100.00%
Falls Within 2 SD of Certified Mean	99.68%
Falls Within 1 SD of Certified Mean	89.32%

Figure 11-3: 2015 Performance Chart for RM SL76

There were two instances (0.3%) where it is suspected that the reference material code was recorded incorrectly. The possibly mislabelled samples were excluded from the calculations in Table 11-4. Results were designated as a “mislabel” in cases where the reported results were similar to other RMs or blanks and were more than three standard deviations from the expected value.

The remaining RMs results are generally within ± 10% of the expected value but on average are biased low by 1%. The laboratory reports acceptable precision for gold assays on the Rocklabs RMs.

There was 1 QC failure (0.2% of the total RMs submitted) (Table 11-6) at the level of ±15% of the recommended value.

11.5.5 Laboratory Performance Based on Blanks and Reference Materials Inserted by the YD Laboratory

A total of 1844 blanks were inserted by YD laboratory personnel with all samples being analysed by the YD laboratory. A summary of results for the blanks is shown in Figure 11-4 and Table 11-7.

The laboratory blanks are assumed to be reagent blanks. Any results over the expected range (0.20 ppm) are not the result of sample preparation carry-over but could be related to a variety of other sources of contamination (for example re-use of high grade pots, solution carry-over at the AAS, etc.).

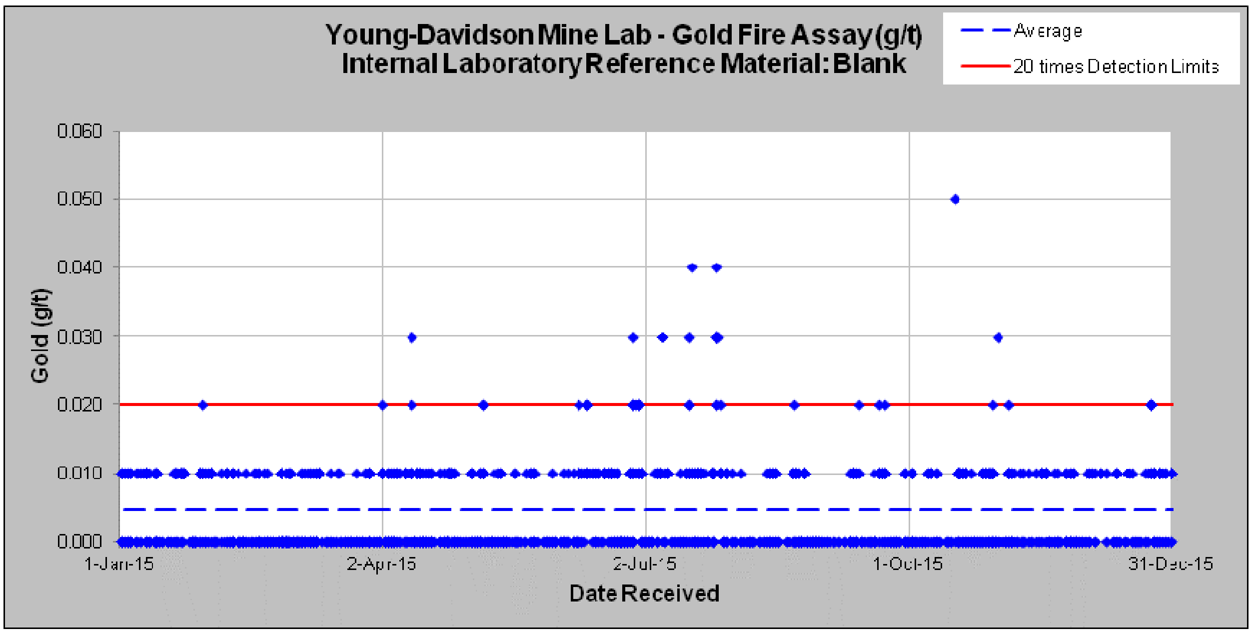


Figure 11-4: Gold Assays for Blanks Inserted by YD Laboratory

Table 11-3: Summary for Blanks Inserted by YD Laboratory

Criteria	Au (g/t)
Count	1839
Min	0
Max	0.050
Average	0.005
Std. Dev.	0.006

There were five mislabels and twenty-one failures at the 20 × detection limit level (Table11-9). The mislabels have been excluded from all charts and analyses. Of the five mislabels, two have assay values similar to reference materials. The other three are likely samples of drill core.

Fifteen of the 21 blank failures occurred on two days suggesting there was a short term issue at the laboratory. The level of contamination, approximately 0.03 g/t Au, does not impact operations given the ore grade but should be monitored.

A total of 4674 RMs were inserted by YD laboratory personnel with samples being analysed by the YD laboratory. The reference materials used by both the YD laboratory and Geology are purchased from Rocklabs.

A summary of the RMs utilized by the lab is provided in Table 11-6.

Table 11-4: Summary of Reference Materials Inserted by YD Laboratory

RM	N	Expected Au (g/t)		Observed Au (g/t)		Percent of Expected
		Average	Std. Dev.	Average	Std. Dev.	
0.599	2728	0.599	0.030	0.581	0.018	96.9%
0.606	241	0.606	0.030	0.593	0.018	97.8%
2.365	531	2.365	0.118	2.279	0.085	96.3%
2.656	1021	2.656	0.133	2.523	0.095	95.0%
8.671	120	8.671	0.434	8.405	0.387	96.9%
Total	**4641**				**Weighted Average**	**96.5%**

There were 33 cases (<1%) where it is suspected that the reference material code was recorded incorrectly. The possibly mislabelled samples were excluded from all charts and analyses. Results were designated as a “mislabel” in cases where the reported results were similar to other RMs or blanks and were more than three standard deviations from the expected value.

The remaining RMs results are generally within ± 10% of the expected value but on average are biased low by 2 to 5%. This is larger than the low bias (~1%) observed for the RMs inserted by the YD laboratory.

Using a tolerance of ±15% of the accepted value, there were eleven QC failures (<0.3% of the total RMs submitted).

11.5.6 Laboratory Pulp Duplicates

Laboratories routinely assay a second aliquot of the sample pulp for their internal quality control monitoring. The assays for pulp duplicates provide an estimate of the reproducibility related to the uncertainties inherent in the analytical method and the homogeneity of the pulps. The precision or relative percent difference calculated for the pulp duplicates indicates whether pulverizing specifications should be changed and/or whether alternative methods, such as screened metallics for gold, should be considered.

The YD laboratory runs a routine pulp duplicate for one in 19 samples. The original and duplicate assays are plotted in Figure 11-5. There were a total of 1,445 laboratory duplicate gold assays from YD provided to ASL.

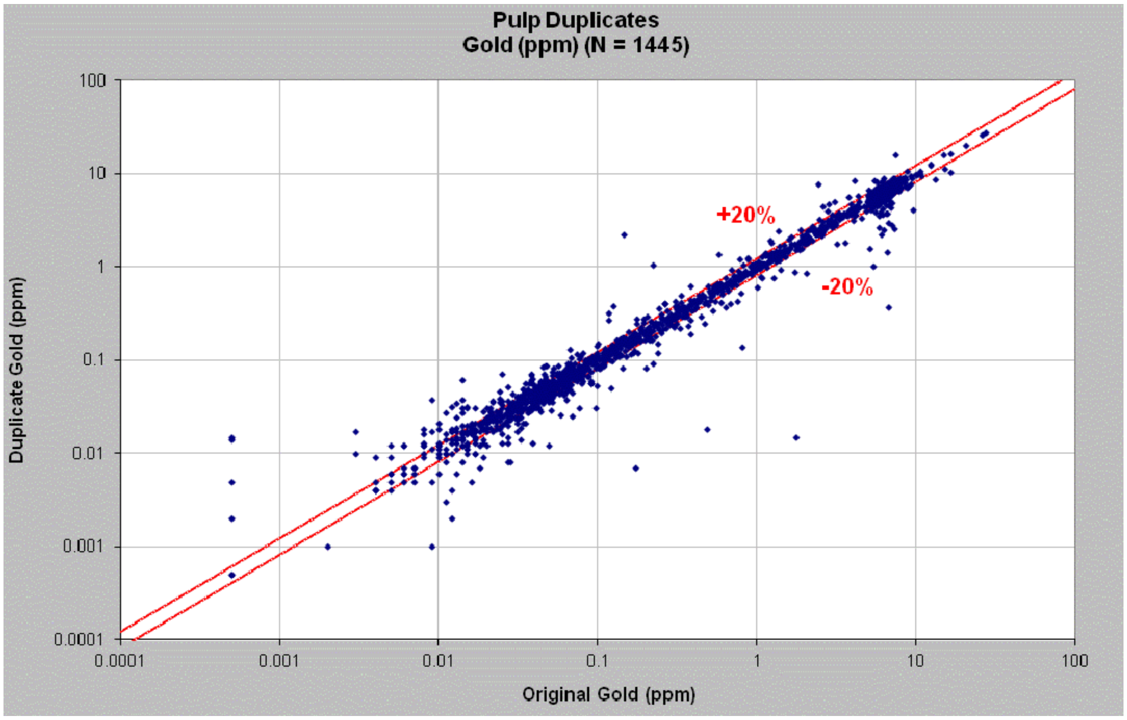


Figure 11-5: Laboratory Pulp Duplicates for Gold at YD

Figure 11-6 shows the relative percent difference (RPD) of the laboratory pulp duplicates for sample pairs. Samples with concentrations less than 0.5 g/t Au have poor repeatability, due to poor precision of the method at lower concentrations. For the 242 samples with Au grades between 1 g/t and 5 g/t, 95% of the samples repeat within ±25%. Reproducibility is similar for samples with greater than 5 g/t Au but it is suspected that a high proportion of these samples have repeat assays (or the fire assays with gravimetric finish have been included with pulp duplicate data).

Visible gold is routinely observed in drill core logging which explains the assay variability associated with pulp duplicates.

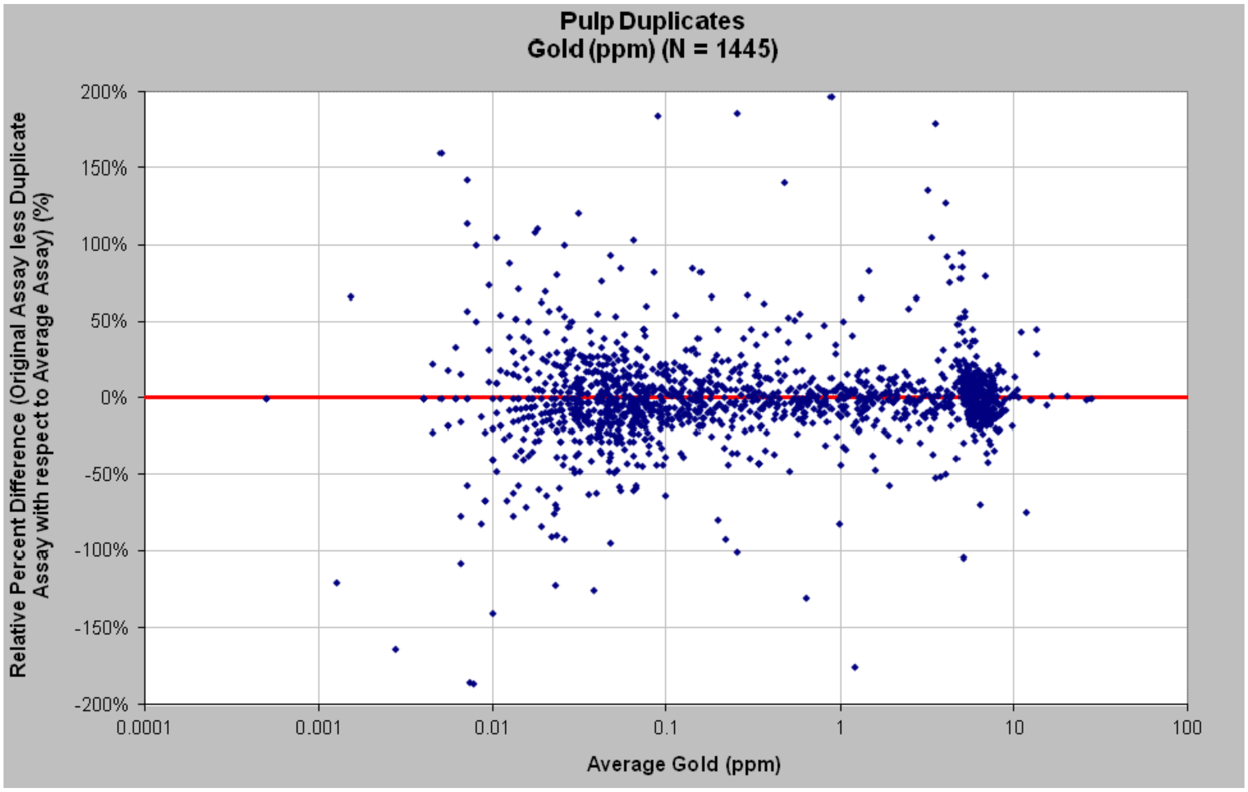


Figure 11-6: RPD Plot of Laboratory Pulp Duplicates for Gold at YD

11.5.7 Repeat Assays with a Gravimetric Finish

Aside from the above routine pulp duplicates, the YD laboratory repeats assays on all samples reporting more than 5 g/t Au by fire assay with a gravimetric finish (FA-Grav). "Original" assays are reported on a certificate and a second "Repeat" certificate is issued with the additional assays on the same pulp.

A total of 246 samples were analyzed by both the FA-AA and FA-Grav methods. The majority of the sample pairs (78%) have assay values within 20% of each other (Figure 11-7). No significant bias was identified.

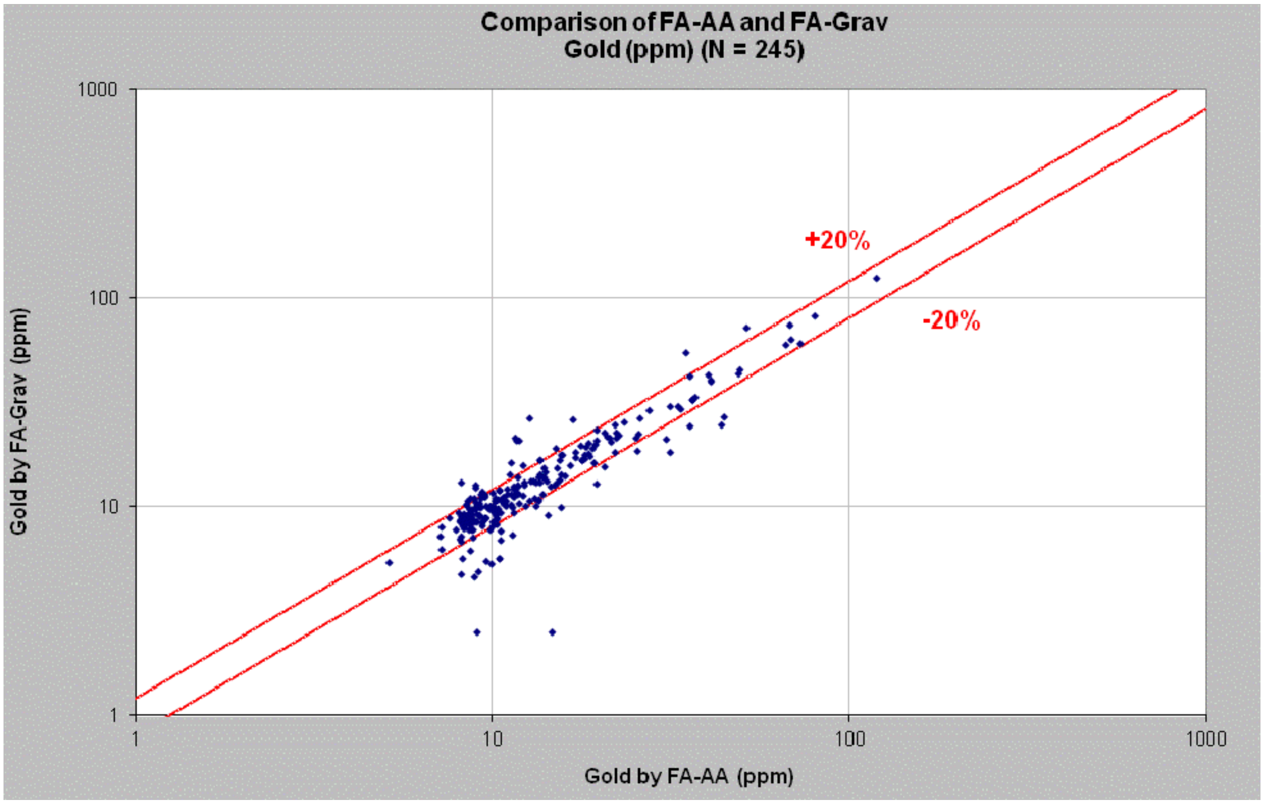


Figure 11-7: Comparison of FA-AA and FA-GRAV Analyses at YD

One assay was identified as a likely error in sample dilution and was excluded from further analysis.

11.5.8 Check Assays at Secondary Laboratory

Check assays are used to augment the assessment of bias based on the RMs and in-house control samples. The same pulp that was assayed originally is submitted to a different laboratory for the same analytical procedures. Reference materials are also inserted with samples submitted to the secondary laboratory to measure whether the secondary laboratory is potentially biased.

A total of 292 samples were submitted to ALS for check assay. A total of 25 reference materials and 15 blanks were submitted to ALS with the samples for check assay. None of the reference materials failed and only a very slight negative bias was observed (Table 11-7). None of the blanks showed any sign of contamination (Table 11-8).

Table 11-5: Summary of Reference Materials Submitted to ALS with Check Assays

RM	N	Expected Au (g/t)		Observed Au (g/t)		Percent of Expected
		Average	Std. Dev.	Average	Std. Dev.	
SE68	4	0.599	0.030	0.583	0.019	97.4%
SG66	10	1.086	0.054	1.081	0.044	99.5%
SL76	11	5.960	0.298	5.851	0.321	98.2%
Total	**25**				**Weighted Average**	**98.6%**

Table 11-6: Summary for Blanks Submitted to ALS with Check Assays

Criteria	Au (g/t)
Count	15
Min	<0.005
Max	0.005
Average	0.004
Std. Dev.	0.001

Most of the assays agreed within ± 30%. This variability is expected based on the precision of the duplicate assays on the same pulp where both assays were performed at the YD laboratory.

More significantly, the YD assays were biased low with respect to the ALS assays. Sixty percent of samples assayed higher at ALS than at YD (Table 11-9). This is consistent with the observation that results from the YD internal laboratory reference materials that show a 2 to 5% negative bias.

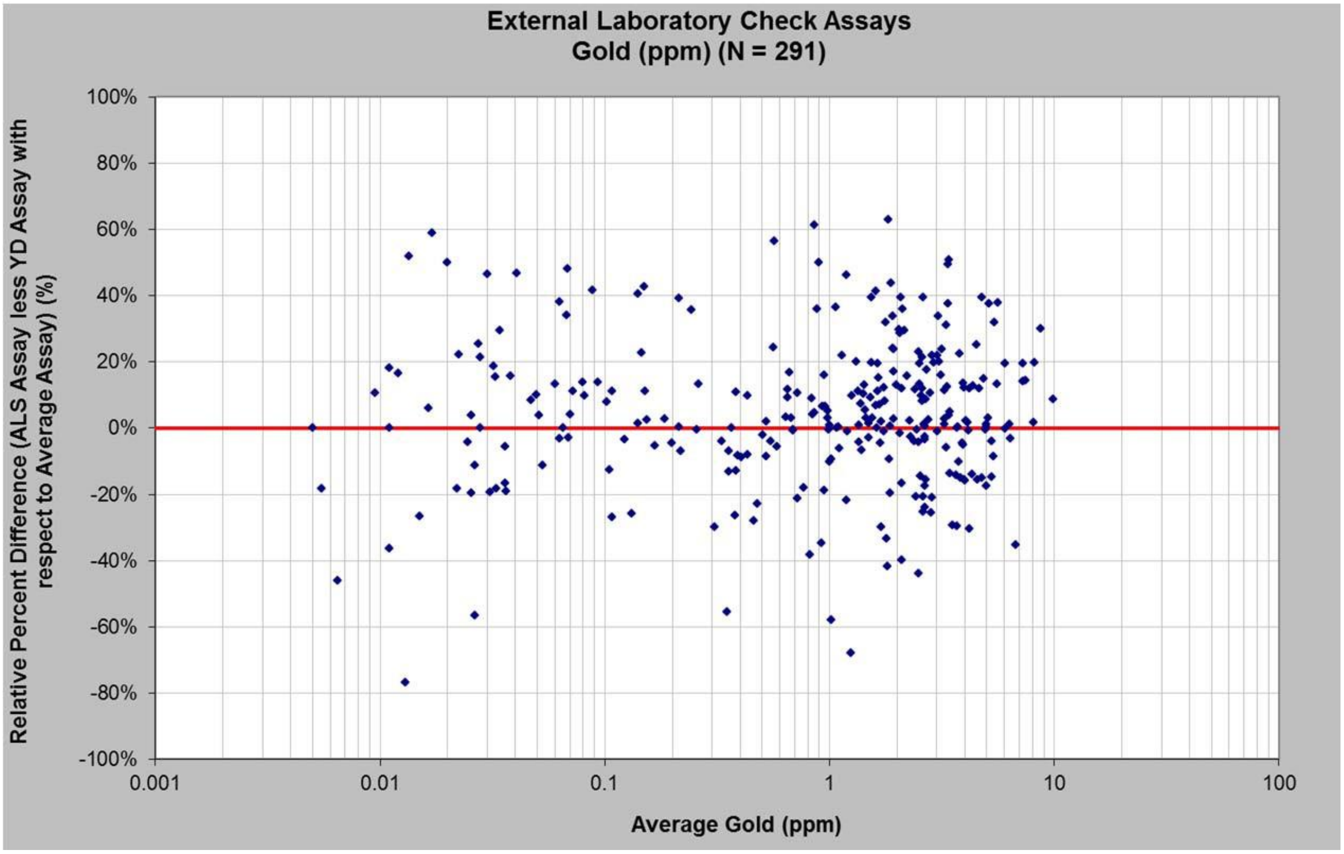


Figure 11-8: Comparison of Samples Analyzed by YD and ALS

In the above graph, where the RPD is positive then ALS assays were higher than YD; where the RPD is negative then YD assays were higher than ALS. The effect is consistent for most grade ranges.

Table 11-7: Analysis of Relative Percent Deviation (RPD) for Samples Analyzed at YD and ALS

	N	Percent of Total
RPD < -20%	34	12%
RPD < -10%	65	22%
RPD < 0 (YD>ALS)	112	38%
RPD > 0 (YD<ALS)	174	60%
RPD > 10%	112	38%
RPD > 20%	59	20%

It is difficult to accurately assess the average relative percent difference. For samples with an average grade less than 1 g/t Au, the difference is nominal. The average difference for samples with greater than 1 g/t Au is about 3%, with ALS reporting higher than YD.

One sample was identified as a clerical error and was excluded from further analysis.

11.5.9 Conclusions

The Young-Davidson operation maintains rigorous assay quality control. Blanks and reference materials are inserted with underground drill core samples on a routine basis. Results are reviewed when received and any issues are discussed with the on-site laboratory. In addition, sample pulps are routinely submitted for check assays to an accredited commercial laboratory.

The mine laboratory also maintains a quality control program that includes reagent blanks, insertion of reference materials and pulp duplicates.

There is no evidence of systematic gold contamination on the basis of the insertion of blanks.

There is a low bias of 2 to 3% for YD gold assays based on results for RMs inserted by the laboratory. This is corroborated by similar low bias for YD gold assays relative to ALS assays. Conversely, RMs inserted by Geology do not demonstrate a bias and the YD laboratory performs well in the international Geostats round robin. This bias has been apparent previously and the laboratory continues to investigate.

It is noted that there are still some cases where assays have been reported that are off by an order of magnitude. This is due to dilutions not being properly recorded in the LIMS and is a recurrent problem.

The Young-Davidson operation maintains a QC program that meets or exceeds industry standards. Sample preparation, security, and analytical procedures are all industry-standard and produce analytical results for gold with accuracy and precision that is suitable for mine operations.

The responsible QP is of the opinion that the Young-Davidson operation maintains a QC program that meets or exceeds industry standards. Sample preparation, security, and analytical procedures are all industry-standard and produce analytical results for gold with accuracy and precision that is suitable for mine operations.

12 DATA VERIFICATION

12.1 Northgate 2008 Data Verification

Collar coordinates for 132 holes with 2,529 down hole survey tests and 2,924 assay intervals were audited and verified for accuracy against a variety of supporting documentation by Company personnel working under the direct supervision of the author from October 2008 through December 2008. The author specifically selected these 132 drill holes because they intersected the underground and open pit Mineral Resource solids forming the subject of this report. Additionally, the data verification extended beyond the limits of the Mineral Resource solids to include a lower grade envelope that may be used for future Mineral Resource modeling.

During the data verification program no significant errors were found. A few minor issues related mostly to collar locations, surveys, and assay intervals were identified and corrected. The most significant revisions occurred with the addition of new survey information in a couple of holes. Six duplicate sample numbers were discovered and resolved but did not involve any grades of consequence. The audit indicated that database accurately reflects available supporting information.

12.2 AMEC 2008 Data Verification

As a background to the preparation of the 2008 Preliminary Economic Assessment completed by AMEC in April 2008, AMEC conducted a database audit of 5% of the total records. AMEC examined 15 drill holes, 60 down hole surveys and 695 assay records concluding that the database at that time was suitable for conducting Mineral Resource evaluation studies on the Young-Davidson property.

12.3 Northgate 2007 Data Verification

Collar coordinates for 351 holes with 1,565 down hole survey tests and 14,851 assay intervals were audited and verified for accuracy against a variety of supporting documentation by Company personnel working under the direct supervision of the Carl Edmunds from October 2007 through December 2007. Mr. Edmunds specifically selected these 395 drill holes because they intersected the underground Mineral Resource solids forming the subject of this report as well as any open pit area Mineral Resources to be reported on later. Additionally, the data verification extended beyond the limits of the Mineral Resource solids to include a lower grade envelope that may be used for future Mineral Resource modeling. The data verification program was designed and supervised by the Mr. Edmunds.

Northgate identified and corrected a large number of minor issues related to downhole survey test distance values, rounding, truncation, and calculation errors. A few significant issues related mostly to collar locations, collar orientations, and assay intervals were also identified and corrected. The most significant revisions occurred with the addition of new interval information for 651 assays which were previously reported as much longer composite or summary results in the historic MCM and Young-Davidson data.

12.4 Northgate 2006 Data Verification

Collar coordinates for 41 holes with 2,360 down hole survey tests and 1,137 assay intervals were verified for accuracy against a variety of supporting documentation by Northgate from September 18 through October 2, 2006. The author specifically selected these 41 drill holes because they intersected the Lower Boundary, Lower YD, and Lucky mineralized zones. The data verification program was designed and supervised by the author.

The Company identified and corrected a large number of minor issues related to downhole survey test distance values, rounding, truncation, and calculation errors. A few significant issues related mostly to collar locations, collar orientations, and assay intervals were also identified and corrected.

The Company made minor adjustments to most of the collar coordinates in the fall of 2006, after the collar locations and grid control points were resurveyed, and minor refinements to the transformation profile for converting NAD83 survey information into “Mine Grid” were made.

12.5 Scott Wilson RPA 2006 Data Verification

In 2006 Scott Wilson RPA verified a small number of collar, downhole survey, and assay records including some of the longer assay intervals and higher gold grades. No significant errors were found. Although, the drill log, assay certificates, downhole survey data, and collar survey data for each hole were well organized and filed together, a number of exceptions were found. Subsequently, Scott Wilson recommended ensuring that the hard copy file for each drill hole be complete thereby expediting future data verification programs.

12.6 Micon 2004 Data Verification

In 2004 Micon verified 7,370 assay records in 51 holes that intersected mineralized zones and found very few errors. None of the errors were considered to be significant (Micon, 2004).

Pre-2004 data verification work is not documented.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The selection of the overall plant configuration was primarily based on typical gold operations, the flowsheet employed by previous operators of a plant processing ore mined from the same deposit, and design guidance by the Company's metallurgical consultant, John Goode. The metallurgical testwork programs, from Phases I through III , were designed and supervised by the Company to confirm and optimize this information and formed the basis for preparing the design criteria, process flow diagrams, mass balance and equipment sizing, used for design of the ultimate plant.

13.1 Introduction

The Young-Davidson project was developed by Northgate and AuRico with feed from a combination of open pit and underground mining operations. The Au/Ag-bearing ores are processed at a nominal rate of 8,000 tpd, with a 91% availability, in a plant recovering the precious metals in bullion. Actual throughput capability will vary over time, along with the average ore hardness characteristic of the plant feed stream.

The design feed grades were 4.2 g/t for gold (Au), 4.2 g/t for silver (Ag) and 1.5% sulphur (S). This last value was largely assumed from metallurgical samples obtained since sulphur was not an element tracked in the geological block model. The design gold grade represented a margin of 25% above the peak year as indicated by the Feasibility Study mine plan, with silver set at an equivalent level.

The expected average gold grade is 2.69 g/t for the material included in current Mineral Reserve from the underground operations, with 0.92 g/t for silver. The equivalent gold content for the open pit ore anticipated to be processed was 1.68 g/t. Insufficient silver assays from the cores of the open pit zone were available to warrant estimation of this metal in the block model and its availability when devising the mine plan.

The process flowsheet includes a single stage of surface or underground crushing, operated by the mine or by a contractor (for open pit ore). This is followed by grinding in a single-stage SAG milling circuit, operated in close-circuit with cyclones and including a gravity recovery circuit coupled with an intensive cyanidation unit. The ground slurry is then fed to a flotation stage, with the concentrate being reground before feeding a carbon-in-leach (CIL) circuit for precious metal dissolution in a cyanide solution. The tailings of this circuit is joined by the flotation tailings and undergoes further CIL treatment. The loaded carbon extracted from the first CIL tank of the combined concentrate-tailings circuit is brought forward into the last tank of the concentrate leaching circuit.

The loaded carbon extracted periodically from the concentrate CIL circuit is brought to an acid washing and stripping circuit, for removal of the precious metals into solution and recovery through an electrowinning (EW) process. The EW step produces a sludge which is dried and then fed to an electric furnace where refining for final production of a gold and silver doré is completed.

The combined CIL circuit tailings are treated for cyanide destruction, prior to release into either the tailings facility or to the paste fill plant. Water is reclaimed from the tailings pond, as make-up process water, the balance of which is obtained from the flotation concentrate and tailings thickener overflows. Excess water, mainly from surface run-off and mine dewatering, is decanted to the mine water pond from where it is discharged to the West Montreal River. Fresh water is provided for specific use within the processing circuits.

The Feasibility Study mine plan called for feeding the mill exclusively with open pit ore for the first two-and-a-half year, then transitioning gradually to 100% underground ore as the underground operation ramped. Low grade stockpiled open pit ore would be used to supplement the mill feed as the underground operation ramped up.

13.2 Mineralogy

Mineralogical examinations were completed during Phase I of the metallurgical testwork by AMTEL. Figure 13-1 presents the result obtained for a master composite metallurgical sample made up from equal portions of three Young-Davidson Pit, two Upper Boundary, two Lower Boundary and one Lucky Zone composite samples. AMTEL assayed the master composite and reported grades of 3.0 g/t gold, 3.2 g/t tellurium and 1.3% total sulphur.

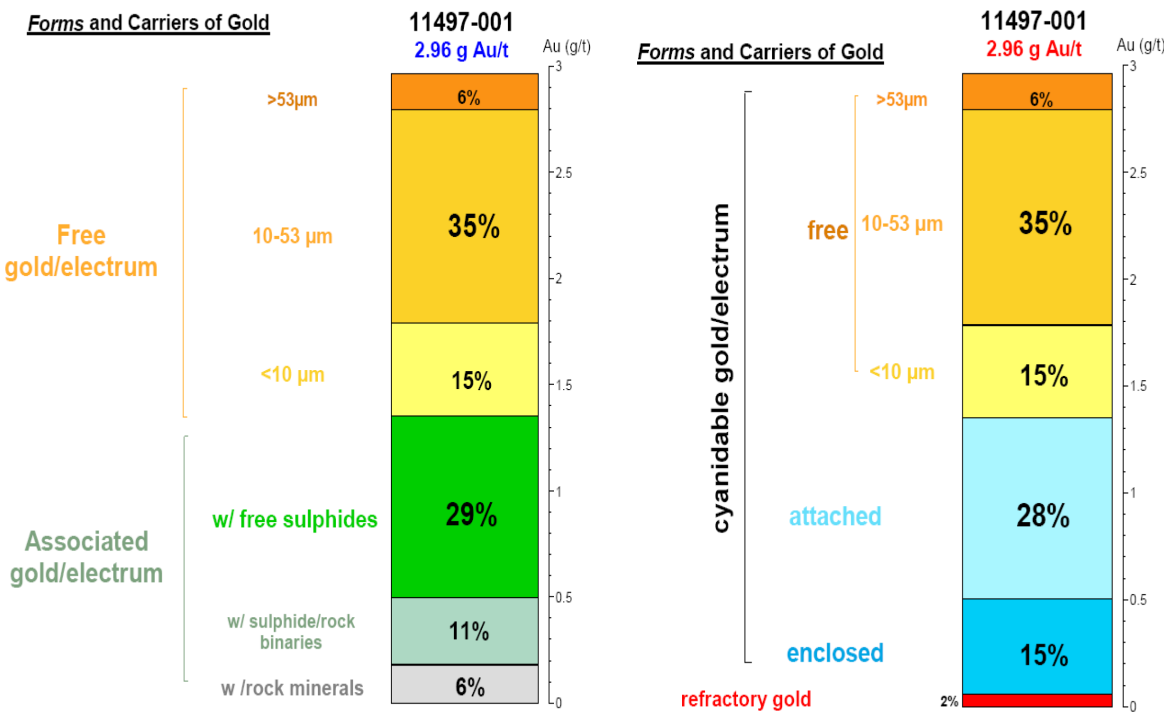


Figure 13-1: Gold Deportment at a P80 of ~ 110 µm Young-Davidson ore

The left column of Figure 13-1 reflects the expected flotation response, while the right column depicts the expected response to cyanidation.

The gold distribution diagram on the left in Figure 13-1 indicates that high gold recoveries should be possible through a combination of gravity concentration and flotation. The diagram on the right suggests that gold recovery by gravity and cyanidation at a 110 µm grind will be around 85%. The proposed gravity-flotation-regrind-cyanidation circuit should attain extractions similar to that shown on the left, i.e. up to 94%.

AMTEL showed that native gold (87% Au, 13% Ag) was the principal form of gold, with all other forms combined representing less than 10% of the gold assay. Sub-microscopic gold locked in pyrite represented only 2% of the total gold content.

The head assays for the Young-Davidson composites indicated tellurium levels about equal to that of gold, which raised a concern regarding the presence of refractory gold tellurides in the ore. The AMTEL study confirmed the presence of hessite (Ag_2Te), altaite (PbTe) and a bismuth telluride but less than 2% of the gold occurred in these compounds. The higher association of silver with tellurides partially explain the somewhat more refractory nature of silver, compared to gold, during cyanidation.

Figure 13-1 indicates that 6% of the gold in the sample was larger than 53 µm with another 35% in the -53+10 µm size range, confirming the viability of a gravity concentration step in the flowsheet.

13.3 Metallurgical Testwork and Design

13.3.1 Introduction

Northgate conducted a program of metallurgical testwork on coarse assay rejects and quartered drill core samples, representing the major ore zones of the Young-Davidson gold deposit, during the period 2007 to 2009. Testwork was generally conducted by SGS Lakefield with specialized studies by AMTEL, Metso, Deswik, and Outotec. The primary purpose of the testwork program was to determine design parameters for subsequent flowsheet development, equipment sizing and cost estimating exercises. A review of the results obtained, with an emphasis on the Phase III program, is presented in this section.

13.3.2 Testwork History

The metallurgical response of Young-Davidson gold ore was tested between early 2007 and early 2009. Various investigative programs were conducted by a number of different experts with JR Goode and Associates (Goode) taking the lead in assisting Northgate in the design and supervision of the testwork programs and with the analyses of results.

Initial scoping level testwork explored the viability of various processing routes and strategies including whole ore cyanidation and either flotation or high-mass gravity concentration followed by cyanidation. This Phase I program included comminution tests on core samples from the individual ore zones, augmented by some variability testing of the comminution test rejects. A Master Composite was used to further explore the metallurgical response of the ore during processing.

In June 2007, after analysing the available scoping level results, Northgate adopted a flowsheet which includes intensive cyanidation (IC) of a gravity concentration circuit product; rougher flotation, followed by flotation concentrate regrinding and carbon-in-leach (CIL); and a combined CIL of concentrate CIL tails and thickened flotation tailings. Loaded carbon is stripped in an elution circuit with the pregnant eluate and intensive cyanidation pregnant solution reporting to separate electrowinning circuits for gold recovery. This high level flowsheet, presented below as Figure 13-2, remained largely intact during the successive study phases of the project and provided a basis for planning the subsequent Phase II and III metallurgical testwork.

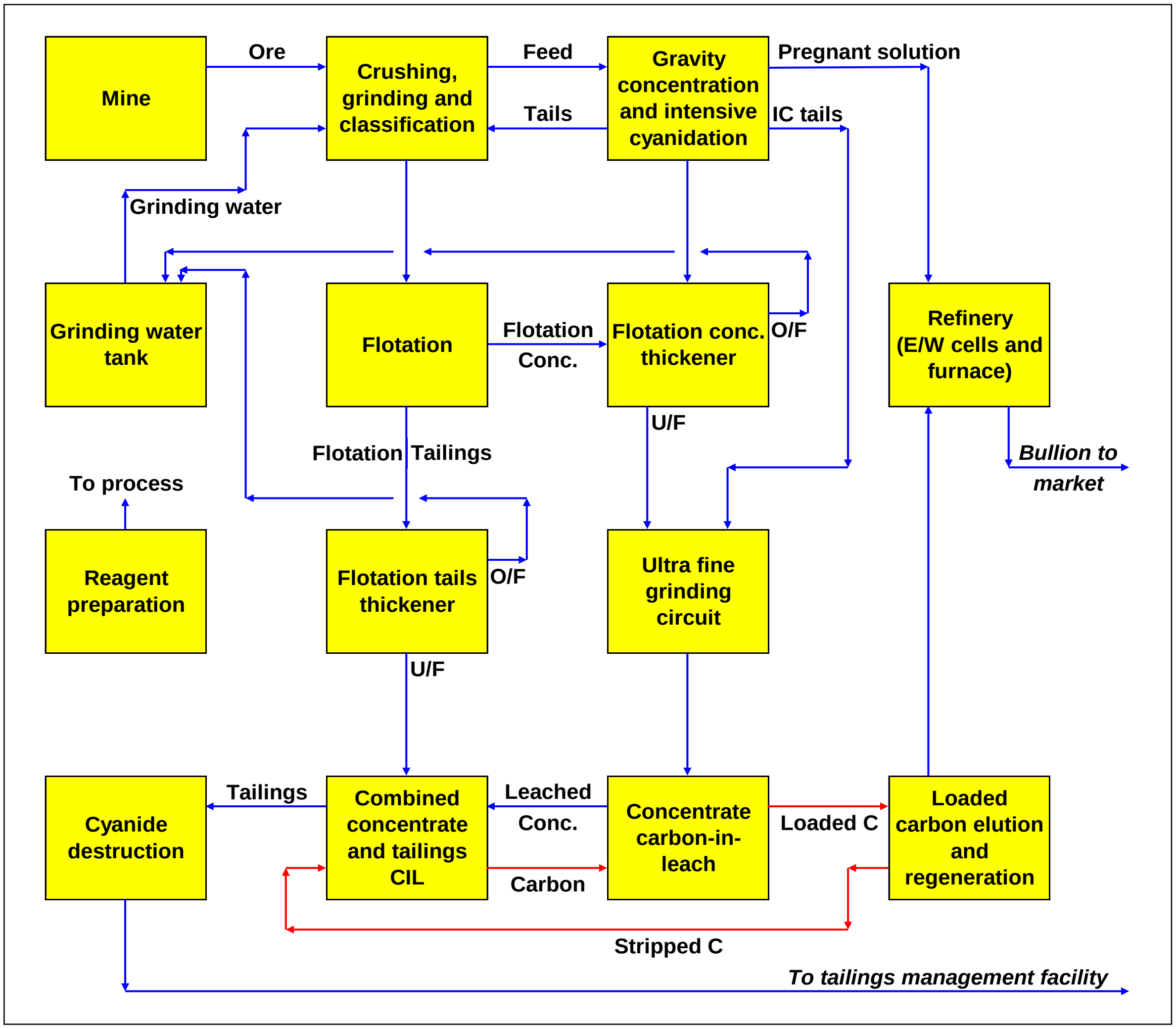


Figure 13-2: Block Diagram of Proposed Young-Davidson Plant

The Phase II testwork program included further variability work on the individual samples used in the Phase I program and work on a second Master Composite. Metal recovery unit operations were explored in more detail. The scope of work was expanded to include cyanide destruction, environmental, ultra-fine grinding, and pulp rheology testing. Results of this program provided the basis for AMEC's Preliminary Feasibility Study.

The Phase III program comprised further variability work on a new suite of samples. These new drill cores were blended to form zone composites, hole composites, and Master Composites 3 and 4. Tests were conducted on all composites mimicking the proposed flowsheet. The Phase III scope expanded to include analysis and tracking of silver, carbon adsorption tests, followed by modelling and simulation of various CIL parameters. Additional cyanide destruction, environmental, solid-liquid separation and ultra-fine grinding programs were also completed. The results and conclusions derived from this third phase of testing formed the basis of the NI 43-101 Preliminary Feasibility report generated by AMEC in 2009.

Reference is made to results from previous phases only where it is required to support the establishment of design criteria or for comparative purposes.

13.3.3 Testwork Contributors

Table 13-1 lists testwork reports received from the various testing agencies during the course of the project studies.

Mineralogical studies were conducted by AMTEL in 2007.

SGS Lakefield undertook the bulk of the testwork scope including ore comminution, gravity concentration, flotation, leaching, cyanide destruction, and environmental testwork. SGS issued five reports as tabulated below.

Three comminution specialist groups were asked to review the SGS comminution data and make recommendations pertaining to power consumption and grinding equipment sizing. These were: Contract Support Services, North American agents for JKTech; Metcom Technologies, Inc., a grinding consultancy; and DJB Consultants, Inc (DJB, 2008).

Outotec was commissioned to do pilot-scale thickening and rheology work on samples of flotation concentrate and tailings.

Ultra-fine grinding of Young-Davidson flotation concentrate samples was investigated by Metso and Deswik. Deswik provided grinding curves in October 2008 and issued a test report as an appendix to a proposal for the supply of an ultra-fine grinding mill dated October 2008 but received January 14, 2009.

Goode (2009) summarised the metallurgical testwork at the end of each of the three phases in memoranda as listed below.

Table 13-1: Listing of Metallurgical Reports

	Title	Issued
AMTEL	MINERALOGICAL CHARACTERIZATION OF YOUNG-DAVIDSON GOLD ORE for J.R. GOODE & ASSOCIATES	April 17, 2007
SGS	GRINDABILITY CHARACTERISTICS OF SAMPLES FROM THE YOUNG-DAVIDSON PROPERTY submitted by NORTHGATE MINERALS CORPORATION. Project 11497-001 – Grindability Report 1	April 18, 2007
SGS	THE RECOVERY OF GOLD AND SILVER FROM THE YOUNG-DAVIDSON DEPOSIT prepared for NORTHGATE MINERALS CORPORATION. Project 11497-002 – Final Report	March 20, 2009
SGS	THE RECOVERY OF GOLD FROM SAMPLES FROM THE YOUNG-DAVIDSON PROPERTY – PHASE I	November 30th, 2007
SGS	THE GRINDABILITY CHARACTERISTICS OF A BULK SAMPLE FROM THE YOUNG-DAVIDSON DEPOSIT	June 16th, 2008
SGS	THE GRINDABILITY CHARACTERISTICS OF A BULK SAMPLE FROM THE YOUNG-DAVIDSON DEPOSIT - APPENDICES	April, 2008
SGS	THE RECOVERY OF GOLD AND SILVER FROM THE YOUNG-DAVIDSON DEPOSIT prepared for NORTHGATE MINERALS CORPORATION. Project 11497-003 – Final Report	February 26, 2009
SGS	THE ENVIRONMENTAL & GEOTECHNICAL CHARACTERISTICS OF THE YOUNG-DAVIDSON CYANIDE DESTRUCT TAILINGS prepared for NORTHGATE MINERALS CORPORATION. Project 11497-002 – Final Environmental Report	April 22, 2009
Contract Support Services, Inc.	Young-Davidson Final Letter Summary Report - 11-01-07	November 11, 2007
Contract Support Services, Inc.	Young-Davidson – Preliminary Simulation Results – Best Fit model of PP-06 Scaled to a 22x36.5 ft AG Mill. Email transmission.	September 24, 2009
Metcom Technologies, Inc.	Young-Davidson Project Comminution Circuit Options	October 30, 2007
DJB Consultants, Inc.	Project No. D000067, Northgate Minerals Corporation, Young-Davidson Project, Pre-Feasibility Study, Assessment of Kemess Grinding Mills in Single-Stage Autogenous Grinding Mode, Final Report	January 2008
Outotec	TEST REPORT TH-0418, “High rate thickening of a flotation concentrate and tailings”	January 2008
Outotec	TEST REPORT TH-0456A “High rate thickening of flotation concentrate and tailings”	December 3-5, 2008
Outotec	INTERPRETATIONS AND RECOMMENDATIONS TEST DATA REPORT TH-0456 B “High rate thickening of flotation concentrate and tailings”	February 12, 2009
Metso	TEST PLANT REPORT NO. 77128 Report No.1	December 13, 2007
Metso	TEST PLANT REPORT NO. 77128, Report No. 2	January 30, 2008
Metso	Test Plant Report No 77128 No 3	September 3, 2008
Deswik	RFQ-0704546, Proposal for the supply of a Deswik 2,000 Mill	October 1st, 2008
Deswik	Testwork Results – grinding curves	2008-10-23
J.R. Goode and Associates	Young-Davidson Project, Interim assessment of Metallurgical Data	2007-05-16
J.R. Goode and Associates	Young-Davidson Project, Assessment of Phase II Metallurgical Data	2008-01-31
J.R. Goode and Associates	Young-Davidson Project, Assessment of Phase III Metallurgical Data	2009-04-29

13.3.4 Samples

The Phase III metallurgical test program was developed to optimize the process flowsheet and to quantify the variability in gold and silver recovery across the Mineral Resource. It was hence important to select drill core which would adequately represent both spatial as well as mineralogical dimensions of the ore body.

The phase III sample population consisted of 357 coarse assay reject samples from 32 holes selected across the known mineralization. These were combined into 32 individual hole composites, five zone composites and a master composite. Flowsheet optimization tests were conducted on the master composite. Once the metallurgical parameters were optimized, the five zone composite and 32 hole composites were used for variability testing. The zone composites were identified as:

- Lower Boundary (LB)
- Young-Davidson (YD)
- Upper Boundary (UB)
- Lucky Zone (LZ)
- Open Pit (OP)

During 2009 the geological block model zones were redefined for the more detailed 2009 mine plan. Table 13-2 provides a comparison between the old zone designations, applicable to the descriptions of testwork samples, and the new zone definitions.

Table 13-2: Zone Designations

New designation	Old designation	Percentage of Mineral Resource
A	Young-Davidson	24.9
F	Young-Davidson	2.0
B	Young-Davidson	3.0
C	Young-Davidson	0.8
H	Young-Davidson	1.2
Q	Young-Davidson	1.0
FF	Lucky	1.9
NN	Lucky	1.1
UBZ	Upper Boundary	14.8
U	Lower Boundary	36.4
Pit	Open Pit	8.4
X	No man’s land	4.5

Figure 13-3 shows the location of the samples selected from the zones deemed by Northgate as representative of the different mineralized lithologies encountered in the deposit.

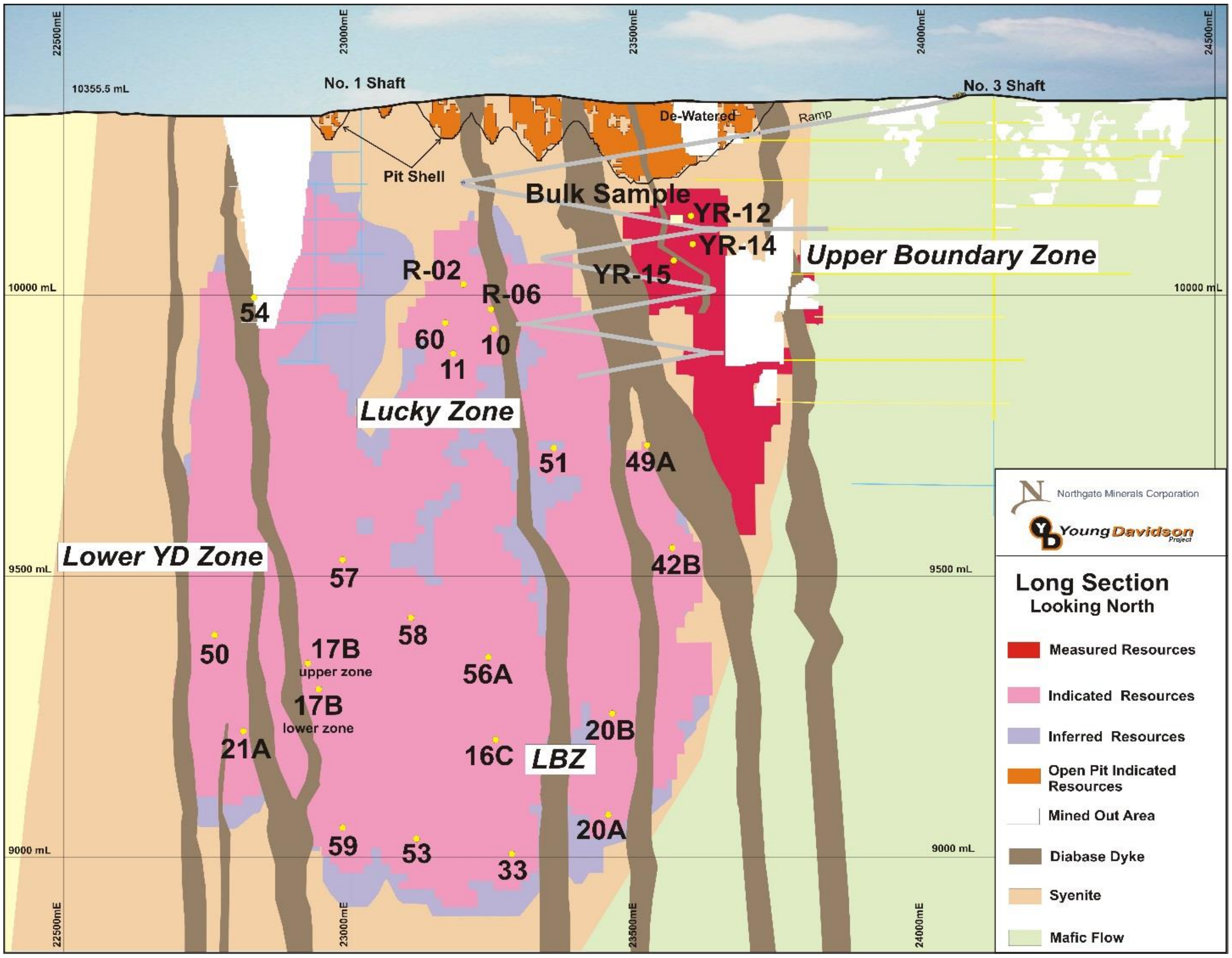


Figure 13-3: Composite Sample Locations

A summary of the principal assays, and pulp and metallics (P&M) precious metals assay results, is provided below in Table 13-3.

Table 13-3: Summary of Assays on Ore Zone Composites and Master Composite 3

Element	Unit	Sample					
		LB	UB	OP	YD	Lucky	Master Comp 3
Au (Pulp & Metallics)	g/t	3.45	4.38	1.81	3.32	3.42	2.60
Ag	g/t	2.94	2.21	7.48	3.61	2.50	3.1
S	%	1.70	1.56	1.84	2.04	1.44	1.73
S=	%	1.42	1.50	1.44	1.96	1.34	1.42
Hg	g/t	< 0.3	< 0.3	< 0.3	< 0.3	< 0.3	< 0.3
Te	g/t	<4	<4	<4	<4	<4	<4
CO_3	%	5.29	8.87	4.10	5.80	7.34	5.59
Pulp and metallics (P&M) details							
Au in +150 mesh	g/t	5.09	23.51	1.81	6.52	20.55	8.83
Au in -150 mesh	g/t	3.40	3.98	1.82	3.24	3.04	2.47
Au in +150 mesh	% total	4.2	11.1	2.9	4.8	13.1	7.0

Higher than average gold grades in the >150 mesh size range (per the Pulp and Metallics assay data) indicate the presence of coarse gold which, in turn, support the decision to include a gravity concentration step in the flowsheet. A large degree of variability with respect to the gravity recovery achieved between ore types can be expected with the proportion of total gold in this size fraction as high as 11% and 13% for the Upper Boundary and Lucky Zone material, compared to around 5% for the other ore types. In fact, the presence of significant quantities of coarse gold implies that variable head grades would be a commonplace occurrence during operations.

SGS also prepared a 50 kg Master Composite 4 to allow generation of flotation concentrate and tailings samples which was shipped to Outotec for pilot thickening work. Master Composite 4 consisted of 20 kg of Master Composite 3 combined with 30 kg of Zone and Hole Composites. Master Composite 4 composition is provided in Table 13-4.

Table 13-4: Composition of Master Composite 4

Zone	Sample	Mass, kg
Lower Boundary	Zone Comp	4
	YD07-53	2
	YD06-16C	1
	YD07-42B	2
	YD07-49A	2
Young-Davidson	Zone Comp	4
	YD07-54	2
	YD07-57	1
Upper Boundary	Zone Comp	3
Lucky	Zone Comp	2
	YD08-60	1
Open Pit	Zone Comp	4
	OP-33	2
Overall	Master Composite 3	20

SGS ran pilot-plant scale AG/SAG and flotation trials in April 2008. Flotation concentrate slurry samples from the pilot plant runs were stored in a refrigerated facility for future use. In July 2008, the samples were blended, as indicated in Table 13-5, to prepare a composite for ultra-fine grind testing by Metso and Deswik.

Table 13-5: Concentrate Composite Sample Used in Ultra-fine Grinding Tests

Sample	Estimated dry mass, kg	K80, µm
PP2A Comb Rougher Conc.	16.9	99
PP2B Comb Rougher Conc.	10.2	103
PP3 Comb Rougher Conc.	33.3	68

13.3.5 Comminution

During Phase I, SGS performed SAG Mill Comminution (SMC) tests, Bond rod mill work index (RWi) tests, Bond ball mill work index (BWi) tests (using 100 and 150 mesh closing screens), and abrasion index (Ai) tests on all six Phase I ore zone samples and two waste samples.

An average Drop Weight Index (DWi) of 7 kWh/m^3 was recorded for all of the samples, with the exception of waste sample YD07-31A (diabase), which yielded a DWi of 11.3 kWh/m^3. These DWi values are related to the energy consumed during impact crushing and hence indicators of the power consumption required for crushing. A DWi value of 7 corresponds to the 75th percentile of hardness in the JKTech database, placing the Young-Davidson orebody in the medium to hard category. The diabase waste rock is in the 95th percentile of hardness and is hence classified as very hard. Table 13-6 summarizes the SMC data.

Table 13-6: Grinding Parameters Determination

Sample Name	Au	Ore SG	SMC Parameters			RWI	BWI, kWh/t		Ai, g
	g/t	g/cm³	A	b	A x b	kWh/t	100 mesh	150 mesh	
Lucky Zone	4.21	2.67	100	0.43	43.0	15.0	16.4	-	0.675
Lower Boundary	5.03	2.69	85.6	0.55	47.1	14.0	15.3	16.1	0.506
Upper Boundary	2.67	2.68	90.2	0.41	37.0	17.5	18.3	17.9	0.690
Pit #6	1.04	2.71	88.5	0.47	41.6	15.5	16.5	-	0.819
YD Zone	3.36	2.69	94.4	0.49	46.2	12.7	14.7	15.8	0.540
Pit Zone	3.13	2.67	96.1	0.38	36.5	16.3	17.7	-	0.687
Waste (YD06-26)	0.02	2.71	86.6	0.45	39.0	15.7	17.5	-	0.696
Waste (YD07-31A)	<0.02	3.02	100	0.27	27.0	21.4	23.0	-	0.395

The SMC results indicate a medium to medium-hard range of hardness for the ore samples and one of the waste samples. However, the second waste sample, YD07-31A, is characterised by a significantly lower JKTech b-value of 0.27 which places this rock in the very hard range (at an A x b < 37). The RWi data are indicative of medium hard ore and waste with the exception, again, of the diabase waste which was very resistant to breakage.

The BWi data depicts hard to very hard ore and waste (average index value of 16.6 kWh/t) except for the diabase waste, which, again, had a very high BWI of 23 kWh/t. Tests on three of the ore samples at closing screens of 100 and 150 mesh showed little dependence between closing screen size and BWi.

Figure 13-4 below shows how the Young-Davidson ore BWi values compare to that of the JKTech worldwide database. The figure also illustrates how much harder the waste sample is than the other ore types (per the lone arrow on the right side of the curve, at a BWi of 23 kWh/t).

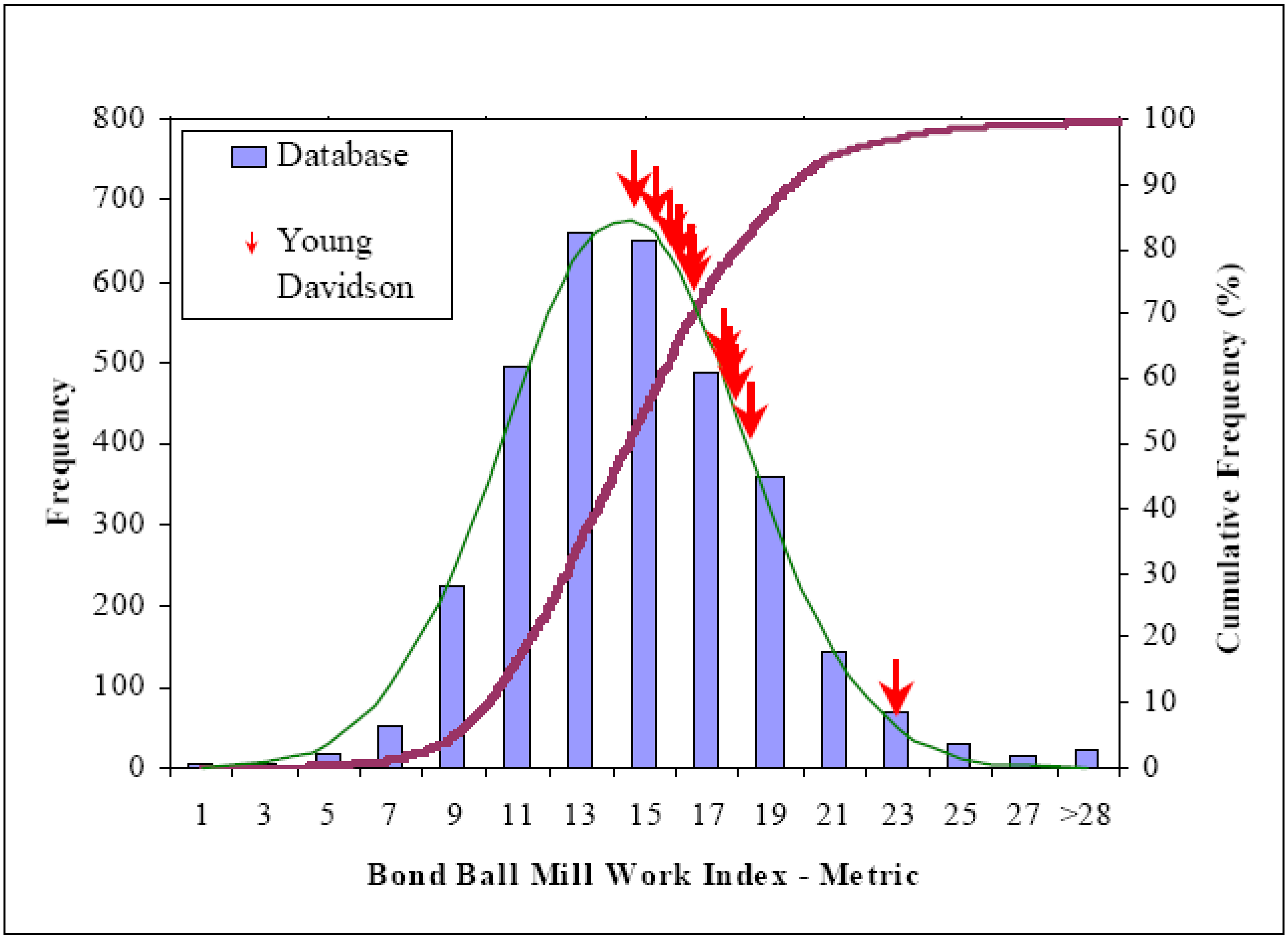


Figure 13-4: Comparison of Young-Davidson Ore BWi against JK Database

The abrasion index (Ai) values for all samples, except the diabase waste, averaged 0.66 g, which classifies these ore types as very abrasive. The diabase waste was the softest of all of the samples tested at an Ai of 0.4 g. Figure 13-5 demonstrates graphically how these ore types compare to other ore types in the JKTech database with respect to abrasiveness. The abrasiveness of all of the ore types tested fall in the top 20% and most in the top 10% of ores tested by JKTech.

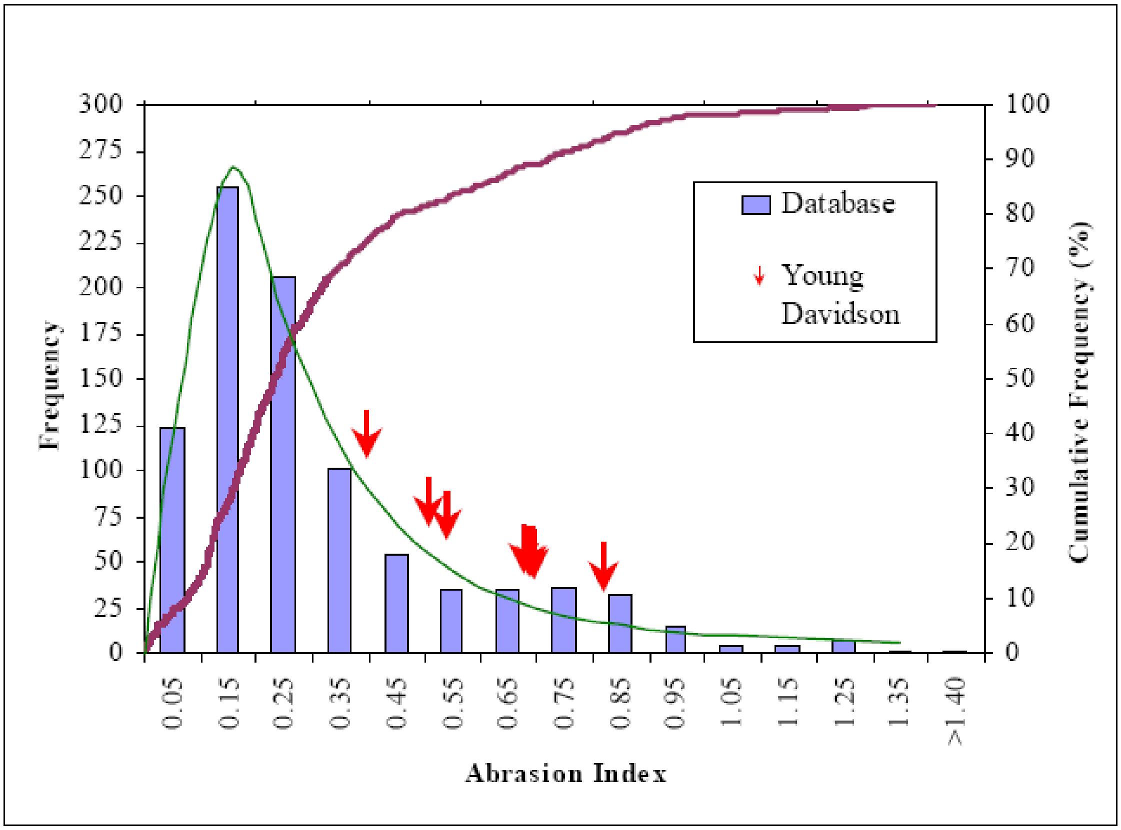


Figure 13-5: Comparison of Abrasion Index Values

The detailed comminution data summarized above were provided to Contract Support Services, Inc. (CSS); DJB Consultants, Inc.; and Metcom Technologies, Inc. with a request to advise on the grinding requirements of the Young-Davidson project. As part of their scope, the consultants were to consider either an ABC circuit (autogenous mill with pebble crushing followed with ball milling), SABC circuit (semi-autogenous mill with pebble crushing followed with ball milling) and the potential suitability of any of the grinding mills that could be made available to the Project from Northgate's Kemess Mine, in British Columbia.

Pilot planting of a bulk ore sample was then completed, at the onset of the Phase III testwork program, mostly as a means to confirm whether or not the Young-Davidson ore would be amenable to autogenous (AG) milling. This approach recognized the uncertainty associated with establishing the power requirements associated with such a circuit solely on the basis of grinding parameters established through standardized laboratory procedures.

Grinding Circuit Pilot Plant

A pilot plant investigation into the grinding characteristics of the Young-Davidson deposit was conducted in April 2008 on a bulk sample collected from the UBZ. Several single-stage grinding configurations were evaluated including autogenous, semi-autogenous and autogenous with pebble crushing.

The pilot plant feed sample was subjected to drop-weight tests to determine the breakage characteristics of this sample. Results include a t_a value of 0.2, SG of 2.70 and an A x b value of 35.1. These values compare well with the SG of 2.68 and A x b of 37 recorded in Table 13-6 for this Upper Boundary zone material.

At this A x b value, the pilot plant sample was shown to be harder than any other material in this orebody, with the exception of the diabase waste. The implication is that the sample was a good choice as the results achieved represent the worst case expectations for grinding performance.

Table 13-7 summarizes the pilot plant operational conditions and results achieved for the SAG and fully AG test runs.

Table 13-7: Pilot Plant Results

Test Type	Load (% v/v)	Speed (%Cs)	Recycle (%)	F80 (mm)	P80 (mm)	Wio (kWh/t)
SAG	30	75	56	96.3	0.129	18.2
AG	33	74	24	96.3	0.095	20.2

The SAG test was conducted with a 4% v/v ball charge.

Achievement of low recycle ratios suggested that a single-stage grinding step should be sufficient to achieve the planned reduction ratio.

The pilot plant results confirmed that the material was amenable to fully autogenous and semi-autogenous grinding. Based on the pilot plant data a single-stage autogenous grinding circuit was selected. Operating work indices in the range of 16-18 kWh/t were expected for the slightly softer average rock type and coarser full-scale target grind of 0.15 mm.

Confirmation of the power requirements for the various major ore blends expected through the mine life was carried out using the JKSimMet grinding simulation package. The detailed data obtained from the pure AG milling circuit tested during the pilot plant trials were used to calibrate said simulations.

13.3.6 Gravity Concentration

Gravity concentration of coarse gold, followed by intensive cyanidation, can lead to improved overall gold recoveries as it allows longer leach times under intensive conditions to ensure complete dissolution of coarse gold particles. High gravity recoveries also reduce overall gold inventories and improves product security as this gold component is removed from the process at an early stage.

Low mass-pull gravity concentration tests were conducted during all three testwork phases to assess the amenability of the Young-Davidson deposit to this processing route. Results from gravity concentration testing are typically highly variable due to the variance in coarse gold distribution in an orebody and the very small mass of feed sample and concentrate produced. This was no different for the Young-Davidson work. As a result of this variability, the results of all three phases, totalling 70 tests, were considered en-masse in the evaluation.

The testwork procedure involved processing ground ore through a 3” Knelson centrifugal concentrator followed by upgrading of the nominally 100 g of Knelson concentrate on a Mozley Mineral Concentrator shaking table to yield an overall mass pull of approximately 0.05%.

Tests were generally done on 2 kg batches. The only exception is the preparation of 50 kg of flotation feed to prepare samples for the Phase III Outotec thickening work. This gravity separation work was done on Master Composite 4, in five 10 kg batches.

The results of all gravity concentration tests are summarized in Table 13-8 and Figures 13-6 to 13-7 presented.

Table 13-8: Summary of Low Mass Pull Gravity Concentration Work

Material tested	*Data*	*No. tests*	*Head grade, g/t*		*Mass pull*	*Concentrate grade, g/t*		*Distribution, %*	
			Au	*Ag*	*%*	*Au*	*Ag*	*Au*	*Ag*
Open Pit	**All phases, weighted av.**	**10**	**1.39**	**3.80**	**0.088**	**254**	**120**	**16.2**	**2.8**
	Phase III holes, weighted av.	7	1.20	4.15	0.066	169	135	9.3	2.2
	Phase III zone composite	1	1.71	6.48	0.344	99	70	19.8	3.7
Upper Boundary	**All phases, weighted av.**	**7**	**3.41**	**2.65**	**0.045**	**2,383**	**431**	**31.6**	**7.4**
	Phase III holes, weighted av.	3	3.85	2.81	0.044	3,112	506	35.2	7.8
	Phase III zone composite	1	3.77	2.37	0.065	2,056	287	35.7	7.9
Lower Boundary	**All phases, weighted av.**	**13**	**4.11**	**5.30**	**0.028**	**3,060**	**1,744**	**21.0**	**9.3**
	Phase III holes, weighted av.	8	3.51	5.22	0.021	3,928	2,253	23.5	9.1
	Phase III zone composite	1	3.06	1.03	0.031	2,596	758	26.1	22.6
Lucky Zone	**All phases, weighted av.**	**13**	**4.43**	**2.81**	**0.047**	**3,678**	**876**	**38.9**	**14.6**
	Phase III holes, weighted av.	7	3.60	2.81	0.017	6,478	1,489	31.1	9.1
	Phase III zone composite	1	13.02	4.20	0.237	4,803	1,026	85.9	56.3
Young-Davidson	**All phases, weighted av.**	**13**	**3.34**	**3.10**	**0.075**	**1,114**	**526**	**24.9**	**12.7**
	Phase III holes, weighted av.	7	3.58	3.35	0.095	1,106	468	29.2	13.2
	Phase III zone composite	1	2.59	2.34	0.111	366	51	15.7	2.4
Composite of all zones	**All phases, weighted av.**	**14**	**3.28**	**3.75**	**0.126**	**508**	**262**	**19.5**	**8.8**
	Phase III zone composite	5	2.85	3.50	0.053	846	265	15.4	3.9
All gravity tests	**All phases and samples, weighted**	**70**	**3.40**	**3.64**	**0.07**	**1,287**	**487**	**26.5**	**9.9**

Results of gold recovery through gravity concentration are presented in Figure 13-6. This graph indicates that gravity recovery of gold will be relatively low during the initial operations phase, due to the lower-than-average recoveries expected for the open pit zone. Both the Upper Boundary and Lucky Zones will yield gravity gold recoveries in excess of 30%, with the Lucky Zone expected to yield very high and also exceedingly variable recoveries and grades. The most reliable estimates of gravity gold recovery are those including the greatest number of samples which would be the "All Phases, weighted average" data, which includes data from Phases I, II, and III and places the expected average gravity gold recovery at around 25%.

Silver recoveries, shown in Figure 13-7, are substantially lower than gold recoveries.

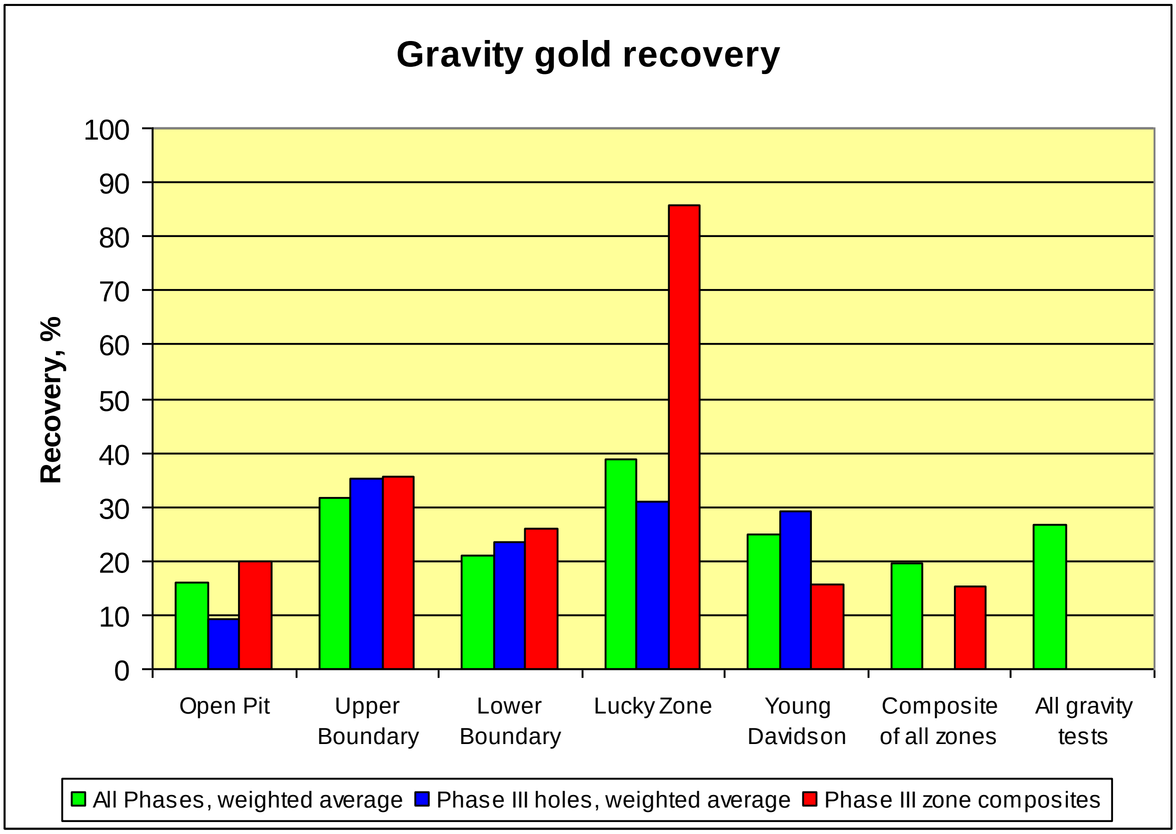


Figure 13-6: Gravity Recovery of Gold by Ore Zone

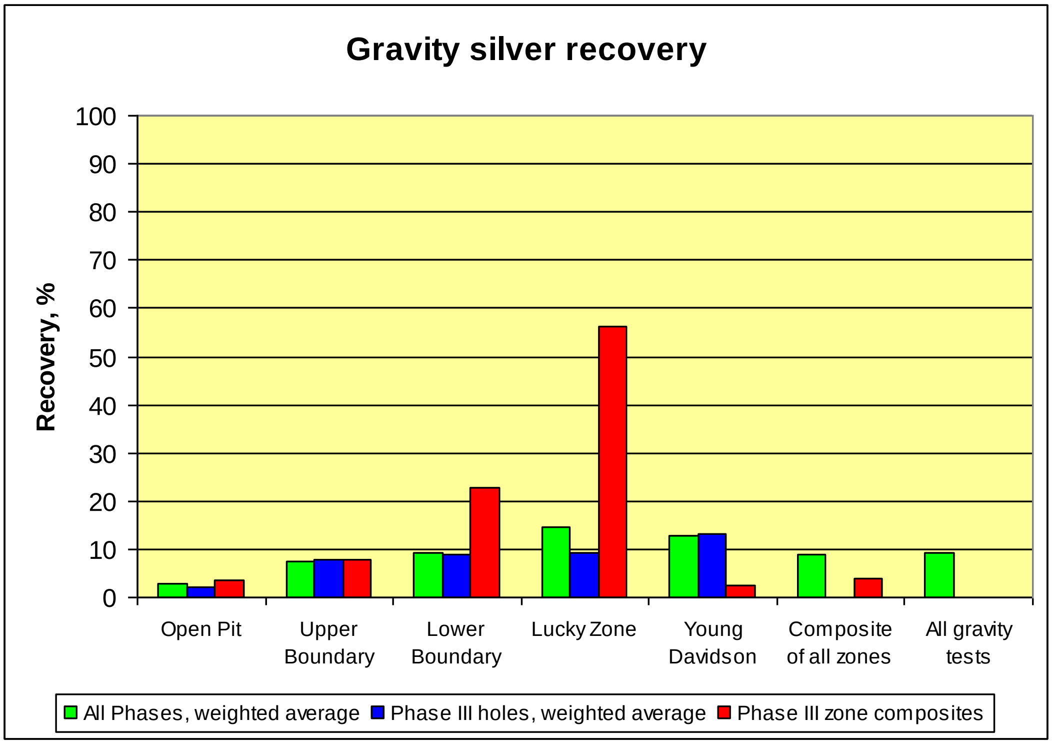


Figure 13-7: Gravity Recovery of Silver by Ore Zone

Figure 13-8 plots the silver versus gold grades of all gravity concentrates produced for all of the tests conducted. The proportion of silver as a percentage of the total gold and silver content of all the concentrates averages 27.4% - probably reflecting an average composition of the electrum in the ore. In the concentrates made from composites, 34% of the precious metal was silver. In tests on material from the individual ore zones, silver varied from 15% of total precious metals in the Upper Boundary to 36% for the Lower Boundary material.

Figure 13-8 also illustrates the degree of variability in concentrate grades with the highest and lowest grades several orders of magnitude apart for both metals. Consequently, monitoring and control will clearly be important in order to maintain high efficiencies in downstream processes.

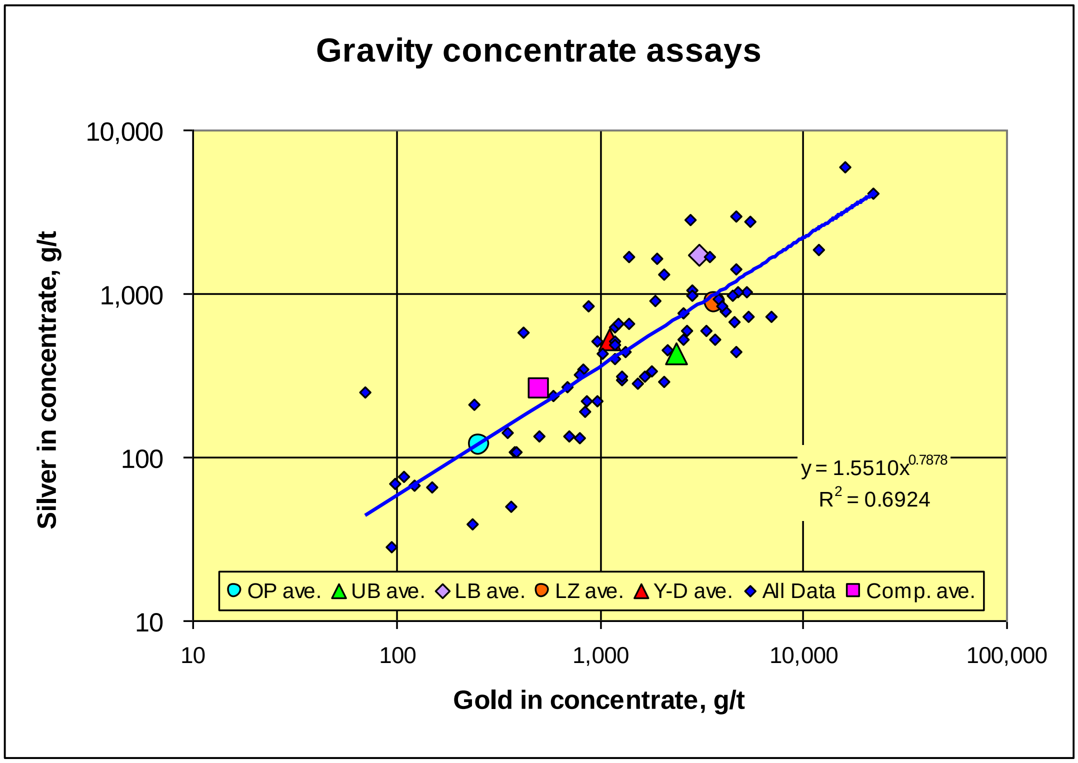


Figure 13-8: Gold and Silver Assays on Gravity Concentrates

The relationship between the distribution of gold between the +150 and -150 mesh fractions in the Pulp and Metallics assay procedure and the recovery of gold during gravity recovery tests from the Zone Composites is shown in Figure 13-9. A positive correlation does exist, as expected, but the number of data points is insufficient to draw a firm conclusion regarding the robustness of such correlation.

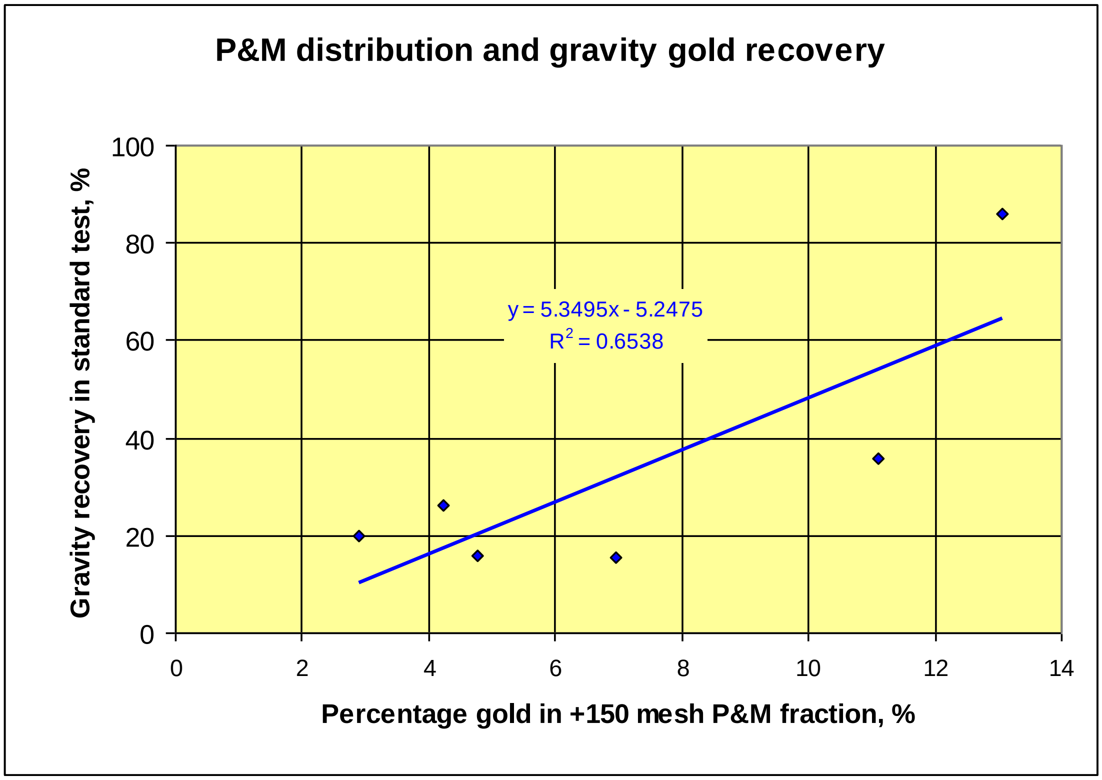


Figure 13-9: Relationship between P&M Assay and Gravity Recovery

Further gravity separation tests were performed to investigate the effect of fineness of grind on gravity recovery. Tests were conducted on 30 kg samples at two K80 sizes. For the first grind a K80 of 150 µm was targeted, while 100 µm was used for the second test. These gravity separation results are summarized in Table 13-9. As expected, the recovery of gold by gravity separation from Master Composite 3 was higher at the finer grind used in test G2.

Table 13-9: Gravity Separation Results for Master Composite 3

Test	Feed	Grind	Gravity Separation Concentrate					Head (calc)	
	Wt	K_{80}	Wt	Assay, g/t		% Recovery		Assay, g/t	
No.	kg	µm	%	Au	Ag	Au	Ag	Au	Ag
G2	20	90	0.07	1186	399	27.4	8.1	3.10	3.5
G3	30	140	0.06	864	220	18.2	4.0	3.00	3.5
F20	50	145	0.05	586	236	11.9	3.5	2.57	3.5
F20B	30	145	0.03	1275	295	13.0	2.6	3.21	3.7
F20En	20	145	0.05	351	141	7.7	2.3	2.36	3.3

In conclusion, test results show that gravity recovery using a centrifugal concentrator in the grinding circuit will remove a significant amount of gold from the mill product stream. At least 20% of the head gold content is expected to report to the gravity concentrate at a mass pull of approximately 0.05%. This lower-than-average value was adopted by AMEC as design criteria for the metallurgical balance as it yields a conservative design for the remainder of the gold recovery steps, from flotation to CIL. However, both gravity recovery as well as mass pull could be significantly higher for some ore zones. The gravity circuit could thus be operated with a variable operating period between concentrate removal, or include incremental units to accommodate these higher grades and concentrate production rates with some of the specific ore types, in order to maximise the opportunity presented by gravity concentration to improve overall gold recovery.

13.3.7 Intensive Cyanidation

Intensive cyanidation tests were performed by SGS using, generally, between 10 and 30 grams of gravity concentrate and leaching parameters of 5% solids in 20 g/L NaCN solution with 500 g/t lead nitrate and elevated dissolved oxygen levels (20 to 25 mg/L) for a duration of 24 hours. Intensive cyanidation leach data are presented in Table 13-10.

Table 13-10: Results of Intensive Cyanidation of Gravity Concentrates

Phase	*Test*	*Ore sample*	*Gravity test*	*Feed assays, g/t*		*Residue assays g/t*		*Extraction %*	
				Au	*Ag*	*Au*	*Ag*	*Au*	*Ag*
II	CN-1	MC2	GS1-3	1,173	512	16	224	98.6	56.3
II	CN-2	MC3	GS4-5	1,974	692	20	266	99.0	61.6
III	CN-2	MC3	G2	1,186	399	17	123	98.6	69.2
III	CN-3	MC3	G3	864	220	24	61	97.2	72.3
III	CN-4	MC3	F20	586	236	4	70	99.4	70.3
III	CN-5	MC3	F20En	351	141	20	27	94.2	80.7
III	CN-6	MC3	F20b	1,275	295	18	60	98.6	79.6
III	CN144	MC4	F58	815	293	19	112	97.7	61.8
Averages				**1,028**	**349**	**17**	**118**	**98.3**	**66.2**

Table 13-10 indicates that gold extraction as high as 98% is achievable, while more modest silver extractions of around 67% can be expected. Figure 13-10, below, shows that gold leaches to a constant residue grade of approximately 17 g/t, irrespective of the gold grade. This constant residue grade was adopted in the design criteria for this section of the plant. However, the silver tails assay is proportional to the grade of the feed to the intensive cyanidation process. This suggests that the observed variance in gravity concentrate gold grade is entirely due to fully liberated gold particles while free silver is recovered in association with refractory forms or possibly only partially liberated. Either way the test results are in agreement with expectations based on the mineralogical analysis by AMTEL, lending further credibility to the results.

Figure 13-10 is indicative of the influence of the gravity concentrate feed grade over the expected intensive cyanidation (IC) tails grade, with no correlation for gold but a very strong one for silver. Figure 13-11 and Figure 13-12 indicate fast kinetics for both gold and silver leaching from gravity concentrates. A residence time of approximately four hours appears sufficient.

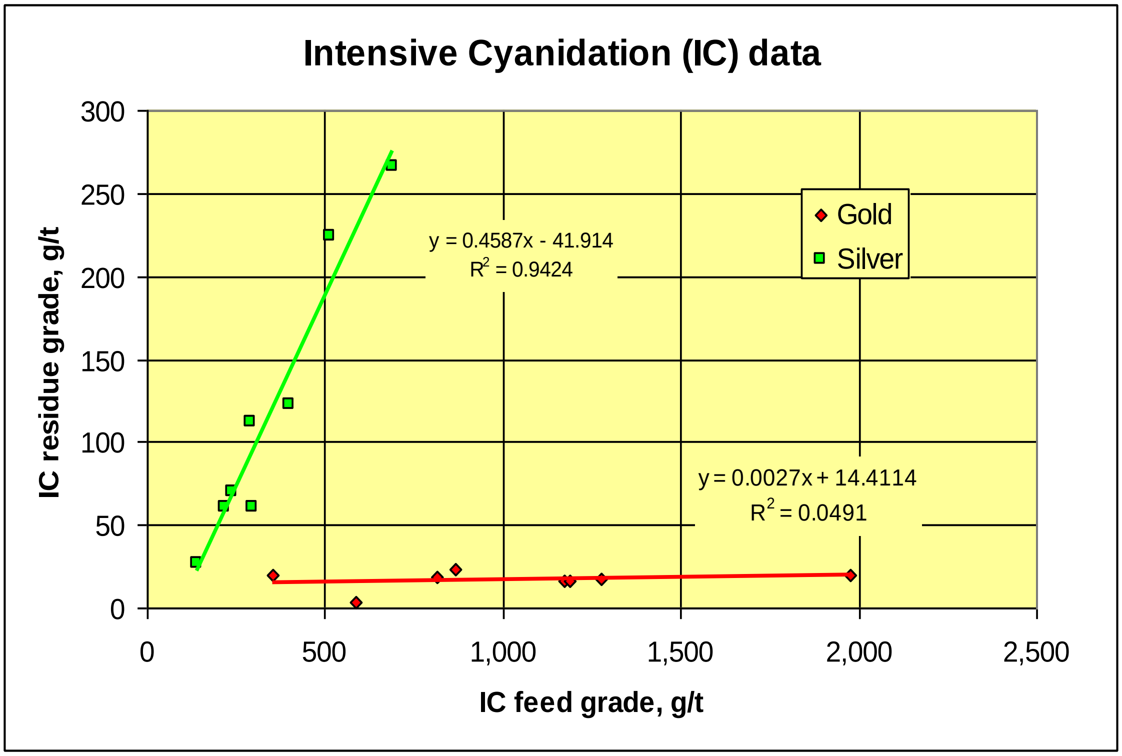


Figure 13-10: Tails Assay at 24 Hours as a Function of IC Feed Grade

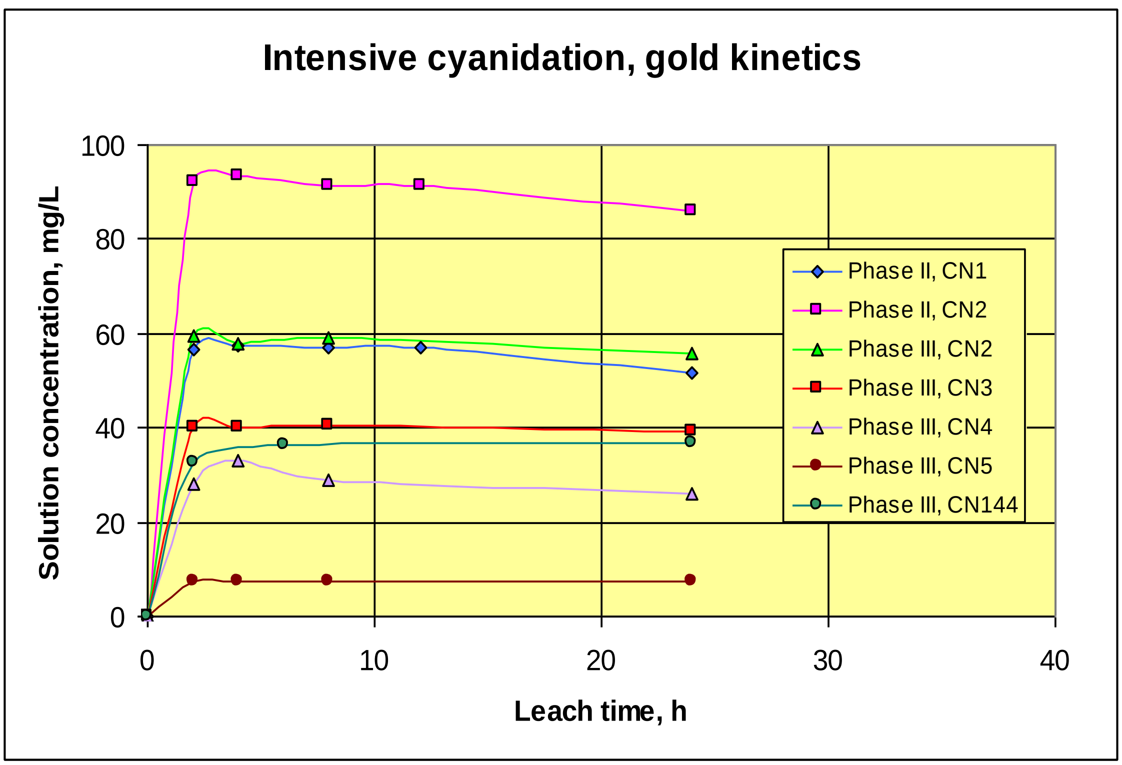


Figure 13-11: Gold Leach Kinetics during Intensive Cyanidation

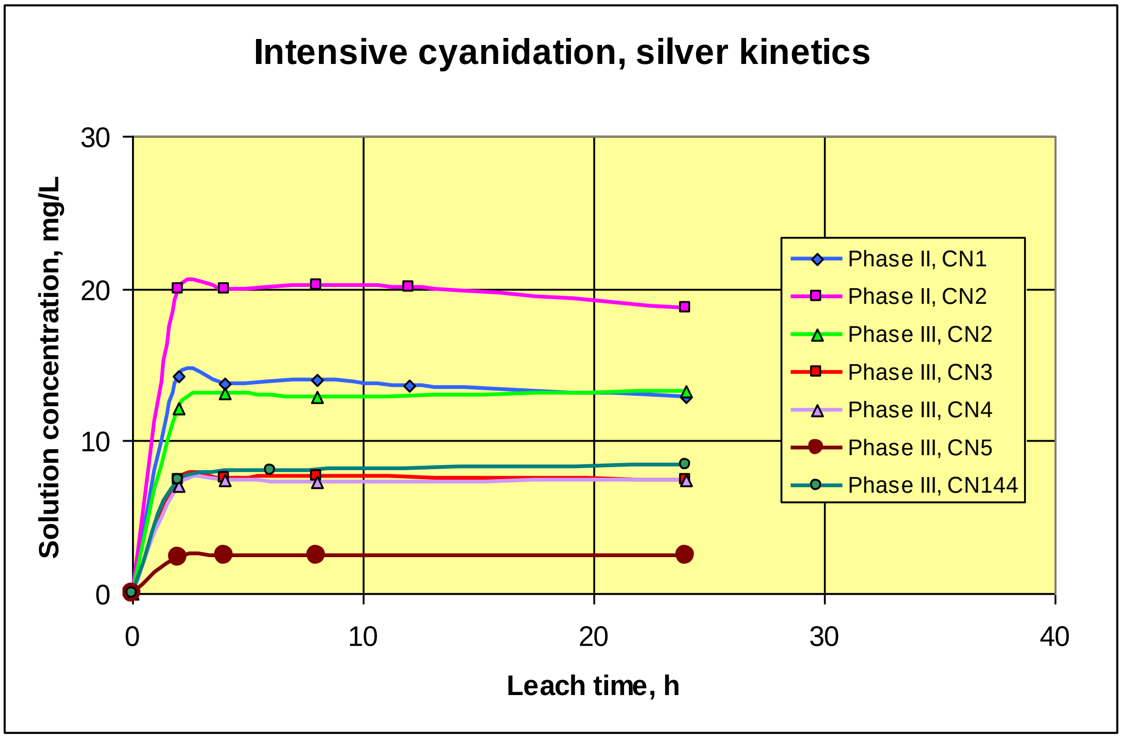


Figure 13-12: Silver Leach Kinetics during Intensive Cyanidation

Figure 13-11 and Figure 13-12 both suggest the presence of some preg-robbing constituents, or re-precipitation of dissolved metals because the concentration of metals in solution decreased after the initial rapid rate of gold dissolution. This is not considered to be a problem as the residue from this step is eventually leached in the CIL step which would negate the effect of any preg-robbing.

In the proposed Young-Davidson plant flowsheet, intensive cyanidation tailings reports to the ultra-fine grinding circuit where it is combined with the flotation concentrate for further liberation ahead of the second leaching step. Tests to measure the additional recovery due to regrinding of the gravity concentrate were performed during Phase III.

A portion of the intensive cyanidation residue from test CN144 was ground to pass a 44 µm screen and then leached in Test CN145 for 48 hours, with 1 g/L NaCN. As the eventual regrind size is significantly finer than this grind, the results obtained here represent a worst-case scenario. This test recovered 78% of the gold from the intensive cyanidation tailings, which improved gold recovery from 97.7% after intensive cyanidation to 99.3% after leaching. Overall silver recovery was not improved and remained at around 66%.

One cyanidation test, CN143, was performed to evaluate whether the intensive cyanidation step can be omitted from the flowsheet, i.e. if it is viable to simply add the gravity concentrate to the flotation concentrate for regrind followed by moderately intense cyanidation.

Test CN143 yielded a residue containing 65 g/t Au and 82 g/t Ag representing just 88% gold extraction and 67% silver extraction. Clearly, the intensive cyanidation process is essential for processing gravity concentrates.

The outcome of Test CN143 affirms that gravity concentration with intensive cyanidation must be a part of the Young-Davidson flowsheet to ensure high recoveries. In this specific test, the gravity concentrate contained 15% of the head feed gold and the indication is that 99% of this would be recovered, or 14.85% of the gold in the feed. Without gravity concentration followed by intensive cyanidation, recovery was 13.2% of the gold in the feed – a loss of gold exceeding 1%.

Table 13-11 below presents the results of intensive cyanidations at different grinds. The concentrate from feed test G2 was produced from a 90 micron primary grind, while the other tests below were performed at a primary grind of 145 microns. Comparison of the residue grades and extractions achieved suggest that the primary grind size does not appear to play an important role prior to intensive cyanidation. More tests, possibly also over a wider range of primary grinds (say 60 to 200 microns), should be conducted to confirm this preliminary conclusion.

Table 13-11: Intensive Cyanidation Results

Test	Feed	Extraction		Residue Assay		CN Feed (calc)	
	Test	%		g/t		g/t	
No	No	Au	Ag	Au	Ag	Au	Ag
CN2	G2	98.6	69.2	16.8	123	1186	399
CN3	G3	97.2	72.3	23.8	61.0	864	220
CN4	F20	99.4	70.3	3.60	70.0	586	236
CN5	F21	94.2	80.7	20.4	27.2	351	141
CN6	F20B	98.6	79.6	17.5	60.3	1275	295

For design purposes an average intensive cyanidation residue gold grade of 17 g/t is assumed and an IC silver extraction of 70%. Cyanide, leach residence time and the addition of an oxidant is also included in the design criteria but it is assumed that a significantly higher pulp density can be used as the 5% w/w solids used during testwork appears exceedingly conservative. Such a low density would dilute the IC pregnant solution unnecessarily, thus affecting both the sizing as well as the efficiency of the downstream electrowinning step.

13.3.8 Flotation

Flotation parameters were investigated in detail during Phases I and II of the testwork campaign. This earlier work established that a simple open-circuit, high mass pull flotation system followed by concentrate regrinding and cyanidation, completed with cyanidation of the flotation tailings is the preferred flowsheet. The selected reagent suite, addition rate and laboratory conditioning and flotation times, as decided upon at the conclusion of Phase II, are listed below. This set of parameters was used in all subsequent (Phase III) testwork.

A407	: 49 g/t
PAX	: 102 g/t
MIBC	: 28 g/t
Conditioning time	: 4 minutes (spread between float campaigns)
Flotation time	: 35 minutes

The effects of the primary grind target and of the flotation stage mass pull on the overall recovery of gold and silver required further testing during Phase III. For this phase, both the flotation concentrate and tailings were leached to establish overall recoveries.

Master Composite 3, containing 2.6 g/t gold and 3.1 g/t silver, was used for a series of gravity-flotation-cyanidation tests to determine the impact of the primary grind target (nominally 80% passing 90 and 150 µm), flotation mass pull (nominally 15% but ranging from about 10 to 17%), and cyanidation conditions on overall recovery.

Figure 13-13 below, shows the effect of primary grind and mass pull on the gold and silver content of flotation tailings. It is evident that the gold content of the flotation tailings was not affected by either primary grind or mass pull across the range investigated. Silver in tailings is slightly reduced as the mass pull is increased but remains unaffected by the fineness of the primary grind.

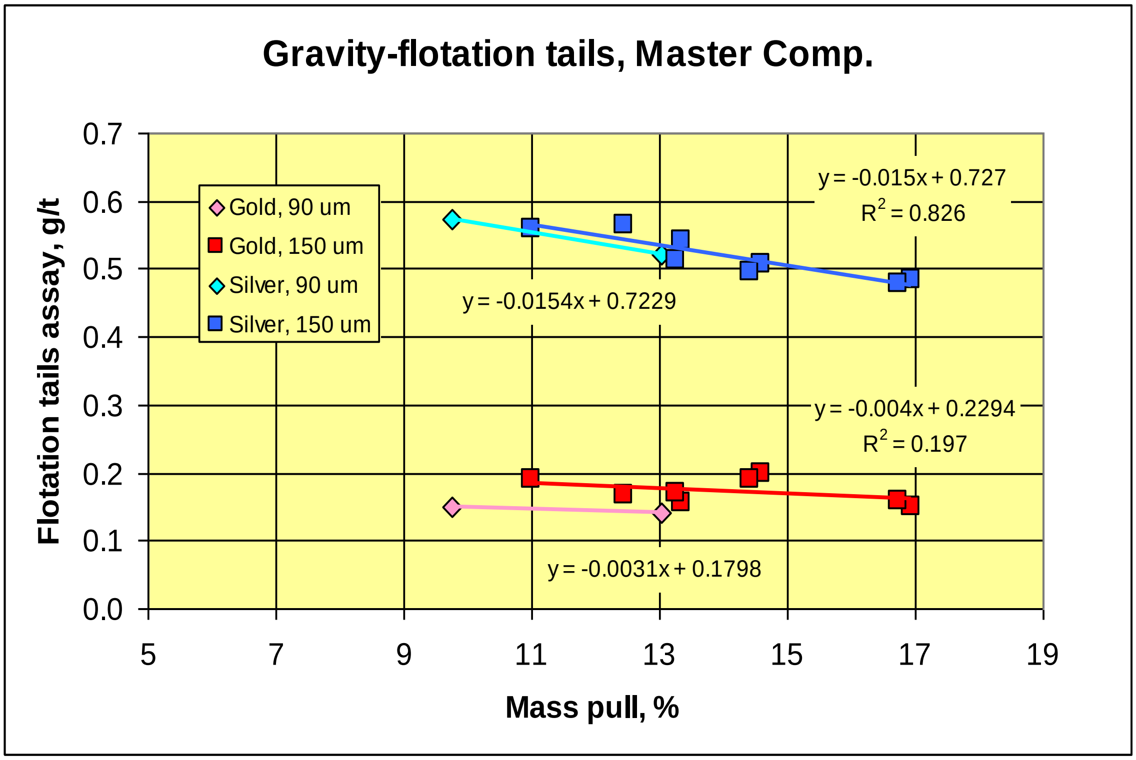


Figure 13-13: Effect of Grind and Mass Pull on Flotation Tailings Grades

It was concluded from these results that the overall gold and silver recoveries will be somewhat insensitive to the operation of the grinding circuit, within the range of P80 tested. This result suggests that some operating flexibility around the grinding circuit could be tolerated and indicates the potential for power cost savings. Alternatively, it can imply that the grinding circuit can be operated at a constant throughput rate with the resulting grind size allowed to fluctuate over a wide range, simplifying the operation of this circuit considerably. Trials with a flotation feed exhibiting a coarser P80 than the 150 µm tested could be performed to establish the limiting size over which recovery losses are not compensated by lower operating costs.

The insensitivity towards flotation mass pull suggested that the operation of the flotation plant will also be somewhat forgiving and that a lower mass pull could be adopted for seeking a finer regrinding circuit product size than adopted in the design criteria, if shown to be beneficial to the overall circuit recovery potential.

13.3.9 Flotation Concentrate Regrinding

Phases I and II showed that regrinding of the flotation concentrate yielded higher metal extractions during subsequent cyanidation. During Phase III, further tests were conducted to evaluate how fine the concentrate needed to be ground, for optimum overall extraction, and established the power input required to reach such targets.

Each concentrate from the gravity-flotation series was split into two portions, which were reground to two different particle sizes (nominally 80% passing either 15 or 30 µm) and then leached.

Figure 6-14 below, presents the resulting metals recovery achieved versus regrind product size used. Gold liberation and subsequent extraction is clearly improved with finer regrinding, with the recovery improving from around 86% to 93% as the grind P80 decreased from 34 to 15 microns. However, silver extraction is not affected by finer grinding of the concentrate.

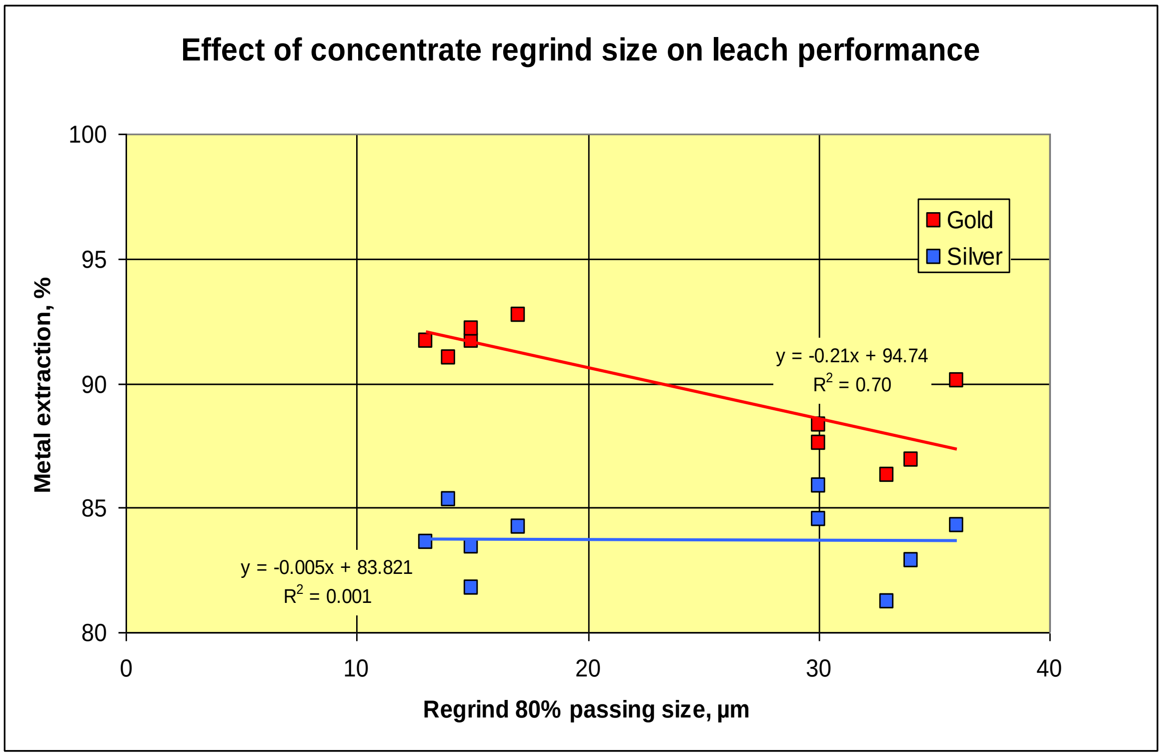


Figure 13-14: Metal Extraction at Different Regrind Sizes

Samples were submitted to vendors to conduct testwork aimed at determining grinding power requirements and to measure the breakage characteristics required to select suitable equipment for this duty.

Three samples were provided to Metso as listed below. Deswik, a South African supplier of ultra-fine grinding mills, was also provided with a sample. The sample origins are as follows:

- Sample 1 was a flotation concentrate produced from Master Composite 2 in test GS4-5 and F4-5 of Phase II. The concentrate mass pull was about 19% and the rougher tails had an F80 of about 130 µm, as measured using a Malvern. 2.5 kg were shipped to Metso, arriving in mid-December 2007, and the balance of about 2.5 kg was tested by Outotec. Metso reported a feed size of 80% passing 102 µm as determined by Malvern size analyzer.

- Sample 2 was another flotation concentrate produced from Master Composite 2 in test GS6-7 and F6-7 of Phase II. The concentrate mass pull was about 21% and the rougher tails had a P80 of about 129 µm, measured with a Malvern. Concentrate samples were shipped to Metso, arriving at the end of January 2008, and grinding tests were done on one of the samples, weighing about 3.5 kg. Metso reported a feed size of 80% passing 76 µm.
- Sample 3 originated from the AG/SAG pilot plant campaign, using a bulk sample of Young-Davidson ore, with which subsequent flotation pilot plant trials were completed. Concentrates from pilot plant runs PP2A (17 kg), PP2B (10 kg), and PP3 (33 kg) had been stored underwater in a refrigerated room. The mass pull in these tests was in the 11 to 16% range. The three flotation concentrates were blended together, 30 kg shipped to Deswik, in South Africa, and 10 kg sent to Metso. Metso reported a feed size of 80% passing 128 µm, Deswik reported 194 µm – all using Malvern (e.g. volumetric distribution) size analysis. SGS had reported sizes corresponding to about 80% passing 100 µm but this was likely a screen (e.g. weight distribution) measurement.

Metso completed a jar mill test for each sample, using a 200 mm diameter by 250 mm high cylinder loaded with 19 mm steel balls. This test generates data required for the sizing of a vertical mill, or VertiMill (VTM). Metso also conducted Sand Media Detritor (SMD) tests, using a scale model Detritor, of a 2 L capacity, loaded with 3 mm diameter ceramic media. A SMD can be used to generate a very fine product from a VTM product.

The results for the jar mill and SMD tests are presented in Figures 13-15 and 13-16.

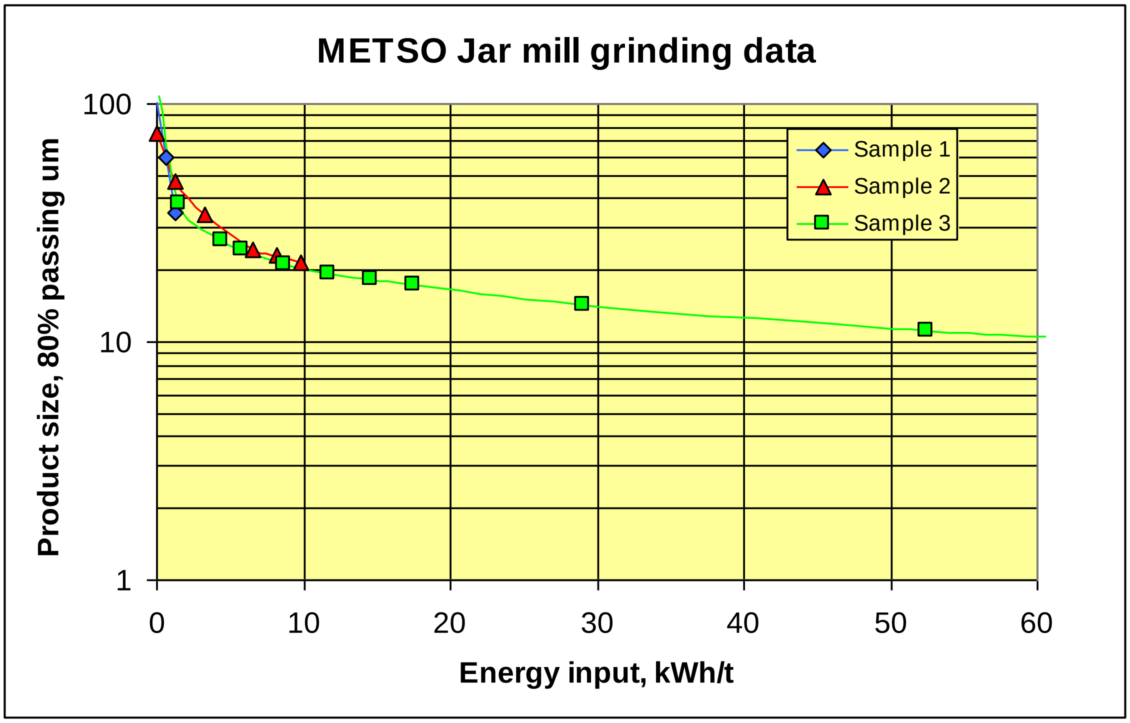


Figure 13-15: Metso Jar Mill Results

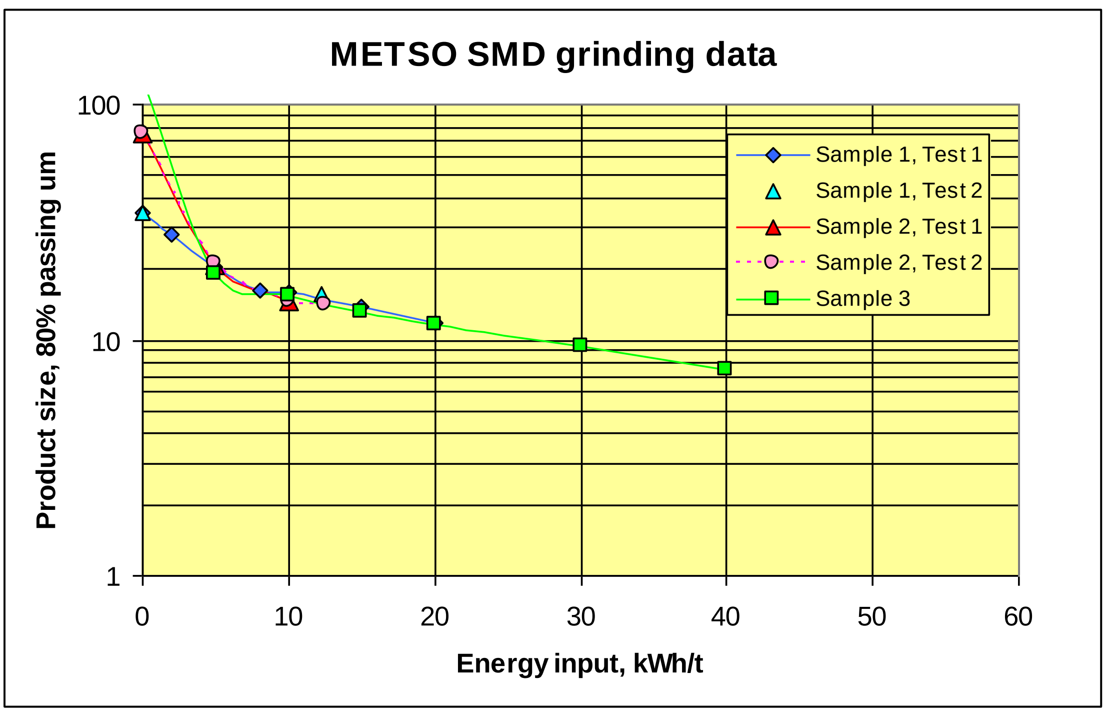


Figure 13-16: Metso SMD Results

Metso provided estimates of the energy required to grind the flotation concentrate from 80% passing 128 µm to various finished sizes as shown in Figure 13-17 below. The recommendations mainly stemmed from the work done on Sample 3.

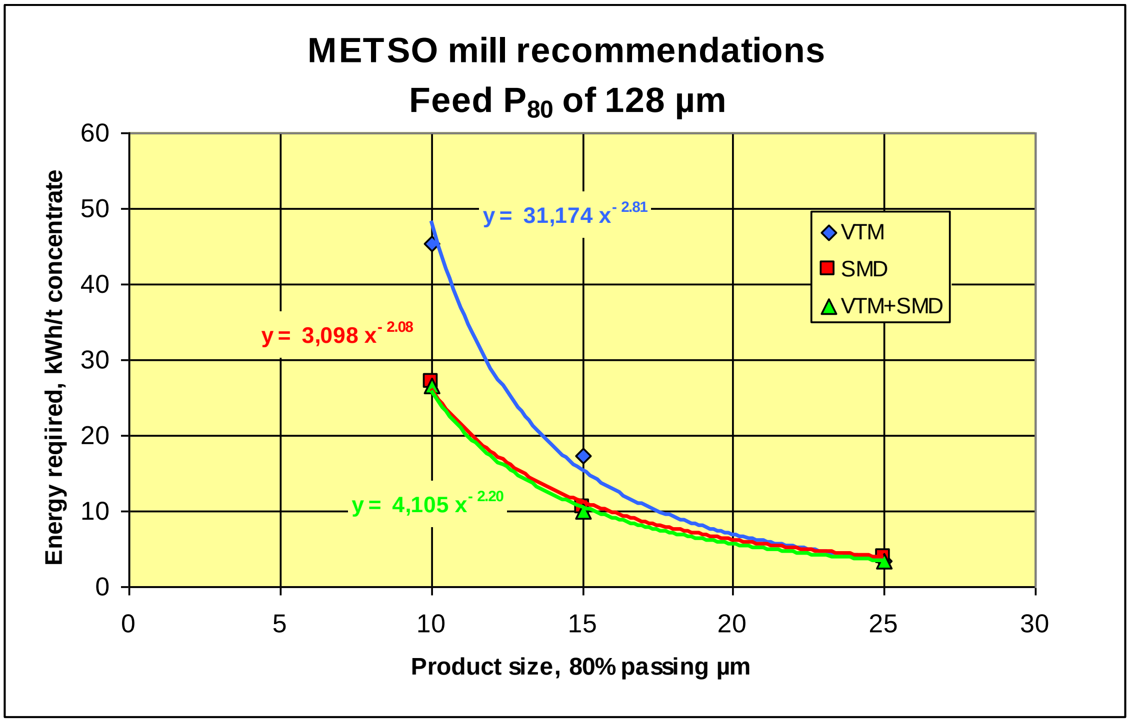


Figure 13-17: Metso Fine Grinding Power Input Recommendations

Per Figure 13-17, the power consumption increases rapidly as the product size decreases below 15 microns. An energy requirement of 17 kWh/t of concentrate was deducted for design and costing purposes from the above graph (assuming the use of a VTM only).

Based on its own testwork, Deswik recommended a single Deswik 2000 mill to grind 50 t/h of flotation concentrate from 80% passing 150 µm to 80% passing 25 to 15 µm. The indicated power consumption for performing this size reduction in the Deswik unit is pegged at 14 kWh/t. This confirmed the power requirement indicated by Metso.

The results suggested that a targeted regrind size of 15 microns would be appropriate for the Young-Davidson ores. Any finer size would require significantly more power, while a coarser grind would yield unacceptable gold losses as indicated in Figure 13-14.

13.3.10 Flotation Concentrate Cyanidation

Initial flotation concentrate leaching testwork, conducted during Phase I and II, yielded a suite of leach parameters which were used during Phase III to evaluate the effect of regrinding. Flotation concentrates obtained from tests F15 - F19 were used for the Phase III testwork.

One portion of concentrate was reground to a P80 ~ 30 µm and the other was reground to a P80 ~15 µm. The concentrates were re-pulped to 40% solids, pre-aerated for one hour at pH 11 and then leached for 48 hours maintaining 1 g/L NaCN and 1 g/L CaO in the presence of 15 g/L carbon. The results are tabulated in Table 13-12. Figure 13-14, in the preceding section, plots the metal recovery achieved for these tests versus the regrind size and shows that finer regrinding is required to achieve desirable gold extractions.

Table 13-12: Summary of CIL Tests on Master Comp 3 Flotation Concentrate

Test	Feed	Size	Reagent Consumption		Extraction		Residue	
	Flot	K_{80}	kg/t of CN feed		%		g/t	
No	Test	µm	NaCN	CaO	Au	Ag	Au	Ag
CIL3	F15	13	2.08	4.19	91.7	83.6	1.00	3.0
CIL8	F15	33	1.25	2.65	86.3	81.2	2.96	5.2
CIL4	F16	15	2.56	5.34	91.7	81.8	1.56	4.9
CIL9	F16	30	1.05	2.42	87.6	85.9	2.91	4.1
CIL5	F17	14	1.48	5.79	91.0	85.3	0.93	2.0
CIL10	F17	34	0.89	2.95	86.9	82.9	2.41	3.7
CIL6	F18	17	1.97	4.89	92.7	84.2	0.88	2.6
CIL11	F18	30	0.96	2.91	88.3	84.5	2.24	3.5
CIL7	F19	15	2.61	5.62	92.2	83.4	1.33	4.0
CIL12	F19	36	1.07	2.92	90.1	84.3	2.36	4.5

Test	CN Feed, g/t		Flotation Distribution		Overall Recovery, %	
	(calc.)		(overall) %		based on ore	
No	Au	Ag	Au	Ag	Au	Ag
CIL3	12.0	17.9	68.9	80.1	63.2	67.0
CIL8	21.5	27.5	68.9	80.1	59.5	65.0
CIL4	18.8	26.9	68.2	78.8	62.5	64.5
CIL9	23.4	28.7	68.2	78.8	59.7	67.7
CIL5	10.3	13.6	77.9	84.2	70.9	71.8
CIL10	18.4	21.4	77.9	84.2	67.7	69.8
CIL6	12.0	16.5	76.2	83.5	70.6	70.3
CIL11	19.2	22.6	76.2	83.5	67.3	70.6
CIL7	17.0	24.1	76.2	83.2	70.3	69.4
CIL12	23.8	28.3	76.2	83.2	68.7	70.1

Figure 13-18, presented below, indicates significant increases in the consumption of both cyanide and lime as the concentrate is ground finer: for cyanide, the increment is around 1 kg per tonne concentrate which equates to 150 grams per tonne of plant feed only (assuming a 15% mass pull to flotation concentrate). While this is still a substantial amount of cyanide, the value of the additional gold thus liberated far exceeds the incremental reagent costs.

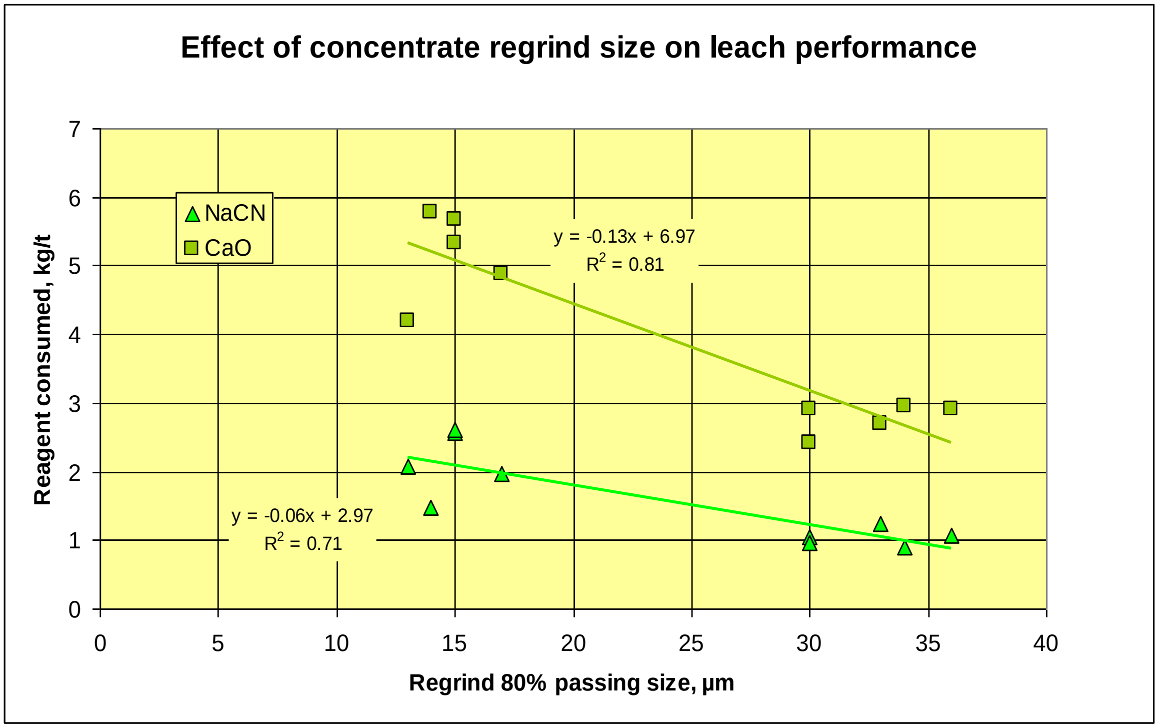


Figure 13-18: Effect of Regrind on Reagent Consumption

Detailed information regarding the cyanide consuming species was not included in the SGS testwork reports. However, based on the head feed analysis for cyanide destruction tests it was deduced that the formation of $Cu(CN)_3$ and SCN are important consumers. This finding supported the decision to include a pre-aeration step in the flowsheet as oxidation of the sulphides ahead of leaching would be expected to decrease subsequent cyanide consumption.

The effect of concentrate leach time on gold and silver extraction and reagent demand was investigated in a series of leach tests (CIL 20 to 25) on reground concentrate (F80 of 16 µm) produced from Master Composite 3 in flotation test F20. The results are presented in Table 13-13 and plotted in Figure 13-19.

Table 13-13: Leach Kinetic Data for Reground Concentrate

Test No.	Leach Time h	Reagent Cons. kg/t of CN feed		Extraction, %		Residue, g/t	
		NaCN	CaO	Au	Ag	Au	Ag
CIL20	2	0.65	1.35	69.3	33.9	4.92	18.4
CIL21	6	0.84	1.42	87.0	70.5	2.17	6.08
CIL22	12	1.21	1.51	90.9	82.8	1.78	4.20
CIL23	24	1.42	1.64	90.3	82.2	1.61	4.01
CIL24	36	1.66	1.63	90.7	82.9	1.50	3.83
CIL25	48	2.11	1.57	91.7	82.2	1.43	4.23

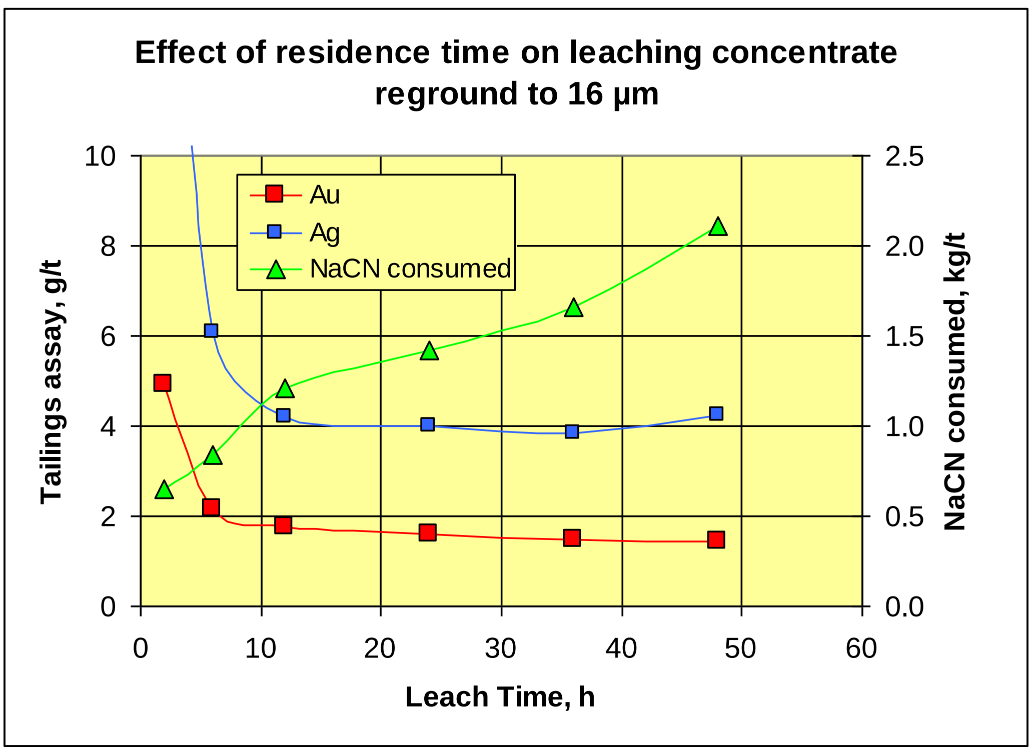


Figure 13-19: Concentrate Leach Kinetics at 16 µm P80

Leaching of the bulk of the gold and silver is fast and essentially completed within 12 hours, as would be expected from material at this very fine size distribution. However, the gold residue grades continue to decrease slowly between 12 and 48 hours suggesting the presence of a slow-leaching gold component or slow desorption of gold from a preg-robbing constituent. The latter explanation is thought to be unlikely as these tests were conducted in the presence of excess activated carbon which would have starved the solution phase of any gold, thus preventing adsorption onto a preg-robbing phase. A slow leaching gold component may indicate that some coarse gold may have escaped the gravity concentration procedure that preceded flotation test F20. If this were the case, the need for a full 48 hour leach residence time is questionable if the full-scale gravity concentration step were to remove all coarse gold prior to flotation.

The rate of cyanide consumption shows an inflection point at around 12 hours. This is thought to be consistent with a shift in the consuming mechanism from winning of valuable metal to other, less valuable, reactions (e.g. the formation of SCN and CNO). Again, it can be argued that extension of the leach duration much beyond 12 hours is of limited impact as it leads to little additional gold recovery and unnecessary incremental cyanide consumption, with about one third of the cyanide consumption saved if leaching were terminated after 24 hours.

In the absence of a definitive explanation for the observed slow kinetics, it was concluded that a simplistic economic analysis of Table 13-13 data suggested an optimum leach time somewhere close to 48 h.

In addition to the stated residence time, a cyanide consumption of 2.1 kg/t concentrate and metal extractions of 94% and 82% for gold and silver respectively was adopted from this testwork for the design and costing of the Young-Davidson Plant.

13.3.11 Flotation Tailings Cyanidation

Tailings from the grind versus flotation mass-pull series of flotation tests, realized with Master Composite 3, were leached under standard conditions determined during previous testwork phases. Leach parameters were chosen to mimic the effect of adding the concentrate leach tailings to the flotation tailings. Specific leach conditions were as follows:

- Solids in slurry 55% solids
- Pre-aeration None
- Residence time 24 h
- NaCN concentration 0.1 g/L (checked & corrected at 3, 6, 24, 32 h)
- pH control 10.5 to 11
- Carbon concentration 10 g/L

The results of these tests are summarized in Table 13-14.

Table 13-14: Cyanidation of Flotation Tails

Test No	Float Test	Size K_{80} µm	Mass pull, %	Reagent Cons. kg/t of feed		Extraction, %		Residue, g/t		CN Feed, g/t	
				NaCN	CaO	Au	Ag	Au	Ag	Au	Ag
CIL13	F15	96	13	0.01	0.25	57.0	27.7	0.06	0.06	0.14	0.7
CIL14	F16	91	9.8	0.03	0.05	53.2	31.6	0.08	0.08	0.15	0.7
CIL15	F17	140	16.9	0.04	0.23	48.4	26.9	0.08	0.08	0.15	0.7
CIL16	F18	132	14.6	0.05	1.01	42.1	30.9	0.12	0.12	0.20	0.7
CIL17	F19	141	11	0.09	0.28	45.8	34.4	0.11	0.11	0.19	0.8

The results presented in Table 13-14 suggest a weak inverse relationship between gold recovery and feed grade. This is conceivable if the head grade variation is due to semi-refractory gold finding its way directly into the low-intensity cyanidation step provided by leaching of coarse flotation tails, after having bypassed the intensive recovery route, comprising of an aggressive cyanidation stage performed on reground material. More tests would be required to confirm this speculative observation, given that the error of the analytical procedures used here is probably not significantly smaller than the absolute values measured.

Design criteria deducted from these tests include projected gold and silver extractions from flotation tailings at 50% and 30% respectively and reagent consumption rates of 0.03 kg/t and 0.3 kg/t for NaCN and lime respectively.

On average, leaching the flotation tailings of Master Composite 3 recovered an additional 0.07 g/t of gold and 0.28 g/t of silver, expressed relative to the plant feed material, providing justification for inclusion of this step in the flowsheet.

13.3.12 Flowsheet Testing: Overall Gold and Silver Recovery

All of the tests conducted using Master Composite 3 allowed to project overall recoveries and reagent consumption for the integrated gravity-flotation-concentrate regrind and leach-flotation tailings leach. Table 13-15 provides such an insight while Figure 13-20 and Figure 13-21 break the overall recovery down into the contribution of each individual processing step.

Table 13-15: Results of Overall Flowsheet Tests on Master Composite 3

Test No.	Primary grind K80, µm	Ro Conc % Wt	Regrind K80, µm	Overall extraction, %		Reagent, kg/t ore	
				Au	Ag	NaCN	CaO
G2/F15	94	13	13	92.6	78.3	0.28	0.8
			33	88.9	76.4	0.17	0.6
G2/F16	94	10	15	92.2	76.7	0.28	0.6
			30	89.4	79.9	0.13	0.3
G3/F17	140	17	14	90.9	79.0	0.28	1.2
			34	87.7	77.0	0.18	0.7
G3/F18	140	15	17	91.1	78.2	0.33	1.6
			30	87.8	78.4	0.18	1.3
G3/F19	140	11	15	91.0	77.8	0.37	0.9
			36	89.4	78.5	0.20	0.6

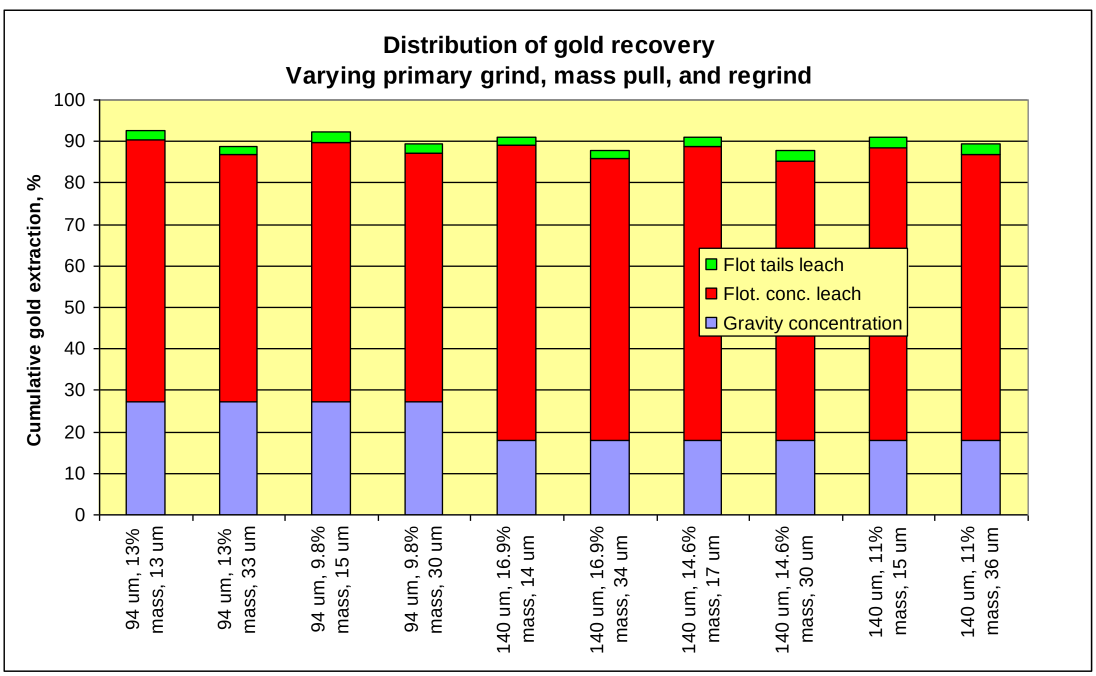


Figure 13-20: Distribution of Gold Recovery from Master Composite 3

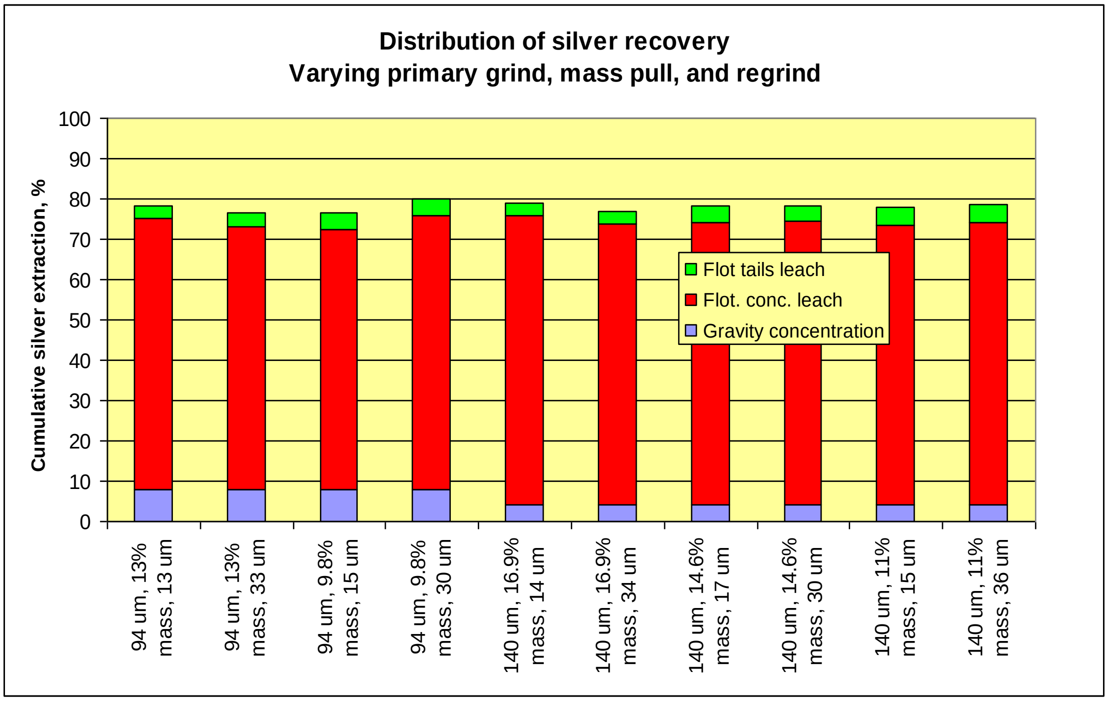


Figure 13-21: Distribution of Silver Recovery from Master Composite 3

The data of Table 13-15 was used in a simple economic analysis, estimating the revenue from gold and silver sales, less the operating cost of each option and it was concluded that the fine regrind option is critical to the operating revenue but that other options should be selected to minimize capital. This meant that, across the range investigated, a coarse primary grind and a low mass pull are acceptable.

A statistical analysis of the data behind Table 13-15 was performed which produced multi-variable regression equations for gold and silver in the final plant tailings with R2 values of 0.89 and 0.1 – statistically significant for gold but not so for silver. The equation for gold in tailings for the Master Composite 3 is:

$$\text{Gold in tailings (g/t)} = 0.076 + \text{Primary grind P80 } (\mu m) \times 0.0029$$
$$+ \text{Mass pull } (\%) \times 0.0038 + \text{Regrind (P80 } \mu m) \times 0.0041$$

While this type of equation may oversimplify the metallurgy and is limited to the range over which it was calibrated, it confirmed that the overall metallurgical performance is most sensitive to regrinding. It also indicated that a small (10 µm) increase in the primary grind target will only "cost" 0.03 g/t of gold recovery and can be easily compensated for by a smaller change to either mass pull or regrind. The implication was that the use of the second-hand Kemess primary mill was justified as it played a less critical role. Applying the same logic to the regrind circuit dictated that it should be conservatively sized to ensure a tightly controlled and very fine regrind product feeding concentrate leaching.

13.3.13 Variability Testing

SGS processed five ore zone composites and thirty-two hole composites through an identical series of consecutive tests aimed at mimicking the overall flowsheet. The purpose of these tests was to quantify the degree of variability that could be expected during operation at a set suite of operating parameters. A secondary objective was to assess the sensitivity of recovery data to fluctuations in variable operating parameters (e.g. mass pull) and feed grades. The conditions of the tests were as follows:

- Primary grind : 80% passing 130 µm
- Gravity concentrate : 0.05% mass
- Flotation mass pull : 13%
- Concentrate regrind : 80% passing 13 µm
- Concentrate leach
  - Pre-aeration : 2 hours at pH 11, 40% solids
  - CIL time : 48 hours
  - NaCN : 1 g/L (Checked at 3, 6, 24, 32 h)
  - CaO : 1 g/L
  - Carbon : 15 g/L
- Flotation tailings leach
  - Pre-aeration : None
  - CIL time : A – 24 h, B – 48 h
  - NaCN : 0.1 g/L (Checked at 3, 6, 24, 32 h)
  - pH : 10.5 to 11
  - Carbon concentration : 10 g/L
  - Assessment : Gold assays only (silver too low for assay)

Gravity and Intensive Cyanidation

Gravity recovery and intensive cyanidation variability data are presented in Sections 13.4.7 and 13.4.8 of this report. Gravity recovery results exhibit orders of magnitude variances with respect to concentrate grades: a range from 100 to 20,000 g/t gold is shown in Figure 13-8. This variance is transferred to the subsequent intensive cyanidation and electrowinning steps. This degree of variability was exacerbated by the fact that the data were the results of both blending (compositing) as well as mathematical averaging of the composite results.

The variability was also not restricted to concentrate grades, with the mass pull also ranging from 0.02% to 0.34%. Mass pull and grades were not perfectly aligned in an inversely proportional relationship.

The implication of this variability was that the gravity, intensive cyanidation and electrowinning steps along this route had to be designed to accommodate peak duties that exceed averages by considerable margins. Failing this, the process has to rely on sustained flotation recoveries to compensate intermittent losses caused by feed characteristics exceeding the gravity circuit's capability.

Flotation

Multiple regression analysis of the flotation data showed that the primary determinant of gold recovery to flotation concentrate across all data was the feed grade and that the mass pull had a generally insignificant impact on recovery with the primary grind being mildly influential. This finding was in good agreement with the results obtained for tests with Master Composite 3, as reported in Section 13.4.9 of this report. Gold recoveries versus mass pull and versus feed grade are shown in Figure 13-22 and Figure 13-23, respectively. Silver recovery data are shown in Figure 13-24.

Figure 13-22 shows a mass pull range from approximately 11 to 19%. All of the zones showed a spread across the entire range i.e. there was no "clustering" of the mass pull data. The results indicated that the flotation concentrate circuit could be designed for an average mass pull of 15%, with an allowance of 20% for peak loads.

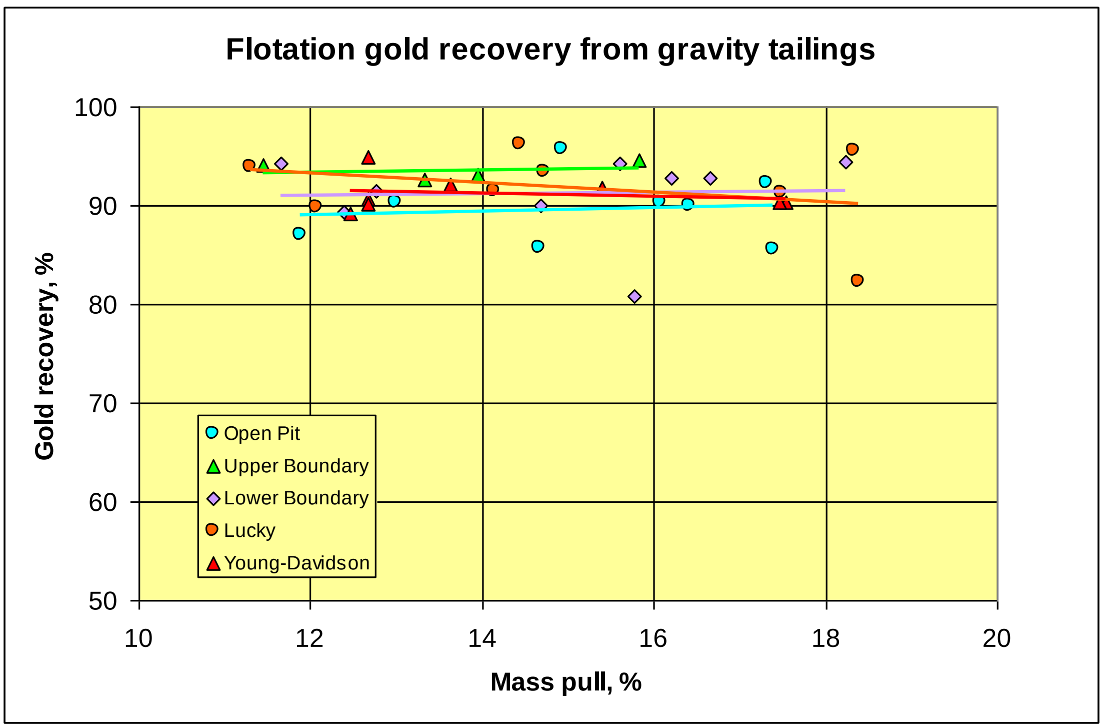


Figure 13-22: Flotation Gold Recovery from Gravity Tailings versus Mass Pull

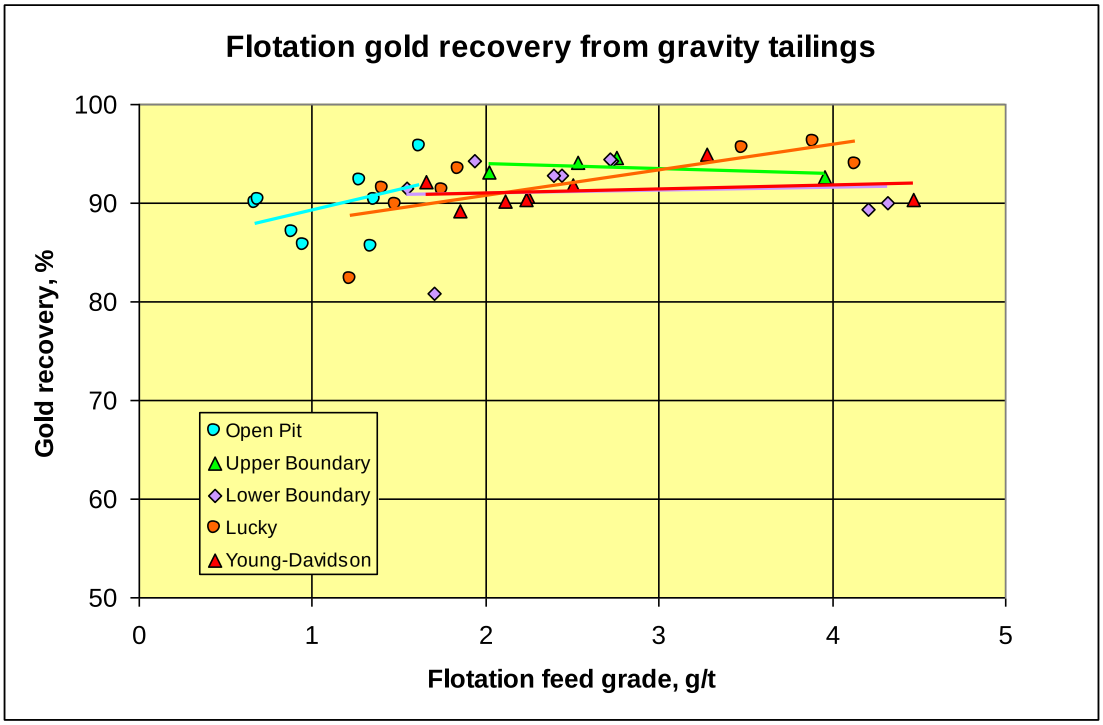


Figure 13-23: Flotation Gold Recovery from Gravity Tailings versus Feed Grade

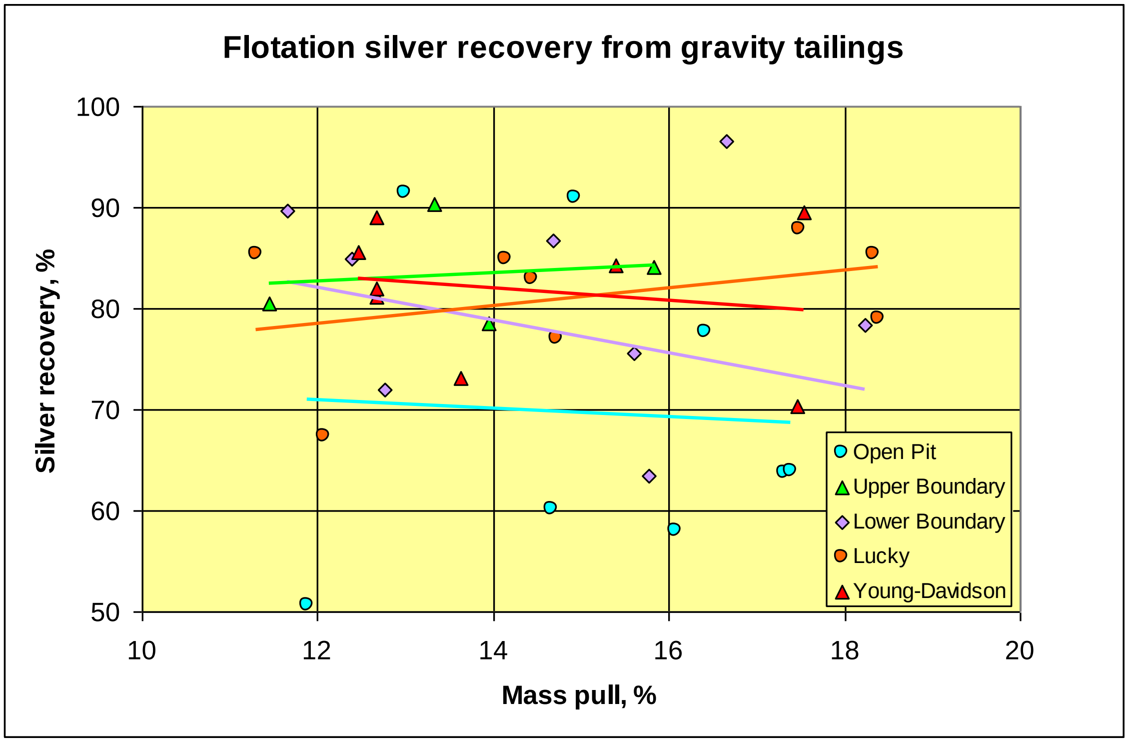


Figure 13-24: Flotation Silver Recovery from Gravity Tailings

With both the flotation tails volume as well as gold grades exhibiting low variability (see Figure 13-13 and 13-22 it was expected that the tailings CIL circuit would operate with minimal disturbances. As this is the section responsible for the ultimate solids residue as well as solution losses, it would absorb any fluctuations that may come its way from the smaller (volume-wise) concentrate leach. This CIL configuration thus negated the effect of any variability in both flotation as well as concentrate leach performance and would ensure a high overall recovery throughout.

A conspicuous feature of the above graphs is that the open pit ore performed less well than the other ore types with respect to gold and silver recovery – although this was probably mainly a function of feed grade. The Upper Boundary ore appeared to be the best performer – although this too may be related to feed grade. The implication was that maximization of the mine earnings potential in the early years of operation would benefit from a rapid contribution of underground ores to the plant feed mix.

Metal Recovery from Flotation Concentrate

The recovery values for gold and silver from flotation concentrate are plotted in Figure 13-25 and Figure 13-26. These figures show some scatter, especially for the silver data. The weighted averages of gold and silver extraction are 92% and 82% respectively at the average regrind size of 80% passing 12 µm. These were deemed reasonable values to apply to all ore types.

Cyanide consumption across all tests averaged 2.1 kg/t of flotation concentrate. Lime addition averaged 5.3 kg/t of concentrate.

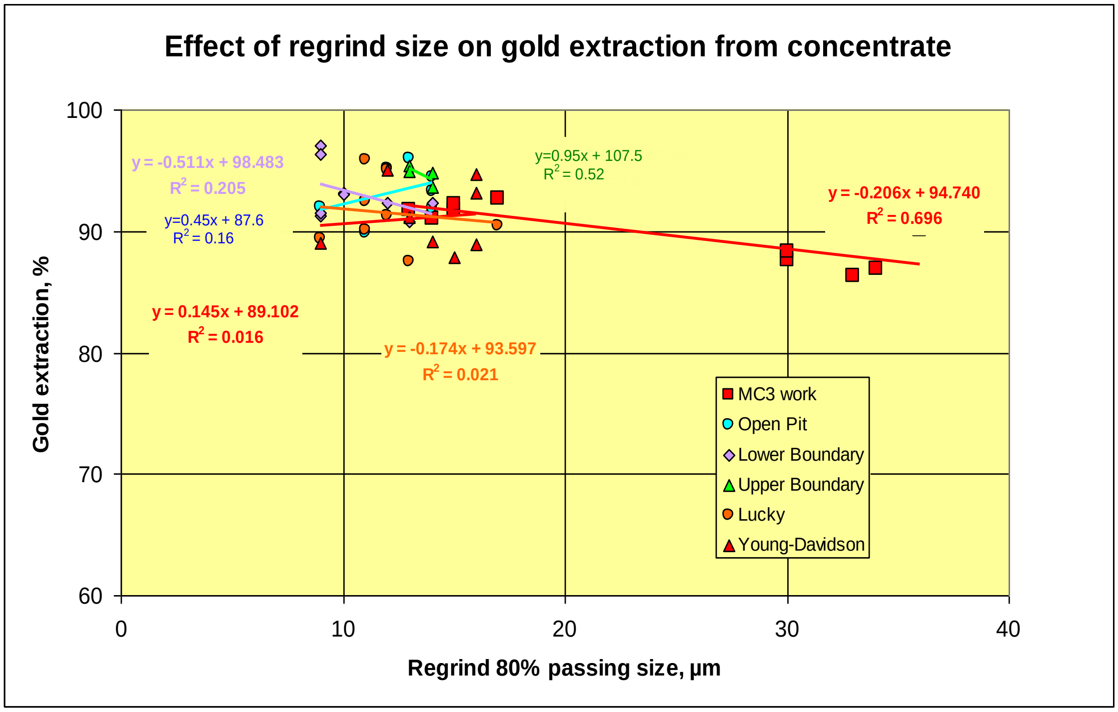


Figure 13-25: Gold Recovery from Flotation Concentrate

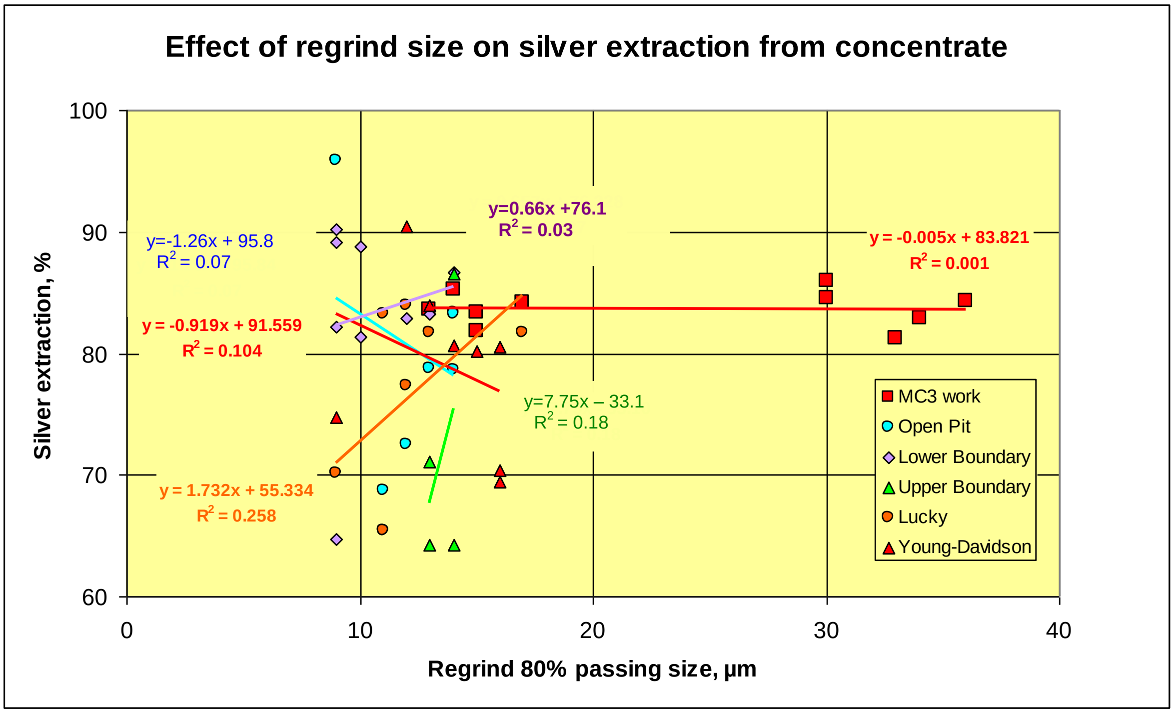


Figure 13-26: Silver Recovery from Flotation Concentrate

Metal Recovery from Flotation Tailings

Leach extraction of gold was measured after 16 hours and 24 hours of leaching time. The 24 hour leach time generally yielded better leach extractions than 16 hours. The average gold assay in the tailings across all flotation tailings leach tests were 0.111 g/t after 16 hours and 0.087 g/t after 24 hours, for a difference of 0.024 g/t of flotation tailings.

Overall Metal Recoveries

The test parameters for the variability testing of the Hole and Zone Composites, and simple average results are summarized in Table 13-16.

Gold and silver recovery distributions, as measured for all 37 tests (including the five zone composites), are presented in Figure 13-27 and Figure 13-28. The relationship between test feed grade and tailings assays are presented in Figure 13-29

Table 13-16: Overall Recovery Summary – Variability

.

Item		***Units***	***Overall***	***Open Pit***	***Upper B'y***	***Lower B'y***	***Lucky***	***YD***
Test feed grade		g/t Au	3.36	1.24	4.16	3.49	4.78	3.51
		g/t Ag	3.81	4.38	3.05	4.84	2.99	3.26
Primary grind		P80, µm	126.7	136.8	148.5	114.2	129.1	117.3
Flotation	Mass pull	%	14.7	15.2	13.6	14.9	15.1	14.3
	Recovery from flotation feed	% Au	91.2	89.6	93.6	91.2	91.7	91.2
		% Ag	78.2	69.6	83.3	77.4	81.3	81.9
		% S	95.4	97.8	97.8	91.7	95.5	95.7
	Gravity+flotation recovery	% Au	93.4	90.7	95.6	93.5	94.7	93.3
		% Ag	80.5	70.5	84.6	81.3	84.3	84.0
Concentrate regrind		P80, µm	12.4	12.6	6.4	10.6	12.0	13.9
Overall Gold Recovery	Gravity recovery	%	25.5	11.2	32.8	24.5	37.6	25.0
	Conc. CIL	%	62.8	74.2	59.5	64.2	52.2	62.3
	Tails CIL	%	3.8	4.9	2.8	4.1	2.7	4.0
	Total	**%**	**92.1**	**90.4**	**95.0**	**92.8**	**92.5**	**91.2**
Overall Silver Recovery	Gravity recovery	%	9.8	2.7	7.4	11.3	14.9	11.3
	Conc. CIL	%	56.1	56.4	55.8	57.7	52.6	57.6
	Total	%	65.9	59.1	63.1	68.9	67.5	68.9

The data set featured a wide variation in gravity gold recovery, ranging from less than 1% (Open Pit) to 86% (Lucky Zone). Despite this difference in gravity recovery potential, the overall recoveries for gold were similar, ranging in a narrow band (+-5%) around an average of 92%.

Silver exhibited more variability, both in individual process recoveries as well as in overall recovery.

A weak grade-recovery relationship was suggested by the data plotted in Figure 13-29. The data in this graph is also grouped in clusters per zone suggesting that a mine plan which excludes blending may result in fluctuations (on a daily or even monthly basis) in the ounces of valuable metals produced. Mitigating this would be that due to the high tonnage rates from underground, multiple mining fronts would be required to be in production simultaneously, producing natural blending of ores, and minimizing large fluctuations in recovery.

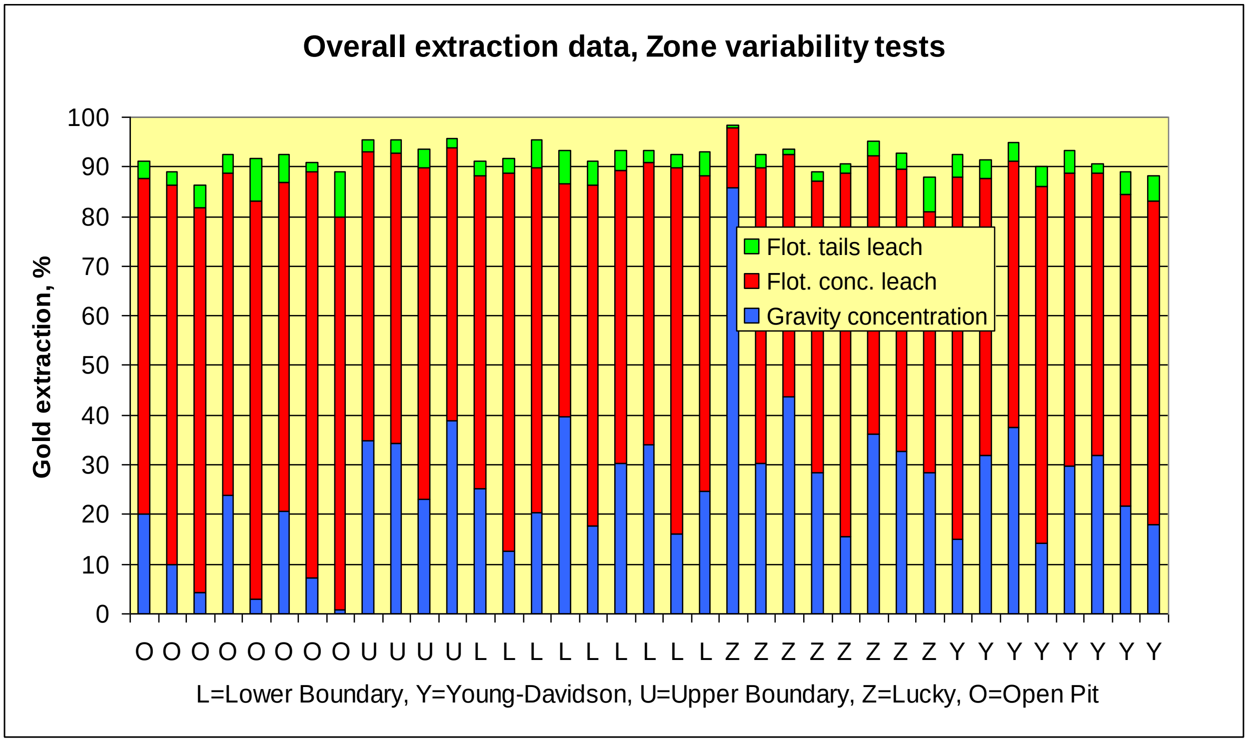


Figure 13-27: Distribution of Gold Recovery from Hole and Zone Composites

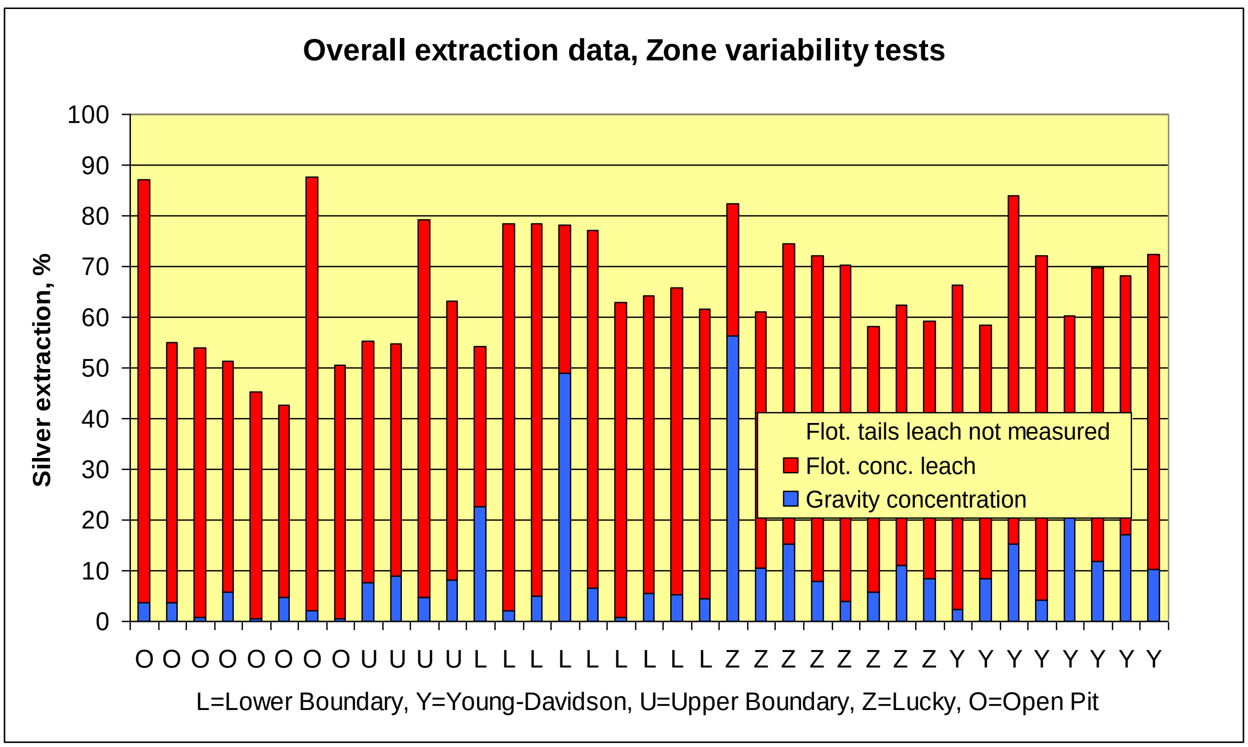


Figure 13-28: Distribution of Silver Recovery from Hole and Zone Composites

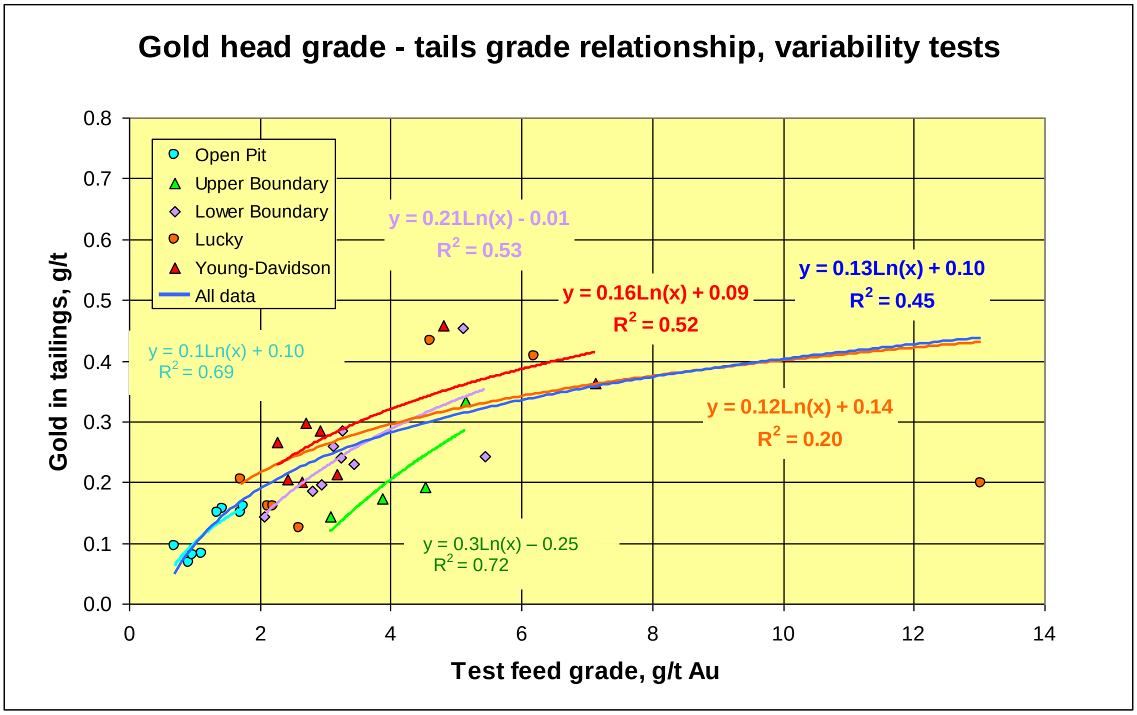


Figure 13-29: Gold Feed Grade versus Final Tailings Grade Relationship

It was difficult to draw any firm conclusions regarding spatial variability as the individual hole samples were still composites, from approximately ten individual intercepts each (dividing 357 samples through 32 hole composites). Any variability with, for instance, depth, would have been masked by the averaging effect of compositing.

13.3.14 Carbon Adsorption Testwork and CIL Modelling

For the purposes of the Feasibility Study stage, the laboratory data produced by SGS was assessed below and used in conjunction with general industry standards to establish a set of CIL design criteria.

To generate the adsorption isotherm, different amounts of pulverized carbon were added to samples of leached concentrate pulp and rolled for 72 hours. The solutions were assayed for gold and silver. Gold and silver loadings on the carbon were calculated and are plotted in Figure 13-30 against the measured equilibrium solution concentrations for these metals. The Freundlich isotherm is expressed by the equation:

$$Y = A * C^n$$

with Y the metal loading on the carbon phase (g/t), C the metal concentration in solution in equilibrium with loading Y (mg/l), and

A and n both model parameters to be determined through curve fitting,

The isotherm equation was fitted to the experimental data and is shown as solid lines in Figure 13-30. The higher "A" value obtained for gold is indicative of the higher affinity of carbon for this cyanide complex. Note that an "A" value of 10,000 for gold is considered excellent and is indicative of a circuit which would allow higher than normal carbon gold loadings.

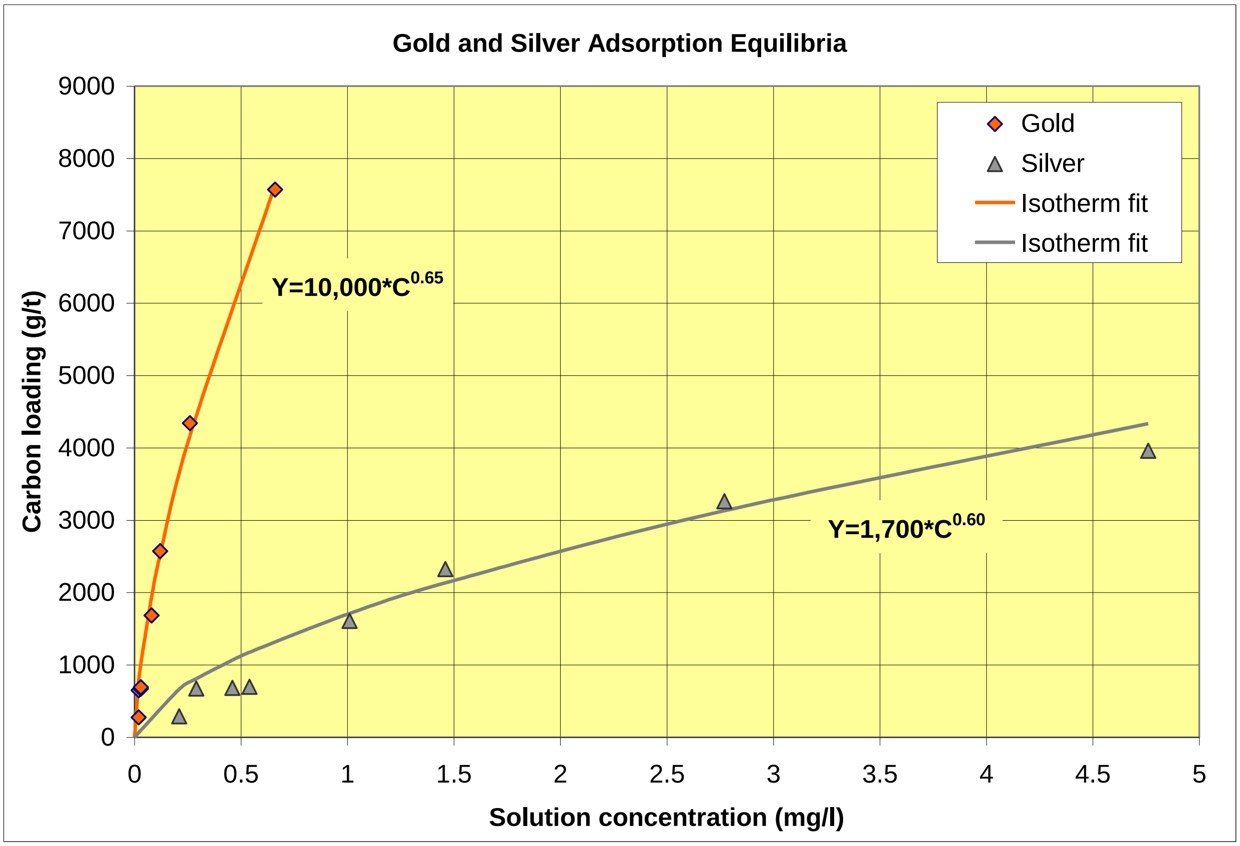


Figure 13-30: Carbon Adsorption Equilibrium Loading Results

The expected or potential loaded carbon metals loading should ideally be established through modeling but can also be estimated using a rule-of-thumb approach.

The approach in question uses the equilibrium equation measured for this carbon-pulp system but with the solution concentration (C) equal to between a quarter and a third of the feed grade. The theoretical maximum carbon loading thus calculated is divided by half to estimate the operational loaded carbon gold loading. This approach yields an estimated loaded carbon value rounded to 12,000 g/t gold for a concentrate grade at approximately 14.3 g/t. A smaller amount of silver would also be expected to load onto the carbon. It was elected to adopt the indicated loading as representative of the total metal loading onto the carbon, thus yielding a conservative design requirement for the carbon stripping circuit.

The logic behind this approach was based on the assertion that, firstly, most of the leaching and adsorption occurs in the first CIL tank hence the solution concentration in this tank can be expected to average around a quarter of that of the head grade. Secondly, in order to maintain a reasonable driving force for adsorption, the carbon loading would not be allowed to exceed 50% of the equilibrium loading at that concentration. It was believed that this rule-of-thumb estimate presented a conservative but reasonable total metal loading capacity for this application.

13.3.15 Cyanide Destruction

SGS completed scoping cyanide destruction tests using the INCO SO_2/Air process during Phase II. The picric acid method was used to assess Weak Acid Dissociable cyanide, CN(WAD), at the end of each test period. The results of these cyanide destruction tests were satisfactory and provided indicative reagent addition rates for the ensuing Phase III tests.

During Phase III, 20 kg of Master Composite 3 was processed by gravity and flotation (F20En), followed by concentrate grinding to 80% passing 16 µm and leaching (CIL27). The leached concentrate was then mixed with the original flotation tailings, in a simulation of the proposed flowsheet, and leached for another 24 h (CIL28). This procedure realized 91.1% gold extraction and 85.2% silver extraction.

SO_2/Air cyanide destruction testwork was conducted on the tailings at a pulp density of 50% solids. The operation started as a batch process then switched to continuous mode after the reaction had been established, as per the normal test procedure. Four tests were conducted at four different SO_2 dose rates. Results of the continuous segment of these tests are presented in Table 13-17 and Figure 13-31.

Table 13-17: Cyanide Destruction Tests on Combined Leach Tailings

Test	*Time min*	*pH*	*Solution assay, mg/L*						*Reagent used g/g $CN_{(WAD)}$*	
			CN_T	*$CN_{(WAD)}$*	*Cu*	*Fe*	*Zn*	*SCN*	*SO_2*	*Lime*
Feed	**-**	**10.2**	**100**	**97**	**59**	**1**	**2.6**	**100**	-	-
CND 2	48	8.7	0.02	<0.01	0.15	<0.01	<0.001	110	5.1	4.6
CND 3	45	8.5	0.33	<0.3	0.29	<0.01	<0.001	110	3.6	2.9
CND 4	47	8.5	4.1	4.1	1.11	<0.01	<0.001	110	3.3	2.0

CND 5	40	8.5	11.7	11.6	30.6	0.83	0.009	110	2.3	1.9

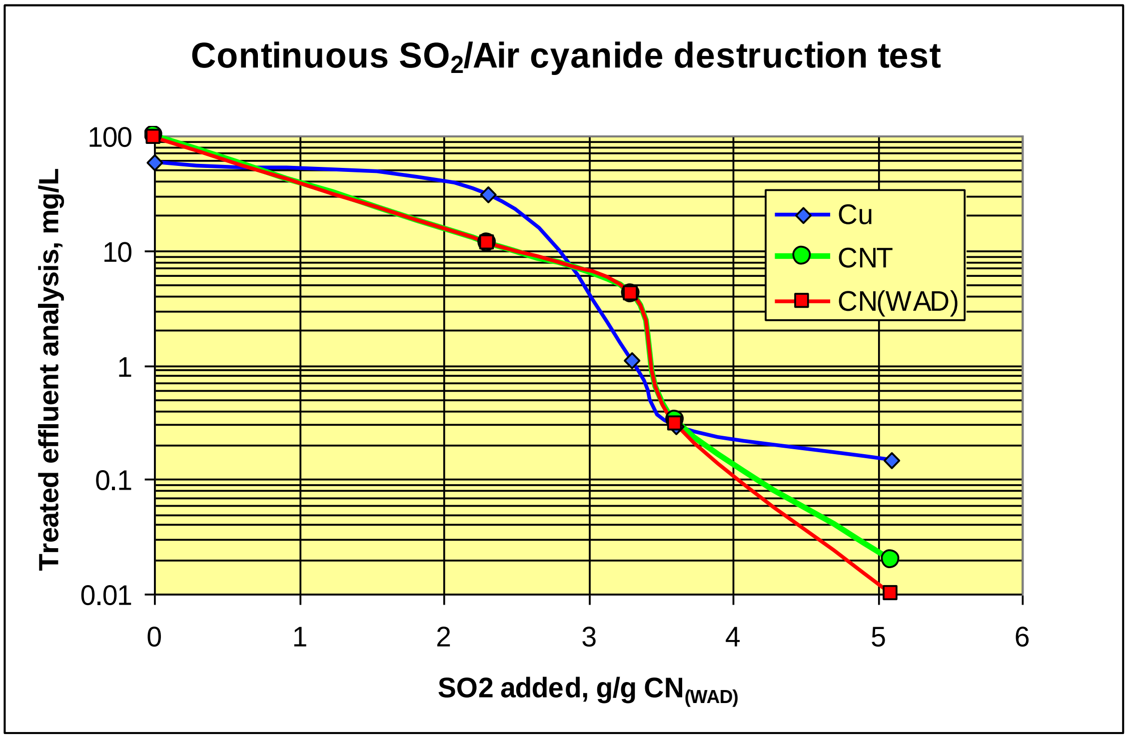


Figure 13-31: Results of Continuous SO2/Air Cyanide Destruction Tests

The data showed that the Young-Davidson effluent is amenable to the INCO SO_2/Air cyanide destruction method. The relationship between SO_2 dose and the treated effluent quality shown in Figure 13-17 can be used to determine the appropriate dosage for a given effluent target.

The cyanide destruction reactor was sized based on the indicated residence time of 45 minutes, assuming leach kinetics is independent of reactor size. Approximately 3.4 g SO_2 and 0.75 g lime is required per gram of CN(WAD) feed to yield an effluent containing only 1 ppm CN(WAD).

13.3.16 Flotation Concentrate and Tailings Thickening

The Young-Davidson flowsheet includes a thickener to partially dewater flotation concentrate before ultra-fine grinding and another thickener on the flotation tailings to recover water and densify this stream ahead of the combined CIL operation.

Following an initial scouting test (Phase I), Outotec was engaged to conduct continuous pilot plant solids-liquid separation tests on flotation tailings and concentrate samples during Phase II and Phase III.

The samples used in the Phase II work originated from Master Composite 2 tests (GS-4,5 and F4,5). SGS reported the tails as 80% passing 120 µm. The samples for Phase III were produced from Master Composite 4 in gravity-flotation test F58 and had a flotation tailings size analysis indicating 80% passing 152 µm.

Outotec used its 94 mm continuous "Supaflo" thickener for its testwork. Prior to testing, Outotec screened various flocculants and, in the Phase II work, selected Magnafloc 351, 15% feed solids, at a nominal dose of 15 g/t for flotation tailings and 25 g/t for concentrate. In the Phase III work, Outotec changed to Magnafloc 10 at doses of 20 and 25 g/t.

Outotec reported that it was difficult to produce underflow densities not exceeding 50% solids. The lowest level attained was 56% solids for the concentrate sample and 63% solids for the tailings sample. Figure 13-32, below, summarises the Outotec results for both phases.

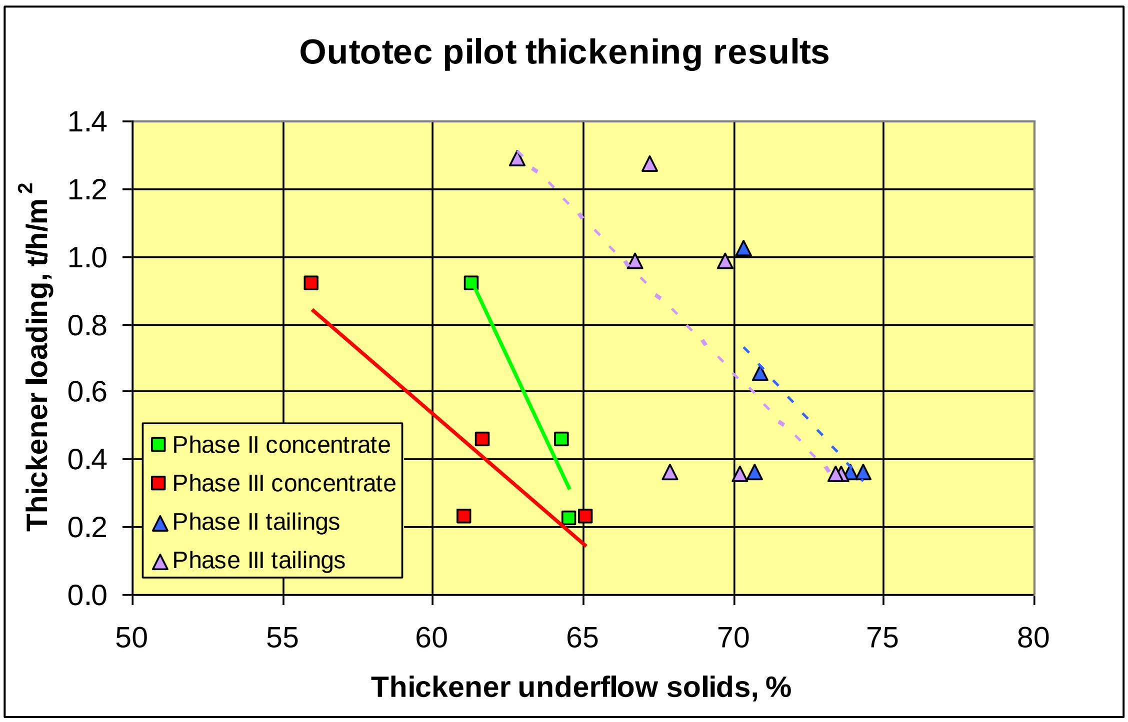


Figure 13-32: Results of Outotec Pilot Thickener Tests

Outotec reported overflow clarities ranging from 55 ppm - 92 ppm and 164 ppm - 1053 ppm total suspended solids for the tailings and concentrate samples, respectively. This meets the design specification of 200 ppm for overflow clarities.

Based on the results of its Phase III testwork, Outotec recommended a design solids loading rate of 0.99 t/h/m^2 for the flotation tailings thickener. At this loading an underflow density of 70% solids is achievable, far exceeding the 55% targeted for the combined CIL plant.

For the flotation concentrate thickener, Outotec proposed a solids loading rate of 0.25 $t/h/m^2$. At this loading rate an underflow product containing 65% solids would be expected. The loading rate was restricted because of concerns over frothing. Outotec offered a solution in the form of its “Frothbuster”, which would allow the design thickener loading to be increased to 0.5 $t/h/m^2$. At this loading rate the expected underflow solids content is 62%. While such a high density would be beneficial to the grinding performance of the regrind mill the classification performance of the cyclones in this circuit would probably decrease appreciably. Given that this density was well above the requirement for this stream this higher loading rate was recommended for design purposes.

A design criterion of 50% solids content was adopted for the regrind cyclone overflow stream reporting to the concentrate CIL and back calculated a solids content of 52% for the thickener underflow. The difference between this target and the expected 62% provided some operational flexibility and would not pose any problems as water could be added to the cyclone feed box if required to achieve a more dilute feed to the CIL than what the thickener produced.

During its first program of work, Outotec measured the rheology of thickened flotation tailings and concentrate. The measured viscosities were low, leading Outotec to conclude that pulp viscosity would not pose any problems, nor would it demand any special pumping requirements.

During Phase II, SGS measured the viscosity of Metso grinding test products ranging in size from 80% passing 14 µm to 80% passing 22 µm. SGS reported that the samples displayed good flow behaviour at solids contents below the Critical Solids Density (CSD). This was determined to be 70% solids for all three samples. Since all of the flowsheet streams would contain significantly less solids than this limit, viscosity constraints would not affect the performance of any of the unit processes.

14 MINERAL RESOURCE ESTIMATES

14.1 Drill Hole Database

Alamos conducted a Mineral Resource estimate of the Young-Davidson underground deposit, which incorporates all drilling data from the Alamos and predecessor drilling programs conducted through November 7, 2015. A database was compiled using data from 2,695 drill holes, with collar, survey, geological and assay information, containing a total of 328,202 metres of assayed gold intervals. Of these data, 157,127 metres (960 holes) are surface drilling, and 171,075 metres (1,735 holes) were drilled from underground. Historic underground drilling conducted by Matachewan Consolidated Mines (MCM) during the 1940’s was not incorporated in the database due to collar location discrepancies with the existing underground. This historic drilling comprises 694 holes with 36,208 metres of assayed gold intervals, and has been largely supplanted/replaced with recent underground drilling.

14.2 Topography

Exisiting topography is based on a 2010 LIDAR survey, but was not used to code the underground Mineral Resource model, as the entire zone of mineralization modeled is below current topo surface. Alamos has reviewed the topography surface in cross sections comparing elevation between the drill hole collars and the topography surface and found close agreement between the two.

14.3 Coordinate System

All data is located in NAD 83 Zone 17T geodetic Datum. The local mine coordinate system uses a truncated version of the above coordinate system, with the first two digits of the Northing and the first digit of the Easting removed.

14.4 Lithology and Alteration Modelling

Currently, no lithologic/alteration model has been completed for the Young-Davidson underground. A manually constructed 1.5 g/t Au grade solid is utilized to constrain the grade estimate. The 1.5 g/t Au grade shell is constructed by site geology personnel and is updated on a continuous basis as drilling results are received. All potentially economic mineralization at Young-Davidson occurs in altered syenite, which has been cut by a series of north-south striking post-mineral diabase dykes. These dykes have been modeled as solids and have been manually cut out of the 1.5 g/t Au mineralized solids. An east-west long section looking north showing the resultant solid is provided in Figure 14-1

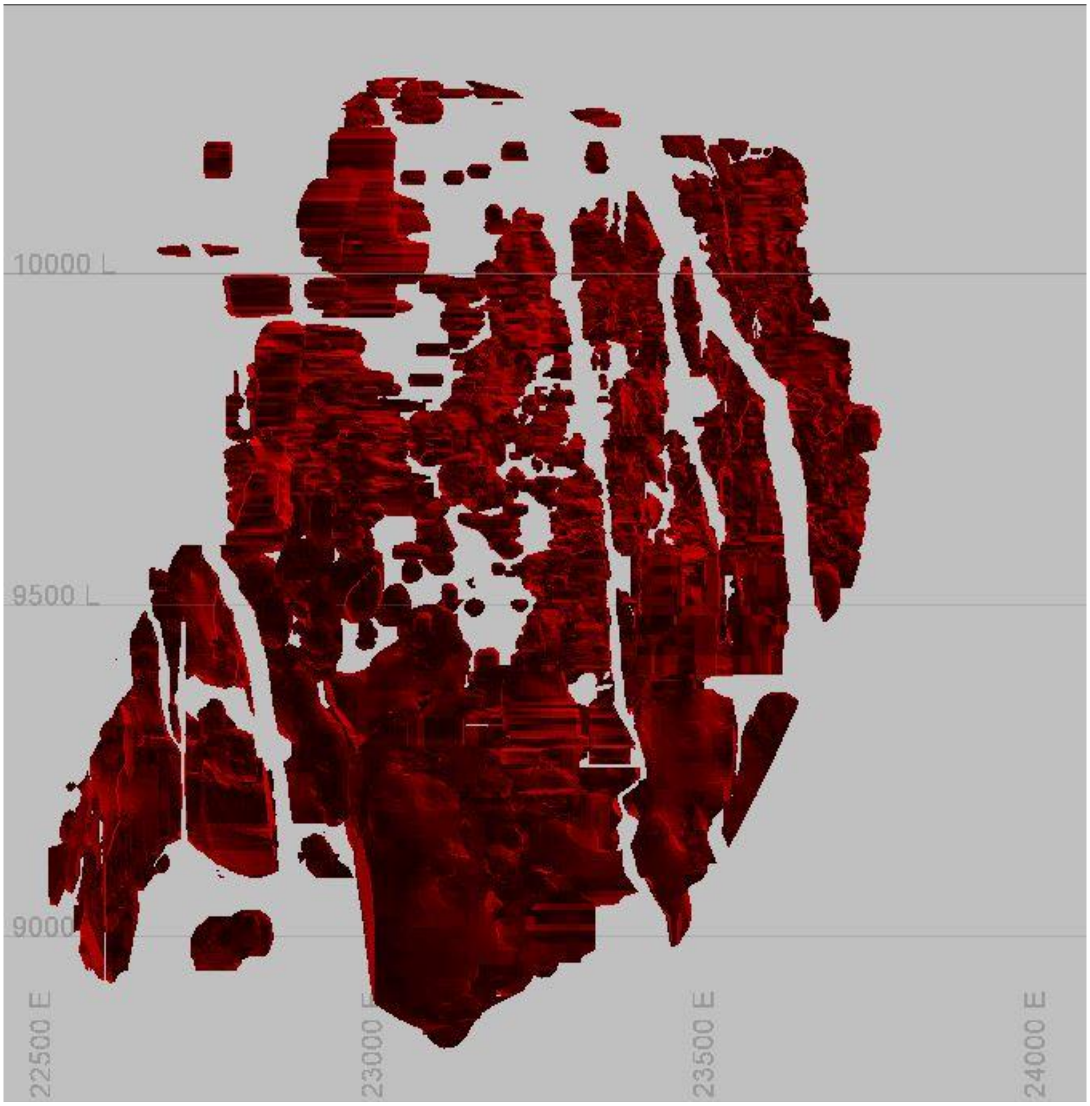


Figure 14-1: E-W Long Section Viewed to the North Showing 1.5 g/t Au Solid

14.5 Exploratory Data Analysis

Exploratory data analysis (EDA) was conducted using the raw gold assays from the Young-Davidson underground project. These data were analyzed above both global and incremental cut-offs in order to assess the distribution and grade ranges of the assays internal and external to the 1.5 g/t Au solid. Basic statistics for the raw gold assays are provided in Table 14.1. It can be observed that on a grade-thickness basis, a significant amount of metal above potentially economic cut-off resides external to the 1.5 g/t solid. Most of the intervals external to the grade solid are not of minable width, however this demonstrates that gold occurs external to the main lode system.

Table 14-1: Summary Statistics – Raw Gold Grades by Domain

Domain	Au Cutoff (g/t)	Statistics above Cut-off						
		Total Meters	Incremental Pct	Max Grade (Au g/t)	Mean Grade (Au g/t)	Grade Thickness	Standard Deviation	Coeff. of Variation
All Data	0.01	311,751	78.27%	552.69	0.66	207,081	2.56	3.86
	0.50	67,735	16.39%		2.73	184,848	4.98	1.82
	3.00	16,640	2.69%		7.08	117,888	8.62	1.22
	5.00	8,260	2.65%		10.39	85,837	11.30	1.09
Internal to 1.5 g/t Au Solid	0.01	43,339	19.02%	552.69	3.03	131,278	5.43	1.79
	0.50	35,095	50.22%		3.70	129,810	5.84	1.58
	3.00	13,332	15.26%		7.11	94,759	8.38	1.18
	5.00	6,718	15.50%		10.33	69,372	10.87	1.05
External to 1.5 g/tAu Solid	0.01	268,413	87.84%	370.05	0.28	75,803	1.35	4.78
	0.50	32,640	10.93%		1.69	55,038	3.56	2.11
	3.00	3,309	0.66%		6.99	23,129	9.53	1.36
	5.00	1,542	0.57%		10.68	16,465	13.00	1.22

14.6 Specific Gravity Analysis

Company geology personnel conducted 19,749 specific gravity tests using the water immersion method utilizing diamond drill core from the project during the 2005-2014 drilling programs. A summary of specific gravity (SG) determinations by rock type is provided in Table 14.2

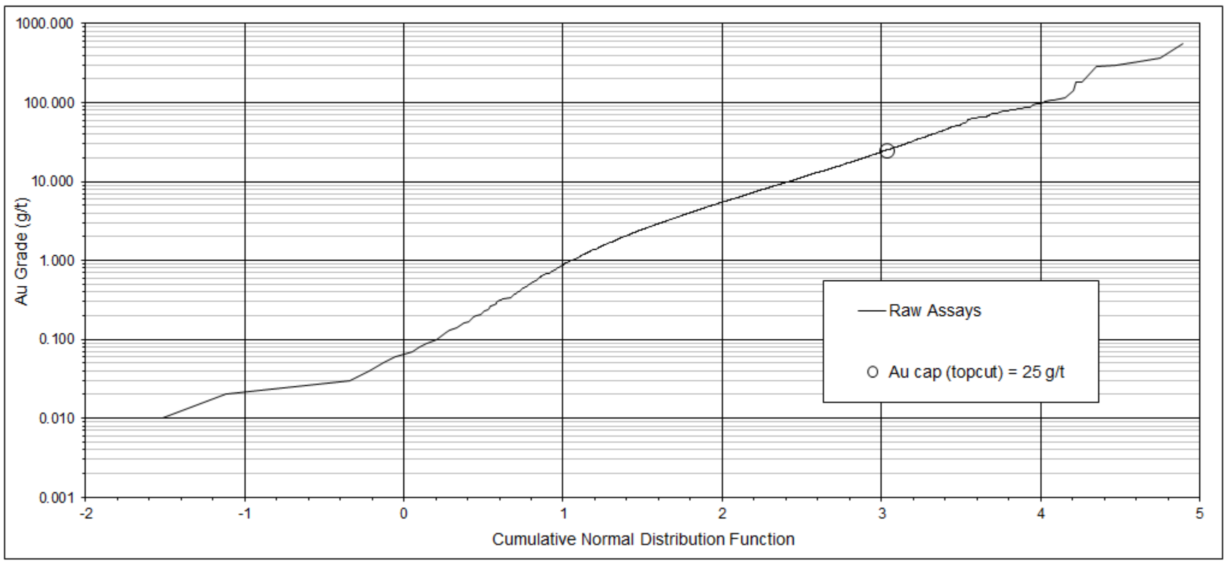
Mineralization at Young-Davidson is largely constrained to syenite rock types. The Company has elected to assign a global SG of 2.69 based on the average specific gravity determinations for the main host rock. Alamos is of the opinion that the average SG assignments are reasonable given the limited range of mineralized lithologies, and believe that the data are sufficient and suitable for use in Mineral Resource tabulation.

Table 14-2: SG Results for Young-Davidson Underground Samples

Rock Type	Number of Samples	Average (g/cm^3)	Maximum (g/cm^3)	Minimum (g/cm^3)
Syenite	459	2.715	3.141	2.503
Diabase	96	2.966	3.110	2.715
Mafic Volcanic	453	2.835	3.204	2.523
Timiskaming	266	2.738	3.110	2.605
Total	*1,274*		*3.204*	*2.503*
Resource Syenite	**67**	**2.692**	**3.051**	**2.563**

14.7 Evaluation of Outlier Data

The raw drill hole gold assay dataset was inspected globally using log cumulative probability plots to assess for the presence of high-grade outlier values that could adversely impact grade estimation. For this analysis, the datasets both external and internal to the 1.5 g/t Au grade solid were combined, as they are one contiguous deposit. After review of log probability plots, all raw gold assays were capped at 25.0 g/t Au prior to compositing (Figure 14-2). This assay cap (topcut) affects 380 meters of sample, and results in a reduction of 3.63% of gold metal on a grade-thickness (GT) basis. Capping levels in the range of 25.0 to 35.0 g/t are typical of deposits found within the Abitibi Belt.



	Untransformed Au Statistics								Log Normal Approximation Model		
	Au Cutoff = 0.01 g/t		Au Cutoff = 0.50 g/t		Au Cutoff = 3.00 g/t		Au Cutoff = 25.00 g/t			Standard	Third
	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Meters	Au (g/t)	Mean	Deviation	Parameter
Raw Assays	311,751	0.664	67,735	2.729	16,640	7.084	380	44.177	-1.12	0.90	0.00
incr. % and grade	78.3%	0.091	16.4%	1.311	5.2%	6.218	0.1%	44.177			
low cut	0.10		25 g/mt percentile		GT lost by capping		percent of GT >= 553 g/mt				
			99.73%		3.63%		0.14%				
Au cap (topcut)	25.00		percent of GT >= 25 g/mt		CV uncapped		CV capped				
			8.36%		3.86		2.80				

Figure 14-2: Cumulative Probability Plot: Raw Gold Assays

14.8 Compositing

All capped raw data were composited into 3 m downhole intervals. Composites were broken across the 1.5 g/t Au grade solid, and short composites were accounted for by using the length weighting process during grade estimation.

14.9 Variogram Analysis

Pairwise relative variograms were constructed for gold in Sage2001© software using the 3 m capped composite data. The resulting variograms exhibited high nuggets and relatively short ranges overall. The resultant fitted variograms for the major, semi-major and minor axes are provided in Figures 14-4 through 14-6, respectively. A summary of variogram parameters is provided in Table 14-3. Although these variogams do not generally reflect geologic field observations, they do suggest multiple controls on mineralization, and appear to loosely confirm a rake to mineralization observed in the plane of the orebody in longitudinal view.

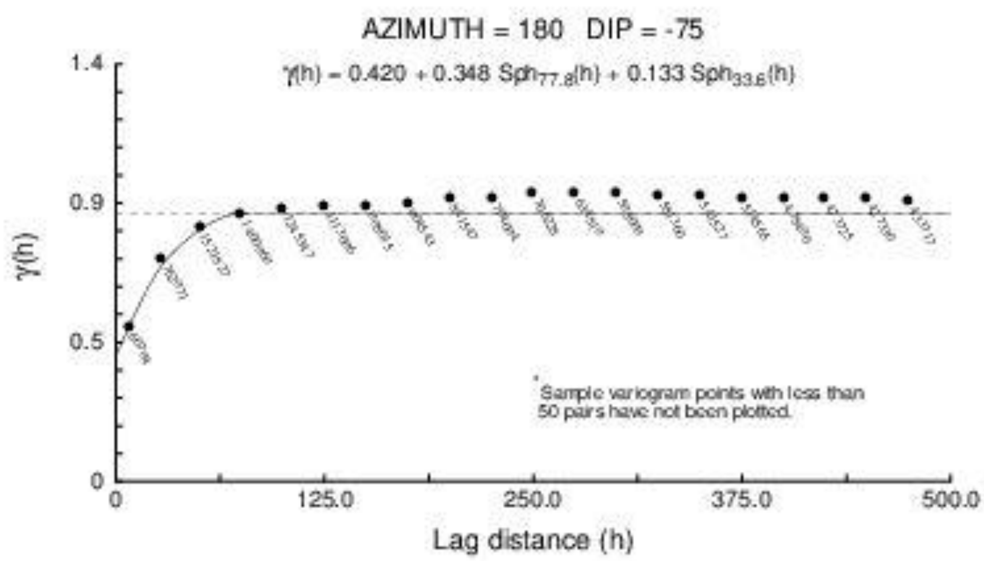


Figure 14-3: Pairwise relative variogram and model: Principle Axis

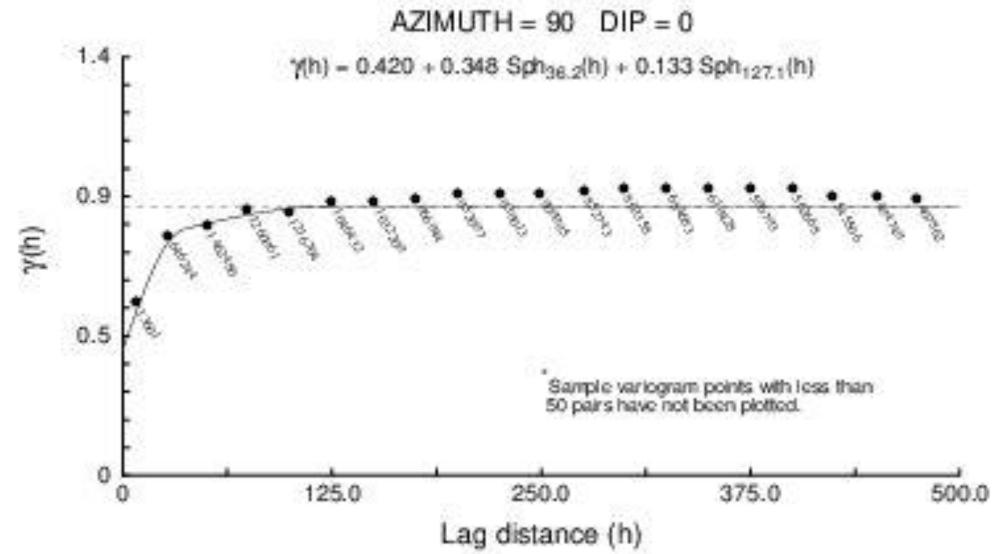


Figure 14-4: Pairwise relative variogram and model: Semi-Major Axis

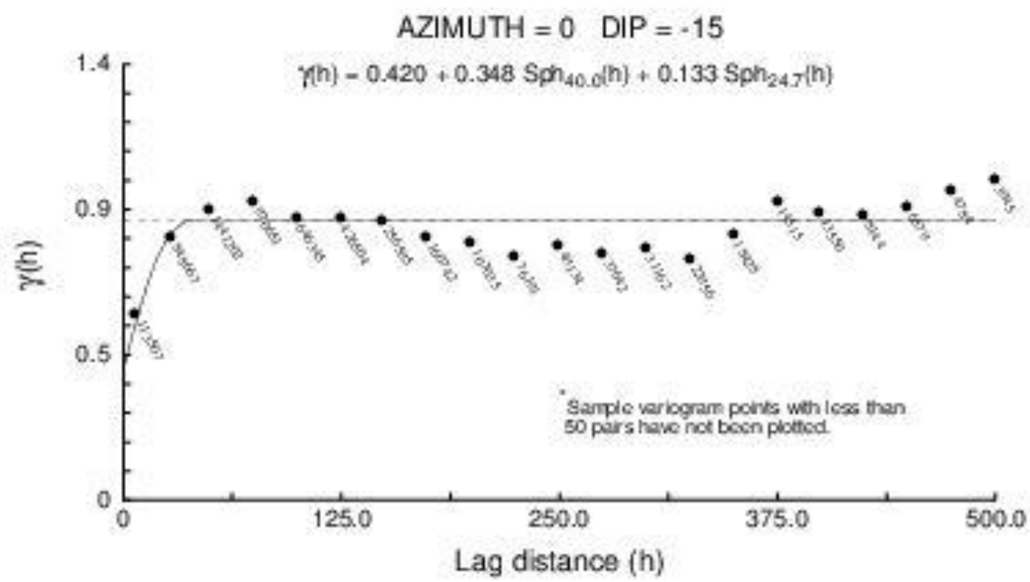


Figure 14-5: Pairwise relative variogram and model: Minor Axis

Table 14-3: Pairwise relative variogram model parameters for Au: 3 m composites

Young-Davidson UG: Variogram Results (Spherical Model)			
Parameters (First Structure)	Principal	Minor	Semi-Major
Azimuth (deg)	292	22	27

Dip (deg)	1	-12	77
Parameters (Second Structure)	Principal	Minor	Semi-Major
Azimuth (deg)	100	10	68
Dip (deg)	-1	-1	89
Nugget Effect C_0	0.420		
1st Structure C_1	0.348		
2nd Structure C_2	0.133		
1st Range A_1 (m)	33	42	73
2nd Range A_2 (m)	436	22	32

14.10 Block Model Construction

A block model was constructed in Vulcan™ for the Young-Davidson deposit using the model limits and extents provided in Table 14.4.

Block model construction utilized sub-celling, with a minimum dimension of 1.5 x 1.5 x 1.5 metres and a maximum dimension of 6 x 6 x 6 metres internal to the 1.5 g/t Au grade. Blocks external to the 1.5 g/t Au grade solid were allowed a maximum dimension of 18 x 12 x 18 metres, in order to assign an external diluting grade.

Table 14-4: Young-Davidson Underground Model Limits and Extents

Mine Grid Origin (m)	Min (m)	Max (m)	Parent Block Size (m) Internal to 1.5 g/t Au Grade Shell	Sub-cell Block Size (m) Internal to 1.5 g/t Au Grade Shell	Parent Block Size (m) External to 1.5 g/t Au Grade Shell	No. of Blocks
East	22,450	23,890	6.0	1.5	18.0	960
North	10,120	10,684	6.0	1.5	12.0	376
Elevation	8,820	10,404	6.0	1.5	18.0	1,056

14.11 Block Model Grade Estimation

The Mineral Resource estimate was undertaken using Maptek Vulcan™ software employing the inverse distance weighting method (ID3) and nearest neighbor (NN) method for use in model validation. The Sub-celled block model and composites were coded by the 1.5 g/t Au grade solid for retrieval during grade estimation. A summary of the search parameters utilized is presented in Table 14.5.

Table 14-5: Search Parameters for Young-Davidson Underground Model

Pass Number	Search Orientation (degrees)			Search Distance (m)			Min Comps	Max Comps	Max per DDH
	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis	Semi-Major Axis	Minor Axis			
1	90	0	-75	30	60	15	3	5	1
2	90	0	-75	60	120	30	2	5	1
3	90	0	-75	100	200	50	1	5	1

14.12 Block Model Validation

Various measures have been utilized to validate the resultant Mineral Resource block model. These measures include the following:

- Comparison of drill hole composites with Mineral Resource block grade estimates by zone visually, both in plan and section;
- Statistical comparisons between block and composite data using histogram and cumulative frequency distribution analysis;
- Generation of comparative Nearest Neighbor (NN) model; and
- Swath plot analysis (drift analysis) comparing the ID3 model with the NN model.

14.12.1 Visual Inspection

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement, which would be expected considering the estimation methodology employed.

Representative north-south cross sections displaying block and composite gold grade, 1.5 g/t Au grade solid outlines and existing development are provided in Figures 14-6 and 14-7 respectively, split at the 9,500 metre level to allow graphical resolution.

An example level plan displaying block and composite gold grade, 1.5 g/t Au grade solid outlines and existing development are provided in Figures 14-8 and 14-9 respectively, split at the 9,500 metre level to allow graphical resolution.

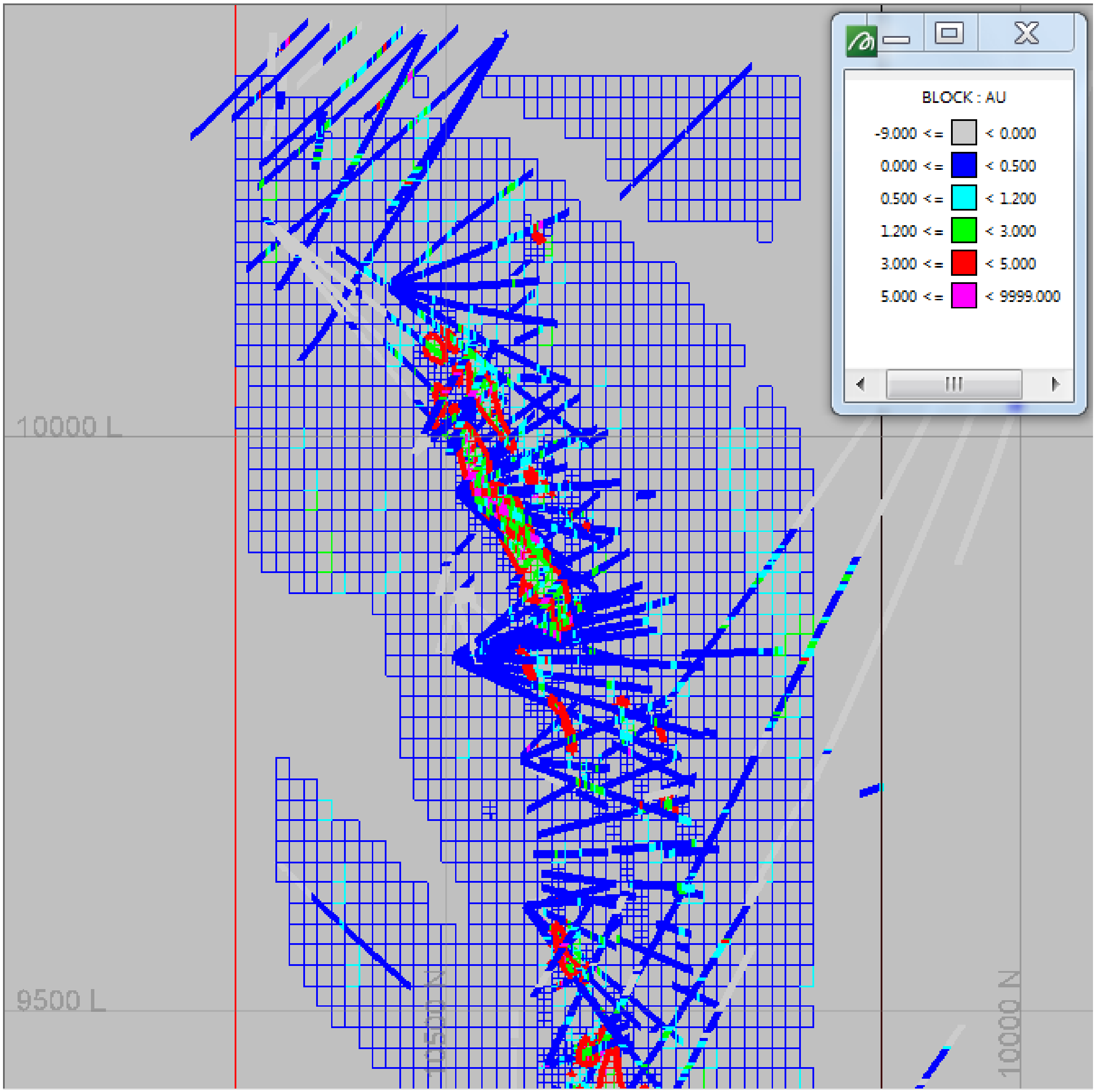


Figure 14-6: Example North-South Cross Section 23,220 East – Above 9,500 Elevation

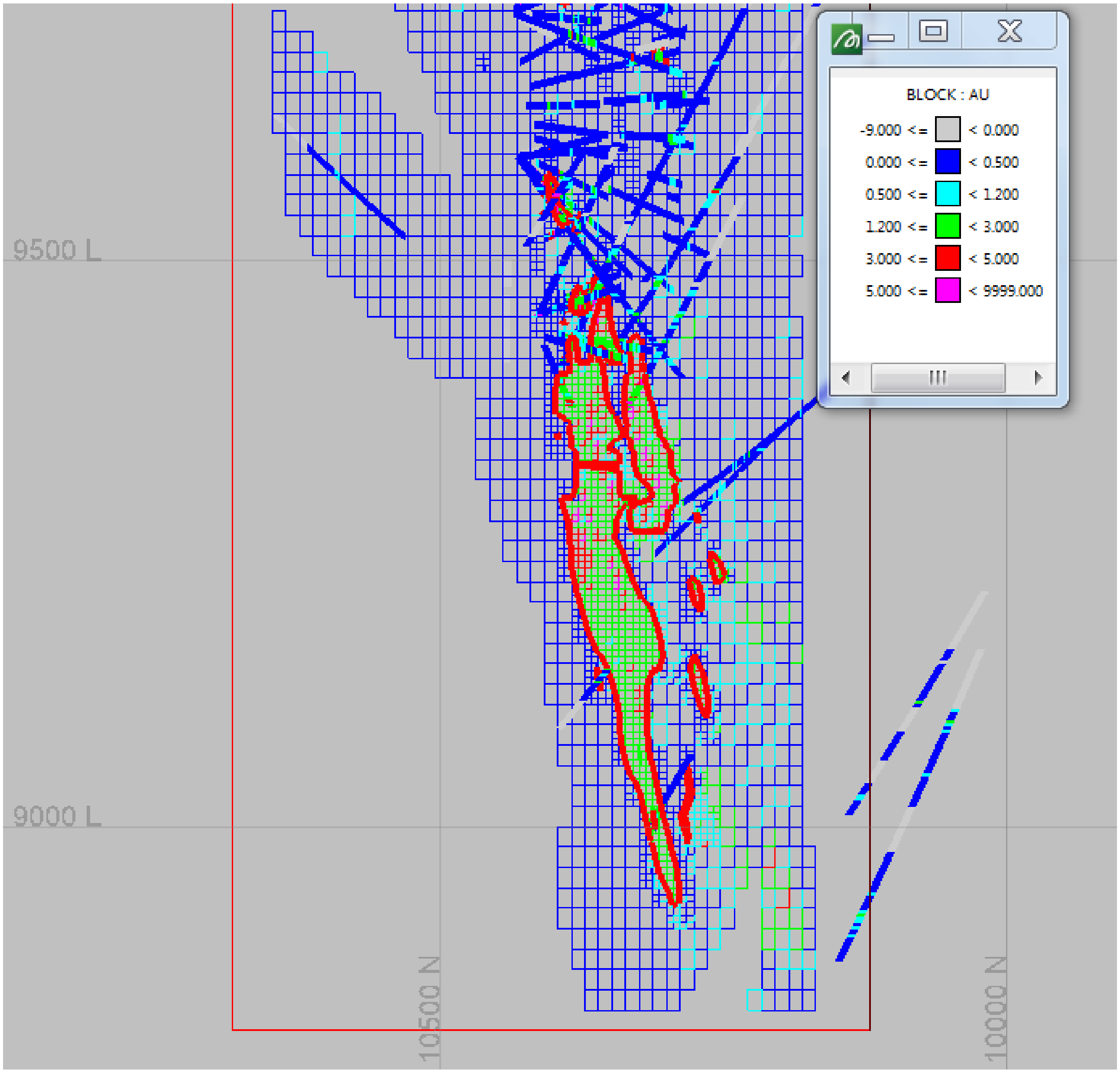


Figure 14-7: Example North-South Cross Section 23,220 East – Below 9,500

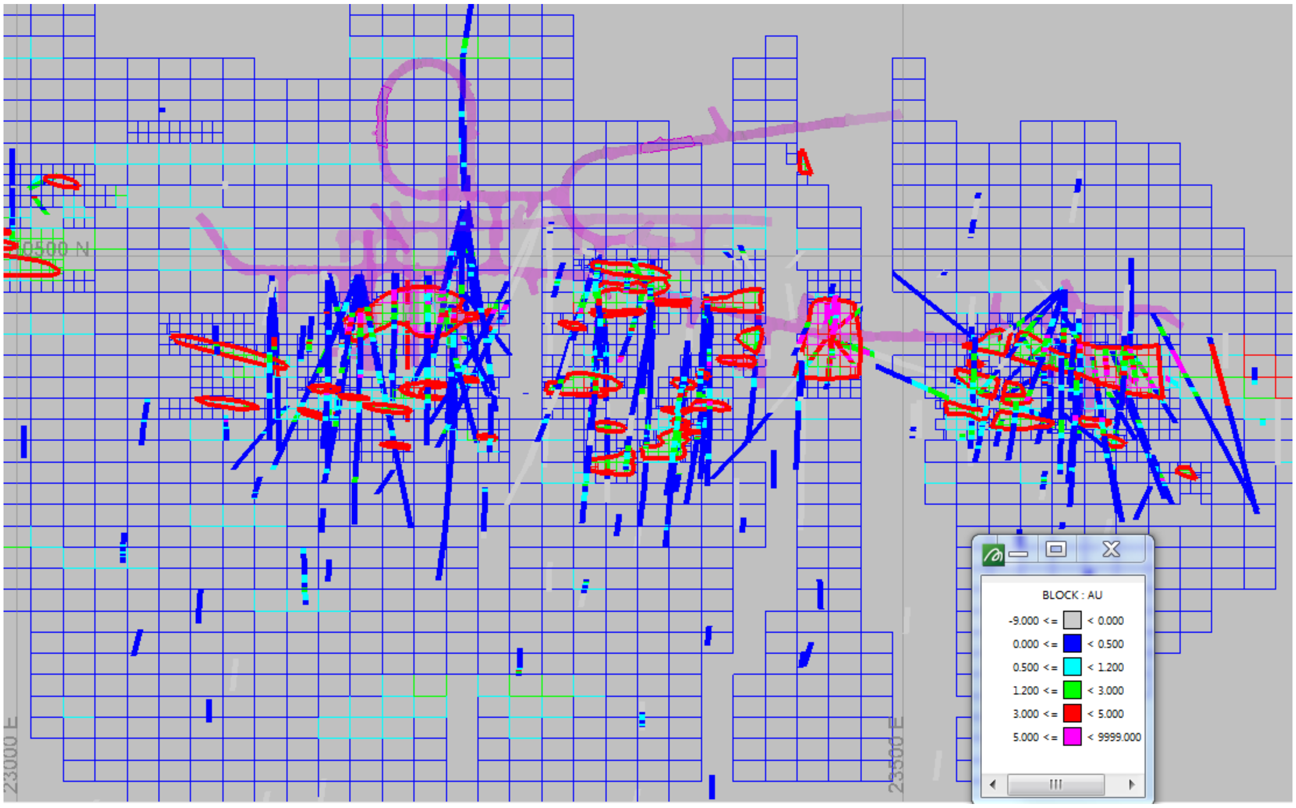


Figure 14-8: Example Level Plan 9960 m – East of 23,050 m

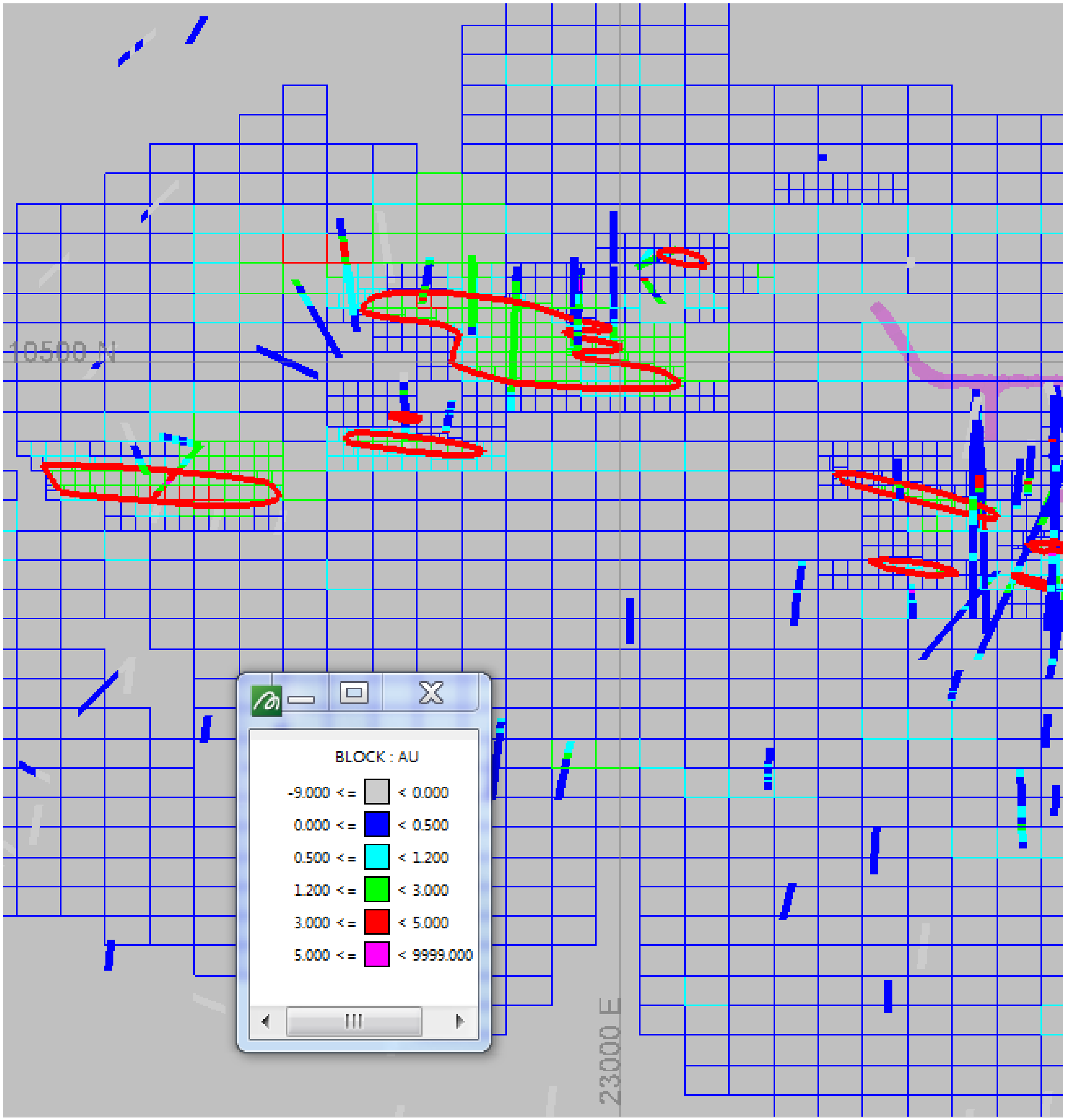
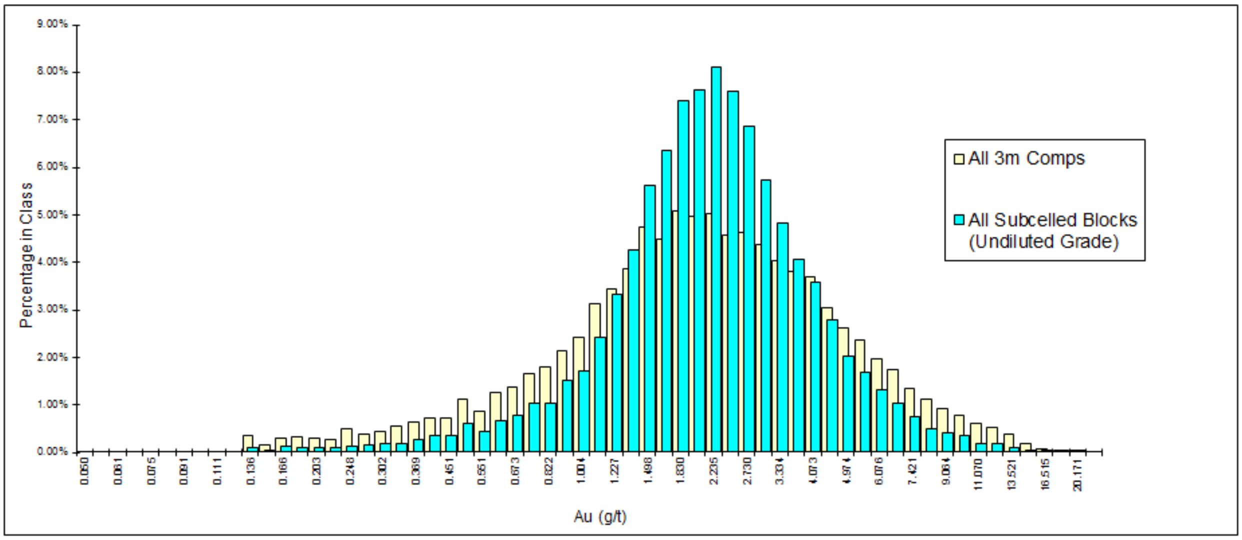


Figure 14-9: Example Level Plan 9960 m – West of 23,050 m

14.12.2 Block-Composite Histogram Comparison

Alamos also conducted statistical comparisons between the grades of the Indicated and Inferred ID3 blocks contained within the 1.5 g/t Au solid and the underlying gold composite grades. A histogram comparison between block and composite gold grades at the Young-Davidson underground mine is provided in Figure 14-10.



Untransformed Au Statistics

	Au Cutoff = 0.010 g/t		Au Cutoff = 0.500 g/t		Au Cutoff = 1.50 g/t		Au Cutoff = 5.00 g/t	
	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)	meters/tonnes	Au (g/t)
All 3m Comps	44,639	2.882	40,426	3.160	30,088	3.890	6,601	7.836
incr. % and grade	9.4%	0.215	23.2%	1.037	52.6%	2.780	14.8%	7.836
All Subcelled Blocks (Undiluted Grade)	69,869,092	2.719	67,940,203	2.788	55,477,668	3.166	6,047,591	7.222
incr. % and grade	2.8%	0.291	17.8%	1.107	70.7%	2.669	8.7%	7.222

Figure 14-10: Histogram Comparison between Block and Composite Grades: Gold

Overall, this comparison shows that the model grade distribution for gold is appropriately smoothed when compared with the underlying composite distribution, and that the comparison of average grades and percentages above geologic absolute and incremental cut-offs show close agreement.

14.12.3 Comparison of Interpolation Methods

For comparative purposes, additional grades were estimated using Nearest Neighbor (NN) interpolation methods. The results of the NN model are compared to the ID3 model at a 0 g/t Au cut-off grade for the measured and indicated blocks in Table 14.6, and Table 14.7 for all inferred blocks, internal to the 1.50 g/t gold grade solid. These comparisons confirm the conservation of metal at a zero cut-off, and shows close agreement on both a tonnage and grade basis within the deposit area.

Table 14-6: ID3 vs. NN Tonnage – All Measured and Indicated Model Blocks

Model	Tonnes (000's)	Au Grade (g/t)	Au Oz (000's)
ID3	65,722	2.74	5,790
NN	65,722	2.73	5,758
% Difference	*0.00%*	*0.55%*	*0.55%*

Table 14-7: ID3 vs. NN – All Inferred Model Blocks

Model	Tonnes (000's)	Au Grade (g/t)	Au Oz (000's)
ID3	4,148	2.47	329
NN	4,148	2.48	330
% Difference	*0.00%*	*-0.4%*	*-0.4%*

14.12.4 Swath Plots (Drift Analysis)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the ID3 model are then compared (using the swath plot) to the distribution derived from the NN grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but on a larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the ID3 model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots have been generated in three orthogonal directions for distribution gold in the model area. Swath plots for gold along the EW, NS and vertical directions are shown in Figures 14-11 through 14-13.

There is good correspondence between all models in all orthogonal directions. The degree of smoothing in the ID3 model is evident in the peaks and valleys shown in the swath plots, however, this comparison shows close agreement between the IDW and NN models in terms of overall grade distribution as a function of X, Y and Z location.

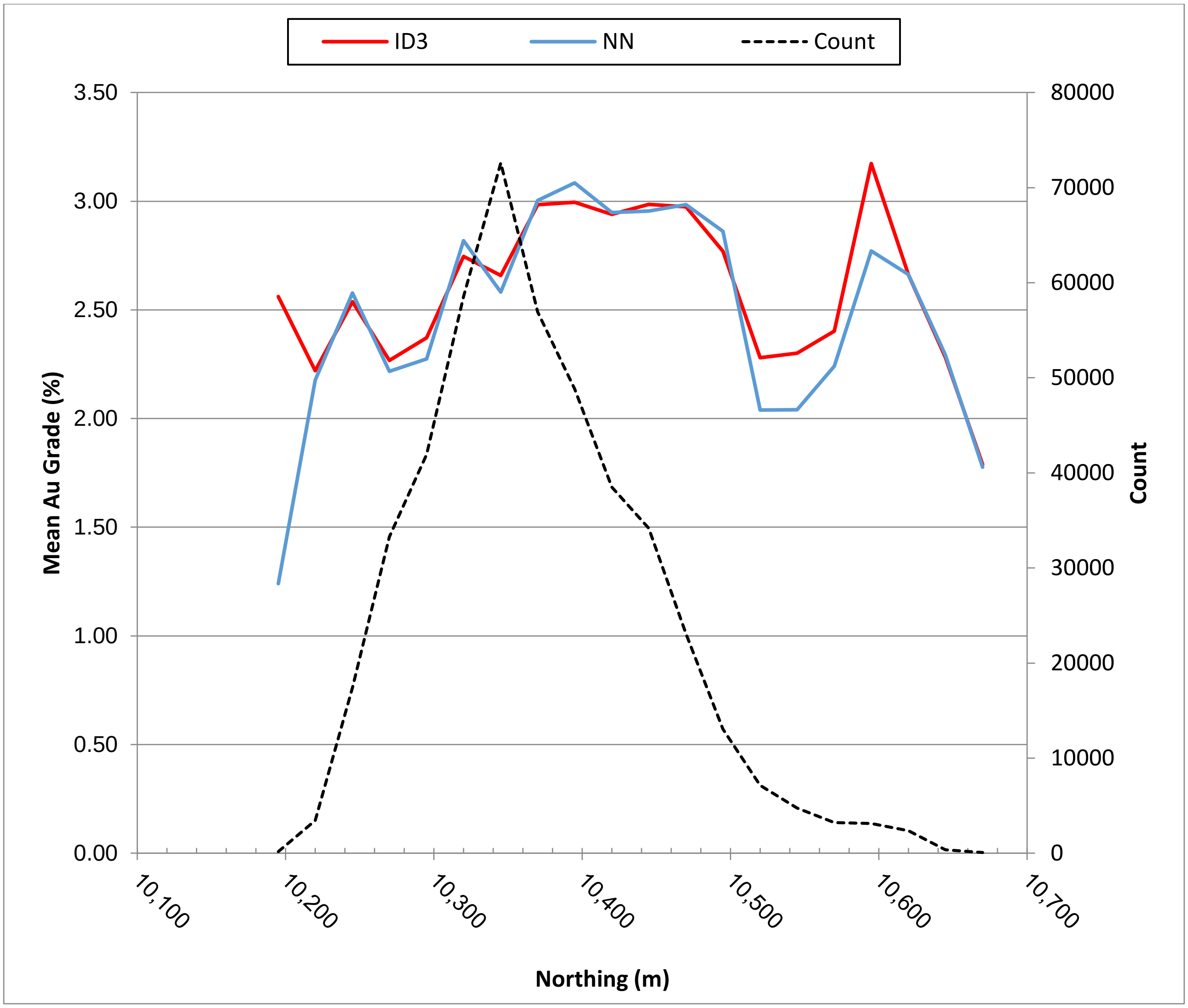


Figure 14-11: E-W Swath Plot, Comparing IDW and NN Model Gold Grades

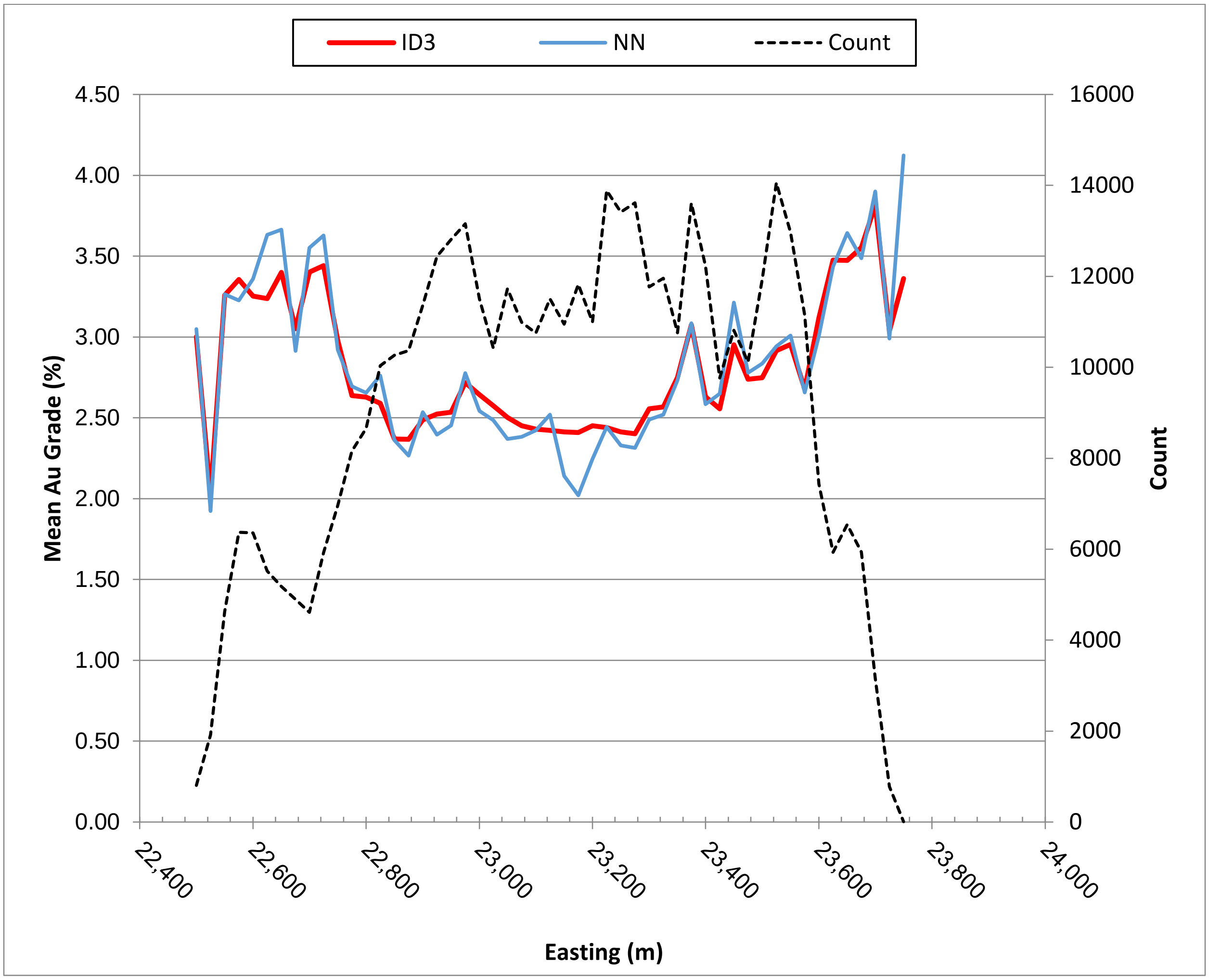


Figure 14-12: N-S Swath Plot, Comparing IDW and NN Model Gold Grades

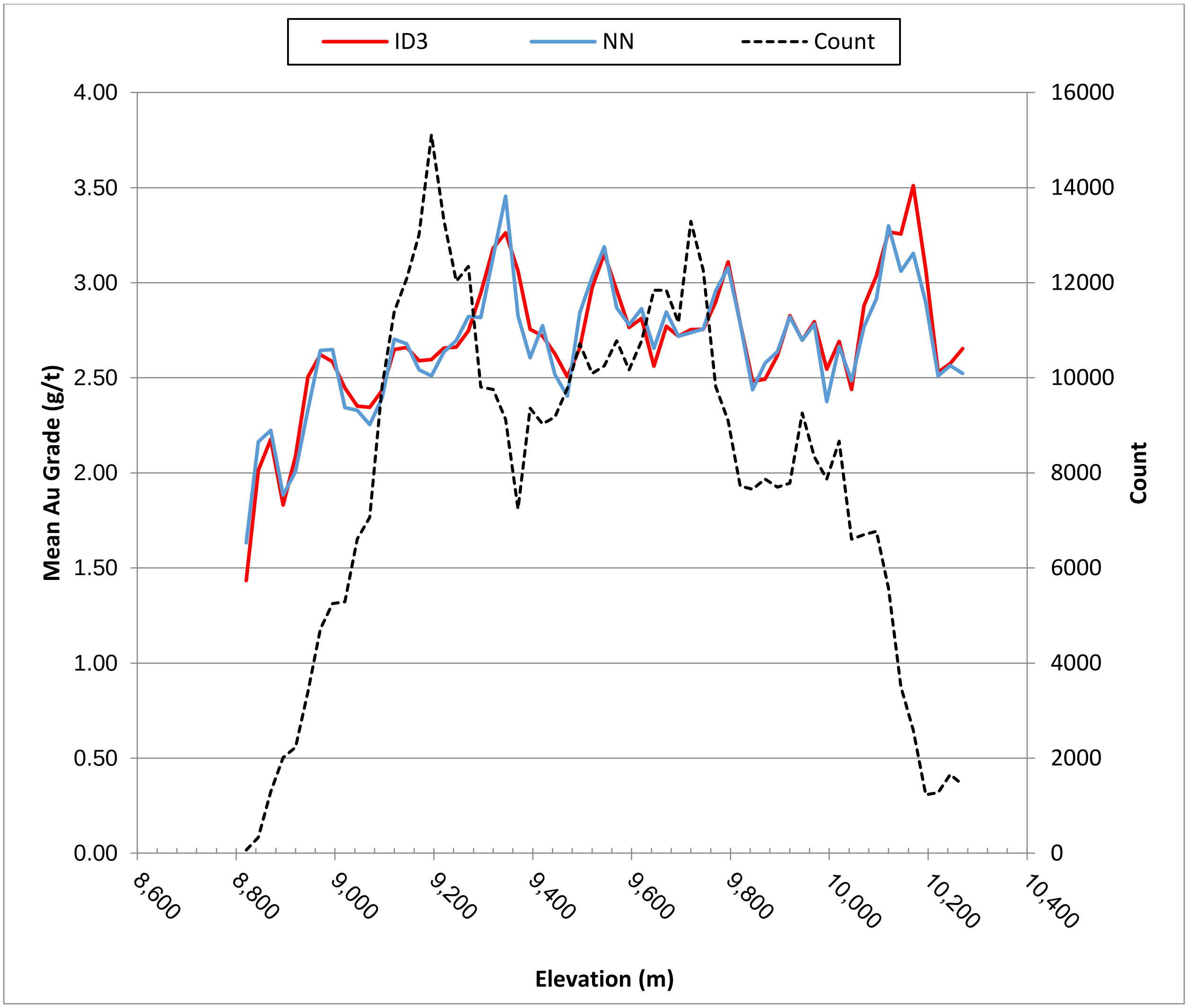


Figure 14-13: Vertical Swath Plot, Comparing IDW and NN Model Gold Grade

14.13 Mineral Resource Sensitivity

In order to assess the sensitivity of the Mineral Resource to changes in gold cut-off grade, Alamos has summarized tonnage and grade above cut-off, within the 1.50 g/t grade shell, at a series of increasing gold cut-offs by Mineral Resource category. The cut-off grade sensitivity analysis for all Measured and Indicated blocks (undepleted for past mining and inclusive of Mineral Reserves) within the Young-Davidson underground deposit are provided in Table 14.8. The cut-off grade sensitivity analysis for inferred blocks within the Young-Davidson underground deposit are provided in Table 14.9. It can be observed that the Mineral Resource is reasonably insensitive to cut-off grades in the increment between 1.50 and 1.80 g/t Au, which is likely the grade range of economic interest. The base case cut-off grade is shown in bold.

Table 14-8: Cut-off Gold Grade Sensitivity – All Measured and Indicated Blocks

Cut-off (g/t Au)	Tonnes (000's)	Au Grade (g/t)	Au OZ (000's)
0.50	63,944	2.81	5,773
0.80	61,915	2.88	5,730
1.00	60,045	2.94	5,676
1.20	57,626	3.02	5,590
1.40	54,224	3.12	5,447
1.50	**52,230**	**3.19**	**5,354**
1.60	49,999	3.26	5,243
1.80	45,008	3.43	4,970
2.00	40,149	3.62	4,673
2.20	35,337	3.83	4,349
2.40	30,920	4.05	4,022

Table 14-9: Cut-off Gold Grade Sensitivity – All Inferred Blocks

Cut-off (g/t Au)	Tonnes (000's)	Au Grade (g/t)	Au OZ (000's)
0.50	3,996	2.55	328
0.80	3,795	2.65	323
1.00	3,727	2.68	321
1.20	3,606	2.74	317
1.40	3,374	2.83	307
1.50	**3,247**	**2.89**	**301**
1.60	3,114	2.94	295
1.80	2,830	3.07	279
2.00	2,550	3.20	262
2.20	2,216	3.36	240
2.40	1,838	3.58	212

14.14 Mineral Resource Classification

The Mineral Resources for Young-Davidson are classified under the categories of Measured, Indicated and Inferred according to the guidelines as defined by the "CIM Definition Standards for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the Mineral Resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data and numerous other factors which influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drill holes used to estimate the block grades and the block-composite separation distance. These classification criteria are intended to encompass zones of reasonably continuous mineralization.

The following classification parameters were applied to the Young-Davidson block model:

Measured Mineral Resources

Blocks in the model which are within the 1.50 g/t Au solid that were informed by a minimum of 3 drill holes on the first estimation search pass. (30 x 60 x 15 metres)

Indicated Mineral Resources – Blocks in the model which are within the 1.50 g/t Au solid that were informed by a minimum of 2 drill holes on the second estimation search pass (60 x 120 x 30 metres) or, all estimated blocks above the 9590 metre elevation that are completely drilled off, from a stope definition perspective, which is targeted at 20 m x 20 m spacing. Below the 9590 metre elevation, any blocks estimated on the third search pass which were informed by 5 composites/drill holes within a 100 m maximum average distance were re-assigned to the indicated category.

Inferred Mineral Resources – Blocks in the model that do not meet the criteria for Measured or Indicated Resources and have been informed by a minimum of one drill hole on the third estimation search pass.

Based on the positive mine reconciliations to date, the block model adequately represents realized tonnes and grade mined through December 31, 2015.

14.15 Mineral Resource Statement - Underground

The Mineral Resources, as at December 31, 2015, for the Young-Davidson Underground deposit have been estimated by Alamos at 7,955 kt grading an average of 3.45 g/t gold classified as Measured and Indicated Mineral Resources; with an additional 3,523 kt grading an average of 2.76 g/t gold classified as Inferred Mineral Resources. The Mineral Resources are stated above a 1.5 g/t gold cut-off and contained within a potentially economically mineable grade solid. Mineral Resources are exclusive of Mineral Reserves.

The Mineral Resources are reported in accordance with NI 43-101 and have been classified in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc. Mr. Volk has over 27 years of operational and consulting experience in the minerals industry, specifically in Mineral Resource estimation, production geology, feasibility studies and economic evaluations. Mr. Volk is a Certified Professional Geologist and a Qualified Person as defined by international reporting codes.

The effective date of this Mineral Resource estimate is December 31, 2015 and is based on drilling data finalized in November, 2015. The Mineral Resource statement for the Project is presented in Table 14.10.

Table 14-10: Underground Mineral Resource Statement, December 31, 2015

Category	Tonnes (000's)	Au Grade (g/t)	Au Ounces (000's)
Measured	4,248	3.47	474
Indicated	3,707	3.43	408
Total Measured and Inferred	**7,955**	**3.45**	**883**
Inferred	3,523	2.76	312

Notes to Mineral Reserves:

- The effective date is December 31, 2015
- The Mineral Resource estimate was completed by Mr. Jeffrey Volk, CPG, FAusIMM, Director of Reserves and Resources for Alamos Gold Inc.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
- Underground Mineral Resources are stated as contained within potentially economically minable gold grade shapes above a 1.50 g/t Au cut-off and as such includes internal dilution within the 1.50 g/t Au solid.
- The gold price used for Mineral Resources was USD $1400 per ounce.
- Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
- Contained Au ounces are in-situ and do not include metallurgical recovery losses.
- Mineral Resources are exclusive of Mineral Reserves.

14.16 Open Pit Block Mineral Resources

Open pit operations commenced in September, 2012, and ceased operation in June, 2014. The current open pit Mineral Resource model was constructed in April, 2008 and formed the basis for the AMEC feasibility study. This model has not been modified since 2008, and the reader is referred to the AMEC document for details, as only a small remnant Mineral Resource remains. A summary of the remaining Mineral Resource is provided in Table 14.11.

Table 14-11: Open Pit Mineral Resource Statement, December 31, 2015

Category	Tonnes (000's)	Au Grade (g/t)	Au Ounces (000's)
Measured	496	1.13	18
Indicated	1,242	1.28	51
Total Measured and Indicated	**1,739**	**1.24**	**69**
Inferred	31	0.99	1

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserve estimate assumed the following mining methods would be employed:

- Underground transverse stoping with backfill
- Underground longitudinal retreat stoping with backfill
- Processing of surface stockpiles of low grade material from the mined out open pit

15.2 Mineral Reserve Statement

Mineralization that had been classified as Measured or Indicated Mineral Resources was used to support estimation of Mineral Reserves. Mineral Reserve estimates were prepared by Tyler Clarke (P. Eng.), the Chief Engineer at Young-Davidson. The Mineral Reserve estimates were reviewed and approved by Chris Bostwick (FAusIMM), Vice-President Technical Services for Alamos Gold. Mr. Bostwick is the Qualified Person responsible for the estimates.

Mineral Reserves for Young-Davidson are tabulated in Table 15-1. Mineral Reserves are estimated using a gold price of US$1,250 per ounce, and assume a long-term exchange rate of CAD$1= US$0.85. The cutoff grade used to report underground estimates is 1.90 g/t gold. The cutoff grade used to report surface stockpiles is 0.50 g/t gold.

Table 15-1: Young-Davidson Mineral Reserves as at December 31, 2015

	Tonnes (x 1,000)	Au Grade (g/t)	Au Ounces (x 1,000)
Underground – Proven	14,282	2.73	1,255
Underground – Probable	30,008	2.68	2,582
Underground - Proven and Probable	**44,290**	**2.69**	**3,837**
Stockpiles – Proven	1,396	0.82	37
Total Proven and Probable	**45,686**	**2.64**	**3,874**

Notes to accompany Mineral Reserve table:

- The estimate has an effective date of December 31st, 2015.
- Chris Bostwick, FAusIMM, Vice President Technical Service, Alamos Gold is the Qualified Person for the estimate.

- The Mineral Reserves are classified as Proven and Probable Mineral Reserves, and are based on the 2014 CIM Definition Standards.
- Mineral Reserve estimates are based on a gold price of USD$1,250 per ounce, and a long-term exchange rate of CAD$1= USD$0.85.
- For underground estimates, a gold cutoff grade of 1.90 grams per tonne is used as an economic indicator only.
- For the surface stockpiles, a gold cutoff grade of 0.50 grams per tonne is used.
- The estimated gold metallurgical recovery rate is 91%.
- Underground mining dilution ranges from 8.4% to 10.9% depending on mining method.
- Underground Mineral Reserves take into account mining recoveries of between 90% and 95% depending on mining method.
- Numbers may not sum due to rounding.

15.3 Factors That May Affect the Mineral Reserve Estimate

The major risk factors that can affect the Mineral Reserves estimates are: metal price and exchange rate assumptions, capital and operating cost assumptions, royalties and taxes, geotechnical stability and dilution assumptions, hydrogeological constraints, geological interpretations, environmental and permitting status, and maintaining a social license to operate.

15.4 Mineral Reserve Estimation Methodology

Mineral Reserve shapes are generated using two different methodologies depending on the area of the mine. The mine is broken into two major sections; 10190 level to 9590 level ("Upper Mine"), and 9590 level to 8850 level ("Lower Mine"). This is because the current producing areas and areas in advanced stages of lateral development are all above 9620 level and these areas require a higher level of detail.

All Mineral Reserve shapes are based off of the Mineral Resource mineralized zones released by the mine geology department. As the deposit is grade constrained, the mineralized zone is better described as a 1.5 gram per tonne grade shell. All shapes are evaluated against the corresponding end of year block model.

15.4.1 Upper Mine Mineral Reserve Shape Generation

The Mineral Reserve shapes above 9620 level are generated using MineRP's Mine 2-4D v12 software. Sub projects are made for each stoping block (between sill levels).

All existing development, stoping voids, passes, and conceptual development is brought in. Stopes in the later design phases are also incorporated, and can be seen in Figure 15-1 as a series of lines near the ends of the existing development. Later phase stopes are designed in section for a final drill and blast shape.

Mineral Reserve stopes are then designed in plan by drawing strings around all existing stopes, existing and conceptual development, as well as around any stopes in later design phases using the mineralized zone (shown as a red outline in Figure 15.1) as a guide. Mineral Reserve shapes are made to approximate drill and blast shapes by matching the planned development and incorporating design dilution as may be necessary. Sections are cut every 2.5 – 5.0 metres in elevation. Strings used to create Mineral Reserve shapes are shown in Figure 15-2.

This process is also undertaken between 9620 and 9590 Levels to tie interface the different wireframing methods used in the Upper Mine and the Lower Mine.

General design criteria are as follow, however, to be practically extracted it is sometimes required to work outside of these parameters to accommodate the shape of the ore body in the area.

- Longitudinal Stoping
- Minimum width of 3 metres
- Maximum strike of 20 metres
- Transverse Stoping
- Maximum strike of 20 metres
- Maximum horizontal thickness perpendicular to strike of 40 metres
- Minimum horizontal thickness perpendicular to strike of 10 metres

Given that the horizontal thickness perpendicular to strike is often greater than 30 metres, transverse stope may be divided into A and B panels with the hanging wall panel mined first in the sequence.

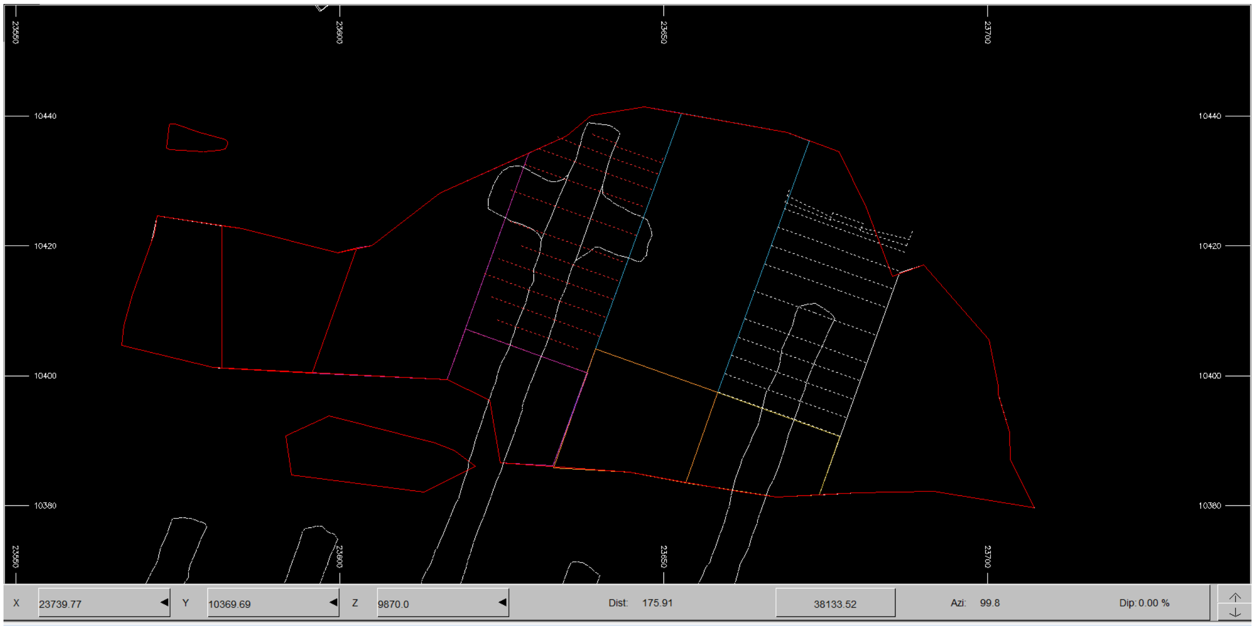


Figure 15-1: 9870 Level in Plan View

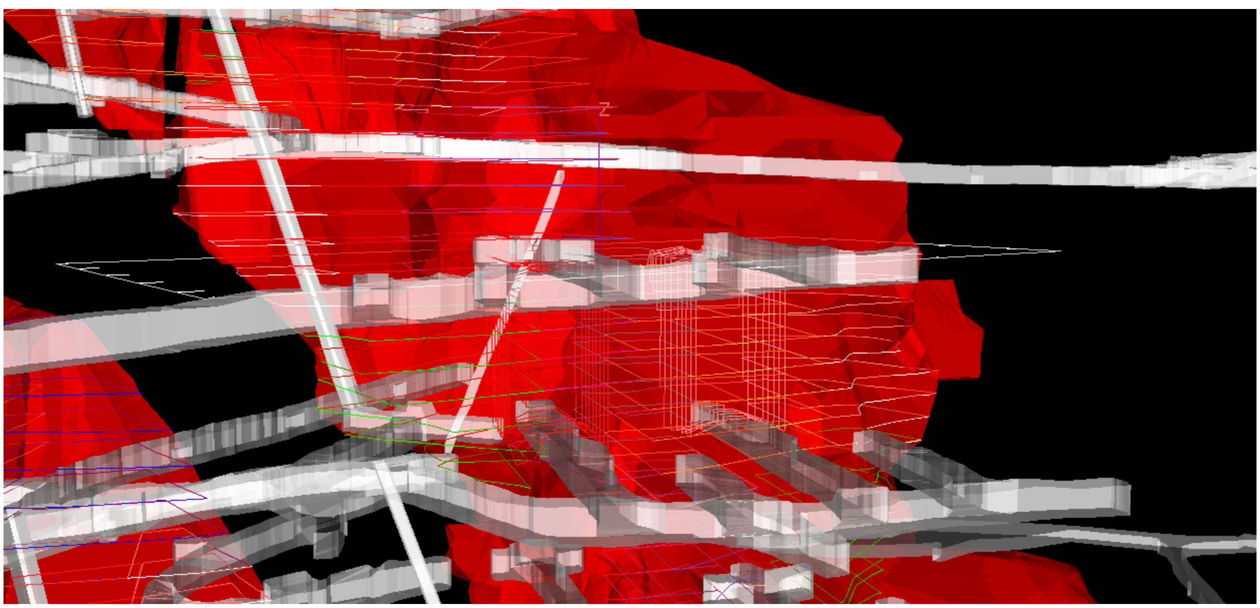


Figure 15-2: Existing Dev., Mineralized Zone Wireframe, and Mineral Reserve Shape

The strings for each stope are given a unique set of attributes through a combination of line style, colour, and point symbol. This is how Mine 2-4D distinguishes which strings belong to an individual stope. Additional attributes are also assigned to the strings to indicate whether it is a transverse or longitudinal stope and the level name of the mucking horizon.

The strings are then made into wireframes, shown in Figure 15-3. The resulting wireframes are then assigned Northings and Eastings using a computer algorithm.

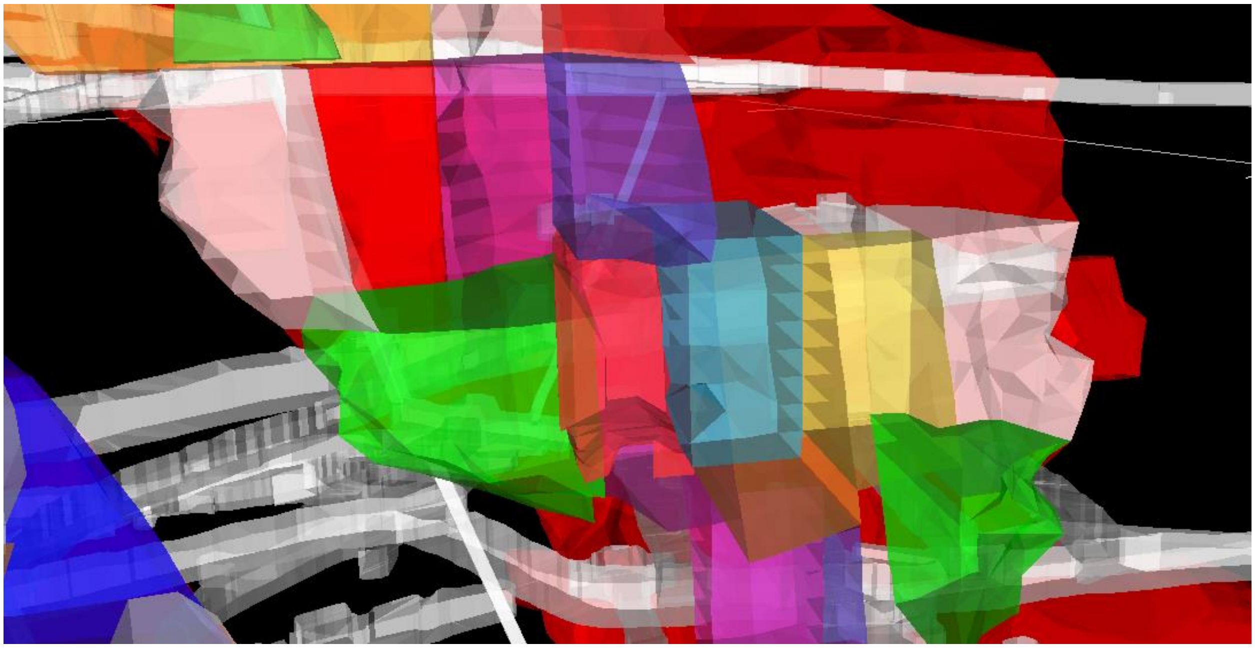


Figure 15-3: Multi-coloured Stope Wireframes with Existing Dev. and Mineralized Zone

After this process has been completed for each of the sub-projects they are combined into a single project. The wireframes in the combined project are then run against the Mineral Resource block model for the purposes of checking for disparities between the Mine 2-4D evaluations and the Vulcan evaluation. The results are then exported to CAE's Enhanced Production Scheduler (EPS). The stope wireframes are exported to an AutoCAD dxf file with the layer field set to the attribute "M4DNUM". This is an attribute unique to an individual stope and is used to tie the attribute values from EPS to the evaluations results from Vulcan when doing Mineral Reserves calculations in Excel.

15.4.2 Lower Mine Mineral Reserve Shape Generation

Mineral Reserve shapes below the 9590 level are generated using Maptek's Vulcan Stope Optimizer software. This is a much less labour intensive method of generating shapes. The Stope Optimizer is used in the lower area of the mine because the detail requirement is not as high as it is for short to medium term mine planning as is the case in the Upper Mine. The two methods have been compared in the same areas in the past, and the variance between the results was always found to be less than 5%.

The Stope Optimizer is run with the following inputs:

- Strike Length
  - Minimum 10 metres
  - Maximum 20 metres
- Height
  - Minimum 15 metres
  - Maximum 30 metres
- Width greater than 3 metres
- Cut-Off Grade 1.7 g/t undiluted (final cutoff grade is applied in the Mineral Reserve calculations)

This generates a series of Mineral Reserve shapes approximating drill and blast shapes. These shapes come with attributes such as tonnes, grade, Northing, Easting, elevation, and average width which are used for calculating the Mineral Reserves.

15.4 Vulcan Evaluations

All stope shapes are evaluated against the Mineral Resource block model in Vulcan. They are evaluated using the Advanced Reserves Editor utility. Evaluations are run twice for each group of shapes; once for total tonnes and grade of the shape, and another breaking tonnes and grade into category (measured, indicated, or inferred). Outputs are saved as .csv files.

All internal dilution, or planned dilution, is included in the total tonnes and grade of the stope when evaluated. This planned dilution consists of individual Mineral Resource block model blocks below cutoff grade, and is considered a part of the undiluted stope shape. Unplanned dilution is later applied using factors, and the resulting tonnes and grade numbers are considered to be the diluted tonnes and grade.

15.5 Mineral Reserve Calculations

Calculations on Mineral Reserves are done in Excel using a series of inputs and the exported data from EPS, Vulcan Stope Optimizer, and the Advanced Reserves Editor utility. Inputs are as follows:

- Cut-Off Grade
- Longitudinal Dilution Factor: 10.9% unplanned dilution
- Longitudinal Dilution Grade: 0.30 g/t unplanned dilution
- Longitudinal Mining Recovery: 95% expected extraction
- Transverse Dilution Factor: 8.4% unplanned dilution
- Transverse Dilution Grade: 0.15 g/t unplanned dilution
- Transverse Mining Recovery: 95% expected extraction

- Sill Pillar Mining Recovery: 85% expected extraction (more difficult mining conditions)

All unplanned dilution, mining recovery, and the dilution grades are considered average global factors without special allowances being made for the varying percentage of dilution which would be expected when mining adjacent to the hanging wall, footwall, unmined stopes long strike, sill stopes, or previously mined paste fill stopes.

The unique identification number of each stope wireframe is then used to link data together in Excel to ensure the correct dilution and recovery factors are used, assign a name in the format of LEVEL_NORTHING, and categorize the stope as measured, indicated, or inferred.

Finally, stopes are evaluated against the selected cutoff grade, with those stopes above the cutoff grade summarized by Mineral Reserve category (Proven or Probable) for Mineral Reserve reporting.

15.5 Cut-off Grade

The underground cut-off grade is calculated using the following inputs:

- Gold price – USD$1,250/oz
- Exchange rate – CAD$1= US$0.85
- Mill Recovery – 91.0%
- Mine Operating Cost – CAD$37.00/tonne mined
- Mill Operating Cost – CAD$14.50/tonne processed
- General & Administration (G&A) – CAD$4.00/tonne processed
- Total operating cost – CAD$55.50/tonne mined
- Royalty – 1.5% NSR

Cut-off Grade = (Operating Costs) / (((Gold Price * Recovery) * (1 - Royalty)) / 31.1035)

= $55.50 / ((($1,250 / 0.85 * 91.0%) * (1 - 1.5%)) / 31.1035)

= 1.31 gram per tonne gold

Despite the 1.31 gram per tonne calculated cutoff grade, Alamos has elected to report Mineral Reserves at an elevated cutoff grade of 1.90 gram per tonne gold.

15.6 Cut-off Grade Sensitivity

Table 15.2 and Figures 15.4 and 15.5 depict the underground Proven and Probable Mineral Reserve sensitivity to a range of gold cut-off grades, with the declared Mineral Reserve cut-off grade highlighted in red. Values were obtained by adjusting the cut-off grade up and down and having diluted stopes added or subtracted to the Mineral Reserve total. It should be noted that this exercise was carried out only within the fixed 1.50 gram per tonne wireframe, used to generate the Mineral Resource block model and the stoping outlines referred to earlier in this section. The use of lower grade (and larger) wireframes would generally increase the tonnes and ounces, but decrease the average grade, at cut-off grade lower than 1.50 grams per tonne.

Table 15-2: Proven and Probable Underground Mineral Reserve Sensitivity to Au Cutoff Grade

Au Cutoff Grade (g/t)	Tonnes (x 1,000)	Au Grade (g/t)	Au Ounces (x 1,000)
1.10	57,409	2.45	4,526
1.30	56,393	2.47	4,486
1.50	54,201	2.52	4,387
1.70	50,195	2.59	4,180
1.90	**44,290**	**2.69**	**3,837**
2.10	35,810	2.86	3,292
2.30	29,738	2.99	2,863
2.50	22,784	3.18	2,329

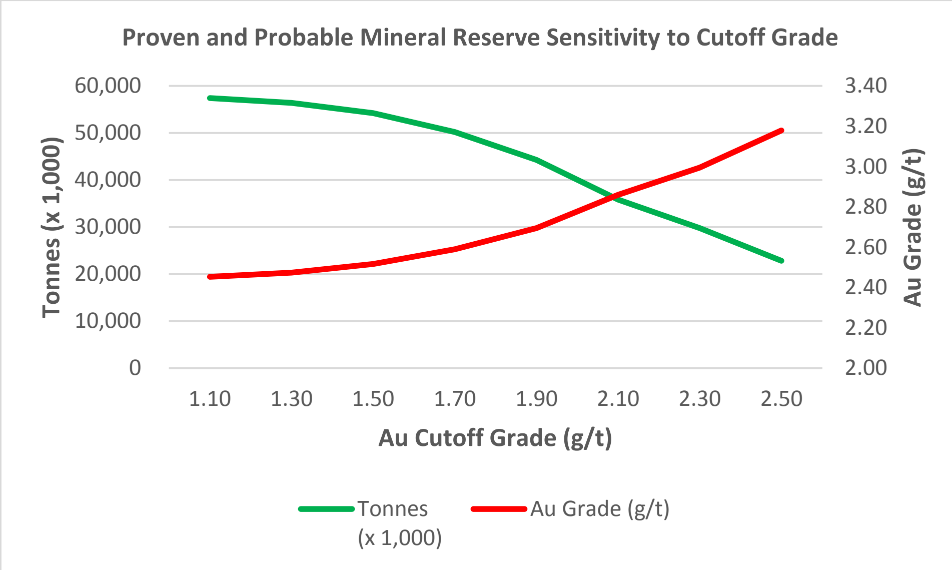


Figure 15-4: Underground Tonnes and Grade Sensitivity to Au Cutoff Grade

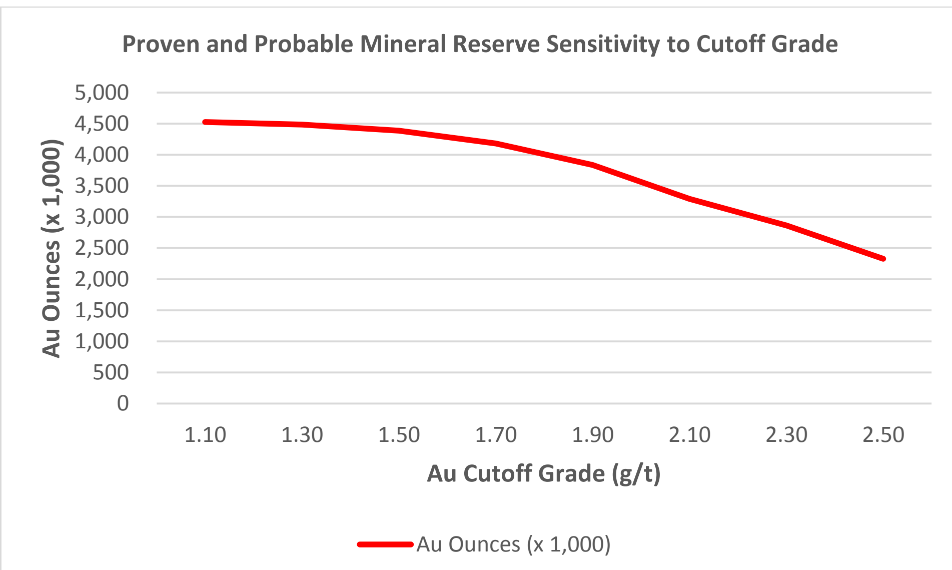


Figure 15-5: Underground Contained Ounce Sensitivity to Au Cutoff Grade

15.7 5-year Mineral Reserve History

Table 15.3 presents the recent Proven and Probable Mineral Reserves history for the Young-Davidson underground. Some observations follow:

- The 2010 Proven and Probable Mineral Reserve reflects those Mineral Reserves outlined in the August 2009 Pre-Feasibility Study and NI 43-101 Technical Report.
- The substantial step change in Mineral Reserves between 2010 and 2011 is attributable to:
    - The inclusion of the YD West Mineral Resource, first reported in August 2011.
    - The increase in the gold price used for Mineral Reserves from USD$750 to USD$1250.
    - And, the change in mining methods from sub-level caving and longitudinal shrinkage to transverse and longitudinal sub-level stoping, brought about by the higher gold process and the introduction of paste backfilling.
- The Young-Davidson underground has either replaced depletion or increased Mineral Reserves in each of the last five years.

Table 15-3: Annual Proven and Probable Underground Mineral Reserves

		2010	2011	2012	2013	2014	2015
	Gold Price (USD/ounce)	$750	$1,250	$1,400	$1,250	$1,250	$1,250
	Au Cutoff Grade (g/t)	1.9-2.3	1.70	1.70	2.05	1.90	1.90
End of Year Proven and Probable Mineral Reserves	Tonnes (x 1,000)	26,209	39,058	39,037	39,296	42,773	44,290
	Au Grade (g/t)	2.96	2.79	2.82	2.81	2.74	2.69
	Au Ounces (x 1,000)	2,494	3,504	3,534	3,556	3,763	3,837
Annual Mining Depletion	Tonnes (x 1,000)	-	-	77	617	1,289	1,854
	Au Grade (g/t)	-	-	3.29	2.83	3.07	2.66
	Au Ounces (x 1,000)	-	-	8	56	127	158

15.8 Stockpile Mineral Reserves

The Young-Davidson mine reports a low grade stockpile of 1,396 Ktonnes grading 0.082 grams per tonne gold. Ore from this stockpile is used to supplement the mill feed while the underground operation is ramping up to 8,000 tonnes per day. The primary source of this stockpile material is from the Young-Davidson open mine that was operated from January 2012 to June 2014. In addition, mill scats are added to the stockpile on a periodic basis. Tonnage in the stockpile is determined by monthly surveying and grade is determined by monthly inventory accounting.

15.9 Comments on Section 15

The responsible QP is of the opinion that the Mineral Reserves for the Young-Davidson Mine were estimated according to industry best practices and conform to CIM (2014) requirements. To the extent known to the QP, there are no known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues that could materially affect the Mineral Resource estimate that are not documented in this Report.

16 MINING METHODS

16.1 Overview

Young-Davidson is mined with longhole sublevel stoping methods, predominately transverse stoping, followed by paste backfilling. The mine has seen a steady ramp of production rates since its startup in 2013 and currently operating at a nominal 7,000 tpd rate. Production is focused in the upper half of the mine ("Upper Mine") where a crushing and loadout facility is located that feeds the Northgate hoisting facility. Development is now focused in the lower half of the mine ("Lower Mine") where infrastructure is being established around the bottom of the completed MCM shaft and the in-progress Northgate shaft deepening project.

16.2 Geotechnical Considerations

16.2.1 Geotechnical Site Characteristics

AMEC conducted a geotechnical site characterization based on laboratory testing and core logging in January 2009 (AMEC, 2009a) and additional underground mapping in November 2009 (AMEC, 2009b). MDEng in August 2015 reviewed the AMEC report and conducted their own mapping campaign. The following sections are based on MDEng's minor revision of AMEC geotechnical site characteristics (MDEng, 2015).

16.2.2 Rock Mass Properties

Geotechnical site characterization found that there were four geotechnical domains:

1. The Timiskaming Sediments (TSED) are found to be predominantly the footwall rock type and also found to be inter-bedded layers within the syenite rock mass. The permanent support is primarily constructed within this domain.

2. Syenite (SYN) hosts the ore body (striking east-west). This domain primarily utilizes temporary support.

3. Larder Lake Volcanics (MAFIC), consisting of inter-bedded Mafic flows and Ultramafic Flows found predominantly within the hangingwall of the deposit. MAFIC domain is not generally considered as mine development as it is the hanging wall.

4. Diabase Dykes zones (DIA) found as an intrusive swarm that generally crosscuts the syenite ore zones perpendicular in a north-south direction. This domain is considered for both permanent and temporary support categories.

The generalized geotechnical domains qualities and intact material properties are summarized in Table 16.1. This table also presents an estimation of friction angle (Read and Stacey, 2009) based on average Jr and Ja values.

Table 16-1: Generalized Geotechnical Domain Qualities and Intact Material Properties.

Domain	RQD	Jn	Jr	Ja	Q'	UCS (MPA)	Φ (°)
TSED	85%	12	1.25	1.25	7.1	155	45
SYN	85%	12	1.5	1.25	8.5	142	50
DIA	70%	12	1	1.25	4.7	175	40
MAFIC	80%	12	1.5	1.25	8.0	99	50

Friction Angle, Φ, was based on average Jr and Ja values (Read and Stacey, 2009)

16.2.3 Joint Sets

The results of geotechnical site characterization identified the major joint sets as summarized in Table 16.2. This table provides joint spacing and the associated factor, Fs, that is used to calculate likelihood of occurrence and subsequent likelihood of wedge instability (Diederichs et al., 2000) in a subsequent section. MAFIC is missing spacing data as it was not reported by AMEC or captured by MDEng

Table 16-2: Summary of Major Joint Sets

Domain	Joint Set	Strike (°)	Dip (°)	Spacing	Fs
TSED	1 (Foliation)	125	73	1-3 m	2.5
	2 (Vertical)	217	87	2-5 m	2
	3 (Horizontal)	291	10	1-4 m	2.5
	4 (Minor Sub-vertical)	071	56	10-15 m	1
SYN	1 (Sub-vertical)	135	68	1-5 m	2
	2 (Vertical)	262	81	0.2-2 m	3
	3 (Horizontal)	018	11	1-5 m	2
	4 (Vertical)	172	85	0.5-1 m	3
MAFIC	1 (Foliation)	088	82	0.5-4 m	2
	2 (Vertical)	056	85		
	3 (Horizontal)	338	02		
	Random				
DIA	1 (Foliation)	310	85	0.5-3 m	2.5
	2 (Vertical)	024	84	0.5-3 m	2.5
	3 (Horizontal)	248	07	0.25-0.5 m	3.3
	Random				

16.2.4 Ground Support Standards

The following subsections provide a brief summary of each of the supports standards used underground at Young-Davidson:

Standard Development Spans

The standard development span is based on Ground Support Standard #1 (GSS1) and is depicted in Figure 16-1. GSS1 consist of 2.4 m resin grouted rebar equally spaced at 1.2 m in the back and 1.8 m resin grouted rebar in each wall and one in each shoulder equally spaced at 1.2 m. Screen (#6 gauge) is installed and held in place with 0.6 m split sets or mechanical rockbolts.

Temporary and permanent support design is the same at Young-Davidson with the exception that side wall support for the temporary support design is required within 2 m of the floor, while permanent support is required within 1.5 m from the floor.

In units TSED and SYN GSS1 is applicable for spans up to 8 m, in DIA and MAFIC spans may not exceed 5.5 and 6.5 m respectively.

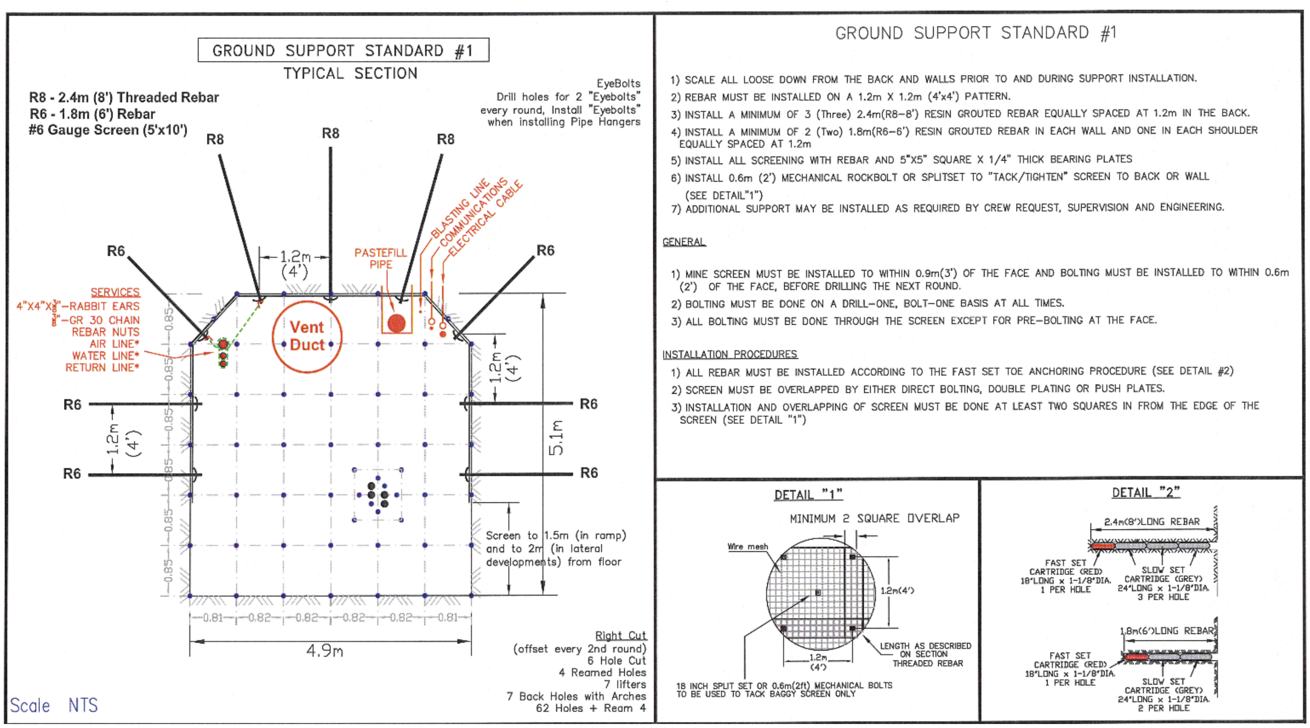


Figure 16-1: Standard Ground Support

Wide Spans and Intersections

Intersections or development with spans ≥8 m require additional secondary support. The current secondary support is an 8 m double stranded bulbed cable bolt on a 2 m x 2 m pattern. This support can be installed until a maximum span of 13 m. The specific support design for wide spans and intersections wider than 13 m are currently designed on a case by case biases. An example of a cable bolt design is presented in Figure 16-2. The standard cable used is double stranded.

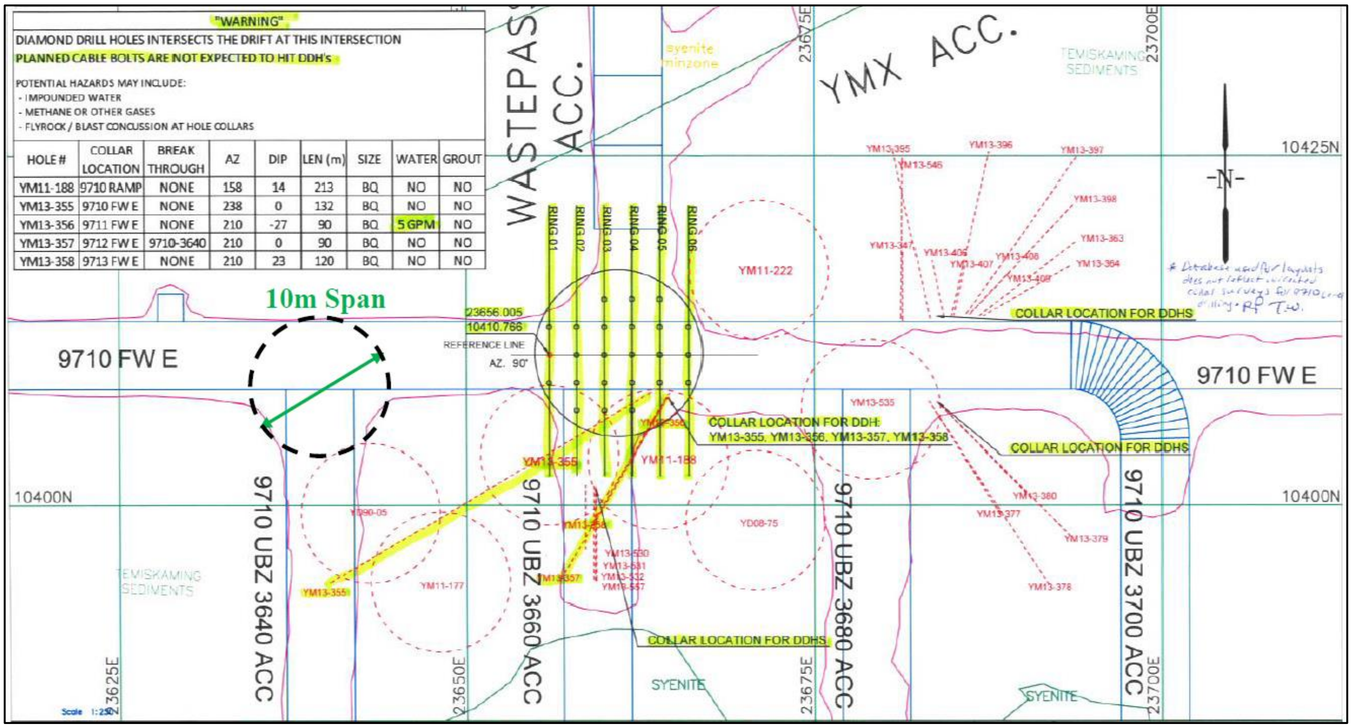


Figure 16-2: Ground Support for Wide Spans and Intersections

Drawpoints

Drawpoints require additional support to ensure the brow is adequately supported during the working life of the planned open stope. Rock within the brow may undergo significant stress change, abrasion from the passage of broken ore and potentially secondary blasting in the event of ore hang-ups. In addition to standard primary support, the current standard for drawpoint support is 3 rings of 8 m long double strand bulbed cable bolts on a 2 m x 2 m spacing, and is depicted in Figure 16-3.

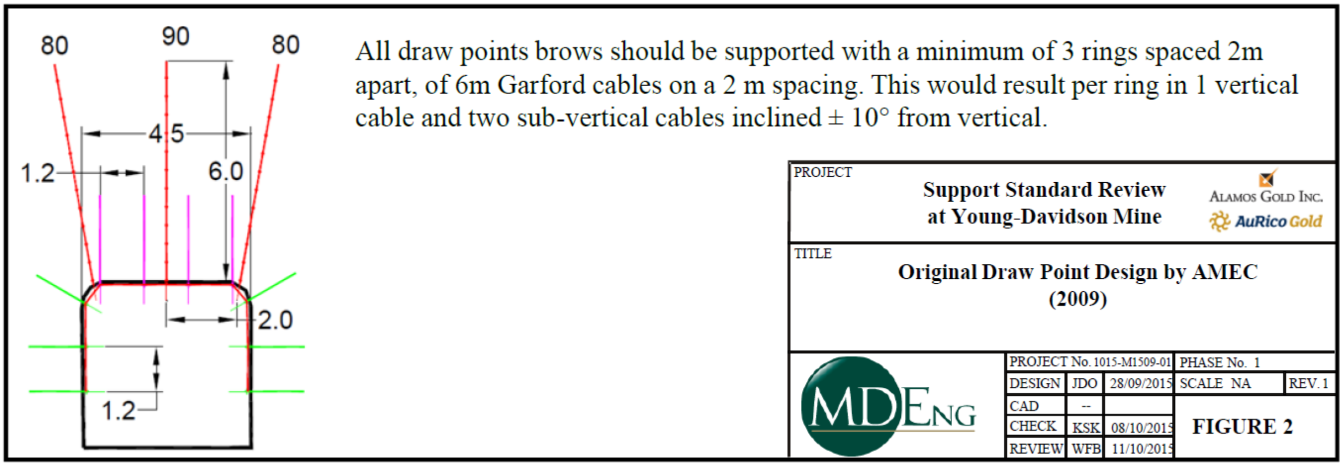


Figure 16-3: Ground Support for Drawpoints

Stope Cablebolting

Original designs by AMEC for stope backs employed single stranded cable bolts on a 2.0 x 2.0 m pattern. Site has employed variations of this pattern up to a 2.0 x 3.0 m spacing as well as switching the cables to double stranded. Figure 16-4 depict typical stope cablebolting. As extraction ratio increases and mining progresses to greater depths it may be necessary to tighten up the cable spacing, this can be evaluated as needed based on stope performance.

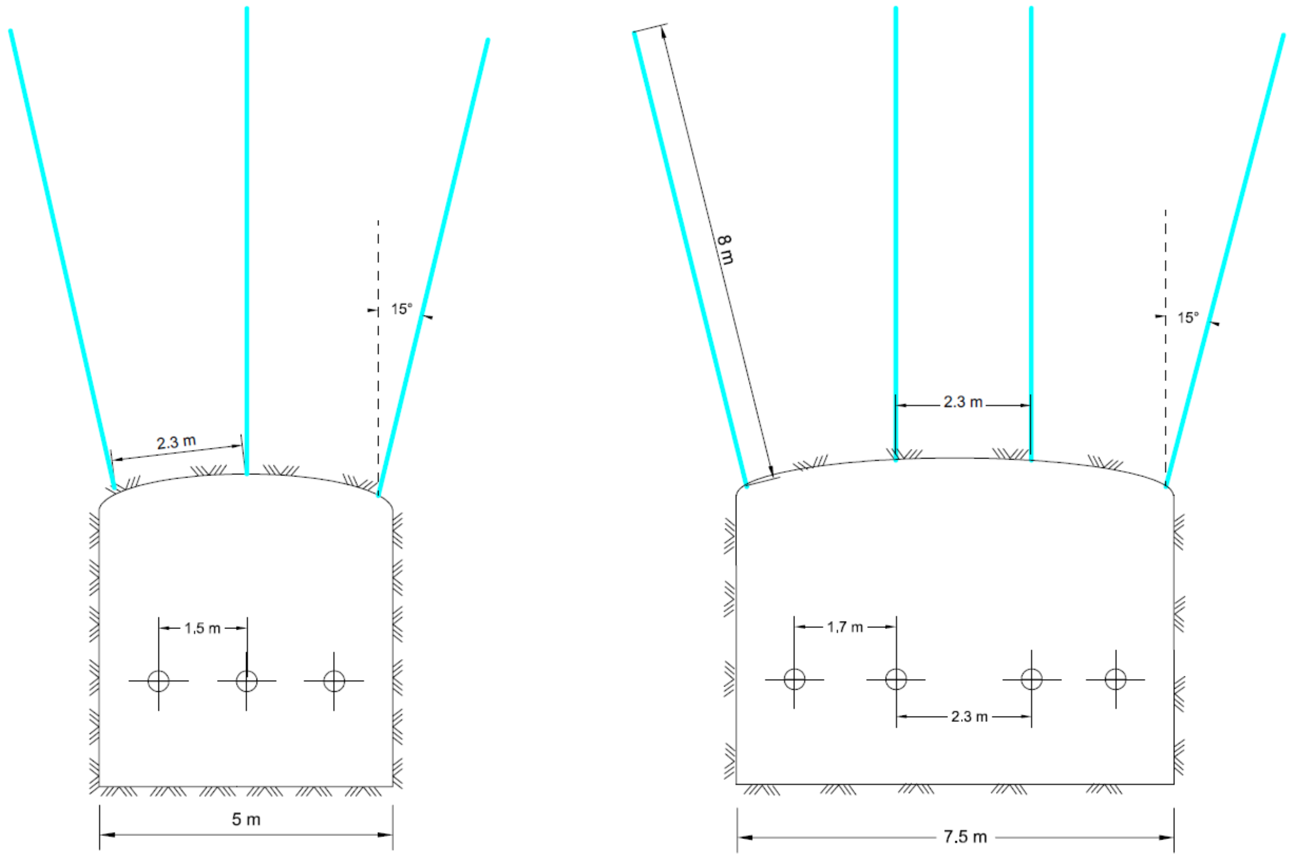


Figure 16-4: Ground Support for Stopes

Other Ground Support

Young-Davidson employs other ground support standards for drifting through paste fill, support in the MCM and Northgate shafts, and for Alimak raise mining.

16.3 Stoping Methods

The mineralized lenses dip steeply from almost vertical to 60° and average approximately 75°. Thickness of the lenses varies from 5 m to over 40 m thick, and the orebody has a strike length of up to 1000 metres and extends from shallow near surface to a depth of 1,400 m at the base of the deposit.

The fundamental basis for the choice of a mining method(s) for the Young-Davidson deposit was to identify safe, low cost and high productivity mining methods suitable for the relatively low grade deposit.

Feasibility Study Mining Methods

As a result of the prevailing gold price (~USD $800/ounce) at the time, the 2009 Feasibility Study contemplated low cost bulk mining methods that would not require cemented backfill. The methods selected were:

- Large dimension sublevel caving
- Longhole shrinkage stoping
- Longitudinal sublevel retreat with sill and rib pillars
- Longitudinal sublevel retreat with Avoca backfilling

It was recognized at the time that these methods in aggregate would result in lower mining recovery and higher dilution. Mining recovery and dilution envisaged in the Feasibility Study are presented in TableTable 16-3. There was also concern regarding the practicality of waste filling, particularly the transfer of waste rock from surface via a pass system.

Table 16-3: 2009 Feasibility Study Mining Recovery and Dilution

Mining Method		Recovery %	Dilution	
			%	Grade(g/t)
SLC	Sublevel Caving	80	18	0.78
LHS1	Longhole Shrinkage stoping - 40m w with 8m barrier pillars	87	15	0.47
LHS2	Longhole Shrinkage stoping - very large stopes	80	14	0.50
LR1	Longitudinal retreat - open stoping with pillars	70	10	0.88
LR2	Longitudinal retreat - Avoca	85	15	0.65
PR1	Pillar recovery	60	15	0.76
PR2	Pillar recovery above LHS2 stopes	40	15	0.76
Weighted Average		77	15	0.63

Present Mining Methods

In 2011, with a more buoyant gold price of USD +$1,300/ounce in play, mine plans and studies were undertaken to evaluate the use paste back fill and longhole stoping in order to recover more of the Mineral Resource and reduce the dilution. This included testwork by Golder Associates which concluded that Young-Davidson tailings are suitable for use as stope backfill with use of a binder. These studies concluded that the additional mining recovery of the expanded Mineral Reserve more than offset the slightly higher mining costs resulting from the use of paste fill and the mine was brought into production in 2012 utilizing transverse and longitudinal stoping. Table 16-4 presents the mining recoveries and dilution currently used for Mineral Reserves.

Table 16-4: Present Mineral Reserves Mining Recovery and Dilution

Mining Method	Recovery %	Dilution	
		%	Grade(g/t)
Transverse Longhole Stoping	90	8.4	0.15
Longitudinal Longhole Stoping	95	10.9	0.30
Sill Pillar Mining	85	8.4-10.9	0.15-0.30

16.3.1 Transverse Longhole Stoping

Based on the geometry of the mineralized zones, the vertical dip and the competency of the rock, the long-hole transverse mining method with primary and secondary stopes is used for the bulk of the mining at Young-Davidson. This mining method involves accessing stopes using two transverse drifts: one above the stope for drilling and loading of bulk explosives, and one below for mucking blasted material.

For transverse longhole stopes with paste fill, the sub-level spacing has been standardized to 30 m (floor-to-floor) and strike length to 25 m (between drawpoint centerlines). Given these dimensions, the maximum recommended stope width is 15-18 m. As such, in areas of the orebody that exceed 20 m width, two or more separate panels will be needed to extract the full orebody width.

Transverse stopes are sequenced as a series of primary and secondary stopes. Primary stopes are mined first and are separated by secondary stopes. After the primary stope is mined it is backfilled with paste fill. When both primary stopes on either side of the secondary are filled the secondary stope can be mined. Secondary stopes, depending on where they lie in the panel sequence, local stability issues, and the availability of waste rock, can be filled as follows:

- 100% paste fill
- Paste fill plug then waste rock
- Paste fill plug then paste comingled with waste rock.
- 100% waste rock

A schematic of a typical development, stoping and fill sequence can be found in Figure 16.5. On a tonnage basis, transverse stopes comprise 90% of the Mineral Reserve.

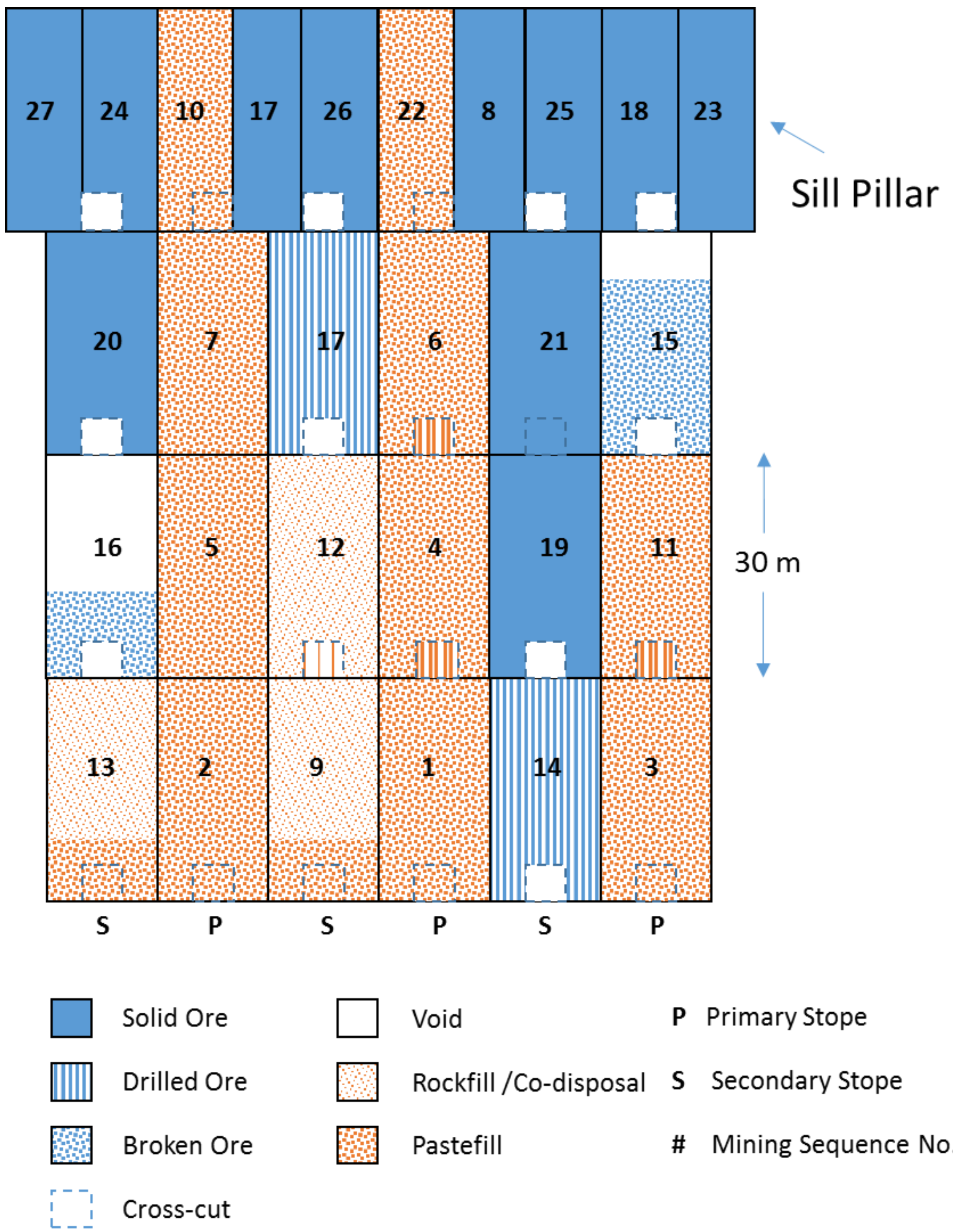


Figure 16-5: Typical Stope Sequence

16.3.2 Longitudinal Longhole Stoping

Longitudinal stoping is a form of longhole stoping used mainly on vein type deposits. During blasting the broken ore is thrown out into the open stope and since this not safe for personnel entry, radio remote control scooptrams are used to advance out into the open stope beyond the safety of the brow, to muck the blasted ore. This technique eliminates the need for production scrams and multiple drawpoints reducing the development costs. This method is planned for ore zones ranging from 5 to 10 m thickness, in other words, for zones that are too narrow to mine with the more productive and lower cost transverse longhole stoping.

For longitudinal longhole stopes, sub-level spacing is the standard 30 m, ensuring compatibility with transverse longhole stopes. Since longitudinal stopes are generally <10 m wide (averaging 6-8 m), the maximum recommended stope length is 80 m above the 9590 level and 30 m below the 9590 level; these lengths were reduced to 50 m and 25 m respectively to tie-in with transverse stope lengths for the reserving process (as stoping methods are only assigned after the original grade shells were split both vertically (every 30 m) and horizontally (every 25 m along strike).

16.3.3 Mine Sublevels

Access to the mining areas is provided by sublevels currently established at 30 m vertical intervals. Beginning at the 9400 level sublevels will be established on 35 metre vertical intervals, as stability analysis has indicated this interval will be acceptable, and the economics of reduced development required per tonne of ore produced make it favourable.

The sublevels are located in the footwall with crosscuts to the ore lenses as required for access. The general design parameters for the sublevels are 4.9 m wide and 5.1 m high to accommodate 20 t LHD's and minimum clearance of 18 m from the edge of the ore lenses. A typical sub level layout is shown on Figure 16-6.

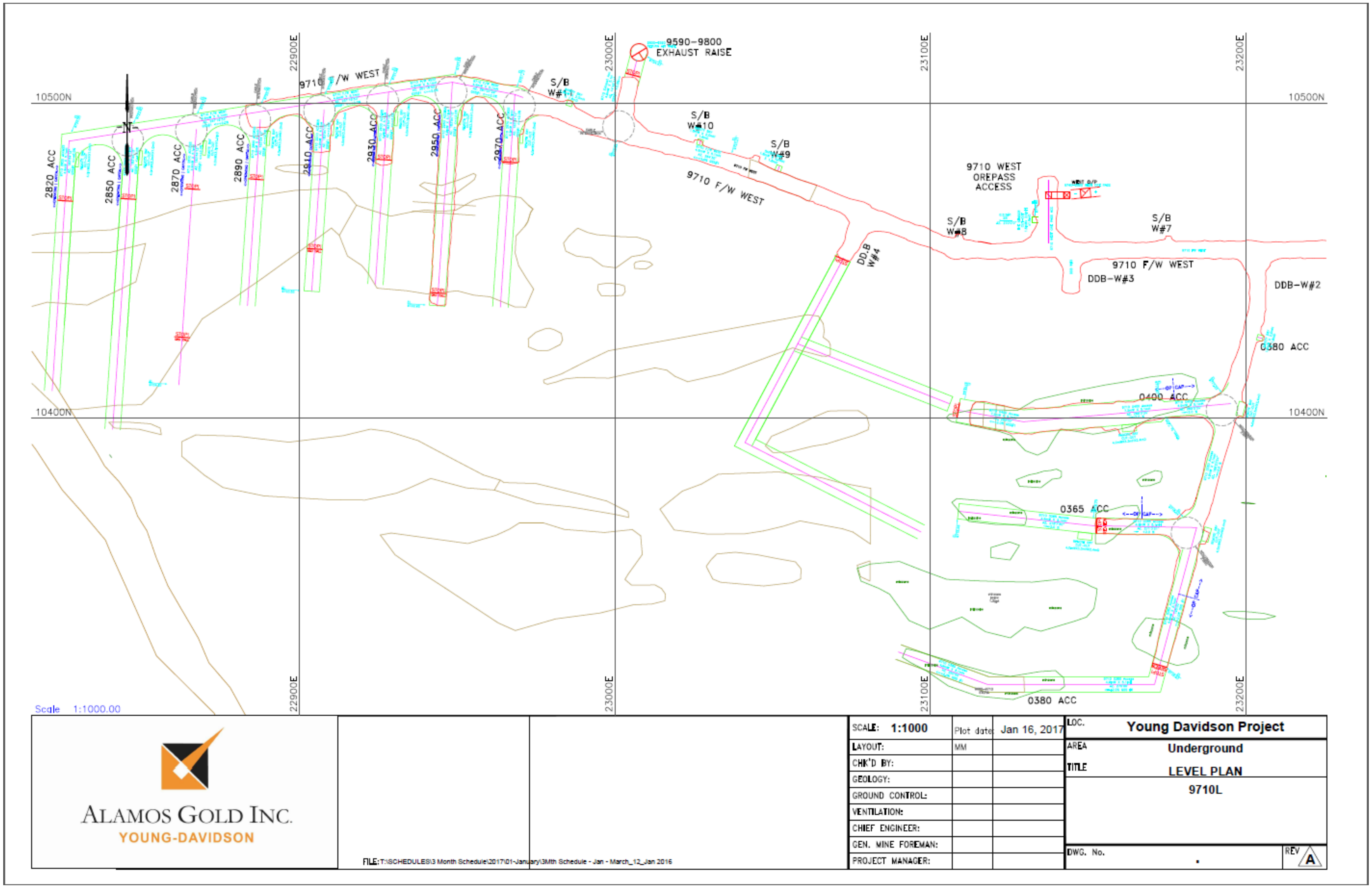


Figure 16-6: Typical Sub-Level Layout

16.4 Paste Fill

16.4.1 Introduction

As part of the current mine plan, paste fill is used as the backfill product in the final step of a stope cycle. Paste fill is produced in the paste fill plant and distributed underground via a network of 8” piping. Draw points for stopes must be barricaded with specially designed ached barricades to support the load of the dense paste fill material. These barricades must be constructed, approved and shotcreted prior to the initial paste fill pour. Each stope is filled in three portions, the plug, body and cap pours. All three portions are required to complete filling of any particular stope. Once a fill is complete paste fill must be allowed to cure until a reasonable strength is attained before proceeding to mine any adjacent stopes. The Mine Engineering Department prepares a paste filling package for each individual stope that provides instruction on how the filling process should be conducted.

16.4.2 Paste Fill Production

Paste fill is produced in the paste fill plant on surface. The paste plant uses tailings from the mill to produce the final paste fill product. The tailings are pumped from the mill to the paste plant through a pipeline on surface, if tailings are not being used for paste they are directed out of the paste plant to the tailing impoundment area (TIA) for long term storage.

Tailings that are being used for paste fill will have most of the water removed in a thickener tank. The thickened tails are directed on to a conveyor that feeds into a mixing tank. Water and binder are added in the mixing tank to produce a specified paste fill product. An example of a paste fill product would be paste with a 5% binder content (10% Portland cement / 90% slag mixture) with a 7” slump. Once the mixing tanks have produced a quality paste with good consistency the paste is directed into a hopper that feeds the underground boreholes.

16.4.3 Underground Distribution

There are two boreholes within the plant on surface that breakthrough on the 10130 level. The paste plant can direct paste into either one of the boreholes or both of them. The underground distribution system (UDS) transfers through the mine using mainline piping, level piping, stope piping and inter level boreholes. Mainline piping is 8” schedule 80 pipe and has the ends striped red, as it is used as the main distribution line for paste underground and has the highest pressure rating. The mainline transfers through the 10130, 10010, 9920, 9800 and 9680 levels. Paste is distributed to other levels as follows;

10130 level transfers to;

- 10100
- 10070

- 10040
- 10010 (mainline transfer)

10010 level transfers to;

- 9980
- 9950
- 9920 (mainline transfer)

9920 level transfers to;

- 9890
- 9860
- 9830
- 9800 (mainline transfer)

9800 level transfers to;

- 9770
- 9740
- 9710
- 9680 (mainline transfer)

9680 level transfers to;

- 9650
- 9620
- 9590
- 9560 (mainline transfer)

Paste fill infrastructure to the 9440 level will be established in first quarter of 2017.

Piping that does not distribute to lower levels or is not part of the mainline is called level piping. Level piping is 8” schedule 40 pipe and has the ends striped green. Level piping is used to distribute paste to a single level only from the bottom of a borehole and in the footwall drifts up to the stope access. At the stope access the piping is switched from level piping to stope piping. Stope piping is a black 8” High Density Polyethylene (HDPE), it is run from the stope access to the pour point into the stope.

There are a few specialty pipes installed in the underground distribution system these include;

- Clean out tee’s – preferred location for blasting plugged lines, capable of installing air and water to clean out line from underground.
- Rupture spools – designed with a groove along the length, used as an engineered weak point in the system to control location of breakage as pipe wear through abrasion begins to be a problem

- Pressure transducer spools – installed with a small blue pressure transmitter, these transmitters send observed pressure readings to plant operator to detect potential plugs or other problems related to paste flow, there is also a readout display on the transmitter for manual observation.

16.4.4 Arched Bulkheads and other Paste Fill Barricades

Bulkhead construction and approval are an important step in the paste fill sequence. The bottom sill of a stope (draw point) must have an arched barricade constructed to withstand the force of the dense paste material pushing up against it before it cures. Before any shotcreteing of a barricade can be done, the steel work must be approved by operations and engineering. The ached bulkhead on the bottom sill of a stope must be shotcreted and allowed to cure for 48 hours prior to the first paste fill pour in an area.

Top sill barricades are required for any stopes requiring a tight fill or a fill that goes above the floor on the upper level of a stope. There are two types of top sill barricades, tight fill and non-tight fill. A tight fill barricade is shotcreted to the back around the pour pipe in a best attempt to fill the stope as close to the back as possible. A non-tight fill fence is constructed with the top of the barricade two feet from the back, this allows paste fill to be poured up to the shoulders of the drift and stope. Tight fill and non-tight fill fences are required only when outlined in the paste fill package distributed by engineering.

16.4.5 Filling sequence

Each stope is filled in three portions, each portion contains different binder percentages and tonnages. Typical paste fill portions are described below.

- Plug pour – the plug pour is the first pour in a stope being paste filled. The plug pour volume is calculated from the floor of the stope to 2 m above the brow. The plug must be allowed to cure until sufficient strength is achieved before the pour can continue in a stope. The plug pour is where the ached barricade on the bottom sill will see pressure, during the time of this pour barricades will be monitored closely to ensure the height of the plug is correct and all valves are closed at the appropriate times.
- Body pour – this is the majority of the paste fill volume, it start from 2m above the brow until 1 m below the floor on the top access. Generally the body pour contains less binder than the rest of the stope.
- Cap pour – the cap pour begins 1 m below and continues until the completion of a stope. If tight filling is required the cap pour will continue until maximum paste has been introduced into the stope. The cap generally has a slightly higher binder content than the body pour to provide more overall strength and allow equipment to travel over the backfill.

Once a stope has been completely filled, paste must be allowed to cure until sufficient strength is attained before blasting in adjacent stopes can be performed.

The mine typically uses 3% binder in the plug, 2% binder in the body and 3% binder for the cap. If the mine needs to cycle a stope faster it will increase the binder to up to 5% for the body and up 7.5% for the cap and plug to enable mining against or over the paste sooner.

16.4.6 Engineering

Engineering distributes a paste fill package for each individual stope with instructions and safety concerns pertaining to the designed pour.

The following items are described in the paste package,

- Stope name
- Expected pressures of paste in the lines
- Pour parameters such as binder percentages, slump and tonnages for different portions of paste fill
- Risk assessment – review of potential risks such as diamond drill holes, ventilation and construction requirements
- Specific pouring instruction
- Barricade design and location of construction
- Plan and cross-sectional drawings of stope with calculated volumes and tonnages
- Paste fill pipeline drawings showing route of travel for paste from surface to discharge point.
- Approval signatures from technical staff and operations management

16.5 Mine Infrastructure

Infrastructure for the Upper Mine is complete and functional. Much of that infrastructure is currently being duplicated for the Lower Mine. A general layout of the built out mine is depicted in Figure 16.7.

.

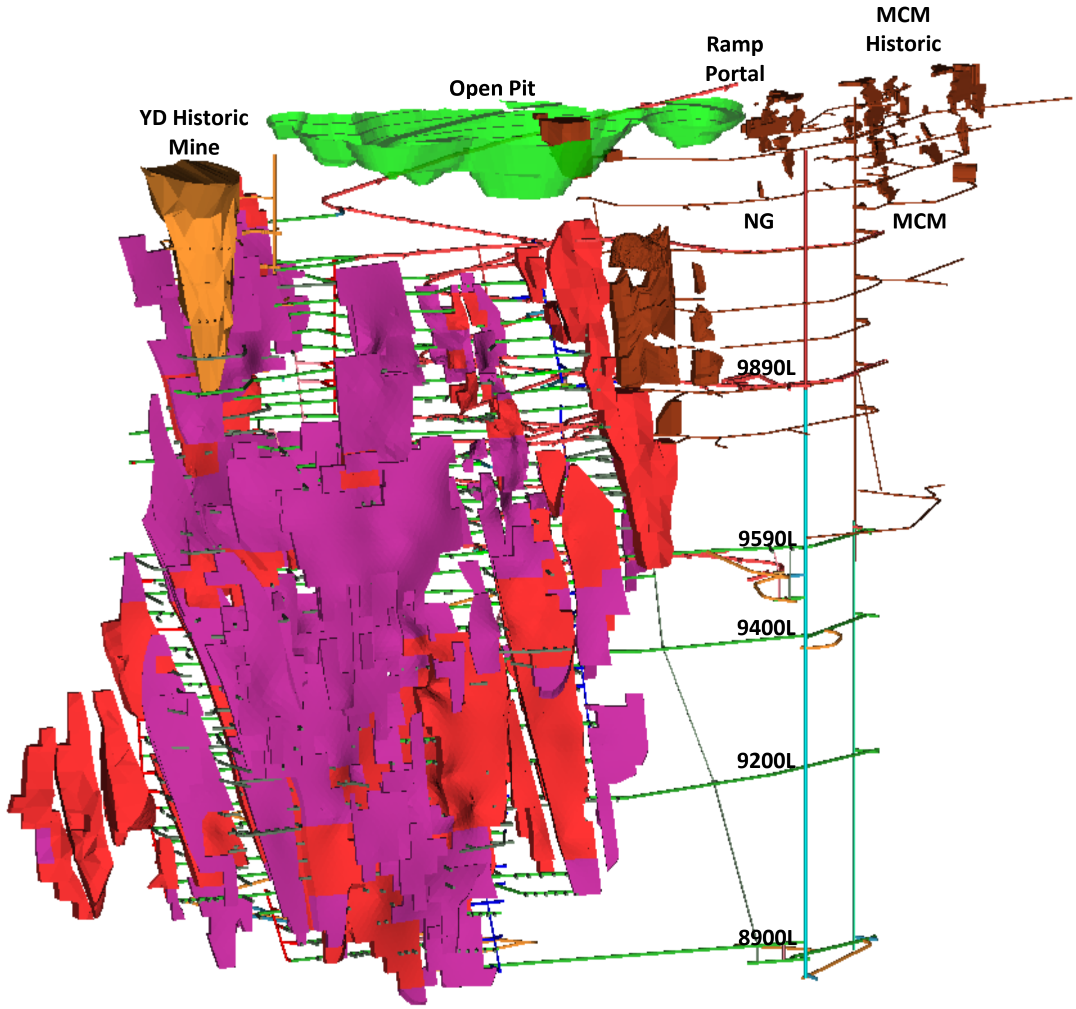


Figure 16-7: General Mine Layout – Long Section Looking North

16.5.1 Northgate Shaft

The Northgate Shaft is a 5.5 metre diameter shaft and is used only for production hoisting of ore and waste and for ventilation. As the Young-Davidson Mine had access to lower levels of the mine via the exploration ramp, the Northgate Shaft was excavated, to the 9440 level via raise boring, in two legs. The first leg was 455 metres (10345-9890) while the second leg was 450 metres (9890-9440).

The Northgate Shaft has a 78 metre concrete headframe topped with a Hepburn friction hoist with a 4.57 m x 1.93 m drum and a 5000 hp drive motor and has a maximum hoisting speed of 15.24 m/s. Four hoist ropes and three trailing ropes are utilized and the two skips are rated at 18 tonnes each, when used at the mid-shaft loading pocket location. The shaft contains four wire guide ropes for each of the two skips and contains no other conveyances or man-ways.

Rock is truck dumped onto either the ore grizzly or the waste grizzly on the 9590 elevation and stored in the ore pass or the waste pass below the grizzlies. Ore is fed to a re-furbished Allis Chalmers 66” x 44” jaw crusher on the 9550 level and then onto the loadout conveyor on the 9500 level. Waste is fed directly to the loadout conveyor on the 9500 level. The loadout conveyor feeds a shuttle car which then feeds one of two measuring flasks for loading the skips. Once hoisted to surface, ore is conveyed to the mill fine ore bins, while waste is conveyed to a temporary stockpile adjacent to the headframe, from where it is periodically hauled to the waste rock dump.

The shaft and loadout facility was commissioned for waste at the mid-shaft location in July 2013 and for ore in October 2013.

The Northgate Shaft is being deepened, again by raise boring, to the bottom of the mine at the 8900 level. The 508 metre pilot hole was completed in the third quarter of 2016, and the reaming is expected to be completed in the second quarter of 2017. The scheduled change to the Northgate shaft bottom skipping is late 2019. This is needed to avoid an impractical number of trucks hauling up to the 9590 level ore grizzly due to the mining centroid deepening each year.

The emergency egress pod will run on a single guide rope.

16.5.2 MCM #3 Shaft

The MCM #3 (MCM) shaft was originally sunk to 750 metres by the Matachewan Consolidated Mine during the original development of the property in the 1934-1954 period. Northgate dewatered and rehabilitated the shaft at the commencement of the modern mine’s development.

From surface to the 9800 metre level the shaft is a rectangular 5.18 m x 2.44 m shaft with three compartments. From the 9800 level to full depth (8840 level) the shaft was expanded to a rectangular 6.54 m x 2.44 m shaft with four compartments. Both sections of shaft have two skip/cage compartments and a manway/services compartment while the new section below 9800 level has an additional compartment for ventilation ducting.

The hoisting plant at MCM is a 3.7 metre diameter x 1.7 metre wide double drum hoist powered by a 2000 hp motor. The hoist is housed in a new hoist house approximately 17 m wide x 16.5 m long x 16 m high.

The shaft conveyances in MCM shaft consist of two 8.5 tonne bottom dump skips with integral cage utilizing bottom dump skip scroll plates in the headframe for dumping hoisted rock. Each single deck cage has been maximized to fit the space available in the shaft and will accommodate nine persons. A loading pocket has been commissioned at the 8870 level to handle muck from the development of the Lower Mine. The hoisting system has been designed to hoist up to 2000 tonnes per day of rock in a 16 hour period. The skips travel at 10.2 m/s on wooden guides with a capacity of 125 t/hr.

Although the MCM shaft can hoist up 2,000 tpd, a significant amount of rock hoisting time will be consumed with movement of men and materials to various levels of the mine, as the MCM shaft is designated as the principal means of access. In addition, heavy equipment will continue to be slung below the skip for the development and construction effort on the 8900 metre level. The MCM shaft will be used for slinging until the main ramp breaks reaches the Lower Mine crusher drift in 2019.

A waste pass is currently being developed between 9440 level and the 8900 level. When completed, in the first half of 2017, the mine will have the ability to hoist development waste from the Upper Mine via the Lower Mine loading pocket in the MCM shaft. This will in turn allow the Northgate shaft to devote more time to hoisting ore and less to hoisting waste.

New shaft stations have been established at the 9500, 9400, and 8900 level. A shaft station will be established at the 9200 level in 2017, as part of the development of the MCM waste pass system.

Ramp access is being established to the shaft bottom for mucking of spillage and dewatering purposes. A sump will be provided at the base of the ramp at shaft bottom and will collect water for settling. The water will be pumped to the 8900 m level where it will be fed into the main mine dewatering system.

16.5.3 Surface Ramp

During the exploration phase of the project an access ramp was driven from surface to a depth of 460 metres, with the face ending up at the 9890 level. After project approval was received the ramp was continued and at the end of 2016 was at a depth of 1,010 metres or at the 9340 metre elevation.

This ramp is 4.9 m wide by 5.1 m high with a grade of 17%. As part of the mine plan, this ramp will be extended at the same size and grade from the current face to the bottom of the mine at the 8,900 crushing and conveying level. Once complete the ramp will extend 1,450 m vertically and will be 8.6 kilometres long.

The ramp connects to each of the sublevels. The ramp provides access to the sublevels for personnel, equipment and supplies. It is not intended as a haulage ramp for ore or and muck haulage on it is generally limited to development waste.

16.5.4 Rock Handling

Ore/Waste Passes

The Upper Mine currently has two ore passes, the West Ore Pass, and the East Ore Pass that connect all sub-levels to the 9590 level. Mucked ore from the stopes is hauled on the sub-level in 12 cubic yard load-haul-dump (LHD's) and dumped into the cone dumps located on each level for each ore pass. The LHD one way haulage distances on the levels average approximately 150 metres, and range up to 300 metres to a maximum of 400 metres in some limited circumstances.

The Upper Mine currently has one waste pass that connect all sublevels to the 9590 level. Development muck is generally transported to the waste pass 8 cubic yard LHD's.

The ore and waste passes are 2.7 by 2.7 metres and were excavated with Alimak raising.

Transfer Level

The main ore passes and waste pass terminate at the 9590 transfer level. Ore and waste are then hauled along a dedicated transfer level to their respective grizzlies.

16.5.5 Communications

Communications is provided by a fibre optic system connected to UG fans, electrical stations and critical infrastructure supplemented by leaky feeder radio system that is installed in the shaft and ramp and all the levels. The system is capable of transmitting both data and voice as well as high speed internet and telephone to the refuge stations and maintenance shops. A mine control system will be installed in 2017 that will allow for the tracking of underground personnel and equipment.

16.5.6 Lower Mine Development

Lower Mine level development and major excavations are being undertaken by Young-Davidson crews. Contractors will be employed for equipment installation.

The loadout facility will be substantially the same as that at the mid-shaft location. The coarse ore bins and crusher will however be located to closer to the horizontal centre of gravity of the orebody. This will allow ore passes from the 9440 level to directly feed the crusher coarse ore bin and the waste bin, as opposed to hauling rock from truck load-outs to a grizzly, as is done on 9590 level of the Upper Mine.

In the Lower Mine waste rock and crusher ore will then be conveyed through a dedicated 700 metre conveyor drift to bins located above the Northgate shaft loadout.

Commissioning of the Lower Mine Northgate hoisting facility will necessitate a change out the all of the conveyance's ropes, including hoist ropes, trailing cables and guide ropes. During this period of time there would be no hoisting of ore and waste with the Northgate shaft. The sequence of events would be:

- Blast the pentice
- Install ground support where the pentice previously was
- Install all ropes

- Commission shaft, loadout, conveyors, and crusher

It is anticipated that the changeover will take approximately 6-8 weeks. During this period of time mill feed will consist of:

- Stockpiled underground ore on surface
- Ore hoisted up the MCM shaft
- Underground ore from the higher levels of the mine trucked to surface, and
- Ore from the current surface stockpile

It is also expected that any major mill maintenance would be scheduled for this period.

Figure 16.8 depicts the built-out Lower Mine infrastructure at the end of 2019, while Figure 16.9 depicts the complete mine development for the life-of mine.

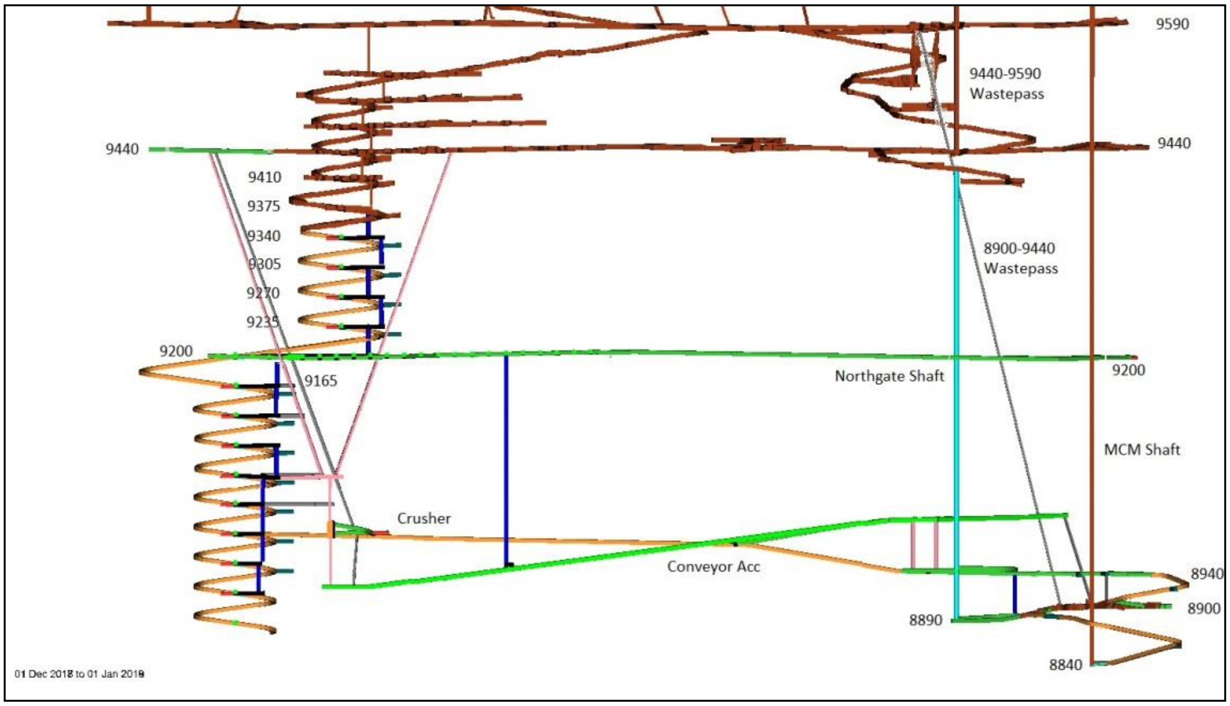


Figure 16-8: Lower Mine Development – End of 2019

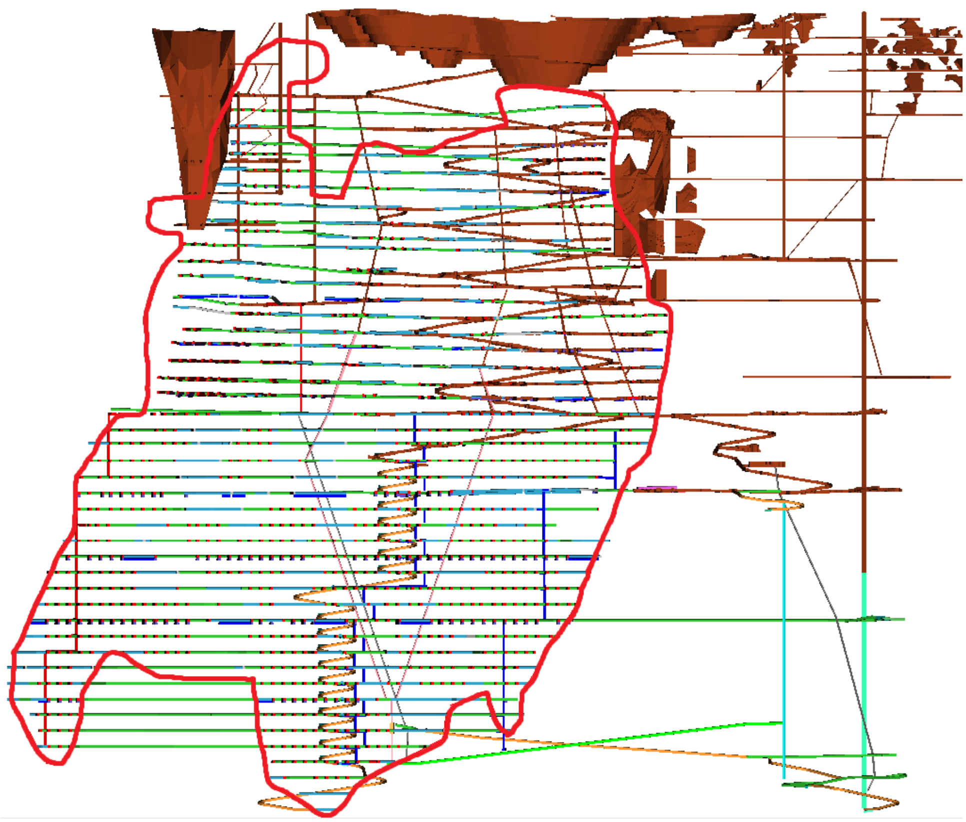


Figure 16-9: All Mine Development - Life-of-Mine

16.6 Production Schedule

16.6.1 Development Requirements and Schedule

During early development phase of the mine, contractor crews were employed for ramp and level development. As the required development rates ramped up a combination of mine and contractor crews were utilized. In April 2016 the contractor was demobilized and all level mine development is currently being undertaken by Young-Davison crews. Contractors are utilized for raise development and shaft sinking, and a small portion of the production drilling and cable bolting.

Mine development is categorized as capital, operating and ore. Capital development includes all infrastructure development such as shafts, ramps, ventilation raises and, all parts of sublevels that serve more than one stope. Operating development is development that serves just one stope such as an individual stope access and drawpoints. Ore development is that development required in ore to provide drill platforms and mucking points for stoping.

Lateral development during operations will average approximately 14,000 metres per year including capital, operating and ore categories between 2017 and 2020. In the following six years lateral development drops to an average off to 10,000 metres per year as most of the sub-levels have been developed. In the last five years of the underground mine life, the development requirements drop off sharply to average 3,200 metres per year as the mine is close to being basically fully developed. Total life-of-mine lateral development is 132,000 metres, inclusive of that development required to put into place the Lower Mine infrastructure.

16.6.2 2017 Production

Production in 2017 will focus on stopes in Upper Mine horizon, as well some stopes on the 9440 level. Production the 9440 level is hauled up to the grizzlies on the 9590 level. Figure 16-10 depicts 2017 production and development, colour coded by quarter of production. Ore production is expected to range from 6,500 to 7,500 tpd with the mining rate weighted towards the second half of the year as the completion of the 9440 level waste pass will allow for the increased hoisting of ore and waste.

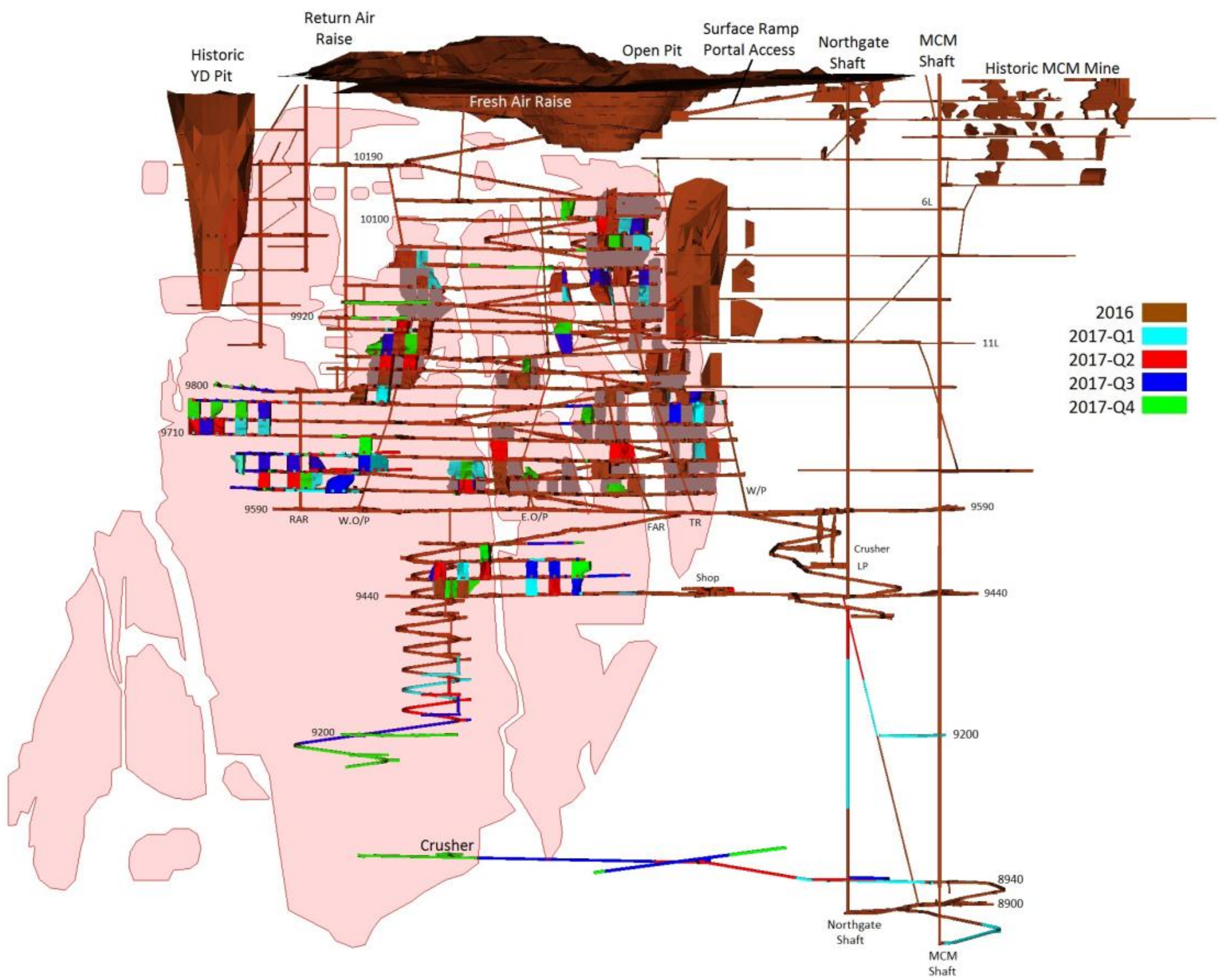


Figure 16-10: 2017 Production and Development

16.7 Ventilation

16.7.1 Summary of Ventilation System

The current ventilation system (Figure 16-11) at the Young-Davidson Mine is comprised of 3 dedicated fresh air sources along with 3 exhaust routes.

All primary intake routes (excluding 10130 Level) have two sets of fans: one on surface and one booster underground. This design has the advantage of allowing for implementation of multiple instances of the same fan model, which simplifies inventory and maintenance.

16.7.2 Primary Fresh Air System

Northgate Shaft

The Northgate (NG) headframe is equipped with 2 x 200hp providing 240 to 260m^3/s to the underground workings with approximately another 20 m^3/s of designed leakage used to keep the headframe from freezing during the winter months. Northern mines often use high-volume, low-pressure fans to force heated air into the headframe during subfreezing temperatures.

The 9590 level is supplied with approximately 80m^3/s from the NG shaft using a 350hp HVT fan. The remaining NG fresh air is supplied to the current shaft bottom on 9440 level with a twin 350hp HVT supplying the level itself (80 m^3/s), MCM shaft (40 m^3/s) and crusher/loading pocket infrastructure (60m^3/s).

Surface Fresh Air Raise to 10130 Level

The fresh air raise (FAR) from surface is transferred across the fresh air transfer level on 10130 level using a 3 m diameter raise bored raised from 11 Level (9890 elevation). The FAR is equipped with twin 600hp fans with a heater providing approximately an additional 170 m^3/s to the upper 9 levels of the mine. The exhaust currently reports back to the main ramp.

No.1 Shaft Fresh Air System

The #1 shaft is equipped with a 350hp HVT fan pushing fresh air through a series of old mine workings and raises. It is aided with a 350hp booster on 11 Level providing approximately 80 m^3/s of fresh air through the 9590-9890 FAR. Some recirculation has been observed or suspected in the ventilation system through the old workings into upper ventilation transfer levels.

It's worth noting the previous #2 pit surface fresh air fan has been decommissioned in the last year due to underground ice buildup and deteriorating historical workings.

Supplying fresh air as close as possible to the active production areas provides a cleaner air supply to most underground personnel by reducing the likelihood that the ventilation air will be contaminated by diesel emissions, dust, or other contaminants upstream of the working area.

16.7.3 Primary Exhaust System

Return Air Raise

The return air raise (RAR) is a series of 5.5m diameter vertical raisebores to surface from the 9590 level. Future development focuses on establishing flow-through ventilation on individual levels. The RAR exhausts roughly 190 m^3/s to surface.

Young Davidson will be installing independent regulators to reduce the majority of auxiliary fans. These consume more power and are more fragile than a properly implemented primary ventilation system.

Auxiliary ventilation is also less efficient at removing/diluting contaminants, as only a fraction of the total available primary ventilation airflow can be applied via auxiliary ventilation without risking recirculation of airflow. Reducing the auxiliary ventilation will also minimize re-entry times after production blasts and reduce overall electrical costs.

MCM Shaft

The MCM shaft is currently supplied fresh air from the 9440 level to near shaft bottom onto 8900 level. The 8900 is a captive level and all exhaust reports back up the shaft exhausting the original provided 40 m^3/s along with an additional ~20 m^3/s of leakage from the old mine workings.

Main Portal Ramp

The main production ramp is currently maintained as an exhaust way which serves to keep the portal free of ice during the long periods of subfreezing weather in Ontario. The majority of the exhaust flows through the portal at 260 m^3/s.

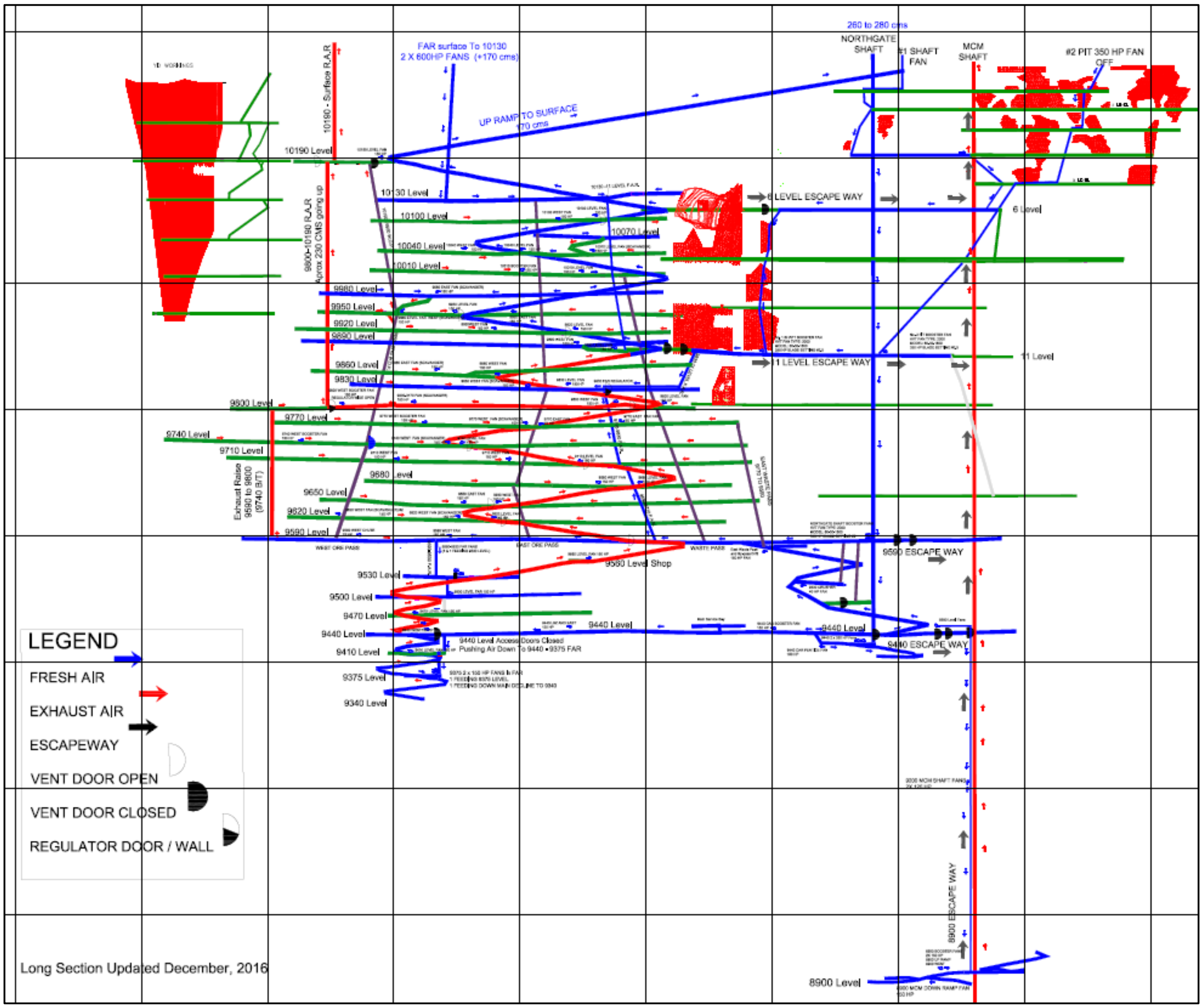


Figure 16-11: Young Davidson Current Ventilation System

16.8 Mining Equipment

Major pieces of development production and support equipment at the end of 2016 are listed in Table 16-5. The only additional equipment expected to be purchased would be one additional production loader an additional Longhole drill. Young-Davidson has been employing a contract production driller and has gradually been phasing the contractor out.

Utilization of the truck fleet will decline after 2019 as all levels will have access to ore and waste pass cone dumps. Currently all production and development from the 9400 level to the 9560 level must be hauled up to the grizzlies on the 9590 level. In mid-2017 it is anticipated that the waste pass from the

9440 level to the 8900 level at the MCM shaft will be operational. This will also take some pressure off of the haulage requirements.

Table 16-5: Major Underground Mobile Equipment, End of 2016

LHD's	CAT R2900G (12 Yd)	9
	CAT 1700 (8 Yd)	8
	CAT 1300 (3.5 Yd)	2
Trucks	CAT AD-45B	3
	CAT AD-30	3
	Atlas Copco MT 42	5
Jumbos	Atlas Copco 282	5
Bolters	MacLean	5
Scissor Lifts	MacLean	10
Longhole Drills	Aries	3
	Orion	1
Light Vehicles	Toyota	30
	Tractors	7
Support Equipment	Shotcrete Sprayers	2
	Transmixers	2
	Boom trucks	5
	Graders	2
	Block Holer	1
	Portable Rock Breaker	1
Total		**104**

17 RECOVERY METHODS

17.1 General Overview

The Young-Davidson mill can be considered a standard flotation/CIL gold mill. The mill was commissioned during Q1 of 2012 and the first gold pour occurred on April 30th, 2012. Ore is currently sourced from two sources, the underground mine and from low grade surface stockpiles, produced by the open pit between 2011 and 2014.

Open pit stockpiled ore is crushed in a jaw crusher to a P_{80} of 150 mm. Crushed ore is then conveyed to the coarse ore bins feeding the processing plant grinding circuit. The surface crusher is capable of crushing over 8,000 tpd and was the sole source of mill feed prior to the commissioning of the underground mine.

The underground mine ore is fed from an ore pass onto a grizzly feeder into a jaw crusher located underground on the 9530 meter level. The crusher product is conveyed to the skip loading system from an underground bin. The underground ore skipped to surface is fed into a bin located in the headframe and a series of feeders and conveyors feed the coarse ore bins located at the process plant.

Feed from the coarse ore bin discharges onto the SAG mill feed conveyor. The grinding circuit is operated in a closed circuit with cyclones and features a single-stage SAG mill. One of Northgate's Kemess ball mills was modified to function as the SAG mill in the Young-Davidson grinding circuit. The SAG mill discharge reports to the cyclone feed pumpbox. The cyclone underflow returns to the SAG mill and the cyclone overflow flows by gravity to the flotation circuit.

The cyclone overflow feeds a trash screen prior to feeding the flotation conditioner. The flotation conditioner overflows into the first flotation cell. The flotation circuit is comprised of four tank cells.

The flotation concentrate is pumped to the concentrate thickener. Thickener underflow is pumped to the regrinding circuit. The thickened concentrate is pumped to the regrinding circuit, comprising a vertical tower mill, where the solids are reduced to a size distribution with a nominal P_{80} target of 15 µm, operated in closed circuit with a cyclone cluster. The cyclone underflow returns to the tower mill box while the cyclone overflow flows directly to the concentrate CIL circuit.

The flotation tailings are pumped to the flotation tailings thickener. The flotation tailings thickener underflow is pumped to the combined CIL circuit. Thickener overflows from both thickeners report to the mill process water tank for reuse.

The flotation concentrate from the regrind circuit reports to a pre-leach tank. The slurry overflows from the pre-leach tank into a series of four CIL tanks providing a total of 48 hours of retention time.

The loaded carbon extracted from the first tank of the concentrate CIL tank is pumped to the carbon stripping circuit for carbon elution.

The flotation tailings, along with the previously leached flotation concentrate are fed to the combined CIL circuit consisting of five leach tanks providing an overall retention time of 24 hours. The combined leach circuit tailings report to the cyanide destruction circuit.

The carbon elution circuit has been sized for processing 4 t/d of carbon. The loaded carbon is pumped from the first flotation concentrate leach tank across a loaded carbon screen. The loaded carbon flows by gravity into the acid wash vessel where the loaded carbon is rinsed by a solution of hydrochloric acid followed by neutralization. The loaded carbon is transferred by a recessed impeller pump to the elution vessel. The stripped carbon is reactivated in an electric kiln and reused in the CIL circuit.

The carbon stripping circuit elutes the precious metals into the pregnant solution. The pregnant solution feeds the electrowinning circuit. Pregnant solution is pumped through the electrowinning circuit to produce sludge on the cathodes containing the precious metals. The cathodes are washed, dewatered and dried. The dried cathode sludge is melted in an electric induction furnace and poured into doré bars.

The combined leach circuit tailings report to the cyanide destruction circuit using the SO_2/air process and copper sulphate solution. The treated tailings slurry is pumped to the tailings impoundment area for settling or to the paste fill plant and a portion of the reclaim water is sent back to the process plant as process water.

The Feasibility Study for Young-Davidson was based on a mine production and mill throughput rate of 6,000 tpd. In late 2011 the Company identified an opportunity to change the underground mining method, in light of higher metal prices, and increase the planned underground mining rate to 8,000 tpd. A mill expansion study was commissioned that in Q1 2012 determined that the mill throughput could be increased to 8,000 tpd for a minimal additional capital expenditure. These capital expenditures primarily involved upgrading several pumps and conveyors and required no purchase and installation of additional process equipment. The mill first achieved a throughput rate of over 8,000 tpd in March, 2014, after the necessary modifications had been made and the permits required to operate at higher levels were received in February 2014.

17.2 Unit Operations

17.2.1 Pit Ore Reclaim

The surface crushing for the open pit will be carried out by an independent contractor and material to the plant will be fed by mine trucks, which will dump the material atop a grizzly fitted with scalping bars set on a 250 mm x 300 mm grid. All the sized material reports into a hopper above on a vibrating grizzly feeder.

The vibrating grizzly feeder drops the open pit ore onto the pit ore feeder conveyor which feeds the coarse ore bin feed conveyor. This coarse ore bind feed conveyor receives the crushed product from both the underground mine and the open pit and is designed for a rate of 550 t/h. A fixed magnet is located at the transfer chute between the two conveyors to capture tramp metal. Each conveyor discharge point is fitted with an extraction hood connected to a dust-collecting bag house. The dust is dropped back on the subsequent conveyor belt through mechanical shaking of the bags.

17.2.2 Underground Ore Crushing Circuit

The mined ore is dropped into an ore pass from which it is metered by a grizzly feeder with 600 mm x 600 mm openings to an apron feeder which drops into the underground jaw crusher with a 160 kW (200 hp) motor size. A fixed magnet mounted above the discharge of the apron feeder to the jaw crusher retrieves any tramp metal. The crushed ore is sent to two underground ore bins each 1500 tonnes in capacity. The product of these two bins reports to two apron feeders which discharge to a conveyor leading to the skip loading system

The material skipped to surface is fed into one of two 500 t surge bins (1 ore, 1 waste) at the headframe. From there, the mill material is transferred via an apron feeder onto the coarse ore bin feed conveyor. This overland conveyor allows filling of a 6,000 live tonnes capacity bin. The bin is insulated to prevent wet material from freezing against the bin wall, while the top of the bin is covered and fitted with a bag house to extract air moisture and prevent freezing. An electric hoist mounted on a beam extending over the edge of the bin outline assists in bringing maintenance supplies to the top of the bin, facilitating the maintenance of the head pulley and idlers at the discharge point of the coarse ore bin feed conveyor.

17.2.3 Coarse Ore Reclaim

Feed from the coarse ore bin discharges onto the SAG mill feed conveyor via two apron feeders fitted below elongated slots at the bottom of the coarse ore bin. Both feeders are normally in operation to achieve the plant design feed rate and to provide some blending capability for managing the SAG mill feed size distribution. In case of a feeder being out of operation, the total ore flow required by the grinding plant can still be obtained with only one feeder in service.

The SAG mill feed conveyor is fitted with a belt scale, providing plant feed tonnage control through the variable speed capability of the apron feeders.

17.2.4 Grinding Circuit

The grinding circuit is in a single-stage SAG mill configuration.

One of the Kemess ball mills was modified to function as the SAG mill in the Young-Davidson grinding circuit. The resulting unit has 6,710 mm ID ø x 11,130 mm (22’ ID ø x 36.5’) of flange-to-flange (FF) length, and is fitted with two 4,474 kW (6,000 hp) synchronous motors taken from the mills at Kemess.

The fresh material from the coarse ore bin feeds the SAG mill at the design rate of 370 t/h, equivalent to 8,880 t per operated day, which meets the average nominal capacity of 8,000 t/d and a plant availability of 91%. Process water is added to the SAG mill feed chute to achieve a pulp density of 75% solids in the mill.

The SAG mill discharge reports to the cyclone feed pump box.

A centrifugal pump (with one stand-by) with a 298 kW (400 hp) VFD drive delivers the slurry from the cyclone feed pump box to a battery of three (two are typically in operation) ø660 mm (26”) hydrocyclones fed from a common manifold. The cyclone battery works in closed circuit with the SAG mill, with the cyclone underflow returning as a circulating load to the mill feed. The cyclone overflow stream proceeds forward to the flotation circuit at a slurry density of 35% solids.

The grinding circuit is designed to operate with circulating loads fluctuating between 100 and 150% relative to the fresh SAG mill feed rate, using the variable speed capability of the cyclone feed pump to maintain the pump box level and the number of cyclones for feed pressure control to the cyclones.

An automatic sampler was installed to collect incremental samples of the cyclone overflow in order to provide a composite shift sample that is used for metallurgical accounting purpose. The secondary sampling stage and collection of the shift composite is achieved by the multiplexer provided with a particle size analyzer (PSA). The PSA is installed at this position to provide feedback to the grinding circuit operator of the grinding product size being achieved. The shift composite sample is also collected for analysis of its precious metal content.

Following sampling, the slurry flows by gravity over a linear trash screen before reporting to the flotation conditioning tank

No reagent additions are required in the grinding circuit.

17.2.5 Flotation, Thickening and Concentrate Regrind Circuits

Flotation

The cyclone overflow obtained from the grinding circuit passes through a trash screen prior to flowing into the flotation conditioner tank. The conditioner tank is agitated with a 7.5 kW (10 hp) drive and provides time for contacting the slurry with the sulphide collectors, potassium amyl xanthate (PAX) and flotation promoter A407 (a mixture of diisobutyl dithiophosphate and mercaptobenzo-tiazol).

The conditioner discharges into the first cell of the rougher flotation where a frother is added (MIBC). The flotation bank is comprised of four cells with 186 kW (250 hp) drives, each with a net volume of 160 m^3, providing an overall average retention time of 50 minutes.

Staged additions of xanthate, A407 and MIBC are made along the bank of rougher cells.

Both the flotation concentrate and tailings streams are fitted with a two-stage sampling system on their respective discharge lines, to provide samples for metallurgical accounting purposes.

Thickening

The rougher concentrate reports to a pump box from which it is transferred to the concentrate thickener via a feed box. The thickener is a high capacity unit with a diameter of 12 m. The flotation tailings stream exiting the last cell is thickened in a high-capacity thickener, 20 m in diameter.

In case of a power shutdown, the thickener drive and underflow pumps are connected to the emergency power system, allowing for recirculation for steady operation.

Flocculant is added to each thickener feed flow which has been diluted with water through an in-line static mixer prior to entering the thickeners. Both thickener overflow streams report to their respective standpipes, from where pumps send the overflow to the mill process water tank for internal reuse.

Each thickener underflow is fitted with a pair of pumps (one stand-by). The flotation concentrate thickener pumps material to the flotation concentrate regrind circuit while the thickened tailings are pumped to the combined carbon-in-leach (CIL) circuit.

Flotation Concentrate Regrind Circuit

The thickened flotation concentrate is pumped to the regrind circuit, comprising of a vertical tower mill, where the solids are reduced to a size distribution with a nominal P_{80} target of 15 µm in closed circuit with a set of cyclones. The cyclone overflow reports directly to the concentrate CIL circuit, while the underflow is returned to the tower mill recycle box for further regrinding.

A dedicated ball bin and bucket are provided for daily ø25-mm ball additions. In addition, a hydraulic screw removal cart and complete spare screw assembly are present to ease screw maintenance duties. Reloading of dumped mill charges is facilitated by a couple of mill charge holding bins.

17.2.6 Leach Circuits

The gold and silver content of the flotation concentrate and tailings are subjected to cyanide leaching in two separate circuits comprised of agitated and aerated leach tanks. Both circuits employ the carbon-in-leach (CIL) method, with the pulp flowing downstream and the carbon transferred upstream for gradual gold and silver loading. The carbon is retained in a given leach tank by using submerged mechanically swept screens. Intermittently, inter-stage recess impeller carbon transfer pumps are activated to transfer carbon upstream to achieve the desired precious metal loadings on the carbon.

Low-pressure compressed air is supplied to each tank, with cyanide and lime solution addition provided along the bank of tanks. The air, cyanide and lime addition is controlled in order to maintain appropriate oxygen, free cyanide concentrations and pH, so as to provide adequate precious metals dissolution and protective alkalinity against cyanide gas evolution.

The circuit configuration allows for bypassing any one of the leach tanks while in operation if the need should arise.

The maintenance of the leach circuits is facilitated by the provision of an overhead crane covering the row of tanks. As well, a pressure washer for the submerged carbon screens, spare screens and a portable submersible pump for tank drainage are present.

Flotation Concentrate CIL Circuit

The reground flotation concentrate reports to a 6,400 mm x 8,620 mm pre-aeration tank with a 22.4 kW (30 hp) agitator. The slurry flows by gravity from this tank into a series of four identical 8,600 mm x 13,900 mm leach tanks each with a 45 kW (60 hp) drive, providing an overall retention time of 48 hours.

All the leach tanks are fitted with a 2.5 m^2 submerged circular and swept carbon screen with a 5.6 kW (7.5 hp) drive (with provision for the installation of a second unit) and a recessed impeller carbon advance pump with a 3.8 kW (5 hp) motor. The loaded carbon extracted from the first tank of the concentrate leach circuit is pumped to the carbon stripping circuit for precious metals elution on a batch-basis.

Combined Leach Circuit

The flotation tailings, along with the previously leached flotation concentrate, enter a series of five 15,500 mm x 16,050 mm leach tanks each with a 93.2 kW (125 hp) drive. Each of these tanks is fitted with two 2.5 m^2 submerged screens with a 3.8 kW (5 hp) drive, with provision for the installation of a third screen. Each tank also contains a recessed impeller carbon advance pump. The carbon from the first of the combined leach tanks transfers into the last concentrate leach tank.

In 2013 it was determined that lower than design recoveries were being achieved due to high solution gold values reporting to the tails dam. To remedy this two additional CIL tanks were commissioned in Q3 2014, binging the current total to five.

The combined leach circuit tailings report to the cyanide destruction circuit.

17.2.7 Carbon Handling Circuits

Based on a design carbon loading of 12 kg/t of a mixture of gold and silver, plus 20% impurities, and the design gold and silver feed grades, the circuits were sized to process a batch size of up to 4.0 t/d of carbon, versus an expected average daily loaded carbon mass of 1.9 t. The extra capacity provides the operator the opportunity to either decrease the stripping frequency or process carbon at a lower loading level.

Carbon Acid Wash Circuit

The loaded activated carbon is pumped from the first flotation concentrate leach tank, via its carbon transfer pump, onto the loaded carbon dewatering screen to remove and wash out the leach slurry. The slurry reporting to the screen underflow goes into a pump box and returns to the back to the first flotation concentrate leach tank of the flotation concentrate leach circuit.

The loaded carbon proceeds by gravity into the acid wash vessel where, on a batch-basis, the accumulated loaded carbon charge is rinsed by a solution of hydrochloric acid. Following the acid wash, the residual acidity is neutralized by replacing the carbon charge moisture with a solution of caustic soda. The rinsed loaded carbon is then mixed with water to produce a slurry and transferred by a recessed impeller pump to the elution, or stripping, vessel.

Carbon Stripping Circuit

The cylindrical carbon stripping vessel receives the carbon on a batch-basis. After the transfer is finished, a solution of cyanide buffered by caustic soda is circulated through the carbon, eluting the precious metals into the solution. The temperature of the system is maintained through heat exchangers on the electrowinning solutions and a steam boiler system. Elution is carried out until recovery plateaus. The precious metals released into this pregnant solution then advance to the eluate solution tank, for feeding the electrowinning circuit.

The stripped carbon is then mixed with water to produce a slurry, drained from the stripping vessel, and passed through a carbon sizing screen.

Carbon Regeneration Circuit

The carbon regeneration circuit operates on a batch and as-needed basis to regenerate the carbon. Regeneration of this carbon is performed on an ‘as-required’ basis, depending on how much fouling has been encountered while in contact with the slurry and residual flotation reagents. If regeneration is not required, the carbon goes directly onto the sizing screen above the carbon fines collection tank. The coarse carbon produced at the discharge of the sizing screen flows to the reactivated carbon holding tank. There the carbon is mixed with water to produce a slurry and introduced into the last tank of the combined leach circuit. When regeneration is necessary, the carbon passes onto a dewatering screen located ahead of the kiln feed hopper. The unit is designed to provide regeneration of 100% of the design carbon load to the elution circuit (4.0 t on a daily basis). When required the plant will strip twice per day for 8.0 t/d.

As required, the carbon flows from the kiln feed hopper into a rotating electrically heated horizontal kiln. The heat and retention time provided in the kiln allow the reactivation of the carbon pores, and burn away most of the fouling contaminants.

The carbon exiting the kiln is cooled in a quench tank prior to being pumped to the carbon sizing screen, where undersized carbon is removed. This fine carbon is collected in the carbon fines collection tank and transferred for dewatering to a plate-and-frame filter. This material is periodically removed from the filter, bagged and shipped to an off-site contractor for the burning and recovery of residual gold content. The coarse carbon retained on the sizing screen gravitates into the carbon holding tank prior to being gradually brought back into the last tank of the combined CIL circuit.

Fresh Carbon Preparation Circuit

Carbon fines generated by attrition within the leach and carbon handling circuits are replaced by introducing new activated carbon to maintain the overall carbon inventory.

To prepare the fresh carbon, it is unloaded from the bulk bags supplied by vendors into the carbon pre-attritioning tank. After a period of agitation, the carbon slurry is pumped onto the carbon sizing screen, sitting above the carbon holding tank, which rejects the fines and replenishes the carbon inventory with the coarse carbon fraction.

17.2.8 Electrowinning Circuits

The EW circuit receiving the pregnant solution from the carbon stripping circuit comprises two EW cells which normally operate in series but are piped to allow for parallel operation as needed. The circuit has a pump box and pumps that receive the solution exiting the EW circuit and re-circulate it back to the pregnant solution tank, again feeding the EW cells in a closed loop until the residual gold in solution is sufficiently low to indicate the end of a cycle, as shown by solution samplers at the discharge of the re-circulating pumps.

Once an EW cycle is completed, the barren solution from the eluate is kept in the barren solution tank, to be used for the next strip cycle. A solution bleed is made as needed to maintain the solution inventory.

A dedicated rectifier powers each EW cell, providing high-amperage DC current for plating the precious metals onto the permanent steel wool cathodes of the EW cells. The cells operate to produce a sludge containing the precious metals. The plated solids remaining on the cathodes at the end of an EW cycle are pressure-washed and report with the surplus sludge deposited at the bottom of the EW cells. The sludge drains from the cells into a common pump box to be transferred by a diaphragm pump into a plate-and-frame filter for dewatering. The dewatered solids are periodically removed from the filter and put onto a cart to be introduced into an oven in preparation for the refining step.

Heat for the EW solution is through the use of heat exchangers and provision is made to control the temperature to prevent flashing prior to EW.

An overhead crane services the EW area, and is used regularly for removal of the EW cells' cathodes.

17.2.9 Gold Refining Circuit

The dried EW sludge is put in a cement mixer, along with the fluxes required for melting it in an electric induction oven. Once the charge is thoroughly mixed, it is introduced into the induction oven.

The molten precious metals are poured into brick moulds set on a mould cart to form the doré bars. A vacuum pin tube is used to collect a sample. The molten slag overflowing from the moulds passes into a slag pot. Low grade material is returned to the primary grinding circuit while higher grade slag is recycled in later pours. The cooled doré bars present in the moulds are emptied, cleaned with a needle gun, punch marked and weighed before being locked up in a vault for storage while awaiting transport to the refinery.

A wet scrubber is provided to clean particulates from the gases released from the induction oven. The scrubber sludge is recycled back to the front of the concentrate leach circuit.

17.2.10 Cyanide Destruction Circuit and Tailings Disposal

The combined leach circuit tailings report to a linear carbon safety screen. From there, they pass through a two-stage sampler arrangement, for collection of the metallurgical accounting sample, before entering the agitated cyanide destruction tank.

In this tank, the residual cyanide content is reduced through oxidation, using the SO2-air process. This involves introducing air, SO_2 and copper sulphate in the destruction tank, with the reagents added in stoichiometric proportions to the residual cyanide content of the slurry. Lime is also added to maintain a target pH.

The treated tailings slurry overflow to a standpipe, from which they are pumped by two centrifugal pumps that operate in series to the tailings impoundment and/or to the paste fill plant, as operating requirements dictate.

17.2.11 Tailings Pipeline

The tailings not required for paste fill are pumped from the plant site to the tailings pond, where the suspended solids settle out before reclaiming the supernatant for reuse in the process plant.

Tailings slurry density is set at 45% solids by weight with 1.40 of specific gravity at 4º C temperature. Two horizontal slurry pumps in series each with approximately 298 kW (400 hp) electrical drives in operation, and two units in series as standby, pump the design flow of 600 m^3/h of tailings via HDPE pipe to the tailings pond. The tailings pipeline is laid on a sand and gravel bedding alongside the service road, to a point at the south-end of the west dam below the dam crest.

The pipeline is ø254 mm (10”), schedule 60 steel, to meet the high pressure requirements over this length. The pipelines have flanged spool pieces at predicted locations, allowing for the connection spigots to discharge into the tailings pond at different locations.

In case of emergency, or during maintenance, low point drain valves are provided to allow the line to be drained into a spill containment pond. The supernatant in the spill containment pond is pumped into the tailings pond and the settled solids is cleaned out and dumped into the tailings pond.

During a planned shutdown, the pipeline is flushed with fresh water; and depending on the outside temperature, the line may need to be drained, if shutdown is anticipated to last an extended period of time.

The pipeline is electro-fused/butt-welded along its entire length, with flange connections only as required. It is anchored with earth berms approximately every 30 m, and with concrete anchors at critical locations to maintain alignment as it expands and contracts due to temperature variations.

The line requires regular checking for erosion along the invert of the pipe, and will be rotated as needed to extend its overall life. To rotate the pipe, it is anticipated that the line will be cut into manageable pieces and fusion-welded back together. As yet, this has not been required.

17.3 Services

17.3.1 Reagents

The reagents used are required for the flotation circuit, dewatering thickeners, leaching, carbon stripping and the cyanide destruction circuits.

Flotation Circuit and Thickeners

The reagents required in these circuits are:

- sulphide collector : potassium amyl xanthate (PAX)
- flotation promoter : Aerofloat 407 (mixture of sodium ditiophosphate diisobutyl and sodium mercaptobenzo-tiazol)
- pH modifier : lime (CaOH)
- activator : copper sulphate
- frother : methyl-isobutyl-carbinol (MIBC)
- flocculant : Magnafloc M-351 (an acrylamide copolymer)

Leach and Carbon Stripping Circuits

The reagents required in these circuits are:

- gold lixiviant : sodium cyanide (NaCN)
- oxidizing agent : LeachAid
- pH modifier – leach circuit : hydrated lime ($Ca(OH)_2$)
- pH modifier – carbon wash and strip : caustic soda (NaOH)
- carbon wash acid : hydrochloric acid (HCl)

Cyanide Destruction Circuit

The following reagents are required in this circuit:

- cyanide oxidizer : sulphur dioxide (SO_2)
- pH modifier : hydrated lime ($Ca(OH)_2$)
- catalyst : copper sulphate ($CuSO_4*5H_2O$)

17.3.2 Water Management

There are four types of water service specifically required by the plant:

- fresh/fire protection water
- process water
- reclaim water
- gland seal water
- potable water.

Fresh Water System

Fresh water is delivered on demand from the Montreal River pump station, based on inventory levels in the fresh/fire water tank.

The fresh/fire water tank is fitted with a pair of pumps (one stand-by) delivering the water under pressure throughout the processing circuits, wherever it is required (reagent preparation, gland seal water, and strip solution make-up). Fresh water can also be used as make-up water, as required, to cover any inventory shortfall in the process water distribution tank. On a normal basis, this is not required since sufficient reclaim water at the tailings pond is available.

The fresh/fire water tank is also fitted with a jockey pump, and an electric and a diesel-driven fire protection pump. All these pumps deliver water to the network of fire hydrants and sprinklers installed in the plant and other surface facilities. The bottom portion of the tank is accessible only to these pumps, with the suction of the pumps delivering fresh water to the process drawing from inventory above a minimum level in the tank. It is necessary to ensure a minimum inventory for fire protection at all times, equivalent to three hours of operation of the system at a flowrate of 175 m^3/h. An additional inventory equivalent to six hours of plant requirements is provided within the tank, as measured assuming a worst-case scenario under which no reclaim water is available.

The tank has a submerged electric heater located in front of the pump suction, ensuring that this area stays ice-free.

Process Water System

The process water system receives the overflow streams of both the flotation concentrate and tailings thickeners as its main feed sources. In addition, reclaim water is sourced from the tailings pond to make up for shortfalls. Fresh water can also be used as a supply source as a last resort if required.

Distribution from the mill water tank is by pumps, at a nominal rate of 604 m^3/h that supply the grinding and flotation circuits. Process water is also used for heat exchangers, etc.

The tank holds an inventory equivalent to two hours of design plant requirements.

Reclaim Water System

Water accumulated within the tailings pond, either as surplus surficial water from tailings deposition or the result of the net precipitation into the pond, is available to cover the process water requirements at the plant.

With most of the process water provided by the thickener overflows, a residual average requirement for reclaim water is approximately 235 m^3/h.

Despite seasonal fluctuations affecting the overall availability of reclaim water, this source has been shown to be sufficiently reliable through dry periods to cover the full residual process water requirements, alleviating the need for make-up water from the fresh water system.

Gland Seal Water System

Gland seal water is obtained from the fresh water distribution ring and delivered to the gland seal water tank. The tank holds an inventory equivalent to one and half hours of plant operation. The gland seal water is distributed through a pressurized ring to all the centrifugal pumps of the plant handling abrasive slurries.

17.3.3 Air Systems

The plant requires five types of air supply:

- dedicated equipment system air
- plant air
- flotation air
- leaching and cyanide destruction air
- instrument air

Dedicated Equipment Air Systems

Some equipment is supplied with a dedicated compressed air system to satisfy the specific needs, including the primary crusher, the SAG mill clutch, and the SO_2 tank pressurizing air.

The crusher and SAG mill clutch compressors and associated air receivers are provided as part of the package installed with these units. The discharge of the surface crusher air receiver branches out to pass through an oil and particulate filter before going into an air dryer, supplying instrument air to this remote plant area.

The associated equipment is specified for a 760 kPa normal pressure duty.

Plant Air

Plant air is supplied to a network of taps located around the plant, for tools used by the operators and maintenance crew, as required. Also, the gold sludge and carbon fines filters draw their air requirements from this source.

A pair of compressors (one stand-by) delivers plant air at a nominal pressure of 860 kPa.

Flotation Air

A multi-stage blower delivers air brought into the flotation cells. The blower operates to yield a pressure slightly in excess to the slurry head found within the cells, or about 45 kPa. The leaching and cyanide destruction compressors provide backups to the single blower provided for the supply of flotation air, utilizing an appropriate pressure-reducing valve implemented within the crossover branch linking the two systems.

Leaching and Cyanide Destruction Air

The CIL tanks and cyanide destruction tank are fed low-pressure (nominally 330 kPa) air from a pair of blowers (one stand-by).

Instrument Air

A dedicated, non-lubricated air compressor provides clean and dry air to the plant instruments. The outflow of the compressor goes into an air receiver, which has an air dryer with self-regenerating desiccant towers installed on the discharge.

17.4 Pebble Crushing

The Young Davidson SAG mill produces pebbles which are extracted at the mill discharge grate. To date these pebbles have not been re-introduced and have been removed and stockpiled. These pebbles have historically accounted for 7-10% of the feed introduced to the SAG mill. It should be noted that reported mill production has excluded the tonnage of pebbles, and only reported tonnages that report to the flotation circuit.

Young-Davidson is currently engineering and constructing a pebble crusher which is being designed to crush and re-circulate the pebbles back into the SAG mill. In addition, the pebble crusher will be capable of crushing the existing stockpile of past produced pebbles. The pebble crushing facility will consist of a:

- reclaim conveyor
- MP200 cone crusher
- feed bin for stockpiled pebbles
- feed conveyor back to SAG mill

It is expected that pebble crushing facility will be operational in the fourth quarter of 2017, and its costs have been included in the 2017 capital budget.

17.5 Mill Performance

The Table 17-1 sets out the Young-Davison mill performance since its commissioning in April 2012. As depicted in the table, the mill feed grade has steadily increased as the underground ramp has progressed, as higher grade underground displaces lower grade open pit stockpiled ore.

Mill recovery increased in late 2015 and through 2016 with the introduction of copper sulphate into the flotation feed. Cold weather, ice, and lack of sunlight in the winter months inhibits the natural degradation of cyanide compounds in the tails dam. This residual cyanide in the mill reclaim water acted to depress the flotation of sulphides in the flotation circuit. Addition of copper sulphate has counter-acted this by activating the sulphide surfaces prior to flotation.

Table 17-1: Young Davidson Project to Date Mill Performance

	Tonnes	Grade (g/t)	Recovery	Produced Ounces
2012	1,483,280	1.36	87%	56,138
2013	2,482,305	1.79	87%	120,738
2014	2,812,954	1.97	88%	156,753
2015	2,753,893	2.02	89%	160,358
2016	2,629,032	2.19	91%	170,041

18 SITE INFRASTRUCTURE

Existing on-site infrastructure includes the following:

- Northgate headframe
- MCM headframe and hoist house
- Surface crusher
- Ore transfer tower that combines the pit ore and underground ore
- Coarse ore Storage Bins
- Mill building housing the Process plant, maintenance areas, supervision offices and metallurgical laboratory.
- Paste fill plant
- Water treatment plant
- Mine maintenance shops.
- Warehouse.
- Administrative offices and mine dry facilities.
- Security gatehouse
- Power switch yard.
- Emergency power generator area.
- Sewage treatment and waste disposal area.
- Fuel storage facilities for diesel and other fuels.
- Propane storage area.
- Water supply and distribution including fire protection system.
- On-site roads and drainage.
- General outdoor laydown areas.
- Explosives magazines.
- Tailings Management Area.
- Waste rock storage
- Water Management Area.

Mining related infrastructure is discussed in Section 16.

18.1 Mine Site Ancillary Facilities

18.1.1 Plant Site Layout

The Northgate production shaft location was the key driver for the location of the ancillary facilities and processing plant. The plant site is located south east of the Northgate Shaft.

The processing plant has overall dimensions of 117 m x 30 m with three different floor levels enabling the plant to take advantage of the topographical elevations to maximize gravity flow as much as possible and minimize rock excavation during its construction.

The building is a steel framed structure with insulated steel roofing and siding founded on concrete piers. It accommodates the process equipment (grinding mills, flotation, carbon stripping and reactivation, electrowinning, gold refinery, cyanide destruction, reagents and utility supply such as compressed air, mill water). Thickening, carbon leach tanks, and water tanks are located just outside the building.

Electrical rooms are provided at two locations, one supplying power for the grinding mill area on the north side and the other serving the refining area on the south side.

The SAG mill grinding area covers approximately 45 m x 30 m with 25 m height to the underside of the roof truss and is serviced by an overhead crane of 10 tonnes capacity. The SAG mill and all other heavy equipment are supported on concrete mat foundations resting directly on bedrock.

18.1.2 Mine Administrative Building and Mine Dry

The administrative offices and mine dry are located in a single two story building adjacent to the MCM Shaft. It houses the administrative, mine operations and engineering offices, first aid, mine rescue, training, shift change area, the dry and locker facilities for underground personnel people.

18.1.3 Maintenance Workshop

The maintenance shop, located adjacent to the mill, has two heavy vehicle repair bays and one welding shops. Overall dimension of the maintenance shop is approximately 38 m x 15 m with a common overhead crane of 10 tonnes capacity to service this area. The shop also includes an office and space for tool crib as well as the compressor for underground.

18.1.4 Warehouse

The warehouse is located between the fuel bay and the mine administrative office. The building size is approximately 32 m x 22 m giving a ground floor area of 700 m^2. One bay is dedicated to receive two containers along the side of the building. A number of other facilities are located around the mine site that house large critical spares under cover.

18.1.5 Laboratory Facilities

The site laboratory facilities is a slab on grade 25.5 by 15.4 m steel clad insulated building housing wet-chemical operations, sample preparation and fire assay; complete with all equipment, electrical connections and ventilation requirements. The core logging facility is situated adjacent to the lab in a 22.2 by 9.2 metre slab on grade building similar to the lab. Both facilities are situated adjacent to the surface maintenance workshop.

18.1.6 Explosive and Cap Magazines

The explosives and cap magazines are located to the north of the open pit, on the east side of the Waste Rock Stockpile. This location permits the storage of up to 39,000kg of explosives (three day storage). Close proximity to explosive producers allows for reduced site storage requirements and does not require on-site explosive manufacturing.

18.1.7 Fuel Storage

Storage requirements on surface for diesel and unleaded fuel storage have been minimized by employing daily fuel deliveries from Kirkland Lake. Dyed diesel and unleaded fuel storage capacities of 80,000 L and 5,000 L respectively have been installed, complete with electric island pumping stations with fuel tracking key card control.

18.1.8 Propane Storage

Storage requirements for propane storage are minimized by assuming weekly deliveries. Propane storage is located three locations; beside the Assay Lab, beside the Paste Plant and beside the Mine Administrative Building.

18.2 Paste Plant

With the adoption of longhole stoping and paste backfill, an 8,000 tpd paste plant was commissioned in January 2014. The paste plant is in a dedicated building to the west of the open pit and directly over the underground mine.

Tailings slurry is received from the mill and thickened in an outdoor high-rate thickener adjacent to the paste plant. The thickener underflow feeds an agitator tank which feeds two disc filters in parallel. The filter cake is fed to continuous mixers and combined with binder from one four binder silos. The continuous mixers feed paste hoppers which in turn gravity feed two cased paste boreholes to the 10130 level underground. Paste distribution underground is via gravity, with boreholes drilled between levels, and steel distribution pipes on the levels.

The paste plant is set up such that either or both disc filters can feed either or both continuous mixers and paste hoppers. This allows the paste plant to supply paste to underground at any rate between zero and +8,000 tpd of paste, and to two stopes at once, depending on the operational requirements of the mine. The paste plant has in the past run at rates of 10,000 tpd.

In Q3 2016 an additional borehole was drilled from the paste plant into the open pit. This borehole will be used to evaluate the supply, on a pilot trial basis, of paste, with a minimum amount of binder, to the mined-out open pit. It is expected that this facility will be operated during the warmer months from May until October, and will potentially give TIA 7 a longer life-span (in years). The pit has a potential tailings capacity of 5 million cubic metres.

18.3 Power Supply and Distribution

18.3.1 Power Supply

Power supply for the Young-Davidson site is 115 kV from a short tap connecting to an existing power transmission line from Hydro One’s Kirkland Lake transformer station approximately 54 km from the site. Utility fault level is 50 kA at 115 kV.

A tap point substation consisting of interrupting breakers, isolation switches and protective relaying was installed at the utility end of the line. The new transmission line from the tap point to the site meets utility standards.

The incoming transmission line is terminated at the mine’s main site substation   The substation has incoming circuit breakers, disconnect switches, power transformers, switchgear and protective equipment for the transformation of power from the transmission voltage level of 115 kV to the site distribution/utilization level of 13.8 kV and 4160 V.

Emergency power is provided by a standby power station sized to provide power to underground mine equipment and process equipment requiring electric power in event of a utility power failure.

The electrical load for the Young-Davidson site is as follows:

- Connected load 32.6 MW
- Average load 27.0 MW
- Load Factor 66%
- Power factor 95%

Note that the average load will increase to approximately 32 MW with the commissioning of the pebble crusher, future installation of additional ventilation, establishment of hoisting from the Lower Mine, and the underground mine ramping up 8,000 tpd.

18.3.2 Main Substation

The main substation is adjacent to the mill, where the largest loads are located, to minimize cabling costs and losses. The substation includes the following equipment:

- incoming dead-end structure
- structures and bus system
- metering transformers
- main incoming circuit breaker
- high voltage isolation switches
- two power transformers
- sub-station electrical room to house the metering, protective relaying and main site distribution switchgear.

The transformer secondaries are connected to a primary distribution centre (PDC) to distribute power to the site.

18.3.3 Main Power Transformers

The main substation consists of two 25 MW power transformers rated as follows:

- 115 kV – 13.8 kV
- 12/16/20 MVA
- ONAN/ONAF/ONAF
- three-phase, 60 Hz
- automatic on-load tap-changer and grounding resistors

The self-cooled and forced-air-cooled ratings are continuous. Each transformer is capable of meeting the total load requirements of the mine in the event of a single transformer outage.

18.3.4 Underground Mine Power Supply

The primary power supply to the underground mine is with two 13.8 kV feeds from the PDC located at the main substation. A third feed provides power to the Northgate Hoist.

The mine distribution is powered from a 13.8 kV shaft cable with taps at the 9590, 9440 and 8900 metre levels. From the "Level Substations" located adjacent to the shaft, cabling is installed on messenger wire throughout the working parts of the mine.

An alternate feed is provided to the underground mine with a 13.8 kV feed installed in the ramp routed down from surface to the 9440 level substation. This is used to provide power to the mine in the event of a failure of the main shaft feeder cable.

Permanent electrical distribution centres is provided at the shaft/hoist areas with lighting and small power receptacles and at the underground crushing and mine dewatering facilities at the 9590 metre level and will be duplicated at the 9100 metre level.

An additional 15 kV line, routed through the exhaust ventilation raise, distributes mine power to strategically located switchgear tap points using 15 kV G-GC cabling. From the tap locations in the main drifts, feeder cables are connected to portable substations at the development and production working faces.

The portable substations are low-profile, heavy-duty, skid-mounted units that are comprised of: main incoming visible fused load break disconnect switch, surge arresters, transformer, neutral grounding resistor, modular LV distribution equipment and grounding and ground check monitoring equipment. Cabling is connected to the distribution gear with approved ground monitoring type receptacles.

The site, including the underground system, has back-up emergency power supply from standby diesel generators located at the Main Substation and rated to meet the total site emergency power requirements. See Emergency Power description below.

18.3.5 Process Plant and Ancillary Services Power Supply

The process plant distribution voltage is 13.8 kV obtained from the 13.8 kV PDC switchgear in the main substation via 15 kV cable. This switchgear feeds the following equipment:

- grinding area, including the SAG mill and all other grinding loads with cable in tray
- crushing area loads via overhead pole line
- flotation area with cable in tray
- leach and carbon handling areas with cable in tray
- tailings, reclaim water, fresh water and ancillary services via overhead pole lines
- paste plant via overhead pole line

18.3.6 Power Quality

Some of the electrical powered equipment generates harmonic distortion. This equipment includes variable frequency drives, rectifiers for battery banks, uninterruptible power supplies for control systems and computer power supplies, and high-efficiency lighting. In addition, the SAG mill motors are variable speed. Significant harmonic distortion can be reflected back onto the Utility power system.

Equipment was selected to meet IEEE Publication No. 519, *IEEE Recommended Practices and Requirements for Harmonic Control in Electrical Power Systems*. This was supplemented with harmonic filters tuned to eliminate higher harmonics as required to meet the Utility standard.

18.3.7 Emergency/Standby Power

Site emergency power is provided with a Standby Power Plant rated for the maximum power required in the event of a Utility power failure or planned shutdown.

The backup power system comprises two diesel generators rated 2.0 MW each, for a total of 4.0 MW, to meet the Young-Davidson Mine emergency power requirement of 3.6 MW.

The main requirements for standby power include:

- Mine ventilation
- Mine dewatering
- Process plant critical loads

The control of the emergency power loads is through the process control system. This system stagger starts, automatically starts and stop loads to keep process tanks properly agitated and run equipment such as kiln drives, lubrication pumps on the large mills and underground mine requirements.

Uninterruptible power supplies are used to provide back-up power to critical control systems. The UPS equipment is sized to permit operations to shut down and back up the computer and control systems to facilitate start-up on restoration of normal (Utility) power.

Emergency battery power packs supply back-up power to the fire alarm system and emergency egress lighting fixtures.

18.4 Water Management

18.4.1 General

This section briefly summarizes the water management and treatment needs for the Young-Davidson site.

18.4.2 Mine Water Settling Pond

The mine water settling pond is an irregularly shaped, single-celled, above grade lined facility, measuring approximately 100 m x 170 m, surrounded by 2 m high berms. The mine water settling pond is located to the north of the mill and to the east of the open pit.

18.4.3 Pipeline to the West Montreal River

The pipeline to the West Montreal River is a 355 mm diameter high density polyethylene pipe of approximately 4.6 km length (with environmental approval for water takings and discharge; Environmental Compliance Approval #6336-ACNH2J and Permit to Take Water for shaft dewatering #6234-AF6JL7).

18.4.4 Plant and General Site Dewatering

Plant site runoff is managed using a system of ditches and catchment ponds as required. This water is pumped to the Stormwater Management Pond #1 settling pond, then pumped to the existing Mine Water Settling Pond for subsequent discharge through the existing pipeline to the West Montreal River.

18.4.5 Mine Dewatering – Underground Workings

Water is pumped from the underground sumps to the tailing dam, or to the Water Treatment Plant and subsequently to the West Montreal River according to the existing environmental approval.

18.4.6 Water Treatment Plant

The Water Treatment Plant (WTP) was commissioned in 2016 to treat water pumped from underground. The WTP treats water for elevated levels of ammonia. The WTP has a capacity of 1,500 m3 per day and utilizes a Veolia Moving Bed Biological Reactor (MBBR) for removal of ammonia and nitrite and a Veolia Actiflo clarifier for removal of suspended solid. The WTP is located within the paste plant building, Water pumped from underground is not continuously supplied as pumping is triggered by level switches in the various sumps. As the WTP performs optimally under continuous supply, a surplus CIL tank in the mill was commissioned in Q1 2017 as a feed / surge tank for the WTP.

Should requirements arise in the future, elevated levels of copper and ammonia can be treated with the addition of Actiflo and MBBR units respectively.

18.4.7 Freshwater Requirements

Potable Water

Potable water is supplied with bottled water

.

Fresh Water and Fire Protection Water

Fresh water for both the process plant and fire water are supplied from a water intake on the West Montreal River. The fire and fresh water reserves are in a combined tank with an active holding capacity of 1,185 m^3.

Fresh water is consumed at an instantaneous rate of 64 m^3/h. The majority of this water is consumed at the gland seals (46 m^3/h), while another 18 m^3/h will be used to make-up a variety of reagents and shower water in the mine and mill dry.

A 6 hour reserve is provided to cover short term interruptions in supply. Adding appropriate design factors, a fresh water surge capacity of 660 m^3 is installed. The tank has level controls to ensure that the water level does not drop below the necessary storage volume for fire protection purposes at any time. Water flow is based on the demand for the largest fire hazard area. A fire water reserve of 525 m^3 is specified to allow for three hours of fire-fighting at a rate of 175 m^3/h.

Process Water

Process water is required for mill operations, and sufficient water storage within the tailings impoundment area is available for recycling. Recycled water from the TIA is stored in the mill water tank and transferred to the processing plant and other required areas as needed.

18.4.8 Tailings Impoundment Area Discharge

The design provides for a single impoundment area constrained by impermeable dams. Tailings are discharged from the west dam in the early stages of the operation with a tailings pond located against the east dam. This arrangement effectively reduces the seepage losses through the west dam foundation. Seepage collection and grouting are provided to control seepage through the west dam towards Mistinikon Lake.

Water is reclaimed back to the process plant from a barge with pumping capabilities located near the east dam. The reclaim water is pumped out and transported via a reclaim water pipeline, which runs along the east dam and then along a dedicated pipeline trench

The existing design assumes that excess run off water coming from the storm water management ponds will normally be pumped to the existing mine water pond, and if there is a shortage in the TIA, it can be directed to the TIA. The TIA east dam has an emergency spillway to pass the runoff resulting from the probable maximum flood.

18.4.9 Domestic Sewage Disposal

Domestic sewage from plant site facilities is collected and transferred in above-ground pipelines to treatment tanks. One 90,000 litre tank is installed and discharges to a raised tile field. Sewage collected from the underground works is transferred to an approved domestic sewage handler for offsite disposal.

18.5 Seepage Pond Pump Station

A seepage collection systems is located below the east and west end of the Tailings Impoundment Area to collect any seepage from the Impoundment Area. All water collected in the seepage collection system is returned to the TIA, via pump stations within the seepage collection wells.

18.6 Site Security

The site has a controlled security entrance on the plant site access road prior to the plant site through which all personnel and material deliveries are directed. Employee parking is located outside of the secured area, and all employees and contractors enter and are security screened in the gate house. All other uncontrolled site access roads are gated and secured.

There are several open pit and historical remnant facilities that are currently fenced with appropriate signage.

18.7 Tailings Impoundment Area

The following sections discuss the existing Tailings Impoundment Area (TIA) facility known as TIA 7.

18.7.1 Tailings Impoundment Area - Design Data

The following were the basic design data underlying the tailings impoundment design:

- TIA 7 will ultimately accommodate approximately 17 million tonnes of tailings
- An additional tailings impoundment and/or management strategy will be developed at a later date to accommodate the remaining tailings beyond the capacity of the existing site location
- The tailings are non-acid generating based on the historic data and available geochemical testing
- A cyanide destruction plant will be operated at the plant site to treat tailings slurry

18.7.2 TIA 7 Site Plan

The storage area TIA 7 is located in a small valley that contained tailings from historic processing operations. The site has a relatively flat bottom with steep side slopes at both the north and south perimeters. The high hills forming the valley comprise primarily bedrock outcrops or bedrock covered with thin overburden.

For the TIA 7 facility, two tailings dam embankments were required. The west dam was located upstream of the original rockfill berm in order to increase the set back from Mistinikon Lake. The east dam was constructed at the east end of the existing tailings deposit. The watershed is approximately 148 hectares.

The site is able to provide a total storage volume of about 15.5 Mm^3 with the top of the dams at an elevation of 365 m. Net storage volume accounting for volume lost in tailings beach slopes, pond and with provision for contingencies is approximately 12.1 Mm^3.

The west and east dams will ultimately be approximately 1,400 m and 800 m long, with maximum heights of about 53 m and 49 m respectively, and average heights of approximately 25 m. A small ‘connecting’ dam (maximum 10 m height) is required between the west and east dams along the north side to contain the tailings. In addition, one small (maximum 5 m height) saddle dams of 200 m lengths are also required on the southern side of the impoundment.

A geotechnical investigation program was carried out to provide basic understanding of the site conditions at the TIA 7, with the following objectives:

- Characterization of the subsurface conditions for the development of designs;
- Determination of the properties of the historical tailings deposit, native soils and upper bedrock through in situ field testing and laboratory testing of recovered samples; and
- Evaluation of the site groundwater conditions.

At TIA 7, eight boreholes each in the west dam and the east dam areas, one borehole each on the north dam and south saddle dam 1 areas were drilled for characterization of subsurface conditions. Standard penetration tests and bedrock hydraulic conductivity tests were carried out in the boreholes.

About thirty test pits were excavated in the area along the west dam alignment and in the impoundment area to define the bedrock surface and to collect tailings and overburden samples.

18.7.3 Tailings Transportation Method

In view of the expected characteristics of the tailings, the sub-aerial method of deposition was considered suitable for the project. The exposure of the tailings to atmosphere maximizes evaporation, resulting in some desiccation and possibly some additional degradation of the residual cyanide compounds that are used in the ore extraction process.

Four methods of tailings transport and deposition were evaluated, including haulage of filtered tailings to construct a dry stack, paste tailings, thickened and conventional slurry. The study concluded that the conventional slurry deposition method presented the preferred option. The conventional slurry disposal method has the lowest operating cost and operating risk, with relatively easy operation and maintenance requirements. Properly engineered tailings/water containment structures coupled with appropriate operating procedures, periodic inspections and maintenance are the main factors underlying the success of this method of tailings disposal.

18.7.4 Design of Dams

The design concepts were developed based on the following considerations:

- The process (tailings slurry) water quality would be close to typical Metal Mining Effluent Regulation discharge limits so that no impact to groundwater is expected during operation. The residual cyanide levels in the tailings pond (after the cyanide destruction in the slurry prior to tailings discharge) were expected to be sufficiently low so that no special measures in this regard would be necessary.
- The tailings would be non-acid generating, with no leaching of metals of consequence expected. Therefore, no long term geochemical impacts to the ground or surface water systems are expected. Note that the mine has, from the start of operations, conducted Acid Base Accounting (ABA) test work on monthly tailings composite. These tests have shown no acid generating potential in the Young-Davidson tails.
- There are limited borrow sources for construction materials available. A design objective was to minimize any new disturbance to the environment. The dams were designed to maximize the use of mine waste material primarily mine rock.
- The dams would be raised in stages to utilize the use of mine rock and reduce the initial capital cost.
- The water in the tailings pond will be recycled back to the processing plant as process water.

West Dam

- The existing tailings in the dam area would be removed due to its low strength and occurrences of several very soft layers.

- The starter dam would be founded on competent native soil or bedrock surface.
- Stage 1 would incorporate a low permeability geomembrane liner to elevation 338.5 m
- After Stage 1, the dam would be raised by centerline method in three or more stages, depending on the availability of mine rock.
- At the dam raise stages, the upstream portion of the dam would be founded on the tailings beach. The downstream shell of the dam would be constructed of mine rock.
- The downstream and upstream side slopes have been assumed, at 2H:1V, which would be re-examined in the following stages of the design.
- The dam would have a crest width of 15 m to provide sufficient space for operating the tailings slurry pipeline.
- Seepage collection ditches downstream of the toe of the dam would drain into seepage collection ponds which will be pumped back into the tailings impoundment.

East Dam

The east dam design concept was essentially the same as for west dam with the following differences:

- A tailings pond would be maintained against the dam.
- The dam would be raised using the downstream method of construction.
- A thick erosion control zone would be provided over the upstream face to handle wind-induced wave erosion.
- The dam crest width would be 20 m during the raises and 10 m upon completion.
- The geomembrane would tie into a grout curtain to reduce seepage through the dam and the bedrock. It would be extended to the top of the final stage dam.

The tailings containment will be confined by a connecting dam along the northern boundary. Along the south perimeter, one saddle dam will be required during the later stages of the operations.

18.7.5 Tailings and Water Management

The Tailings Impoundment Area is close to Mistinikon Lake and adjacent to an existing public boat launch. While the dams and TIA do not encroach on the lake / boat launch, the TIA covers a portion of the public access road to the launch area and thus a new access road was constructed.

Tailings were discharged from the west dam in the early stages of the operation and a tailings pond is located against the east dam. This arrangement effectively reduces the seepage losses through the west dam foundation and toward Mistinikon Lake. In the later stages of the operation, the tailings will also be discharged along the north and south perimeters of the impoundment to form desired tailings beach geometry, adequate for closure. The impoundment is designed to store runoff resulting from a 1:100 year spring runoff event during the operation.

Tailings water is reclaimed and returned back to the plant for use in the process. A barge with pumping facilities is located in the tailings pond close to east dam. The reclaim water is pumped out and transported via a reclaim water pipeline, which runs along the east dam and then on the north access road pipeline bench to the plant. The pumping facilities and the reclaim pipeline are sized to allow excess water to be pumped through the reclaim pipeline and directed to the Mine Water Pond. From the Mine Water Pond, water can be discharged to the Montreal River.

The tailings impoundment has an emergency spillway to pass the runoff resulting from the design extreme hydrologic event, being probable maximum flood (PMF). The dam is designed to have sufficient freeboard to safely pass the PMF peak flows.

The watershed north of the North Perimeter (‘connecting’) Dam will be directed to the natural creeks.

18.7.6 Current Status of TIA 7

TIA 7 is being built in stages, with five stages having already being constructed and final stage to be constructed in 2017 and 2018. Table 18-1 presents staged construction of the TIA 7 facility.

TIA 7 has a fixed capacity that, in combination with placement of tailings underground as paste, and the placement of tailings in the open pit as paste, will necessitate the permitting, construction and commissioning of an additional TIA at some point in the future. Figure 18-2 depicts cumulative tonnages of mill tailings expected to be sent underground, expected to be deposited in the open pit, with the remainder having to be deposited in surface TIA’s. As can be seen, with an ultimate capacity of 17 million tonnes, the final TIA 7 facility will be reaching ultimate capacity in 2023.

Young-Davidson is currently permitting another TIA know as TIA 1, located to the southwest of the mine site in an area that previously was a site for tailings disposal. The TIA 1 location is depicted in Figure 18-2. While there is no guarantee that this proposed location will ultimately be permitted, the Company is confident that permits will be granted, and has alternatives, albeit less desirable, available to it should permitting of this location not be successful.

Table 18-1: Stage Construction of TIA 7

Stage	Construction	West Dam Elev (m)	East Dam Elev (m)	Spillway Elev (m)	Design	QA/QC
1A	Aug 2010 - Nov 2011	339	345	338.5	AMEC	AMEC
1B	Sep 2012 - Dec 2012	344	345	343	AMEC	Golder
2A	Jun 2013 - Nov 2013	352	350	349	Golder	Golder
2B	2015	355	355	354	Golder	Golder
3A	2016	360	360	359	Golder	Golder
3B	2017-2018	365	365	364	Golder	TBD

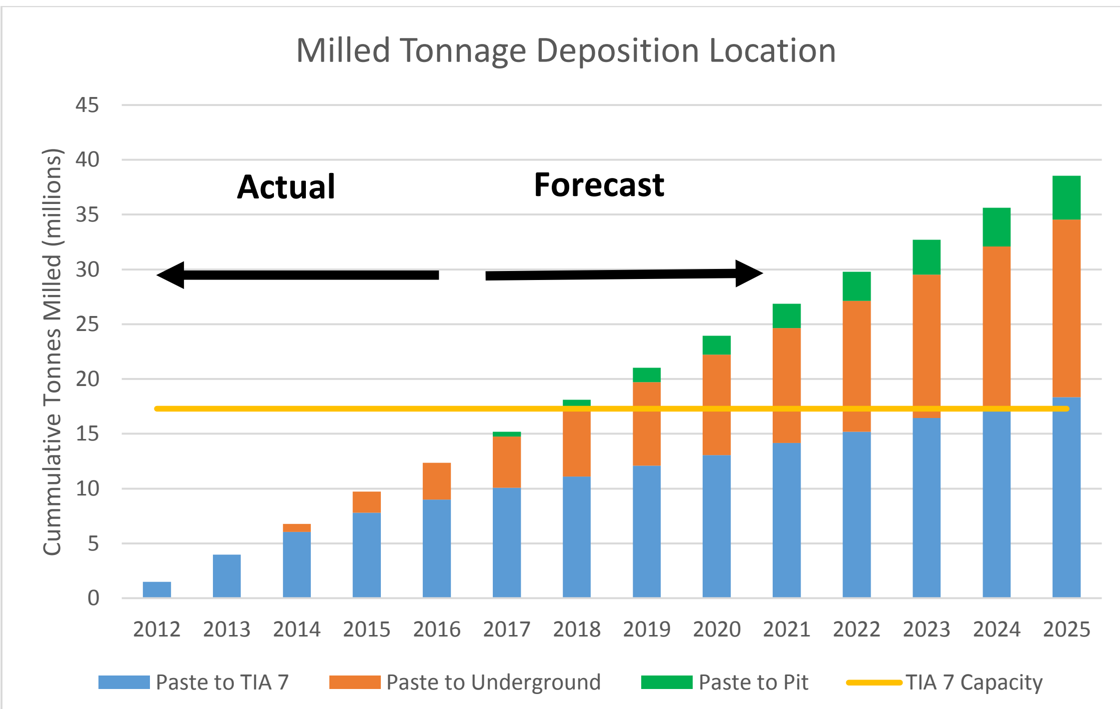


Figure 18-1: TIA 7 Capacity and Milled Tonnage Deposition Location

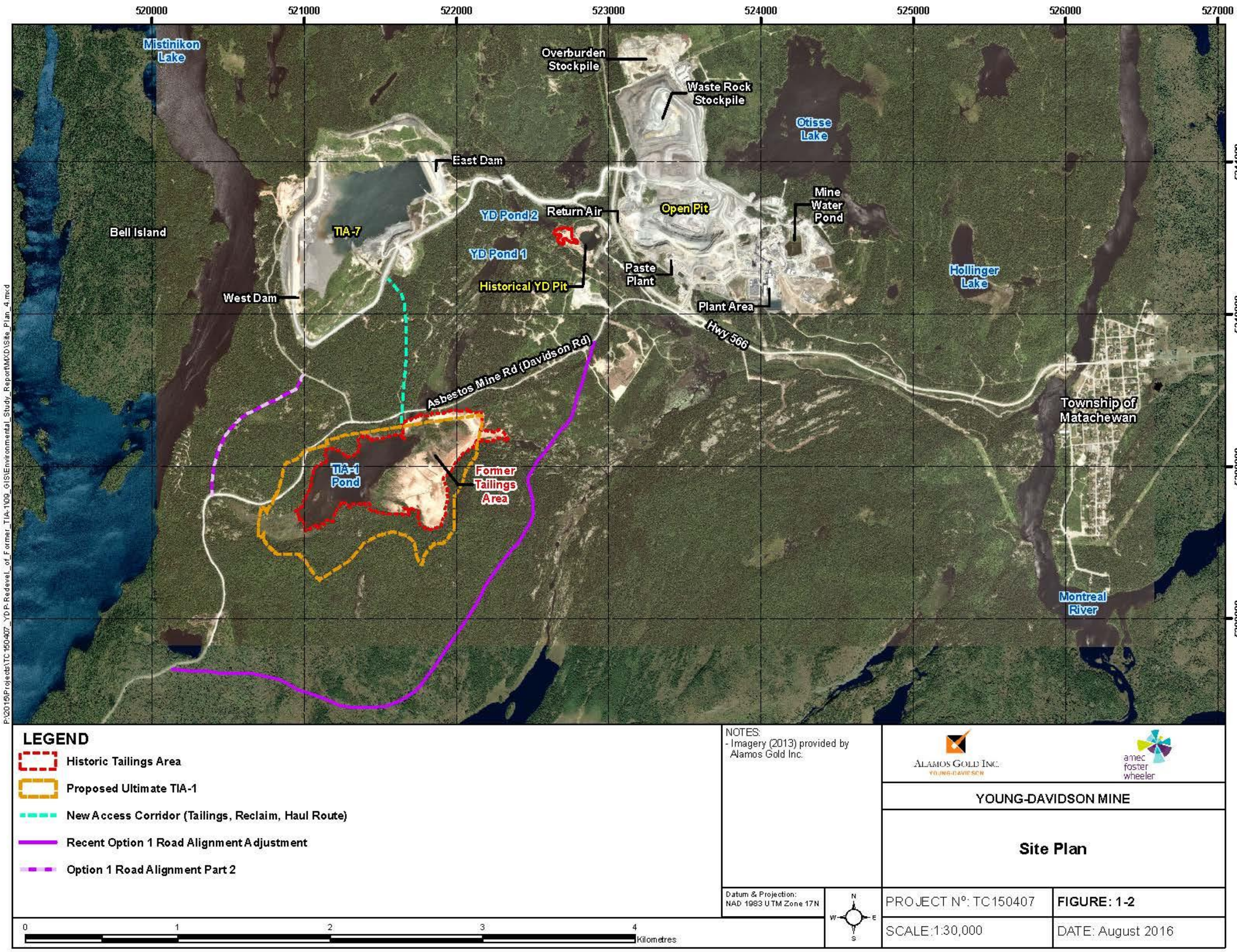


Figure 18-3: TIA 1 Location

18.8 Mine Rock and Overburden Stockpiles

18.8.1 Mine Rock Storage Requirements

Mine rock was produced from the open pit operation and continues to be produced from the development of the underground ramp, shafts and underground level development. Underground mine rock is either hoisted to surface in the Northgate shaft or the MCM shaft, or hauled to surface with underground haulage trucks. Some mine rock is left underground and is co-disposed with paste fill in mined stope voids. The maximum amount of mine rock to be brought to surface in the current life of mine plan is 7.8 million tonnes, assuming none is co-disposed underground. Some of this mine rock will be used to build subsequent TIA lifts.

ARD testwork undertaken in 2010 by AMEC indicates that mine rock and tailings will have little potential to generate acid or produce leachable metals at concentrations of concern. Results for Acid Base Accounting (ABA) based on these samples indicate that the mine rock will have excess capacity to neutralize any acidity produced by oxidation of the sulphides present.

18.8.2 Overburden Storage Requirements

Overburden from pit stripping and material from grubbing around the tailings dams will be used for cover for closure of existing tailings areas and for closure of TIA 7 and subsequent TIA's.

The Mine Rock and Overburden Storage Facility area is used as a stockpile area for this material.

18.8.3 Location

The Mine Rock Stockpile is located north of the mine site, south of the public access road to Otisse Lake, west of Otisse Lake and east of Highway 566. It covers an area of about 26 ha.

The Overburden Stockpile is located north of the public access road to Otisse Lake.

18.8.4 Stockpile Design

The waste rock and overburden stockpiles were designed for long term stability to facilitate final closure should it become necessary to leave the material in place. The stockpile design assumed a maximum height of approximately 30 m comprising three lifts. A bench of 15 m width for each lift has been used. The side slopes of each lift are between 1.3H:1V and 1.5H:1V. The overall constructed rock pile side slopes are 2.5H:1V to 3.0H:1V, depending on the site foundation conditions

The stockpiles are constructed by dumping from haul trucks from appropriately designed access ramps. Safety berms along the access ramps and runoff ditches along the toe of the stockpile are provided for safety reasons and ensure proper drainage. Runoff is captured in ditches and directed to the mine water settling pond if needed prior to release to the environment to remove suspended solids.

18.9 Access Roads and Slurry Pipeline Benches

Access to TIA 7 is via an 8 m wide main access road from the Open Pit. Along the south side of the Tailings Dam, the access road alignment is purposely located along the connecting dam alignment. The connecting dam will be built in the later stage of the operation, incorporating the existing access road as part of the dam. The tailings and recycle water pipelines are located on the tailings pond side of the access road. Construction access limbs take off from the main access road to the required location of the dams during lift construction.

19 MARKET STUDIES AND CONTRACTS

Gold and silver are the principal commodities at Young-Davidson and are freely traded, at prices that are widely known, so that prospects for sales of any production are virtually assured. Prices are usually quoted in US dollars per troy ounce.

There are numerous contracts in place at the mine to support mining or processing that augment Young Davidson's efforts. Currently, there are contracts in place to provide supply for all major commodities used in mining and processing, such as equipment vendors, power, explosives, cyanide, binder, tire suppliers, raise boring, ground support suppliers, long hole drilling contractors and diamond drilling contractors.

The terms and rates for these contracts are within industry norms. These contracts are periodically put up for bid or negotiated to ensure the rates remain favorable to Alamos.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL AND COMMUNITY IMPACT

20.1 Environmental Considerations

The Young Davidson Mine has been designed to maximize utilization of lands previously altered by historic mining-related activities and, where practical, reclaim brownfield areas progressively during operations; and, minimize overall Project footprint and impacts.

Environmental permits were required by various Federal, Provincial, and municipal agencies for the construction and operation of the Young-Davidson Mine, and are in place. A legal registry is used to maintain a list of active environmental permits covering operation of the Young-Davidson Mine.

Amending existing permits is required from time to time, and in 2015, applications for amendments have been submitted regarding increasing maximum tonnage throughput along with a minor amendment for the design of the septic tile field. In 2016, an amendment to the Environmental Compliance Approval for Industrial Sewage Works was received to allow for the operation of an Effluent Treatment Plant for the treatment of underground water.

Waste rock and ore are routinely sampled for acid rock drainage (ARD) potential as per the internal programs for ARD and metal leaching. Since there are no significant ARD issues related to the waste and ore from the Young-Davidson Mine, waste rock materials can be used for construction purposes.

Active tailings facilities for the operations were designed by third-party consultants. Annual inspections are conducted by these firms. In addition, engineering assessments and investigations to enhance tails storage strategies are performed as required. Permitting is underway for the redevelopment of the historical Young-Davidson Tailings Area (TIA 1) for additional storage for the life of mine.

Water treatment processes are in place at the processing facilities to address the destruction of cyanide in solution. The Young-Davidson Mine recently commissioned its biological reactor to reduce the ammonia concentration from the mining process along with a treatment to precipitate heavy metals. All effluent discharges to the environment are in compliance with all applicable laws.

20.2 Environmental Regulations

The Mine is required to comply with applicable environmental legislation and regulatory requirements. A list of existing permits for the operation of the mine and required permits for redevelopment of TIA-1 are displayed in Table 20-1.

20.2.1 Required Environmental Approvals

Additional permitting will be required for the redevelopment of the historic TIA 1 Tailings area to accommodate the remaining tailings storage requirements for the Project to end of mine life. With an expanded Mineral Reserve and Mineral Resource of approximately 57 Mt, and only 40 Mt of existing storage at site (20 Mt to be returned underground as paste backfill and a total capacity of approximately 17 Mt in the existing TIA 7 tailings area); there is a tailings storage shortfall of approximately 20 Mt that will be addressed through redevelopment of former tailings area TIA 1 and deposition of tailings as paste into the open pit.

Redevelopment of TIA 1 as a tailings storage area would encroach upon a portion of the existing Asbestos Mine Road. Asbestos Mine Road is a local resource access road that connects to Highway 566, and also provides recreational access to Mistinikon Lake. The road is used by a number of local cottage owners, as well as by other resource users. Development of TIA 1 would therefore require a realignment of Asbestos Mine Road. The construction and operation of TIA 1 would also require a pipeline and road corridor between the new TIA 1 and the existing TIA 7, to facilitate the transfer of tailings slurry.

It was confirmed in February 2016 that the proposed tailings project would not trigger the requirement to prepare an Environmental Assessment (EA) pursuant to the *Canadian Environmental Assessment Act 2012*.

It has been determined through consultation with ECCC that the redevelopment of TIA 1 will require a listing under Schedule 2 (Appendix B). The Schedule 2 process requires a stand alone alternatives assessment for mineral waste disposal which has been submitted to the federal agency. Fish compensation plans are required to achieve a no net loss to fisheries productivity. A draft fish habitat compensation plan describing these compensatory measures has been reviewed and accepted by the Department of Fisheries and Oceans.

A Provincial class EA is required for completion pursuant to Ministry of Natural Resources and Forestry's (MNRF) *A Class Environmental Assessment for MNRF Resource Stewardship and Facility Development Projects*

- A Work Permit (or permits) pursuant to the Public Lands Act for realignment of Asbestos Mine Road; and
- Forest Resource Licenses for tree clearing in relation to TIA construction, realignment of Asbestos Mine Road, and development of the construction and operations access corridor between TIA 7 and TIA 1;

Table 20-1: Permits Granted

Permit/Licence/Assessment	Agency Responsible	Description
Closure Plan Amendment *Mining Act*	Ministry of Northern Development and Mines	For mine production

Comprehensive Environmental Compliance Approval – Air and Noise *Environmental Protection Act*	Ministry of the Environment and Climate Change	Approval to discharge air emissions and noise
Environmental Compliance Approval – Industrial Sewage Works *Ontario Water Resources Act*	Ministry of the Environment and Climate Change	Approval to treat and discharge effluent (mine / pit water, TIA, septic field, oil water separator)
Permits to Take Water *Ontario Water Resources Act*	Ministry of the Environment and Climate Change	Water takings from surface or ground water
Forest Resource Licence (Cutting Permit) *Crown Forest Sustainability Act*	Ministry of Natural Resources and Forestry	Clearing of Crown merchantable timber
Plans and Specifications Approval *Lakes and Rivers Improvement Act*	Ministry of Natural Resources	Dams and dikes in watercourses, including tailings impoundment area
Work Permit *Public Lands Act*	Ministry of Natural Resources	Work / construction on Crown land
Land Use Permit *Public Lands Act*	Ministry of Natural Resources	Pipeline corridors
Leave to Construct *Ontario Energy Board Act*	Ontario Energy Board	Approval to construct a transmission line
Approval of Works in Navigable Waters *Navigable Waters Protection Act*	Transport Canada	Construction of transmission line crossing over the West Montreal River.
Letter(s) of Advice *Fisheries Act*	Fisheries and Oceans Canada	Disruption to creeks and ponds supporting fish populations; approval for groundwater dewatering effects.
Licence for a Magazine for Explosives *Explosives Act*	Natural Resources Canada	Construction and operation of an explosives magazine.

20.3 Closure Planning

The objective of the Mine Closure Plan is to present a decommissioning strategy that reflects the current and expected site conditions and defines a program which ensures the long-term chemical and physical stability of the site. The ultimate goal of the Closure Plan is to ensure that chemical and physical impacts to the site are minimized during operations, and that the site is returned as close as possible to pre-development conditions at close-out.

The Closure Plan has been developed using data collected during pre-development (baseline) physical, chemical, and biological studies of the site and the surrounding environment.

20.3.1 Proposed Rehabilitation Measures

At the conclusion of the Mine life, the following close-out rehabilitation measures will be implemented:

- Cap the ventilation raises and shafts in accordance with Ministry of Northern Development and Mines (MNDM) requirements and best practices;
- Utilise excess mine waste rock for backfill or use for rehabilitation;
- Flood the open pit and underground mine workings;
- Secure the portal in accordance with MNDM requirements and best practices;
- Removal of surface buildings, associated infrastructure and equipment;
- Break up and bury concrete structures;
- Remove all chemicals, petroleum products and explosives;
- Remove or treat contaminated soil;
- Deepen spillways in Tailings Impoundment Area, cover and seed beached tailings and dams;
- Drain Settling Pond and Stormwater Ponds, re-grade and cover;
- Re-establish natural drainage; and
- Scarify, cover and seed disturbed areas.

20.3.2 Expected Final Site Conditions

The intent of the final closure of the Young-Davidson Mine is to ensure that the site is safe and stable and represents minimal concern with respect to public health/safety or the environment.

The benign nature of the waste rock, and responsible operating and close-out practices, will minimize potential chemical stability concerns associated with the Mine. Contouring and revegetation of the site will maintain physical stability, prevent erosion and return the land to a condition that is suitable for the return of native plants and animals. Upon closure of the site, it is anticipated that conditions will return to their pre-development state.

20.3.3 Estimated Mine Closure Costs

The estimated costs of mine closure and ongoing monitoring are $10 million. A closure bond is in place to cover this amount. The cost of closure has been accounted for in the cash flow at the end of the LOM.

Alamos Gold is responsible to provide the full amount of the financial assurance, subject to any required or approved changes to the closure plan. Financial assurance will be submitted to the MNDM prior to the beginning of the associated activities of the phase.

20.4 Engagement

Alamos Gold is committed to the ongoing development and operation of the Young-Davidson Mine while engaging in an open and respectful dialogue with Aboriginal people, having an interest in the Mine. Alamos has committed to fostering positive working relationships with the First Nation (FN) groups in an effort to better understand each other's perspectives and shared interests in the environment. Using varied and culturally appropriate engagement activities Alamos will continue to engage and share information in an open, honest and transparent manner.

Impact Benefit Agreements (IBA) have been developed with both the Matachewan First Nation and Temagami First Nation to ensure that the communities derive positive benefit from Young-Davidson Mine operations as it is in part of their traditional territories. Several committees have been implemented, as part of these agreements, to share and gather input on all aspect of the mining operations.

Alamos Gold has engaged with the Métis Nation of Ontario (MNO) since September 2008. Ongoing discussions were carried out with representatives of the MNO on environmental related matters on a regular basis. To date, no impacts from the Young-Davidson Mine has been identified through the MNO Consultation Protocol process and notifications continue to be provided.

Engagement methods with Aboriginal groups include:

- Meetings and workshops with Aboriginal groups, by working with Matachewan First Nation and Temagami First Nation's Environmental Committees and through notification with the MNO;
- Making the drafts and final permit application documents available for review in electronic and hard copy format;
- Posting notices in First Nation offices advising of upcoming community meetings and workshops;
- Maintaining regular liaison with FN leaderships;

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Capital costs are based on the latest mine construction data and budgetary figures and quotes provided by suppliers. Capital cost estimates include funding for infrastructure, mobile equipment, development and permitting, and miscellaneous costs. Infrastructure requirements were incorporated into the estimates as needed. Table 21.1 reflects capital costs budgeted for 2017 as well as capital costs estimated for the life of mine.

Table 21-1: 2017 Budget and LOM Capital Costs

$CDN	2017 Budget	2017-2031 LOM
Surface		
Pebble Crusher	$5,200,000	$5,200,000
Tailings	$6,954,000	$30,193,000
Environmental & Reclamation	$600,000	$10,427,000
Underground		
Capital Development	$42,778,000	$336,283,000
Underground Infrastructure	$8,045,000	$40,518,000
Lower Mine Infrastructure	$13,658,000	$52,753,000
New Equipment	$8,861,000	$13,194,000
Replacement Equipment	$2,383,000	$35,311,000
Equipment Rebuilds & Misc. Maint.	$7,404,000	$64,281,000
Diamond Drilling	$1,554,000	$11,630,000
Total Young-Davidson	**$97,437,000**	**$599,790,000**

21.1.1 Underground Development

Ongoing underground capital development comprises the majority of the life-of-mine project capital. Capital development comprises a portion of the overall horizontal development the other portion being operating development which is included in the mining cost per tonne. Horizontal development consists of the ramps, footwall development and the stope accesses. Capitalized development consists of 100% of the ramps and footwall development and the first 15 metres of the stope access. The remainder of the stope access drives are expensed.

The LOM include 132,000 metres of horizontal development of which approximately 47% is capitalized. All vertical development is capitalized and is included in Underground Infrastructure.

21.1.2 Underground Infrastructure

Underground Infrastructure includes ongoing projects required for the development of the mine's infrastructure. Specific items included within Underground Infrastructure include:

- Ore and waste pass cone dumps
- Advancement of the paste fill distribution system
- Ventilation infrastructure including raises, doors, man fans, and auxiliary fans
- Dewatering sumps and pumps

21.1.3 Equipment

Having reached close to design capacity, the equipment fleet for Young-Davidson peaks in 2017 and as such, requirements for additional new pieces of equipment diminishes beyond 2017. After 2017, the majority of equipment capital consists of rebuild and replacement capital. Young-Davidson has assumed industry standard life cycles for rebuild and replacement of major pieces of equipment and has adopted maintenance practices to ensure that these life cycles are achieved. As Young-Davidson had been operating its own equipment since 2010, the first replacement cycles are beginning in 2017.

21.1.4 Environmental and Reclamation

Annual capital expenditures are assumed for ongoing environmental projects and commitments. Young-Davidson has assumed an environmental liability of $10 million at the end of the mine's for reclamation of the property. This is in concurrence with the properties current bonding.

21.1.5 Major Capital Projects

In 2017, and through the life of the mine, a number of major capital projects are to be undertaken, which are summarized in the following sections:

TIA 7 Dam

Construction of the final lift, 3B, of TIA 7 will commence in 2017 and will be completed in 2018. Expected costs over the two year period are $10 million. Construction is by local contractors under the mine's supervision with Golder Associates supplying the design and QA.

TIA 1 Dam

It is anticipated that the proposed TIA 1 dam would begin construction in 2019. A number of dam raises would be undertaken through the life of the mine on a regular basis. Permitting for TIA 1 is currently in progress.

Pebble Crusher

In 2017 a pebble crusher is being installed to crush pebbles discharged at the grates on the SAG mill and re-introduce them back to the SAG mill. The crusher will also have the ability to be directly fed from the existing stockpile of pebbles via a bin and feeder. Installation of the pebble crusher is expected to increase overall mill throughput but this has not been modeled in the mine's cash flow model. Expected costs for this project are $5.2 million and completion is expected in the fourth quarter 2017.

Lower Mine Development

The Lower Mine development project is being undertaken to provide access and hoisting capabilities for Mineral Reserves below the 9590 level. The project began in earnest in 2016 with the completion of the MCM shaft and its ability to gain access to the 8900 level, the setting off point for development of the bottom of the mined. The project will continue through 2019 and will be undertaken by Young-Davidson mine development crews with contractors installing major equipment. Major tasks to be undertaken include:

- Completion of the waste pass from the 9440 level to the 8900 level
- Raise boring and ground support of the third leg of the Northgate shaft
- Development of conveyor drive and access drive
- Construction of the loading pocket and the ore and waste pass feeders on the 8950 level
- Installation of the crusher, and the ore pass feeders above the crusher
- Raising of the ore and waste passes from 9590 level to the 9100 level
- Excavation of the coarse ore bin, fine ore bin, and waste bin at the crusher
- Excavation of the fine ore bin and waste at the Northgate loadout.
- Installation of the main conveyor 740 metre from the 8920 level to the 9000 level.
- Removal of the pentice, installation of ground support, rope changeover, and shaft commissioning

Costs outlined for Lower Mine Infrastructure in Table 21.1 above are only for shaft sinking, bins and other major excavations, passes, and equipment procurement and installation. Costs for lateral development are included in Capital Development within the table.

21.2 Operating Costs

Table 21-2: 2017 Budget and Life of Mine Operating Costs

	2017 Budget	2017-2031 LOM
Mining ($/t mined)	44.84	36.80
Milling ($/t milled)	14.95	14.34
Admin ($/t milled)	4.30	3.94
Refining ($/ounce)	1.30	1.50

Mining costs are expected to fluctuate year over year. Inputs such as labour, supplies, power, diesel are expected to remain flat, however rates of capitalization of mine development and mining rate will impact the mining cost per tonne.

21.2.1 Mining Costs

Given the same inputs underground mine operating costs will vary from year to year depending on the proportion of mine development that is capitalized.

Major underground mining cost items as a percentage of total underground mining costs in the 2017 budget are presented in Figure 21.1.

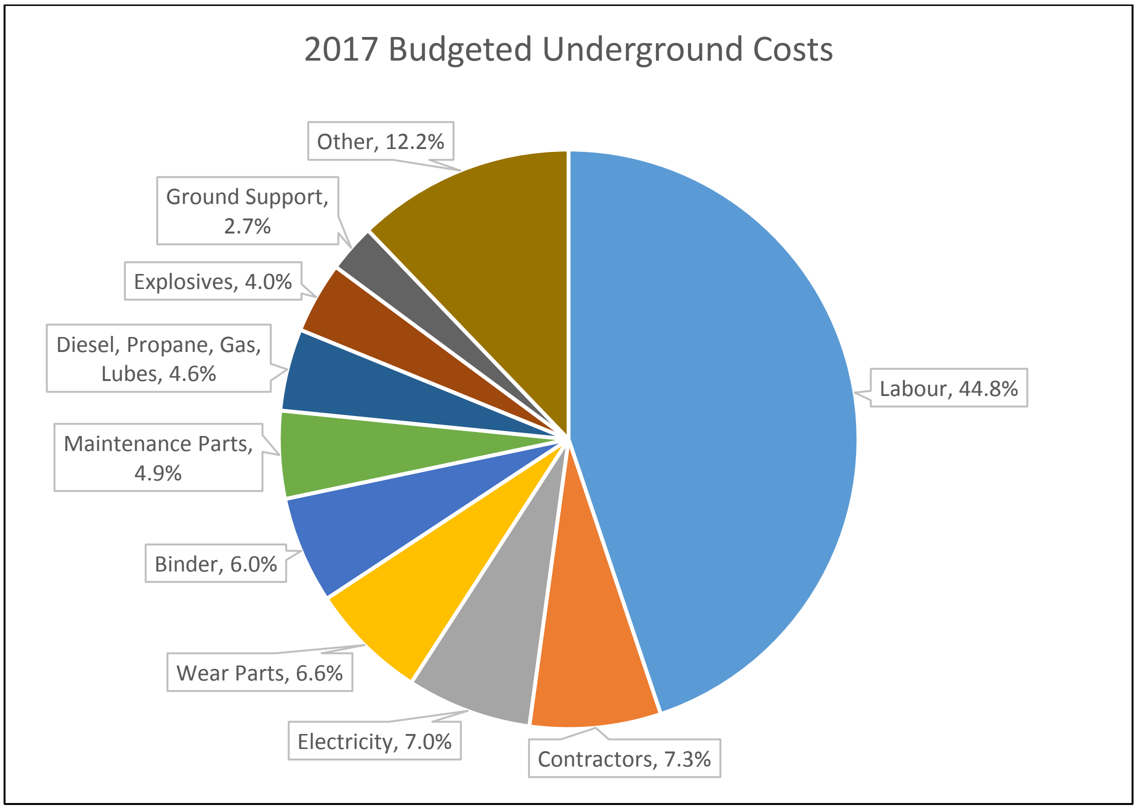


Figure 21-1: 2017 Underground Budget Cost Breakdown

21.2.2 Milling Costs

Major milling cost items as a percentage of total milling costs in the 2017 budget are presented in Figure 21.2.

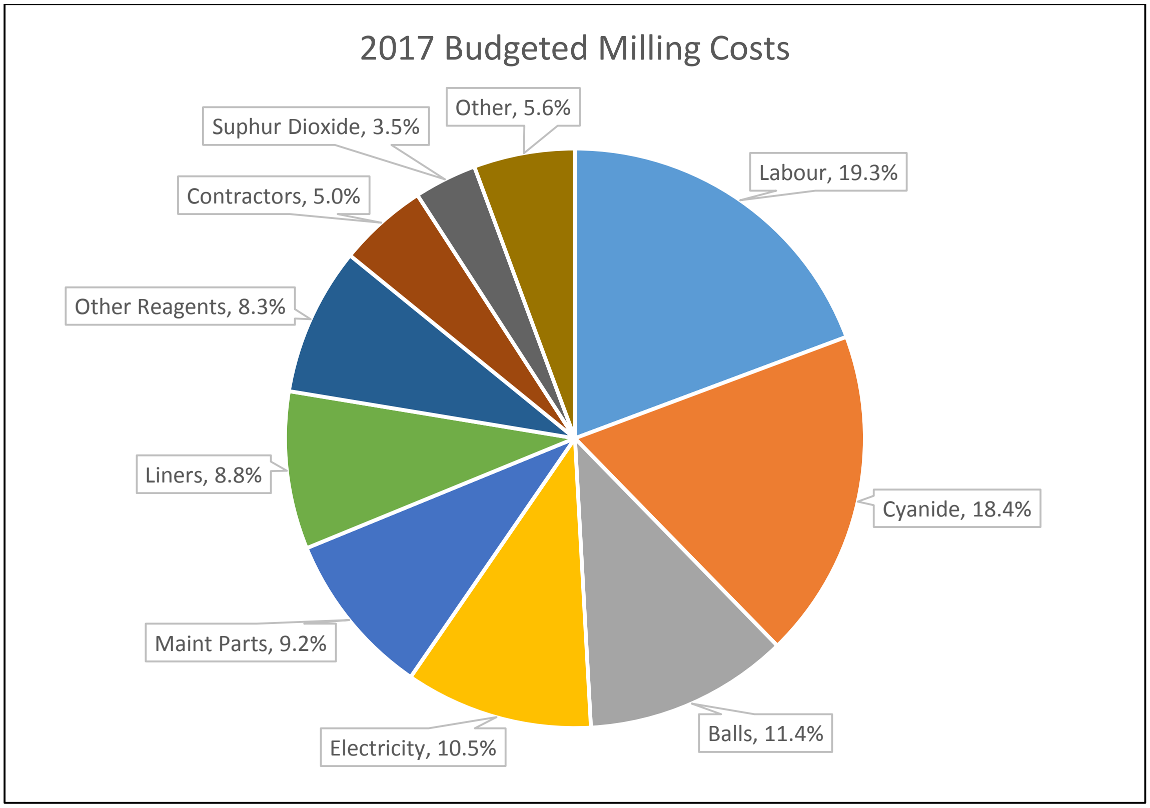


Figure 21-2: 2017 Mill Budget Cost Breakdown

21.2.3 Administrative Costs

Major administrative cost items as a percentage of total administrative costs in the 2017 budget are presented in Figure 21.3.

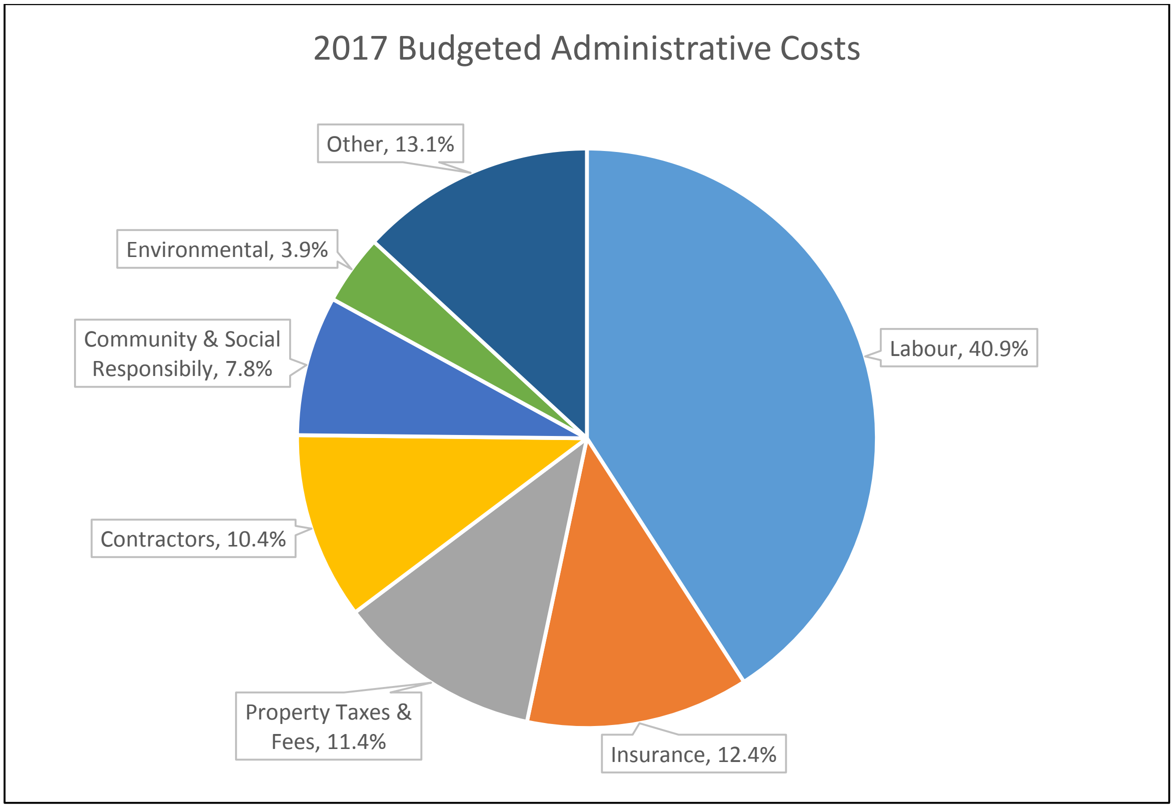


Figure 21-3: 2017 Administration Budget Cost Breakdown

22 ECONOMIC ANALYSIS

Under NI 43-101, producing issuers may exclude the information required in this section on properties currently in production, unless the Technical Report includes a material expansion of current production. Alamos is a producing issuer, the Young-Davidson Mine is currently in production, and a material expansion is not being planned. Alamos has performed an economic analysis of the Young-Davidson Mine using the estimates presented in this report and confirms that the outcome is a positive after tax cash flow and net present value at a 5% discount rate at USD $1250 per ounce of gold price that supports the statement of Mineral Reserves.

23 ADJACENT PROPERTIES

The Young-Davidson Mineral Resources and Mineral Reserves are centered on a large claim block owned by Alamos. There is currently no significant gold mineralization located in adjacent properties.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that is material to this report

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary

The pertinent geological, mining and metallurgical data and information from the Young-Davidson Mine was reviewed to obtain a sufficient level of understanding to assess the status of the project as of the effective date of this report.

Based on the information gathered during the preparation of this report the QPs' conclude that ongoing and operation and continued development of the Young-Davidson Mine is supported by positive economic factors.

25.2 Geology and Mineral Resources

The Young-Davidson deposit is a syenite hosted gold deposit hosted by granodioritic to syenitic stocks and dykes.

The sampling, sample preparation, analyses, security, and data verification meet or exceed industry standards and are appropriate for Mineral Resource estimation.

The parameters, assumptions, and methodology used for Mineral Resource estimation are appropriate for the style of mineralization.

Mineral Resources are reported exclusive of Mineral Reserves and are estimated effective December 31, 2015.

Underground Mineral Resources Young-Davidson Mine, are:

- Measured Resources – 4,248 thousand tonnes, grading 3.47 g/t Au, containing 474,000 oz Au
- Indicated Resources - 3,707 thousand tonnes, grading 3.43 g/t Au, containing 408,000 oz Au
- Inferred Resources – 3,523 thousand tonnes, grading 2.76 g/t Au, containing 312,000 oz Au

25.3 Mineral Reserves

Mineral Reserves at Young-Davidson are estimated in a sound, consistent and well documented manner. Mineral Reserves are backed up by a mine plan and cash flow model and demonstrate positive economics. Economic inputs including metal prices, operating costs and capital costs are re-evaluated on an annual basis and are used in the generation of Mineral Reserve estimates.

25.4 Mine Development and Mine Plan

An appropriate mine plan has been developed, and is being implemented, to effective and efficiently mine the Mineral Reserves outlined in this report. The mine should continue to optimize the schedule so as to maximize the value of the project.

25.5 Metallurgy and Processing

The process plant is currently operating at design rates of 8,000 tpd and with recoveries of 91%. No anticipated long term changes in ore characteristics are anticipated that might cause the process plant to deviate from these operating levels.

25.6 Environmental Considerations

The mine has all necessary permits in place to continue operations. The mine is the progress of permitting the proposed TIA 1 tailing facility, and no delays are anticipated. The mine has regularly engaged, and expects to continue to engage, the local communities in operating and present and future permitting issues.

25.7 Cost Estimates and Financial Analysis

Estimations of budgetary and life-of-mine operating costs are carried out on a first principles basis based on annual reviews of major inputs and productivities. Alamos has performed an economic analysis of the Young-Davidson Mine using the estimates presented in this report and confirms that the outcome is a positive after tax cash flow and net present value at a 5% discount rate at USD $1250 per ounce of gold price that supports the statement of Mineral Reserves.

26 Recommendations

26.1 Geology

The current reserve and resource base extends to an elevation of ~8,900, and is based on the limits of surface drilling. Geologically, there is no evidence that mineralization should not continue down dip at depth, and may represent a potentially significant additional mineral resources. It is recommended that YD geologic staff develop a program and budget to explore this potential, which would include a dedicated exploration drill and additional step out drilling, with infill drilling requirements based on the results of the wider spaced drilling. Preliminary analysis supports development of ~ 600m of exploration drift with an initial 6,000m of diamond drilling. Based on the results of the wide space drilling, additional drilling will be required to classify resources in the indicated and inferred categories. The total cost of this program is estimated at ~$2M.

There are numerous structural zones that are becoming increasingly understood with detailed underground mapping as depth of development continues. These structural zones not only have an impact on mining, but also exert controls on mineralization. It is recommended that mine geologic staff evaluation the practicality and data requirements to construct a simplified 3-D structural model.

26.2 Mining and Mineral Resources

Mineral Reserves are currently reported at a cut-off grade of 1.9 g/t. As demonstrated in Section 15.5 the cut-off grade would be 1.31 gpt using expected LOM operating costs, recoveries and a gold price of USD $1,250 per ounce. A lower cut-off grade could potentially yield additional Mineral Reserves as demonstrated in Section 15.6. As most of the additional material is already adjacent to existing or planned infrastructure and development, including level drives, additional mine life and cash could be realized.

The mine should develop a long range planning scenario whereby the cut-off grade is reduced below 1.9 g/t. Net after tax cash flow analysis should then should be undertaken to determine if the mine life can be increased and the value of the property enhanced.

27 References

AMEC, 2008, NI 43-101 Technical Report and Preliminary Assessment of the Young-Davidson Property, Matachewan, Canada, April 1, 2009. Technical Report prepared for Northgate Minerals Corporation by P. Rocque, W. Hamilton, A. Simon, R. Welyhorsky, and S. Daniel.

AMEC, 2009, NI 43-101 Technical Report and Preliminary Feasibility Study on the Young-Davidson Property, Matachewan, Ontario, August 27 2009. Technical Report prepared for Northgate Minerals Corporation by G. Taylor, C. Edmunds M.Sc., L. Magumbe J. Melnyk, and S. Daniel

AMEC 2009a, Appendix C1 YD Laboratory Rock Testing Final, January 15, 2009

AMEC 2009b Appendix C2 – YD Joint Mapping and Rock Mass Properties, Dec 4, 2009

CIM, 2004, CIM Definition Standards on Mineral Resources and Mineral Reserves, Prepared by CIM Standing Committee on Reserve Definitions, Adopted by CIM Council, November 14, 2004.

CIM, 2000, CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines, CIM Bulletin Vol. 93, No. 1044, October 2000.

CIM, 1996, CIM Mineral Resource/Reserve Classification: Categories, Definitions and Guidelines, CIM Bulletin CIM Bulletin Vol. 89, No. 1003, September 1996.

Corfu and Andrews, 1986. A U-Pb Age for Mineralized Nipissing Diabase, Gowganda, Canadian Journal of Earth Sciences, v. 23, p. 107-109

Corfu, F., Krogh, T.E., Kwok, Y.Y., Marmont, S., and Jensen, L., 1989, U-Pb Zircon Geochronology in the Southwestern Abitibi Greenstone Belt, Superior Province, Canadian Journal of Earth Sciences 26, p. 1747-1763.

CIM, 2004. CIM Definition Standards on Mineral Resources and Reserves, Prepared by the CIM Standing Committee on Reserve Definitions.

Derry, D.R., Hopper, C.H., and McGowan, H.S., 1948, Matachewan Consolidated Mine in Structural Geology of Canadian Ore Deposits: CIMM Jubilee Volume I, p 638-643.

Diederichs et al. (2000) A Semi-Empirical hazard assessment approach to wedge instability in underground mine openings.

DJB Consultants, Inc. Project No. D000067, January 2008. Assessment of Kemess Grinding Mills in Single-Stage Autogenous Grinding Mode Final Report.

Edmunds, Carl, 2007, Young-Davidson Regional, Local and Property Geology, January, 2007., 5 p.

Edmunds, Carl, 2008. Technical Report on the Underground Mineral Resource Estimates, Young-Davidson Property, Matachewan, Ontario. NI 43-101 Report filed on SEDAR.

Evans, L., 2006, Technical Report on the Lower Boundary Zone, Lucky Zone, and Lower YD Zone Mineral Resource Estimates, Young-Davidson Property, Matachewan, Ontario. Prepared For

Northgate Minerals Corporation Scott-Wilson RPA Evans, L., 2008, January 2008 Young-Davidson Resource Estimate Review. Internal Memo for Northgate Minerals Corporation Scott-Wilson RPA

Edmunds, C, 2012, AuRico Gold, Resource Estimate

Fairbairn, H. W., Hurley, P. M., Card, K. D., and Knight, C. J., 1969, Correlation of Radiometric Ages of Nipissing Diabase and Metasediments with Proterozoic Orogenic Events in Ontario, Canadian Journal of Earth Sciences, v. 6, p. 489-497.

Heaman, L.M., 1988, A Precise U-Pb Zircon Age for a Hearst Dyke, Geological Association of Canada V13:A53.

JORC, 1996, Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves, report of the Joint Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia, Joint Ore Reserves Committee (JORC).

J.R. Goode and Associates, April 29, 2009. Young-Davidson Project – Assessment of Phase III Metallurgical Data.

Konst, Ron, 2008a. Matachewan Gold Project 2006-2007 Drilling Program Sample Preparation and Analytical Quality Assurance Protocols. Internal document prepared for Northgate Minerals Corporation.

Konst, Ron, 2006, Young-Davidson Project 2006 Drilling Program Sample Preparation and Analytical Quality Assurance Report as of September 25.

Konst, Ron, 2008, Young-Davidson Project 2007 Data Audit. Internal Memo Jan 21 2008.

Konst, Ron, 2009, Young-Davidson Project 2008 Data Audit. Internal Memo In prep Jan 2009.

Konst, Ron, 2006, Young-Davidson Project 2007 Drilling Program Sample Preparation and Analytical Quality Assurance Report as of December 20 2007

Lovell, H.L., 1967, Geology of the Matachewan Area: ODM Geological Report 51, 55 p.

MDEng, 2015 Technical Memo #1015-M1509-01: Support Standard Review at Young-Davidson Mine, October 2, 2015.

MERQ–OGS, 1983, Lithostratigraphic map of the Abitibi Subprovince: Ontario Geological Survey / Ministère de l'Énergie et des Ressources, Québec, Map 2484/DV83-16 Scale 1:500 000.

Metso Minerals; Test Plant Report No. 77128-Report No. 3, September 3&4 2008. Jar Mill Testing for Vertimill Sizing and Stirred Mill Detritor (SMD) Testing.

Micon, 2004, Technical Report on the Mineral Resource Estimate for the Matachewan Property Powell, Cairo & Yarrow Townships Larder Lake Mining Division, Ontario, Canada. Project NTS 41 P/15. NI 43-101 Technical Report prepared by Micon International Limited for Northgate Mineral Corporation, August 2004.

Northgate Minerals. Matachewan Consolidated Property – Royalty Agreement, June 2000.

Northgate Minerals. Welsh Property – Royalty Agreement, 1986.

Northgate Minerals Corporation, Project Alex Property, Property Interests and Royalty Agreement - Matachewan Consolidated Property, Royalty Agreement between Pamour Porcupine Mines Limited and Matachewan Consolidated Mines Limited. Dated June 1, 1979 and amended on July 9, 1981, December 13, 1984 and April 27,1988 and extended on June 13, 2000.

Northgate Minerals Corporation, Project Alex Property, Property Interests and Royalty Agreement - Welsh Property, Royalty Agreement between Pamour Porcupine Mines Limited and George S. Welsh. Dated April 26, 1979 and amended on March 11, 1986.

Outotec; December 2008. TH-0456-A High Rate Thickening of a Flotation Concentrate and Tailings.

Outotec; December 2008. TH-0456-B High Rate Thickening of a Flotation Concentrate and Tailings.

Panterra Geoservices Inc., 2003, Structural Study of the Surface Outcrops of the Matachewan Consolidated Mine, Powell Township, Ontario, Prepared for Young-Davidson Mines, Limited, by David A. Rhys, December 17, 2003.

Powell, W.G., Kilbourne, M.W., and Hodgson, C.J., 1991, Gold-Related Geology of the Matachewan Camp in Archean Gold Deposits of the Matachewan-Kirkland Lake-Larder Lake Area: Soc. Econ. Geol. Guidebook series, Volume 2, Lovell, H.L., Robinson, D.R., and Guindon, D.L., eds., p 72-88.

R.J. Longley et al Minerals Engineering 16 (2003) 41 1-419.

Read, J. Stacey, P. (2009) Guidelines for Open Pit Slope Design. Cairo Publishing.

Robert, Francois, 2004. Syenite-associated disseminated gold deposits in the Abitibi greenstone belt, Canada. Mineralium Deposita, Volume 36, Number 6 / September, 2001.

Skeeles, Brad E., J., 1989, Open Pit Potential for the Matachewan Deposit (a Pre-Feasibility Report): Unpublished Internal Company Document, 44 p. Sinclair, W.D., 1980, Gold Deposits of the Matachewan Area, Ontario in Geology of Canadian Gold Deposits: CIMM Spec. Volume 24, Hodder, R.W., and Petruk, W., eds., p 83-93.

SGS Lakefield, April 16, 2007. Grindability Characterisation of Samples from the Young and Davidson Deposit –Project 11497-001.

SGS Lakefield, February 26, 2009. An Investigation into the Recovery of Gold and Silver from the Young and Davidson Deposit- Project 11497-003.

SWRPA, 2007. Technical Report of the Lower Boundary Zone, Lucky Zone, and Lower YD Zone Mineral Resource Estimates, Young-Davidson Property, Mathewan, Ontario. NI 43-101 report prepared by Scott Wilson Roscoe Postle Associates Inc. for Northgate Minerals Corporation, January 12, 2007.

Appendix A – Young Davidson Claims

Alamos Gold Inc.

Young-Davidson Project, Ontario

Client No. 412844 — **Revised: December 31, 2016**

Land Tenure Holdings by Property

Property	Tenures	Area (ha)	Area (acres)
Yarrow Twp. Property (Schaus, Clark & Shirriff)	28	466.23	1,152.04
Kiernicki & Fekete Property	3	178.07	440.01
Welsh Property	2	24.54	60.64
MCM Property	25	374.57	925.57
Shirriff Property	37	597.11	1,475.45
OKA Property (SEDEX JV)	18	354.01	874.77
Young-Davidson Property	48	2,115.86	5,228.29
Walker Property	6	94.25	232.89
Young-Davidson North Property (Opawica)	33	529.03	1,307.23
82 Balsam Ave, Kirkland Lake	1	0.05	0.13
	201	4,733.66	11,696.88

Davidson Mine location - NTS: 041P/15; UTM NAD83 ZONE 17N - 5310000N; 523000E, Long. 80 41', Lat. 47 56'

Claim Maps: Larder Lake Mining Division, Plans G3218, G3209, M0260, M0237

Townships: Powell, Cairo, Yarrow, Teck

Temagami Provincial Forest, District of Timiskaming, Province of Ontario

MNDM - Ministry of Northern Development & Mines

CLR - Crown Land Registry, Ministry of Natural Resources

LR - Land Registrar, Ministry of Gov't & Consumer Services

Yarrow Twp. Property (Schaus, Clark & Shirriff)

Alamos Gold Inc. 100%, subject to Letter Agreement between Royal Oak Mines Inc. and Robert B. Schaus, Donald L. Clark, John F. Shirriff dated September 9, 1992 (property tenures listed below are subject to 2% NSR Royalty held by Dorothea Mae Stamm and Susan Arlette Otterstrom)

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Plan Reference No.
YARROW	L494591		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L494592		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L494593		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L494594		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L494595		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L495895		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L495896		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L495897		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L495898		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L495899		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Mar-2020	
YARROW	L523116		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523117		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523118		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523119		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523141		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523142		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523143		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523144		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	L523145		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	1-Mar-2020	
YARROW	MR 29666	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	start ↓		N	Y		30-Nov-2034	54R-1120
YARROW	MR 29667	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1121
YARROW	MR 29668	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1122
YARROW	MR 29669	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1123
YARROW	MR 29670	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1124
YARROW	MR 29671	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1125
YARROW	MR 29672	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1126
YARROW	MR 29675	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	↓		N	Y		30-Nov-2034	54R-1127
YARROW	MR 29676	61262-0005 (LT)	4938 SEC LTIM	109500	21	Alamos Gold Inc.	158.653		N	Y		30-Nov-2034	54R-1128
	28						466.23						

Kiernicki & Fekete Property

Alamos Gold Inc. 100%, subject to Purchase Agreement between Young-Davidson Mines, Limited and Fred S. Kiernicki and Mark A. Fekete dated May 8, 2003 (property tenures listed below are subject to 2% Net Smelter Returns Royalty)

Twp.	Claim No.		Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type		Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Plan Reference No.
CAIRO	L1199662		Active Claim	Mining claim		Alamos Gold Inc.	80.94	5	N	N	Y	26-Aug-2020	
CAIRO	L1199663		Active Claim	Mining claim		Alamos Gold Inc.	64.752	4	N	N	Y	26-Aug-2020	
CAIRO	L1199664		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	26-Aug-2020	
	3						**178.07**						

Welsh Property

Alamos Gold Inc. 100%, subject to terms of Mining Lease Agreement between Pamour Porcupine Mines Limited and George Welsh dated April 26, 1979 and Amendment dated March 12, 1986 (below listed property tenures are subject to Royalty of C$1.50/ton, divided and payable to Mary Beth Burke and John Horward Welsh in Trust)

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)		Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Plan Reference No.
Powell	L316523	61257-0053 (LT)	5397 SEC LTIM	107382	21	Alamos Gold Inc.	14.524		Y	Y		30-Jun-2022	54R-2240
Powell	L511097	61257-0054 (LT)	5454 SEC LTIM	107440	21	Alamos Gold Inc.	10.016		Y	Y		30-Apr-2024	54R-2240
	2						**24.54**						

Matachewan Consolidated Mines, Limited ("MCM") Property

Alamos Gold Inc. 100%, subject to Purchase Agreement between Matachewan Consolidated Mines, Limited and Northgate Minerals Corporation dated February 2011. Property tenures listed below are subject to a MCM held Royalty of US$1.00/ton.

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Date & Status of Lease Renewal Request
CAIRO	T 18264	61256-0032 (LT)	1172 SEC SST	Crown Patent No. TP 6102		Alamos Gold Inc.	14.41		Y	Y		N/A	T.18264
Powell	MR 5379	61257-0055 (LT)	3193 SEC LTIM	19631	10	Alamos Gold Inc.	16.106		Y	Y		30-Sep-2018	Survey Plan Feb 14, 1918
Powell	MR 5380	61257-0064 (LT)	3194 SEC LTIM	19632	10	Alamos Gold Inc.	19.951		Y	Y		30-Sep-2018	Survey Plan Feb 15, 1918
Powell	MR 5402	61257-0083 (LT)	4901 SEC LTIM	109443	21	Alamos Gold Inc.	19.142		N	Y		30-Sep-2033	Survey Plan Feb 16, 1918
Powell	MR 5396	61257-0084 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	start ↓		N	Y		30-Sep-2035	
Powell	MR 5412	61257-0085 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
Powell	MR 5414	61257-0086 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
Powell	MR 5415	61257-0086 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
CAIRO	MR 5417	61256-0109 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
CAIRO	MR 5454	61256-0109 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
CAIRO	MR 5455	61256-0109 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
CAIRO	MR 5707	61256-0109 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2035	
Powell	MR 5712	61257-0084 (LT)	5125 SEC LTIM	109554	21	Alamos Gold Inc.	139.321		N	Y		30-Sep-2035	
Powell	MR 5397	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	start ↓		*N	Y		31-Dec-2035	
Powell	MR 5398	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	↓		*N	Y		31-Dec-2035	
Powell	MR 5401	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	↓		*N	Y		31-Dec-2035	
Powell	MR 5403	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	↓		*N	Y		31-Dec-2035	
Powell	MR 5406	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	↓		*N	Y		31-Dec-2035	
Powell	MR 5408	61257-0087 (LT)	5126 SEC LTIM	109580	21	Alamos Gold Inc.	100.664		*N	Y		31-Dec-2035	
Powell	MR 5413	61257-0088 (LT)	5127 SEC LTIM	106953	21	Alamos Gold Inc.	19.142		N	Y		31-Mar-2015	Renewal Request to CLR - Dec 15, 2015
CAIRO	MR 9655	61256-0110 (LT)	5128 SEC LTIM	106954	21	Alamos Gold Inc.	10.927		N	Y		30-Apr-2015	to CLR - Dec 15, 2015
Powell	MR 5991	61257-0089 (LT)	5287 SEC LTIM	108685	21	Alamos Gold Inc.	15.726		*N	Y		31-Dec-2016	to CLR - Dec 14, 2016
CAIRO	L537314		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2020	
						subtotal:	**371.577**						
Powell	MR's 5401, 5403, 5397, 5398, 5406, 5408, 5412, 5413, 5991	61257-0103 (LT)	Part of CL 15753 (146.72 ha.)	**SRO # 1 Crown Lease: DT 25289 Lease No. 109581**	4.5	Alamos Gold Inc.	N/A		Y	N		31-Dec-2035	54R-5457
CAIRO	T 18264		Mining Licence of Occupation with Undersurface rights & Surface rights (Montreal River area)	MLO 1007		Alamos Gold Inc.	2.995		Y	Y		N/A	T.18264
	25						**374.572**						

Note: Mining Licence of Occupation - MLO 1007 enables holder to remove all ores & minerals from land covered by water. Licence area included in Freehold Patent (T18264).

Shirriff Property

Alamos Gold Inc. 100%, subject to Royalty Deed among AuRico Gold Inc., Nan Shirriff Muloney, Adele Shirriff and Shirriff Mining LLC dated Janaury 16, 2012 superseding Agreement between Thomas J. Obradovich and John F. Shirriff dated September 25, 2001

Below listed property tenures are subject to 2% Net Smelter Returns Royalty

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Date & Status of Lease Renewal Request
Powell	MR 5386	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	start ↓		Y	Y		31-Mar-2035	
Powell	MR 5400	61257-0035 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5568	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5569	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5570	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5657	61257-0033 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5659	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 5922	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 6032	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	↓		Y	Y		31-Mar-2035	
Powell	MR 9835	61257-0034 (LT)	4982 SEC LTIM	109520	21	Alamos Gold Inc.	173.08		Y	Y		31-Mar-2035	
Powell	MR 33919	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	start ↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 33920	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 33921	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 33922	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 33923	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 33924	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 34250	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 34251	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 34252	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	↓		Y	Y		31-Oct-2035	54R-1237
Powell	MR 34253	61257-0029 (LT)	5060 SEC LTIM	109562	21	Alamos Gold Inc.	139.843		Y	Y		31-Oct-2035	54R-1237
Powell	MR 34242	61257-0043 (LT)	5365 SEC LTIM	107257	21	Alamos Gold Inc.	start ↓		N	Y		30-Jun-2020	54R-5504
Powell	MR 34243	61257-0043 (LT)	5365 SEC LTIM	107257	21	Alamos Gold Inc.	20.598		N	Y		30-Jun-2020	54R-5504
Powell	MR 34242 & MR 34243	61257-0117 (LT)	CL 16176	SRO # 2 108694	9	Alamos Gold Inc.	N/A		Y	N		30-Jun-2020	SRO #2 54R-5504
Powell	MR 35807	61257-0031 (LT)	5390 SEC LTIM	109544	21	Alamos Gold Inc.	start ↓		Y	Y		31-Jul-2035	TER-795
Powell	MR 38931	61257-0030 (LT)	5390 SEC LTIM	109544	21	Alamos Gold Inc.	↓		Y	Y		31-Jul-2035	TER-795
Powell	MR 39023	61257-0028 (LT)	5390 SEC LTIM	109544	21	Alamos Gold Inc.	63.831		Y	Y		31-Jul-2035	TER-795
Powell	MR 35902	61257-0032 (LT)	5392 SEC LTIM	109547	21	Alamos Gold Inc.	start ↓		N	Y		31-Jul-2035	TER-796-98
Powell	MR 39022	61257-0027 (LT)	5392 SEC LTIM	109547	21	Alamos Gold Inc.	29.684		N	Y		31-Jul-2035	TER-796-98
Powell	MR 40066	61257-0091 (LT)	5391 SEC LTIM	109546	21	Alamos Gold Inc.	start ↓		N	Y		31-Jul-2035	TER-800-01
Powell	MR 40067	61257-0091 (LT)	5391 SEC LTIM	109546	21	Alamos Gold Inc.	32.65		N	Y		31-Jul-2035	TER-800-01
Powell	MR 40068	61257-0026 (LT)	5389 SEC LTIM	109545	21	Alamos Gold Inc.	30.52		Y	Y		31-Jul-2035	TER-802-03
Powell	MR 40071	61257-0026 (LT)	5389 SEC LTIM	109545	21	Alamos Gold Inc.	11.04		Y	Y		31-Jul-2035	TER-802-03
Powell	MR 50439	61257-0090 (LT)	5145 SEC LTIM	106876	21	Alamos Gold Inc.	start ↓		N	Y		31-Dec-2015	Renewal Request to CLR - Apr 29, 2016
Powell	MR 50440	61257-0090 (LT)	5145 SEC LTIM	106876	21	Alamos Gold Inc.	47.296		N	Y		31-Dec-2015	TER 805
Powell	L512587	61257-0105 (LT)	CLM 470	108522	21	Alamos Gold Inc.	16.188	1	Y	Y		31-Aug-2031	54R-5460
Powell	L512588	61257-0105 (LT)	CLM 470	108522	21	Alamos Gold Inc.	16.188	1	Y	Y		31-Aug-2031	54R-5460
Powell	L512589		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2020	
	37						**597.11**						

OKA Property (SEDEX JV)

Alamos Gold Inc. - 80%, SEDEX Mining Corp. - 20%,

Property held under Joint Venture Agreement between Northgate Minerals Corporation & Sedex Mining Corp. with below listed mining claims and mining lease 108683 are subject to a 1% - 3% NSR as noted below.

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type		Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Subject to Royalties Noted Below
Powell	L1205862	61257-0116 (LT)	CLM 472	108683	21	Alamos Gold Inc.	↓ CLM472	1	Y	Y		30-Apr-2032	
Powell	L1206077	61257-0116 (LT)	CLM 472	108683	21	Alamos Gold Inc.	↓ CLM472	1	Y	Y		30-Apr-2032	(iii), (vi)
Powell	L1206081	61257-0116 (LT)	CLM 472	108683	21	Alamos Gold Inc.	↓ CLM472	1	Y	Y		30-Apr-2032	(ii), (vi)
Powell	L1206147		Active Claim	Mining claim		Alamos Gold Inc. - 50% Canadian Royalties Inc. - 50%	16.188	1	N	N	Y	4-Apr-2017	(i)
Powell	L1206148		Active Claim	Mining claim		Alamos Gold Inc. - 50% Canadian Royalties Inc. - 50%	16.188	1	N	N	Y	4-Apr-2017	(i)
Powell	L1206150		Active Claim	Mining claim		Alamos Gold Inc. - 50% Canadian Royalties Inc. - 50%	16.188	1	N	N	Y	4-Apr-2017	(i)
Powell	L1213838		Active Claim	Mining claim		Alamos Gold Inc.	48.56	3	N	N	Y	27-May-2020	(vi), (v)
CAIRO	L1223270		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	17-May-2020	(i)
Powell	L1223271		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	10-Apr-2020	(i)
Powell	L1223281		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	17-May-2020	(i)
Powell	L1223283	61257-0116 (LT)	CLM 472	108683	21	Alamos Gold Inc.	78.822	1	Y	Y		30-Apr-2032	(i)
Powell	L1223284		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(i)
Powell	L1223285		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(i)
Powell	L1223286		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(i)
Powell	L1223287		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(i)
Powell	L1223288		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(i)
Powell	L1224878		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	10-Apr-2020	(iv), (vi)
Powell	L3009961		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	20-Sep-2020	(i)
	18						**354.01**	21					

NSR Royalty: as per Schedule E of Option & Joint Venture Agreement dated October 1, 2002

(i) 2.5% NSR - Tom Obradovich, Canadian Royalties Inc., Gino Chitaroni

(ii) 2.0% NSR - Fred Kiernicki

(iii) 2.0% NSR - Don Campbell & Steve Stanwick

(iv) 2.0% NSR - Alcanex Ltd.

(v) 2.0% NSR - Fred Kiernicki & Gino Chitaroni

(vi) 1.0% NSR - Tom Obradovich & Canadian Royalties Inc.

Young-Davidson Property

Alamos Gold Inc. - 100%, subject to 3652378 Canada Inc. a holding 0.5% NSR on 3 mining claims (L1248827-L1248829)

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)	Claim Units	Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Claim Expiry or Renewal Date	Plan Reference No.
CAIRO	L537315		Active Claim	Mining claim		Alamos Gold Inc.	8.094	1	N	N	Y	30-Sep-2020	
CAIRO	L537316		Active Claim	Mining claim		Alamos Gold Inc.	2.023	1	N	N	Y	30-Sep-2020	
CAIRO	L537317		Active Claim	Mining claim		Alamos Gold Inc.	8.094	1	N	N	Y	30-Sep-2020	
Powell	L1207501		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Apr-2020	
Powell	L1207502		Active Claim	Mining claim		Alamos Gold Inc.	259	16	N	N	Y	6-Apr-2020	
Powell	L1207503		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	6-Apr-2020	
YARROW	L1207504		Active Claim	Mining claim		Alamos Gold Inc.	259	16	N	N	Y	6-Apr-2020	
Powell	L1207505		Active Claim	Mining claim		Alamos Gold Inc.	194.256	12	N	N	Y	6-Apr-2020	
YARROW	L1207506		Active Claim	Mining claim		Alamos Gold Inc.	242.82	15	N	N	Y	6-Apr-2020	
YARROW	L1207507		Active Claim	Mining claim		Alamos Gold Inc.	259	16	N	N	Y	6-Apr-2020	
Powell	L1207508	61257-0118 (LT)	CLM 473	108643	21	Alamos Gold Inc.	34.65	3	Y	Y		30-Apr-2032	54R-5461
Powell	L1207509	61257-0105 (LT)	CLM 470	108522	21	Alamos Gold Inc.	29.813	4	Y	Y		31-Aug-2031	54R-5460
Powell	L1207510		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Apr-2020	
Powell	L1207511		Active Claim	Mining claim		Alamos Gold Inc.	48.564	3	N	N	Y	6-Apr-2020	
Powell	L1207512	61257-0105 (LT)	CLM 470	108522	21	Alamos Gold Inc.	14.9515	1	Y	Y		31-Aug-2031	54R-5460
Powell	L1207513		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Apr-2020	
Powell	L1207514		Active Claim	Mining claim		Alamos Gold Inc.	64.752	4	N	N	Y	6-Apr-2020	
YARROW	L1207515		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	6-Apr-2020	
YARROW	L1207516		Active Claim	Mining claim		Alamos Gold Inc.	129.504	8	N	N	Y	6-Apr-2020	
CAIRO	L1207518	61256-0161 (LT)	CLM 473	108643	21	Alamos Gold Inc.	54.06	5	Y	Y		30-Apr-2032	54R-5461
Powell	L1207521	61257-0116 (LT)	CLM 472	108683	21	Alamos Gold Inc.	8.1347	1	Y	Y		30-Apr-2032	54R-5464
Powell	L1207522	61257-0105 (LT)	CLM 470	108522	21	Alamos Gold Inc.	14.9515	1	Y	Y		31-Aug-2031	54R-5460
Powell	L1207550		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	6-Apr-2020	
CAIRO	L1248827		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	7-Jun-2020	
CAIRO	L1248828		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	7-Jun-2020	
CAIRO	L1248829		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	5-Jun-2020	
CAIRO	L3004550		Active Claim	Mining claim		Alamos Gold Inc.	48.564	3	N	N	Y	16-Sep-2020	
CAIRO	L3004551		Active Claim	Mining claim		Alamos Gold Inc.	32.376	2	N	N	Y	18-Sep-2020	
Powell	MR 12506	61257-0046 (LT)	3858 SEC LTIM	19775	10	Alamos Gold Inc.	22.237		Y	Y		30-Apr-2023	Survey Plan Feb 17, 1940
Powell	MR 12507	61257-0047 (LT)	3857 SEC LTIM	19772	10	Alamos Gold Inc.	15.362		Y	Y		30-Apr-2023	"
Powell	MR 12508	61257-0045 (LT)	3856 SEC LTIM	19773	10	Alamos Gold Inc.	12.93		Y	Y		30-Apr-2023	"
Powell	MR 12509	61257-0042 (LT)	3861 SEC LTIM	19768	10	Alamos Gold Inc.	9.741		Y	Y		30-Apr-2023	"
Powell	MR 12510	61257-0040 (LT)	3860 SEC LTIM	19769	10	Alamos Gold Inc.	14.225		Y	Y		30-Apr-2023	"
Powell	MR 12511	61257-0041 (LT)	3859 SEC LTIM	19770	10	Alamos Gold Inc.	13.124		Y	Y		30-Apr-2023	"
Powell	MR 12512	61257-0044 (LT)	3855 SEC LTIM	19771	10	Alamos Gold Inc.	12.351		Y	Y		30-Apr-2023	Survey Plan Feb 17, 1940
Powell	MR 12610	61257-0039 (LT)	3854 SEC LTIM	19774	10	Alamos Gold Inc.	3.849		Y	Y		30-Apr-2023	Survey Plan May 16, 1940
Powell	MR 5371	61257-0114 (LT)	3104 SEC LTIM	108699	10	Alamos Gold Inc.	12.3684		Y	Y		30-Sep-2017	54R-5520
	"	61257-0115 (LT)	3104 SEC LTIM	"		Alamos Gold Inc.	2.5645		N	Y		30-Sep-2017	54R-5520
Powell	MR 5372	61257-0108 (LT)	3105SEC LTIM	108696	10	Alamos Gold Inc.	12.003		Y	Y		30-Sep-2017	54R-5520
	"	61257-0109 (LT)	3105SEC LTIM	"		Alamos Gold Inc.	1.473		N	Y		30-Sep-2017	54R-5520
Powell	MR 5374	61257-0050 (LT)	3106 SEC LTIM	19576	10	Alamos Gold Inc.	10.522		Y	Y		30-Sep-2017	
Powell	MR 5375	61257-0056 (LT)	3107 SEC LTIM	19577	10	Alamos Gold Inc.	15.216		Y	Y		30-Sep-2017	
Powell	MR 5376	61257-0112 (LT)	4215 SEC LTIM	108698	10	Alamos Gold Inc.	15.9371		Y	Y		1-Sep-2016	Renewal Request to MNDM - Jul 28, 2016
	"	61257-0113 (LT)	4215 SEC LTIM	"	10	Alamos Gold Inc.	1.8286		N	Y		"	54R-5520
Powell	MR 5383	61257-0110 (LT)	3108 SEC LTIM	108697	10	Alamos Gold Inc.	8.9086		Y	Y		1-Sep-2017	54R-5520
	"	61257-0111 (LT)	3108 SEC LTIM	"	10	Alamos Gold Inc.	2.4631		N	Y			54R-5520
Powell	MR 5399	61257-0051 (LT)	4314 SEC LTIM	19560	10	Alamos Gold Inc.	21.893		Y	Y		31-Mar-2017	
Powell	MR5375, MR5376, MR5383, L316523	61257-0120 (LT)	SRO only part of CL 16342	108829		Alamos Gold Inc.	7.029		Y	N		1-Jun-2020	SRO#3 54R-5520
	44						**2,115.86**						

Surface Rights Only - Fee Simple Absolute Lots lying within MR5408, MR5398 & MR5991 along HWY 566, Davidson Creek area

Patent		PIN No's	Parcel No.	Part No.									
Powell	CL4871/MR5408	61257-0063 (LT)	23381 SEC SST	6 of CL4871		Alamos Gold Inc.	0.228		Y	N		N/A	54R-3030
Powell	CL4871/MR5398	61257-0059 (LT)	23394 SEC SST	4 of CL4871		Alamos Gold Inc.	0.267		Y	N		N/A	54R-3030

Surface Rights Only - Fee Simple Absolute Lots lying within MR34252 & MR39023 on the east side of Mistinikon Lake

Powell	TP11992/MR34252	61257-0037 (LT)	11647 SEC SST	Location WB44		Alamos Gold Inc.	0.32		Y	N		N/A	54R-1247
Powell	TP11900/MR34252	61257-0038 (LT)	11572 SEC SST	Location HS2274		Alamos Gold Inc.	0.959		Y	N		N/A	54R-1247
	4						**1.774**						

Note: Above SRO fee simple parcel or SRO lease areas are not reported in held area, since their held areas are included in undersurface right area held by the mining leases. The number of SRO tenures are included in the total number of held project tenures.

Walker Property

Alamos Gold Inc.- 100%; Northgate acquired property through Letter Agreement dated May 29, 2008 with Opawica Explorations Inc. and Underlying Property Option Agreement dated January 24, 2007 between Reginald (Rick) Walker and Opawica Explorations Inc. (All below listed property tenures are subject to 2.5% NSR Royalty)

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)		Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Clm Expiry or Renewal Date	Plan Reference No.
Powell	L372902	61257-0096 (LT)	5546 SEC LTIM	108281	21	Alamos Gold Inc.	start ↓		N	Y		30-Sep-2029	54R-2852
Powell	L372903	61257-0096 (LT)	5546 SEC LTIM	108281	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2029	54R-2852
Powell	L373507	61257-0096 (LT)	5546 SEC LTIM	108281	21	Alamos Gold Inc.	↓		N	Y		30-Sep-2029	54R-2852
Powell	L367170	61257-0097 (LT)	5546 SEC LTIM	108281	21	Alamos Gold Inc.	70.516		N	Y		30-Sep-2029	54R-2852
Powell	MR37455	61257-0065 (LT)	4636 SEC LTIM	108116	21	Alamos Gold Inc.	10.607		Y	Y		30-Sep-2028	Survey Plan May 20, 1965
Powell	MR37456	61257-0066 (LT)	4635 SEC LTIM	108117	21	Alamos Gold Inc.	13.128		Y	Y		30-Sep-2028	Survey Plan May 20, 1965
	6						**94.25**						

Young-Davidson North Property (Opawica)

Alamos Gold Inc. - 100%; Pursuant to Offer Purchase Agreeement dated June 11, 2010, Northgate purchased an undivided 100% interest from Opawica Explorations Inc. in the below listed property tenures.

Twp.	Claim No.	PIN No's	Parcel No. or Claim Status (Active)	Mining Lease No. & "R" for ML's being Renewed & Tenure Type	Term (Yrs)	Registered Tenure Holder	Area (ha.)		Surface Rights	Mining Rights	Exploration Rights Only	Lease or Unpatented Clm Expiry or Renewal Date	Plan Reference No.
Powell	L1206306		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	28-Feb-2018	
Powell	L1206307	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	↓ CLM 471	1	Y	Y		31-Mar-2032	54R-5468
Powell	L340615		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L340616		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374013		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2019	
Powell	L374014		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374015	61257-0104 (LT)	CLM 462	108510	21	Alamos Gold Inc.	CLM 462 ↓	1	Y	Y		30-Jun-2031	54R-5434
Powell	L374016	61257-0104 (LT)	CLM 462	108510	21	Alamos Gold Inc.	CLM 462 ↓	1	Y	Y		30-Jun-2031	54R-5434
Powell	L374017	61257-0104 (LT)	CLM 462	108510	21	Alamos Gold Inc.	CLM 462 ↓	1	Y	Y		30-Jun-2031	54R-5434
Powell	L374235	61257-0104 (LT)	CLM 462	108510	21	Alamos Gold Inc.	CLM 462 ↓	1	Y	Y		30-Jun-2031	54R-5434
Powell	L374236	61257-0104 (LT)	CLM 462	108510	21	Alamos Gold Inc.	95.758	1	Y	Y		30-Jun-2031	54R-5434
Powell	L374237		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374238		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374239		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374240		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374241		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374242		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L374243		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L387779		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L387780		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L511486		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L511487		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L511488	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	↓ CLM 471	1	N	N		31-Mar-2032	54R-5468
Powell	L511489		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L511490		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	4-Sep-2018	
Powell	L531566	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	↓ CLM 471	1	N	N		31-Mar-2032	54R-5468
Powell	L531567	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	↓ CLM 471	1	N	N		31-Mar-2032	54R-5468
Powell	L531568		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2018	
Powell	L531613	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	↓ CLM 471	1	N	N		31-Mar-2032	54R-5468
Powell	L531614	61257-0119 (LT)	CLM 471	108632	21	Alamos Gold Inc.	77.133	1	N	N		31-Mar-2032	54R-5468
Powell	L531615		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2018	
Powell	L531815		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2018	
Powell	L531816		Active Claim	Mining claim		Alamos Gold Inc.	16.188	1	N	N	Y	30-Sep-2018	
	33						**529.03**						

Young-Davidson Property - 87 Balsam Ave, Kirkland Lake

Alamos Gold Inc. - 100%

Surface Rights Only - Fee Simple Absolute (see Crown Grant TP875)

Twp.		PIN No's	Plan No.	Part No.									Reference No.
Teck		61408-1003 (LT)	M-120TIM	Lots 2 & 3, 54R-5446 & SRO parts in 54R-5481		Alamos Gold Inc.	**0.0533**		Y	N		N/A	54R-5446 54R-5481

Alamos Gold Inc.

Young-Davidson Project, Larder Lake Mining Division, Ontario

Date: December 31, 2016

Land Tenure Type by Property

	Mining Leases	Number of Mining Claims in Mining Leases	Unpatented Mining Claims	Surface Leases	Patents	Fee Simple	Licence of Occupation	Total Tenures	Area (ha)	Area (acres)
Yarrow Twp. Property (Schaus, Clark & Shirriff)	1	9	19	0	0	0	0	28	466.23	1,152.05
Kiernicki & Fekete Property	0	0	3	0	0	0	0	3	178.07	440.01
Welsh Property	2	2	0	0	0	0	0	2	24.54	60.64
MCM Property	8	21	1	1	1	0	1	25	374.57	925.57
Shirriff Property	11	35	1	1	0	0	0	37	597.11	1,475.46
OKA Property (Sedex JV)	1	4	14	0	0	0	0	18	354.01	874.76
Young-Davidson Property	19	21	22	1	0	4	0	48	2,115.86	5,228.29
Walker Property	4	6	0	0	0	0	0	6	94.25	232.89
Young-Davidson North Property (Opawica)	2	11	22	0	0	0	0	33	529.03	1,307.23
82 Balsam Ave, Kirkland Lake	0		0	0	0	1	0	1	0.05	0.13
Totals:	48	109	82	3	1	5	1	201	4,733.7	11,697.0

Notes:

(i) The number of mining leases (48) are not reported in Total Tenures (201), since mining claims that comprise each Mining Lease & their held area are included in the Total Tenures and Area columns.

(ii) Surface leases (SRO#1, #2, #3) are included in the Tenure Total and their held areas are included in the area held by the mining claims.