Project No. 169514568

Original Date: July 15, 2015
Amended Date: September 15, 2015

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CONCERNING:

Bradshaw Gold Deposit
Timmins, Ontario

NI 43-101 Technical Report and Prefeasibility Study

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NI 43-101 Gowest Gold – Bradshaw Deposit Technical Report and Pre-Feasibility Study Cautionary Statement

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

Certain information and statements contained in this report completed by Stantec for Gowest Gold are "forward looking" in nature. All information and statements in this report, other than statements of historical fact, that address events, results, outcomes, or developments that Gowest Gold and/or the Qualified Persons who authored this report expect to occur are "forward-looking statements". Forward-looking statements are statements that are not historical facts and are generally, but not always, identified by the use of forward-looking terminology such as "plans", "expects", "is expected", "budget", "scheduled", "estimates", "forecasts", "intends", "anticipates", "projects", "potential", "believes" or variations of such words and phrases or statements that certain actions, events or results "may", "could", "would", "should", "might" or "will be taken", "occur" or "be achieved" or the negative connotation of such terms. Forward-looking statements include, but are not limited to, statements with respect to anticipated production rates; grades; projected metallurgical recovery rates; infrastructure, capital, operating and sustaining costs; the projected life of mine; proposed development and potential impact on cash flow; estimates of Mineral Reserves and Resources; the future price of gold; government regulations; the maintenance or renewal of any permits or mineral tenures; estimates of reclamation obligations that may be assumed; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted.

Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this report, the forward-looking statements in this report are subject to the following assumptions: (1) there being no signification disruptions affecting the operation of the mine; (2) the availability of certain consumables and services, and the prices for diesel, propane, cyanide, electricity and other key supplies being approximately consistent with current levels; (3) labour and materials costs increasing on a basis consistent with current expectations; (4) that all environmental approvals, required permits, licenses and authorizations will continue to be held on the same or similar terms and obtained from the relevant governments and other relevant stakeholders within the expected timelines; (5) no significant changes will be made to tax rates and no new taxes, royalties or other fees will be levied by applicable governments; (6) the timelines for exploration activities will proceed in accordance with estimates; (7) assumptions made in Mineral Resource and Reserve estimates, including geological interpretation, grade, recovery rates, gold prices, foreign exchange rates, and operational and capital costs, will hold true; and (8) general business and economic conditions will remain substantially the same.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by forward-looking statements. These risks, uncertainties and other factors include, but are not limited to: a decrease in future gold prices; costs of labour, supplies, fuel and equipment rising; adverse changes in anticipated production, including discrepancies between actual and estimated production, Reserves, Resources and recoveries; exchange rate fluctuations; title risks; regulatory risks, and political or economic developments in Canada; changes to tax rates; risks and uncertainties with respect to obtaining necessary permits, land use rights and other tenure from the Crown and private landowners or delays in obtaining same; risks associated with maintaining and renewing permits and complying with permitting requirements, and other risks involved in the gold exploration, development and mining industry; as well as those risk factors discussed elsewhere in this report, in Gowest Gold's latest Annual Information Form, Management's Discussion and Analysis and its other SEDAR filings from time to time. All forward-looking statements herein are qualified by this cautionary statement.

Accordingly, readers should not place undue reliance on forward-looking statements. Gowest Gold and the Qualified Persons who authored this report undertake no obligation to update publicly or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.
CAUTIONARY NOTE TO U.S. READERS CONCERNING ESTIMATES OF MEASURED, INDICATED AND INFERRED MINERAL RESOURCES

Information concerning the Gowest Gold – Bradshaw Deposit Technical report Mine has been prepared in accordance with Canadian standards under applicable Canadian securities laws, and may not be comparable to similar information for United States companies. The terms "Mineral Resource", "Measured Mineral Resource", "Indicated Mineral Resource" and "Inferred Mineral Resource" used in this report are Canadian mining terms as defined in accordance with National Instrument 43-101 ("NI 43-101") under guidelines set out in the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Mineral Reserves adopted by the CIM Council on May 10, 2015. While the terms "Mineral Resource", "Measured Mineral Resource", "Indicated Mineral Resource" and "Inferred Mineral Resource" are recognized and required by Canadian securities regulations, they are not defined terms under the rules and regulations of the United States Securities and Exchange Commission applicable to mining companies. As such, certain information contained in this report concerning descriptions of mineralization and resources under Canadian standards is not comparable to similar information made public by United States companies subject to the reporting and disclosure requirements of the United States Securities and Exchange Commission. An "Inferred Mineral Resource" has a great amount of uncertainty as to its existence and as to its feasibility. It cannot be assumed that all or any part of an "Inferred Mineral Resource" will ever be upgraded to a higher category. Readers are cautioned not to assume that all or any part of an "Inferred Mineral Resource" exists, or is mineable.

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1.0 SUMMARY

The preparation of this technical report was authored and supervised by Noris Del Bel Belluz (P.Geo) who is also a QP for the report, with specific technical sections authored by the following individuals who are considered Qualified Persons (QPs) under National Instrument (NI) 43-101 definitions: Michel St-Laurent (P. Eng.), Peimeng Ling (P. Eng.), David Brown (P. Geo.) and Neil N. Gow (P. Geo.). This technical report for the Bradshaw Project (Bradshaw Deposit) conforms to NI 43-101 Standards of Disclosure for Mineral Resource Projects.

The purpose of this technical report is to provide a full description of study work completed on the mine design, cost estimate, and economic evaluation of the indicated mineral resources. This study work has been completed at a prefeasibility study level [as defined under Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines] for the Bradshaw Project.

The prefeasibility study is preliminary in nature and does not include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability and have not been included for this study.

Gowest Gold Limited (Gowest) is the sole owner of the Bradshaw Deposit, formerly known as the Frankfield East Deposit. The property is located approximately 32 kilometres (km) northeast of Timmins in the southwest part of Tully Township, Ontario. The Kidd Creek Mine is approximately 15 km west-southwest of the property. Surface access to the property is easily gained from Timmins via Highway 655 and an all-weather gravel road that turns east off Highway 655. This 13.5 km long all-weather road ends at the Texmont Deposit. The site of Gowest's Bradshaw Deposit and drilling program is approximately 1.5 km further east along a drill road.

1.1 Mineral Resources

The mineral resources for the Bradshaw Project have been estimated and classified using the November 27, 2010, CIM standards and definitions for estimating resources, as required by Canadian National Instrument 43-101.

The resource is based on a three-dimensional block modelling approach. Using a statistically determined mineralization envelope cut-off grade of 0.1 g/t Au, the solid representing the deposit was modeled. Waste zones within the 0.1 g/t Au envelope/solid were modeled and discounted from the resource. Grade interpolation was conducted using the nearest neighbor technique. A summary of the resource is given in Table 1.1 at cut-off grades of 3.0 g/t Au for underground resources.

Table 1.1: Mineral Resources Summary at January 12, 2015

Category	Depth	Zone	3 g/t Au Cut-Off Tonnes	Au Grade	Ounces
Indicated	500 m	MZ1	412,503	6.14	81,429
	500 m	MZ2	634,583	5.88	119,963
	400 m	HWZ1	345,637	6.35	70,563
	400 m	HWZ2	299,258	5.33	51,281
	400 m	HWZ3	194,029	6.93	43,230
	400 m	HWZ4	127,096	6.16	25,171
	400 m	HWZ5	53,094	8.01	13,673
	400 m	HWZ6	55,666	9.36	16,751
		Total	**2,121,866**	**6.19**	**422,059**
Inferred	below 500 m	MZ1	331,752	8.64	92,153
	below 500 m	MZ2	1,078,096	4.36	151,121
	below 400 m	HWZ1	693,934	5.16	115,119
	below 400 m	HWZ2	566,913	6.45	117,559
	below 400 m	HWZ3	443,788	11.85	169,073
	below 400 m	HWZ4	514,614	6.62	109,527
		Total	**3,629,097**	**6.47**	**754,553**

Notes

1. *CIM (Canadian Institute of Mining, Metallurgy and Petroleum) definitions were followed for Mineral Resources.*
2. *Mineral Resources are estimated at a cut-off grade of 3 g/t Au.*
3. *Mineral Resources are estimated at a long-term gold price of US$1,200/oz., and a US$/C$ exchange rate of $0.80.*
4. *A minimum downhole length of 2 m was used.*
5. *Bulk density of 2.89 g/cm^3 was used.*
6. *The Mineral Resource estimate is based on drilling up to December 2014.*

Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources follows the CIM Definitions that were approved and issued on May 10, 2014. There has been insufficient exploration to define the inferred resources as an indicated or measured mineral resource. It is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

1.2 Mineral Reserves

A mine design was created for the mineralized areas that had a cut-off grade of 3.0 g/t Au over a minimum 2 metre down-hole width, down to a depth of 495 metres below surface. Cut-off grades for 2.5 and 3.5 g/tonne were also investigated and after evaluating costs, revenues and optimization of the resource, a cut-off grade of 3.0 g/t Au was selected. This cut-off grade was applied to all of the mineralized horizons present within the resource and targeted only measured and indicated resource. Inferred resources were not included. The mine design was completed using Studio 5 and the minimum stope width allowed in the stope shapes was 2 meters.

A dilution grade was estimated from the resource model and an average grade of 0.7 g/t was applied up to 1 meter outside of the mineralized boundaries. A recovery rate of 95% was applied. The dilution rate applied was 15% and is derived outside of the stope boundaries (called unplanned dilution). Dilution within the stope envelope was included with the overall reserve within the stoping boundary (called planned dilution), with all stopes having a minimum grade of 3 g/t including the planned dilution. The Mineable Reserves derived from the mineral resources are represented in Table 1.2.

Table 1.2: Mineable Reserves Breakdown

	STOPES				DEVELOPMENT Ore			
LEVEL	Insitu Tonnes	Recoverable Tonnes	Grade	Au ounces	Total Tonnes	Grade	Au ounces	Total Oz
45 (CROWN)	69,417	63,840	6.25	12,838	15,027	5.54	2,678	15,515
75 Total	143,919	138,327	5.24	23,291	23,732	3.74	2,852	26,143
105 Total	118,544	112,247	4.83	17,432	11,255	3.72	1,348	18,780
135 Total	66,471	62,575	4.29	8,632	8,628	4.36	1,209	9,841
165 Total	70,967	66,760	4.33	9,295	12,866	3.95	1,634	10,929
195 Total	102,325	95,679	4.21	12,954	13,790	4.66	2,068	15,023
225 Total	171,208	161,928	5.33	27,734	30,398	4.17	4,077	31,810
255 Total	133,302	128,134	5.33	21,954	21,202	4.33	2,950	24,904
285 Total	83,999	81,846	5.56	14,623	7,950	5.42	1,385	16,008
315 Total	50,706	48,293	4.37	6,779	8,480	4.09	1,114	7,893
345 Total	87,719	82,208	4.64	12,273	20,655	3.32	2,205	14,477
375 Total	149,432	142,000	5.12	23,366	30,389	4.09	3,993	27,358
405 Total	128,214	124,553	5.31	21,261	2,878	7.26	671	21,933
435 Total	35,733	32,729	3.61	3,803	5,915	3.07	584	4,387
465 Total	92,508	91,836	3.88	11,455	10,807	4.07	1,414	12,869
495 Total	118,689	118,457	4.63	17,624	11,880	4.19	1,601	19,225
Grand Total	1,623,153	1,551,412	4.92	245,314	235,855	4.19	31,782	277,096

By adding the Recoverable Stope Tonnes (1,551,412 tonnes @ 4.92 g/tonne) and the Total Development ore (235,855 tonnes @ 4.19 g/tonne), the recoverable Mine Reserves are; **1,787,295 tonnes at 4.82 g/tonne Au in the probable category.**

1.3 Mining Method and Design

The mineable portion of the Bradshaw deposit extends from surface to a depth of 500 metres. A 45 metre crown pillar will be established and the ore reserves from 45 level (45 metres below surface) to 495 level (495 metres below surface) will be mined by underground methods.

The primary access to the underground mine will be via a single portal and main ramp from surface to the working levels. All active production levels, spaced in 30 metre intervals, will be accessed via the ramp (i.e. no captive levels). Personnel, materials, ore and waste rock will be transferred via the ramp.

The ore zones are dipping (60 to 85 degrees) with a nominal thickness of 2-3 metres, with a maximum 5 metres width. The underground deposit is comprised of two main zones and six hanging wall zones over a strike length of one kilometre. The ore zones are not continuous along the strike length but the presence of mineralization is continuous between the ore zones and have a low gold content and will be separated on surface as mixed development for future ore sorting. For the purpose of this study and given these parameters, longitudinal longhole stoping, with both unconsolidated and consolidated rockfill, has been selected as the primary mining method.

Sublevels have been designed at 30 metre vertical intervals to mine the ore deposit. On each sublevel, the ore will be accessed from the centre and developed east and west along strike in the mineralized horizons to a minimum mining width of 4 metres. Longitudinal mining will retreat from the outer limits back to the centre access point. The minimum stope width will be 2 metres, and a typical stope length will be 11-13 metres, depending if mining occurs in the main and hanging wall zones. Mining will progress from the bottom of each stope upwards. All stopes will be backfilled with a mix of cemented and unconsolidated waste rock derived from the reject pile of the sorter located on surface.

Mining will be conducted by contractors. There will be two development crews required throughout the duration of the project and operating period. The first development crew will complete ramp access to the bottom of the first stope (75 level) and continue the ramp downward to access the remaining stopes. The second crew will complete level development to support production. A total of 8,080 metres of lateral and vertical capital development (project and sustaining period) will be required, plus over 19,472 metres of operating (waste and silling) lateral development for the life of mine.

The bulk of the lateral development will follow one of eight mineralized horizons which will contain mineralized gold bearing material. This material that has to be hauled up the ramp to surface will be processed through the ore sorter to recover the gold bearing mineralization only if it is economically feasible by recouping the cost of crushing, sorting, transport to the mill and grinding/flotation. This material is shown as “mixed” or incremental material and is not included as part of the ore reserve, but gold recovered from this material is included with revenue calculations since it reduces the cost of the lateral development.

The Life of Mine Production Profile is summarized in Table 1.3.

Table 1.3: Bradshaw Deposit Life of Mine Plan

	Production Profile LOM									Total
	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	
Dev Tonnes (t)	1,1334	31,630	37,798	53,264	50,332	51,497				235,855
Dev Grade (g/t Au)	5.52	3.63	4.30	4.36	3.82	4.35				4.19
Stope Tonnes (t)	39,562	49,695	144,019	188,695	250,448	197,468	302,046	302,550	76,957	1,551,440
Stope Grade (g/t Au)	5.78	5.49	4.56	5.31	4.64	4.48	4.95	4.88	5.89	4.92
Stope and Dev Prod (t)	50,896	81,325	181,817	241,959	300,780	248,965	302,046	302,550	76,957	1,787,295
Stope and Dev Grade (g/t Au)	5.72	4.77	4.50	5.10	4.50	4.45	4.95	4.88	5.89	4.82
Incremental Dev (t)	27,444	112,871	123,841	133,920	148,877	119,700	0	0	0	666,253
Incremental Dev Grade (g/t Au)	1.31	1.31	1.31	1.31	1.31	1.31	1.31	0	0	1.31
Gold from Dev and Stope Production (Au Oz)	9,366	12,467	26,327	39,666	43,520	35,648	48,036	47,492	14,579	277,101
Gold from Incremental Dev (Au Oz)	1,152	4,736	5,197	5,619	6,230	5,023	0	0	0	27,957
Total Au Oz	10,518	17,203	31,523	45,286	49,751	40,671	48,036	47,492	14,579	305,058
Avrg Stope and Dev Production (tpd))	139	223	498	663	824	682	828	829	855	675

1.4 Milling and Processing

Metallurgical studies were completed on the Bradshaw Deposit and it was found that the great majority of the gold is associated with arsenopyrite with a small remainder associated with pyrite. The gold can be recovered through grinding and flotation to achieve a 96% overall recovery of the gold. The resultant concentrate can then be treated at facilities that can extract the gold from a refractory type of concentrate.

Gowest has approached milling facilities in the Timmins area and have signed a memorandum of understanding for the treatment of their ore on a toll milling basis that will be transported from the Bradshaw Mine site to the respective mills. The Bradshaw ore will be treated through a toll milling arrangement to avoid the capital cost of building a milling and flotation facility. The resultant concentrate will be shipped to a facility that has the ability to treat the refractory concentrate and extract, and refine the gold further. Studies have indicated treating the concentrate and refining has a recovery of 98% at the refinery.

Unique to the Bradshaw Deposit is the planned use of an ore sorter where the ore hauled out of the mine at the portal will be crushed down to 2 inch size and then sorted using technology that selectively retains the gold bearing sulphide material using x-ray diffraction and rejects about 47% of the barren waste material. After testing, using Bradshaw mineralized material, a high recovery of 99% was obtained. In this way, the mill-feed grade will be upgraded by a factor of about 2, reduce the volume of the mill-feed to lower transportation costs to the toll milling facility, and negate or greatly reduce crushing costs at the milling facility. The sorter also allows the retrieval of mineralized material from rock that normally would be disposed as waste. Material that has grade below cut-off, and has to be hauled up the ramp to surface can be salvaged for gold bearing material economically as long as the crushing, sorting, ore transportation to the mill and milling costs can be re-couped. This “Mixed” or incremental development material is not part of the ore reserve since it cannot stand as economically viable on its own, but has been included in revenue calculations since it has value and serves to partially recoup development costs, since the bulk of the mine development will be placed in uneconomic mineralized areas. Table 1.4 tabulates the amount of this material that will be produced.

Table 1.4: “Mixed” Incremental Development Material

SOURCE	"Mixed" or Incremental Material		
LEVEL	Total Tonnes	Au g/tonne	Au Oz
45 (CROWN) Total	48,898	1.33	2,094
75 Total	61,065	1.35	2,646
105 Total	56,487	1.52	2,761
135 Total	45,388	1.25	1,819
165 Total	49,764	1.20	1,922
195 Total	54,459	1.55	2,709
225 Total	60,681	1.24	2,422
255 Total	39,741	1.12	1,435
285 Total	38,532	0.99	1,229
315 Total	39,561	1.22	1,550
345 Total	38,993	1.65	2,074
375 Total	24,544	1.51	1,195
405 Total	47,348	1.09	1,665
435 Total	18,352	1.20	710
465 Total	23,246	1.40	1,045
495 Total	19,194	1.12	691
Grand Total	**666,253**	**1.31**	**27,965**

1.5 Project Infrastructure and Sustaining Capital

The Bradshaw Deposit has only had diamond drilling performed at site and is considered a green field project. Mine site infrastructure will be kept to a minimum and there will be relatively little infrastructure required to put this site into operation.

Project capital will include surface and underground construction, as well as 18% contingency for the pre-production period and Year 1. Powerline installation costs used a higher contingency of 30% in the pre-production year.

The project capital (C$) required to support the operation for the first year with the bulk sample (year 0) and then the following 8 years consists of:

- Surface road upgrades
- Power line grid Installation
- Surface infrastructure installations
- Ventilation infrastructure installations
- Development costs for the bulk sample ($10.74 million)
- Contingency @18% (power line installation has a 30% contingency included)
- Closure costs in the final year (year 8)

Total Project Capital, is $27.25 million (C$) for the above items. Excluding the bulk sample development costs, the total capital is $16.52 million. The bulk sample will have revenue produced from an expected 8,817 ounces, which will reduce the cost of the bulk sample to a net cost of $12.5 million after deducting the development costs, crushing and sorting, transport to the mill, milling, smelting and refining. This cost is carried throughout the mine life on a pro rata tonnage basis over the 8 year mine life. Revenue is generated assuming a 93% process recovery rate from bench testing, a gold price $1,200.00 USD/ounce and an exchange rate of 0.80 USD = 1.00 C$.

The Total Initial Project Capital in the first two years is $26.90 million (C$) which excludes the $0.35 million for closure costs.

Sustaining capital includes all underground ramp, level access development, and infrastructure required to support the operation. Underground infrastructure will be kept to a minimum and will include a dewatering system of 4,500 m^3/day (840 USgpm) capacity, refuge stations, electrical substations, materials storage areas, and other ancillary installations. A total of $37.2 million of sustaining capital has been estimated for this project, which includes ramp development, level access, ventilation development and raise development.

1.6 Environmental, Permitting and Social Impact Summary

Gowest initiated environmental baseline studies and the permitting process on the Bradshaw Project early during the exploration stage. The environmental studies were carried out from 2009 to 2014 and consisted of the following: waste and mineralized rock geochemistry, hydrogeology, meteorology and hydrology, water quality, terrestrial ecology, aquatic ecology, and archaeology (Stage 1 and 2). All of the studies were co-ordinated by Golder Associates Ltd. on behalf of Gowest. They have been compiled and form the basis of a Closure Plan for an Advanced Exploration underground bulk sample program that has been submitted by Gowest to Ontario Ministry of Northern Development and Mines. Several other permit applications in parallel with the Closure Plan have been submitted by Golder Associates Ltd. on behalf of Gowest to various government regulatory agencies. The key permit applications include an Environmental Compliance Approval (ECA) and Permits to Take Water (PTTW) being submitted to the Ontario Ministry of the Environment; as well as a Work Permit that requires Lakes and Rivers Improvement Act Approval and a Forest Resource License being submitted to the Ontario Ministry of Natural Resources.

Gowest has been actively engaging and consulting with the local communities in the vicinity of the Bradshaw Project since 2010. The consultation process initiated by Gowest has involved the general public, five First Nations, their associated Tribal councils, and the Métis community. Through meetings, site tours and regular communications, Gowest has strived to ensure engagement with all members of the local communities. Gowest has also carried out stakeholder consultation activities including communications and meetings with the City of Timmins, ongoing consultation with relevant Ontario government regulatory agencies, and a public Open House for the advanced exploration program held in Timmins on December 1, 2014.

1.7 Capital and Operating Costs

A construction and development schedule, production profile, and underground mine design were prepared as a basis of estimate for the capital and operating costs. Key outcomes of the study show that the mineral resources will support a nominal nine year mining plan, including the pre-production period (bulk sample), at a nominal production rate of 675 tonnes per day (365 days per year).

Surface building facilities and construction is scheduled to begin during the pre-production period and will be completed at the end of Year 1. Steady state production will be achieved in Year 3. The capital cost during the project period will be $16.52 million (including 18% contingency). Sustaining capital costs throughout the mine life will amount to $37.23 million (excluding contingency and closure costs).

The underground ore reserves will be extracted at a total operating cost of $248.9 million (excluding capital costs), or an average operating cost of $139.26 per tonne mined, sorted and milled. The Life of Mine (LOM) costs including smelting, refining and royalties will be $295.0 million (not including financing, taxes, capital or depreciation). A total of 56 people at the mine site will be required during the operating period, including direct and indirect hourly, technical, and administration personnel.

1.9 Financial Analysis

Table 1.5 presents a summary of the project economics and associated parameters.

Table 1.5: Bradshaw Project Economics (C$)

Item	Value
Forecast Gold Price (C$)	$1,500
Mine Tonnes	1,787,295
Exchange Rate	C$1.00 = US$0.80
Mined Grade	4.82 g/t Au
Mill, refining and Ore Sorting Recoveries	93%
Mined Available Ounces (including "mixed" incremental ounces)	305,058
Mill Recovered Ounces	284,129
Total Revenue to Operations (C$)	$376,586,486
Operating Costs (C$)	$291,523,680
Total Capital Costs (C$)	$53,740,688
Net Cash Flow (C$)	$73,941,414
Net NPV (5%) – Before Taxes (C$)	$49,750,509
IRR (%) – Before Taxes	32.0%
Net NPV (5%) – After Tax (C$)	$36,495,879
IRR(%) – After Tax	27.3%

The life of mine cash flow summary is illustrated in Figure 1.1.

Figure 1.1: Bradshaw Project – Cumulative Cash Flow Graph (C$ Pre-tax)

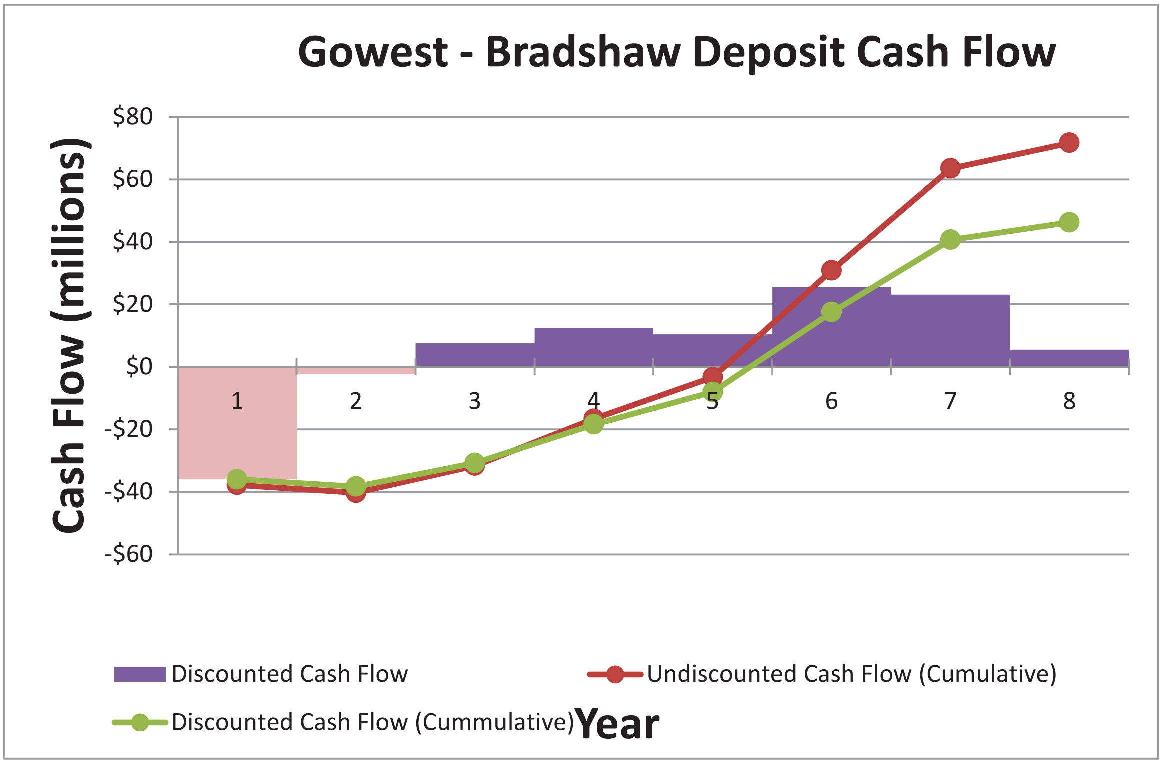
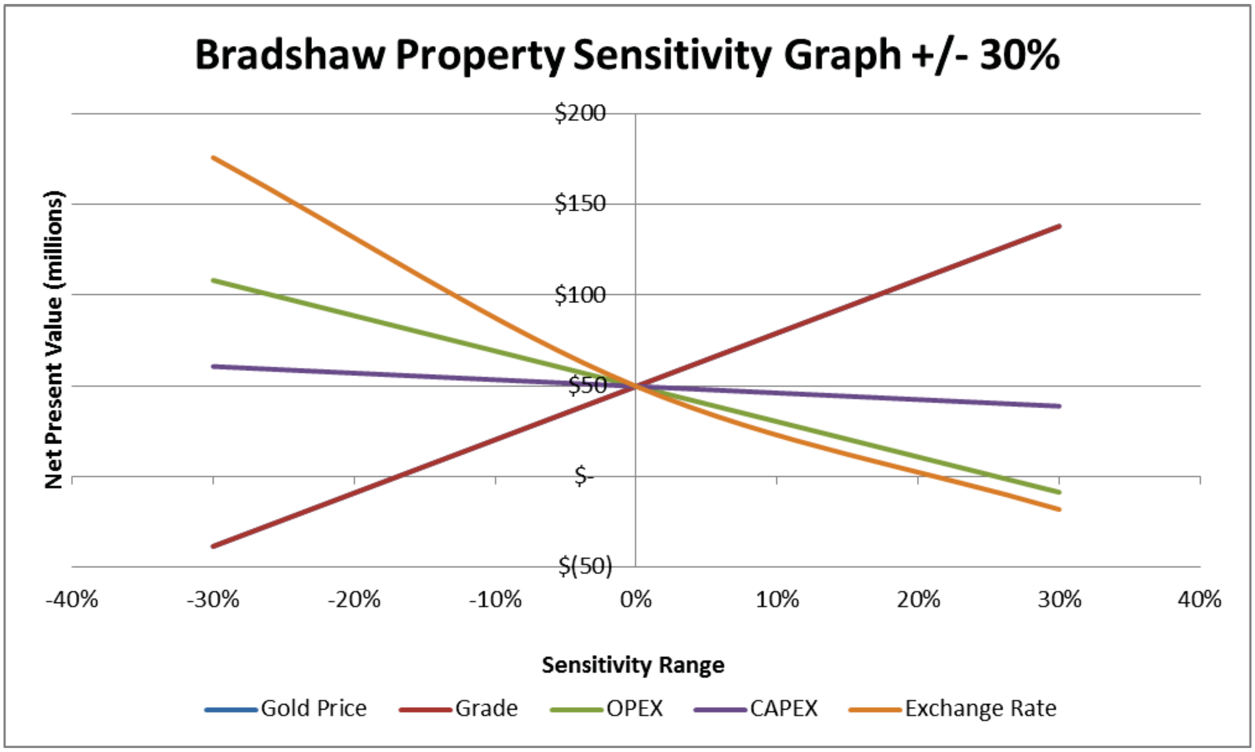


Figure 1.2 illustrates the sensitivity and influence for the factors of the capital and operating cost, forecast gold price, and average mine grade parameters on the net present value (NPV) of the project.

Figure 1.2: Change in Project NPV (5%) versus 10% Change in Variables (C$ Pre-tax)



**Note that the curve for gold price and grade overlap each other and appear as one line.*

1.10 Recommendations

Stantec recommends proceeding to the next phase of work. This will involve a bulk sample program that will verify the operating and recovery parameters used in this study, followed by a feasibility study.

Prior to proceeding with the feasibility study, conversion of additional resources from the inferred category to a higher measured/indicated classification will improve the viability of the project [in accordance with NI 43-101 standards, inferred resources are not eligible for consideration to be converted to recoverable reserves at the prefeasibility stage and should not be considered as economically viable].

The mineral resources reported in this study have been estimated based on the information provided from the sampling of diamond drilling core. There is some risk related to the grade continuity of the mineralization within the accuracy of the current interpretation. This risk would be reduced by an underground bulk sampling program. During this program, a portion of the ore reserves between the 45 and 75 levels would be mined and processed. The mined grade may then be compared to the predicted ore reserve estimates. After completion of the bulk sampling program that is planned following this phase of study, and after evaluating the results, the next phase of study can then proceed assuming favourable results that confirms the data used for the Pre-Feasibility Study and assuming the project financial analysis is still positive at that stage.

The capital cost for the underground bulk sampling program is estimated to be $25.7 million and after deducting revenue from the gold recovered, the actual cost will be $12.5 million. This cost has been carried over the remaining 8 years of the mine life and pro-rated on a tonnage basis for the ore produced for that particular year. The bulk sampling program will include surface and underground infrastructure including development to access and mine nominally 50,900 tonnes of the ore reserve (a total of 15 stopes in MZ2, HWZ1, HWZ2, HWZ3 and HWZ4 including sill development). The bulk sample is envisioned to be extracted in the upper part of the mine, on the east side of the resource between 45 and 75 Levels.

The proposed feasibility study and bulk sample program should include:

- Collection of geotechnical data to produce more definitive design parameters for specific areas of the mine. Currently, it is expected that the areas near the ultra-mafic Footwall contact (MZ1 and MZ2) will have the poorest rock mechanics conditions, which are assumed for all areas. There is potential for better conditions in the Hangingwall zones (HWZ1-4)
- Confirmation of dilution and mining recovery data.
- Confirmation of ore distribution and grades.
- Confirmation of metallurgical recoveries and ore sorting test results.
- Confirmation of smelting and recovery test results.
- Confirmation of ground water assumptions.
- Updated block model with new drilling information.

Additional recommendations which may improve the economics of the project include:

- Connection of the Bradshaw Deposit to the local electrical grid. Gowest is currently in talks with Hydro One to determine the feasibility and final costs. Revisions to the economics will be required when associated capital costs are determined.
- The parameters used to determine the block model cut-off grade should be updated for the next level of study. Optimizing the cut-off grade used to define the resource may improve the rate of return for the project.
- Milling facilities exist within trucking distance of the Bradshaw project. Trucking and, milling costs will require revisions once a facility is chosen and a milling contract has been finalized.

- The Bradshaw project does not include the Sheridan Zone (formerly Texmont) Deposit. Future studies related to the Bradshaw project may assess the economic benefits of including the two deposits.
- Capex has been minimized through the use of contractor equipment and rentals. Trade off studies should be undertaken to review capital requirements to purchase versus rental/leased costs for mining equipment and surface buildings.
- The bulk sample will allow a more definitive larger sample for the ore sorter and metallurgical testing regarding confirmation of testing the performance of the ore sorter, metallurgical recoveries, and operating costs.

2.0 INTRODUCTION

This report has been prepared for Gowest, a publically traded company listed on the TSX venture exchange and trading under the symbol GWA, with their head office located in Toronto, Ontario. Gowest is a Canadian mining exploration company focused on precious metal properties.

This technical report, originally dated July 15, 2015, was amended on September 15, 2015 to comply with the independence requirements of NI 43-101 and to limit the volume of appendices. The mineral resource and mineral reserve estimates remain unchanged as do the financial results of the prefeasibility study.

This Gowest Bradshaw Project Technical Report is authored by Noris Del Bel Belluz(P.Geo.), Michel St-Laurent (P. Eng.), Peimeng Ling (P. Eng.), David Brown (P. Geo) and Neil N. Gow (P.Geo.).

The authors have prepared this report using a combination of publicly available and confidential information. Key documents that have been relied upon in the preparation of this report are listed in Section 27.0 References.

A site visit to the Bradshaw Deposit was undertaken in the week of 09 – 15 November 2014 by Noris Del Bel Belluz and Michel St-Laurent. During the visit to the Bradshaw project, exploration diamond drill core of the mineralization was examined along with the overall condition of the site. A review of the site was held by Gowest personnel in their Timmins office, as well as a possible location of the underground ramp portal. Neil N. Gow visited the property March 11, 2011 and August 28, 2015.

2.1 List of Qualified Persons

This technical report has been prepared by the staff of Stantec and Gowest under the supervision of five Qualified Persons as defined by the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101").

Each Qualified Person retains the responsibility for his contribution as noted below:

- Noris Del Bel Belluz, P.Geo., Senior Consultant for Stantec Consulting Ltd. is responsible for overall project management, assimilation and editing of the report, and contributing to sections 1, 2, 15, 22, 25 and 26.
- Michel St-Laurent, P. Eng., Mining Consultant for Stantec Consulting Ltd. is responsible for Sections 1, 2, 3, 15, 16, 18, 21, 22, 25, 26, and 27.
- Peimeng Ling, P. Eng., Peimeng Ling and Associates Ltd is responsible for Section 13, 17, and 19 (mineral process related).

- Neil Gow, P. Geo., Associate Consulting Geologist for RPA is responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 19, 20, 23 and 24.
- David Brown, P. Geo., Principle for Golder Associates is responsible for Section 20.

2.2 Units and Currency

Metric and Imperial units are used throughout this report. Canadian dollars ("C$", "$") is the currency used unless otherwise noted. On 26 January 2015, Gowest advised to use a gold price of $1,200.00 US/ounce and the exchange rate of C$1.00 to US$0.80.

Common conversions used include converting one ounce of gold to grams gold with a factor of 31.104 grams/troy ounce.

2.3 List of Abbreviations

Table 2.1: List of Abbreviations

Unit or Term	Abbreviation or Symbol
Above mean sea level	amsl
Advanced Exploration Project	AEP
Atomic absorption	AA
Arsenic	As
Arsenopyrite	aspy
Azimuth	AZ
Billion years ago	Ga
British thermal unit	Btu
Carbon in leach	CIL
Carbon in pulp	CIP
Centimetre	cm
Copper	Cu
Cubic centimetre	cm^3
Cubic feet per second	ft^3/s, cfs
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Degree	°
Degree Celsius	°C
Degrees Fahrenheit	°F
Diamond bore hole	dbh, DBH
Diamond drill hole	ddh, DDH

Unit or Term	Abbreviation or Symbol
Dollars Canadian	C$
Dollars American	US$
Dry metric tonne	dmt
Dry metric tonne per day	dmtpd
Foot	ft
Gallon	gal
Gallon per minute	gpm
Gold	Au
Gold equivalent grade	AuEq
Gram	g
Gram metres	m.g/t
Grams per litre	g/l
Grams per tonne	g/t, gpt
Greater than	>
Hectare (10,000m^2)	ha
Hour	h (not hr)
Inch	in, "
Kilo (1,000)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilowatt hour	kWh
Less than	<
Lead	Pb
Life of mine	LOM
Litre	L
Litres per minute	L/m
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne) (2,000 kg) (2,204.6 pounds)	t
Micrometre (micron)	µm
Miles per hour	mph
Milligram	mg
Milligrams per litre	mg/L
Milliliter	mL
Millimetre	mm
Million	M
Million grams	M g
Million tonnes	Mt
Million Troy ounces	M oz
Million years	Ma

Unit or Term	Abbreviation or Symbol
Minute (plane angle)	min, ‘
Minute (time)	min
Month	mo
National Instrument 43-101 (Canadian)	NI 43-101
No Personal Liability	N.P.L.
Ounces	oz
Page	p, pg
Parts per billion	ppb
Parts per million	ppm
Percent	%
Percent moisture (relative humidity)	% RH
Potassium	K
Pound(s)	lb
Pounds per square inch	psi
Preliminary Economic Assessment	PEA
Pyrite	py
Pyrrhotite	po
Quality Assurance/Quality Control	QA/QC
Quart	qt
Revolutions per minute	rpm
Rock Quality Description	RQD
Run of Mine	RoM
Second (plane angle)	sec, “
Second (time)	s
Short ton (2,000 lb)	st
Short ton (US)	t (US)
Short tons per day (US)	tpd (US)
Short tons per hour (US)	tph (US)
Short tons per year (US)	tpy (US)
Silver	Ag
Sodium	Na
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d, tpd
Tonnes per hour	t/h
Tonnes per year	t/a
Volt	V
Week	wk
Weight/weight	w/w
Wet metric tonne	wmt

Unit or Term	Abbreviation or Symbol
Yard	yd
Year (annum)	a
Year (US)	yr

2.4 Definitions

The following definitions of Mineral Resources and Mineral Reserves have been prepared by the CIM Standing Committee on Reserve Definitions and Adopted by the CIM Council on 10 May 2014.

2.4.1 Mineral Resource

Mineral resources are subdivided, in order of increasing geological confidence, into Inferred, Indicated, and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but a lower level of confidence than a Measured Mineral Resource. A Measured Mineral Resource provides the highest level of confidence.

A “Mineral Resource” is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust referring to such items as diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, geological characteristics, and continuity of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling.

2.4.2 Inferred Mineral Resource

An “Inferred Mineral Resource” is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and limited sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be convereted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Inferred resources are not used in this study to assess economics or the conversion to a mine reserve and should not be considered as economically viable.

2.4.3 Indicated Mineral Resource

An "Indicated Mineral Resource" is that part of the mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence sufficient to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An indicated Mineral Resource has a lower level of confidence that that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

2.4.4 Measured Mineral Resource

A "Measured Mineral Resource" is that part of the mineral resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail, to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

A measured Mineral Resource has a higher level of confidence that that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

2.4.5 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could be reasonably be justifies..

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing,metallurgical, infrastructure, economic, marketing, legal, environemnatl, social and governmental factors.

2.4.6 Probable Mineral Reserves

A "Probable Mineral Reserve" is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource.The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower that that applying to a Proven Mineral Reserve.

2.4.7 Proven Mineral Reserves

A "Proven Mineral Reserve" is the economically mineable part of a Measured Mineral Resource.A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

2.4.8 Preliminary or Pre-Feasibility Study

A Pre-Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral resource may be converted to a Mineral Reserve at the time of reporting. A Pre-Feasibility Study is at a lower confidence level than a Feasibility study.

2.4.9 Feasibility Study

A feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any releveant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher that that of a Pre-Feasibbility Study.

2.5 Glossary

Table 2.2 summarizes common technical words accompanied by a simple explanation of the term or word as the term pertains to this report.

Table 2.2: Glossary

Term	Explanation
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-Off Grade (CoG)	The grade of mineralized rock, which determines whether or not it is economic to recover its gold content by further concentration (also referred to as “break even” grade).
Dilution	Unwanted waste, which is mined with ore.
Dip	Angle of inclination of a geological feature / rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of “gold” within mineralized rock.
Hanging wall	The overlying side of an orebody or stope.
Haulage	A horizontal underground excavation which is used to transport mined material.
Igneous	Primary crystalline rock formed by the solidification of magma.
Level	Horizontal tunnel with the primary purpose to transport personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life of mine plans.
Material Properties	Mining properties.

Term	Explanation
Metamorphism	Process by which consolidated rock is altered in composition, texture, or internal structure by conditions and forces of heat and pressure.
Milling	A general term used to describe the process in which the ore is crushed, ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Asset	Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserves.
Project Period	Also called the Capital Period. Project Period is completed when 60% of the steady state production rate is achieved for three consecutive months.
RoM	Run of Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulphur bearing mineral.
Sustaining Period	Or Operating Period. Capital Costs during the Operating Period are classified as Sustaining Capital Costs.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operation and capital nature.

2.5.1 General Glossary

Not applicable.

2.5.2 Gowest Mine Site Terminology

Not applicable

3.0 RELIANCE ON OTHER EXPERTS

This report utilizes information and data contained in the NI 43-101 compliant technical report on the Frankfield Gold project completed for Gowest by ACA Howe International Ltd. (filed August 3, 2011) and a NI 43-101 Technical Report and PEA by Peimeng Ling and Associates Ltd (filed February 22, 2012). Specifically, report sections related to the property history, land tenure, exploration activities and resource estimates have been included in their entirety as they were presented in the Howe and Peimeng report. Additional exploration and infill drilling has been completed on the Bradshaw (Frankfield East) Deposit subsequent to the completion of the Howe technical report but it is the opinion of PL&A and Gowest that these activities have not resulted in significant changes to the data/analysis for the included sections as presented by Howe.

Stantec has relied on information provided by Gowest, which may or may not be in the public domain. This includes but is not necessarily limited to the following:

- Preparation methodologies and analyses for metallurgical testwork composites.
- Detailed metallurgical testwork results from programs completed at SGS Canada (Lakefield), Goldfields, McGill University and others.
- Previous hydrological and environmental studies conducted by Golder and their local partner Blue Heron Environmental.
- Tests completed for the sorter study by the vendors of the sorting equipment.
- Costing obtained for sorter operation and leasing, toll milling costs, transportation, smelting and refining costs from various vendors.
- Local site conditions, power-line costs and infrastructure.

Stantec has made every attempt to accurately convey the content of those files, but cannot guarantee either the accuracy or validity of the work contained within those files. However, Stantec believes that the preparation of these reports and data were completed with the objective of presenting the results of the work performed without any promotional or misleading intent. In this sense, the information presented should be considered reliable, unless otherwise stated, and may be used without any prejudice by Gowest.

Stantec has relied on the experience of Mr. Kevin Small (P.Eng.) to assist in the preparation of a conceptual plan and costing for the Bradshaw Deposit. Mr Small has been previously involved with the Bradshaw mine development portion of the current PEA (described in Section 16). Mr. Small currently works for St Andrews Gold in Timmins and has served as Manager – Technical Services for DMC Mining Services (formerly Dynatech) and as a Project Manager for Dumas Mining.

Neil N. Gow relied on reports covering environmental matters and prepared by Golder Associates (2013, 2014, and 2015) and by Davidson (2014) and Seyler (2014). Further, Mr. Gow has relied on advice from Gowest regarding ongoing relations and negotiations with various First Nations groups.

The authors have sourced the information for this report from an amalgamation of several reports listed in Section 27 – References.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Bradshaw Project (also known as the North Timmins Gold Project and formerly the Frankfield East Project) of Gowest is located approximately 32 kilometres north-northeast of the City of Timmins, the nearest largest city in Northeastern Ontario.

4.1 Mineral Tenure

The Bradshaw Project is comprised of one patented mineral claim, 11 leased mineral claims and 56 unpatented mineral claims variously located in Prosser, Wark, Tully, Gowan, Little and Evelyn Townships (Figure 4.1). The central part of the Project located in Tully Township lies at about 490000 East and 5397500 North (UTM Zone 17N NAD 83). The total area of the Gowest holdings is 10,942 ha. Within that area, the area of the Transition Metals Corp. (Transition) joint venture is 3,302 ha.

All unpatented mining claims are recorded in the name of Gowest Gold Ltd., save and except those unpatented mining claims set out within the 'Transition Option' described in more detail below in table 4.1. As of the effective date of this technical report all the Project unpatented mining claims are in good standing and have sufficient work assessment credits available for several years. Gowest has advised the authors that all municipal realty and provincial mining land taxes applicable to the patented claim and leased mineral claims are in good standing.

Details of the claims are set out in Table 4.1and were provided to the authors by Gowest.

Table 4.1: Bradshaw Project - Gowest Gold Ltd. Claims

Division	Project/Property	Township	Claim Number	Recording Date	Claim Due Date	Lease Expiry
Porcupine - 60	GW Orphan Tully (G-3985)	Tully	4240049	2010-Mar-03	2016-Mar-03	-
Porcupine - 60	GW Orphan Tully (G-3985)	Tully	4254623	2010-Mar-03	2020-Mar-03	-
Porcupine - 60	GC Tully East Block-1	Tully	1207009	1996-Mar-19	2017-Mar-19	-
Porcupine - 60	GC Tully East Block-1	Tully	1244809	2001-Mar-30	2017-Mar-30	-
Porcupine - 60	Gowest Tully East	Tully	4277620	2014-Aug-28	2016-Aug-28	-
Porcupine - 60	Gowest Tully East	Tully	4277624	2014-Aug-29	2016-Aug-29	-
Porcupine - 60	Guidoccio Tully East	Tully	4269722	2012-Mar-08	2016-Mar-08	-
Porcupine - 60	Guidoccio Tully East	Tully	4269723	2012-Mar-08	2016-Mar-08	-
Porcupine - 60	Transition Pipestone East	Evelyn	4253001	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone East	Evelyn	4253002	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone East	Evelyn	4253003	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone East	Evelyn	4253004	2010-Feb-02	2016-Feb-02	-

Division	Project/Property	Township	Claim Number	Recording Date	Claim Due Date	Lease Expiry
Porcupine - 60	Transition Pipestone East	Evelyn	4253005	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone East	Evelyn	4253006	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone East	Evelyn	4257022	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone East	Evelyn	4257023	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone East	Evelyn	4257024	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone East	Evelyn	4257025	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone East	Evelyn	4257027	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone West	Gowan	4253015	2010-Feb-02	2017-Feb-02	-
Porcupine - 60	Transition Pipestone East	Little	4257021	2010-Jul-12	2016-Jul-12	-
Porcupine - 60	Transition Pipestone West	Prosser	4253014	2010-Feb-02	2017-Feb-02	-
Porcupine - 60	Transition Pipestone West	Prosser	4255012	2010-Mar-09	2016-Mar-09	-
Porcupine - 60	Transition Pipestone West	Prosser	4255234	2010-Apr-26	2017-Apr-26	-
Porcupine - 60	Transition Pipestone West	Wark	4252998	2010-Apr-27	2017-Apr-27	-
Porcupine - 60	Transition Pipestone West	Wark	4252999	2010-Apr-26	2017-Apr-26	-
Porcupine - 60	Transition Pipestone West	Wark	4253007	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4253009	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4253010	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4253011	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4253012	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4253013	2010-Feb-02	2016-Feb-02	-
Porcupine - 60	Transition Pipestone West	Wark	4255013	2010-Mar-09	2017-Mar-09	-
Porcupine - 60	Transition Pipestone West	Wark	4255233	2010-Apr-26	2017-Apr-26	-
Porcupine - 60	Transition Pipestone West	Wark	4255235	2010-Apr-26	2017-Apr-26	-
Porcupine - 60	GW Pipestone East	Little	4270230	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270231	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270232	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270233	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270234	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270235	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Little	4270236	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Evelyn	4270237	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Evelyn	4270238	2012-May-04	2016-May-04	-

Division	Project/Property	Township	Claim Number	Recording Date	Claim Due Date	Lease Expiry
Porcupine - 60	GW Pipestone East	Evelyn	4270239	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Evelyn	4267266	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Evelyn	4267267	2012-May-04	2016-May-04	-
Porcupine - 60	GW Pipestone East	Evelyn	4262511	2011-Jun-15	2017-Jun-15	-
Porcupine - 60	GW Pipestone East	Evelyn	4262512	2011-Jun-15	2017-Jun-15	-
Porcupine - 60	GW Pipestone East	Little	4262513	2011-Jun-15	2017-Jun-15	-
Porcupine - 60	GW Pipestone East	Little	4270356	2013-Apr-08	2017-Apr-08	-
Porcupine - 60	GW Pipestone East	Little	4270357	2013-Apr-08	2017-Apr-08	-
Porcupine - 60	GW Pipestone East	Little	4270358	2013-Apr-08	2017-Apr-08	-
Porcupine - 60	GW Pipestone East	Tully	4270359	2013-Apr-08	2017-Apr-08	-
Porcupine - 60	GW Pipestone East	Little	4261682	2013-Apr-22	2017-Apr-22	-
Porcupine - 60	GW Pipestone East	Little	4261683	2013-Apr-08	2017-Apr-08	-
Division	**Project/Property**	**Township**	**Lease or License**	**Claim No.**	**Start/ Anniversary**	**Lease Expiry**
Porcupine - 60	Dowe/Frankfield	Tully	107242	101372	1999-Feb-01	2020-Jan-31
Porcupine - 60	Dowe/Frankfield	Tully	107242	101373	1999-Feb-01	2020-Jan-31
Porcupine - 60	Dowe/Frankfield	Tully	107242	101374	1999-Feb-01	2020-Jan-31
Porcupine - 60	Dowe/Frankfield	Tully	107242	101375	1999-Feb-01	2020-Jan-31
Porcupine - 60	Texmont/Frankfield	Prosser	107280	508392	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Prosser	107280	508394	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508389	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508395	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508396	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508398	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508397	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508399	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508400	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508401	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107280	508402	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Prosser	107281	508391	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Prosser	107281	508393	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107281	508390	1999-Dec-01	2020-Nov-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97938	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97941	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97942	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97943	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97939	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97940	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97948	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107335	97949	2000-Oct-01	2021-Sept-30

Division	Project/Property	Township	Claim Number	Recording Date	Claim Due Date	Lease Expiry
Porcupine - 60	Texmont/Frankfield	Tully	107336	97944	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107336	97945	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107336	97947	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107336	97946	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107360	99286	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107360	99287	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107360	99289	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107360	99288	2000-Oct-01	2021-Sept-30
Porcupine - 60	Texmont/Frankfield	Tully	107361	100440	2001-Jun-01	2022-May-31
Porcupine - 60	Texmont/Frankfield	Tully	107361	100437	2001-Jun-01	2022-May-31
Porcupine - 60	Texmont/Frankfield	Tully	107361	100441	2001-Jun-01	2022-May-31
Porcupine - 60	Texmont/Frankfield	Tully	107361	100438	2001-Jun-01	2022-May-31
Porcupine - 60	Texmont/Frankfield	Tully	107361	100442	2001-Jun-01	2022-May-31
Porcupine - 60	Texmont/Frankfield	Tully	107361	100439	2001-Jun-01	2022-May-31
Porcupine - 60	White Star/Frankfield	Tully	107310	501057	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107310	501058	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107310	501062	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107310	501063	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107310	515807	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501055	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501056	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501059	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501060	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501061	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501064	2000-Jun-01	2021-May-31
Porcupine - 60	White Star/Frankfield	Tully	107311	501065	2000-Jun-01	2021-May-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101255	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101256	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101257	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101258	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101259	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101260	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101261	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101262	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101948	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101949	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101950	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101951	2003-Sept-01	2024-Aug-31
Porcupine - 60	GC Tully North Block-1	Tully	107484	101952	2003-Sept-01	2024-Aug-31

Division	Project/Property	Township	Claim Number	Recording Date	Claim Due Date	Lease Expiry
Porcupine - 60	GC Tully East Block-1	Tully	109337	1160197	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207001	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207003	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207004	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207005	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207007	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207010	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207701	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207702	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1207703	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1212880	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1244810	2013-Aug-01	2034-Jul-31
Porcupine - 60	GC Tully East Block-1	Tully	109337	1245331	2013-Aug-01	2034-Jul-31
Division	**Project/Property**	**Township and Location**				
Porcupine - 60	Boudreau purchase	Tully	SE1/4 andSW1/4 N1/2 and S1/2 of Lot 1, Conc 1			

Other than, the claims labeled with the Transition name Table 4.1 that are discussed below, all of these claims are wholly owned by Gowest.

The law firm Weaver Simmons of Sudbury prepared a title opinion dated December 19, 2012 for a private placement. Some minor additions have been made since that time. The unpatented mining claims were independently verified by Neil N. Gow on September 4, 2015 through the MNDM website http://www.mndm.gov.on.ca/en/mines-and-minerals/applications/mining-claims-information.

4.2 Underlying Mineral Agreements

The Bradshaw Project consists of several blocks of claims acquired through mineral agreements since 2008.

On December 19, 2008, Gowest Amalgamated Resources ("GWA") entered into an option acquisition agreement with New Texmont Explorations Ltd. ("New Texmont") for a 50% interest in the Texmont/Frankfield project (Table 4.1). The New Texmont Acquisition closed on March 6, 2009. In consideration of the New Texmont Acquisition, GWA issued 15,000,000 Common Shares to New Texmont and granted New Texmont a net smelter returns royalty (a "NSR Royalty") on the transferred properties equal to 1.0% at gold prices less than US$950 per ounce or 1.5% at gold prices equal to or greater than US$950 per ounce. GWA has the right to purchase the NSR Royalty at any time upon payment of $1,000,000 for each half-percent (0.5%)

of the NSR Royalty and it also has a right of first refusal to purchase the NSR Royalty upon any offer made by a third party to purchase the NSR Royalty. GWA also agreed to make a one-time payment to New Texmont, at New Texmont's option, of $500,000 or 2,500,000 Common Shares upon a positive decision being made by the Company to place a mine into production at the Frankfield Block and subject to satisfactory financing being committed to fully-fund such mine development. The New Texmont NSR was transferred to SPG Royalties under the same terms on February 18, 2015.

On December 23, 2009, GWA entered into an agreement with Goldcorp Canada Ltd., manager of the Porcupine Gold Mines Joint Venture (the "PGMJV"), and Goldcorp Inc. (together with Goldcorp Canada Ltd., "Goldcorp"), for the purchase of all of Goldcorp's properties in Tully Township adjacent to the Frankfield Block (the "Goldcorp Acquisition"). The Goldcorp Acquisition was completed on February 12, 2010. Pursuant to the Goldcorp Acquisition, the Company acquired a 100% interest in 15 unpatented mining claims (GC Tully East Block) contiguous to the eastern boundary of the Frankfield Block. In addition, the Company acquired a 100% interest in 13 leased mining claims (GC Tully North Block), with both surface and mining rights. GWA also acquired pursuant to the Goldcorp Acquisition an extensive exploration database for all of Tully Township which was previously compiled by the PGMJV.

In consideration of the Goldcorp Acquisition, Goldcorp was paid $100,000 in cash and retained a 2.0% NSR Royalty from future production from the (GC Tully North Block) and a 1.0% NSR Royalty from future production from the (GC Tully East Block). GWA maintains an NSR Royalty buyout option for both blocks valued at $500,000 for each half-percent (0.5%) of the NSR Royalty; provided, however, Goldcorp may elect not to sell (and is under no obligation to sell) the final half-percent (0.5%) of each applicable NSR Royalty.

On July 13, 2010, GWA entered into an agreement with Thomas Trevor Kurt Dowe and Thomas Melvin Dowe for the purchase of a 100% interest in the mining rights of four leased mining claims, contiguous to the Frankfield Block (the "Dowe Acquisition"). The Dowe Acquisition closed on December 1, 2010. In consideration for the Dowe Acquisition, GWA paid an aggregate of $16,000 in cash, issued an aggregate of 70,000 Common Shares to the vendors and granted to each vendor a NSR Royalty on the transferred properties equal to 0.5% at gold prices of less than US$950 per ounce or 0.75% at gold prices equal to or greater than US$950 per ounce. GWA has the right purchase the NSR Royalty from the vendors at any time upon payment of $125,000 for each quarter-percent (0.25%) of the NSR Royalty and it also has a right of first refusal to purchase the NSR Royalty upon any offer made by a third party to purchase the NSR Royalty.

Gowest Amalgamated Resources (GWA) changed its name to Gowest Gold Ltd. on March 29, 2011.

On December 13, 2013, Gowest received a $750,000 royalty payment from Gold Royalties Corporation ("Gold Royalties") for the purchase of: (i) a 1.0% gross royalty interest on future gold production from Gowest's Bradshaw Project ("NTGP"), including the Bradshaw Deposit; and (ii) a right-of-first refusal agreement with respect to future gold streams associated with the NTGP.).

On March 3, 2014, Gowest entered into an agreement with J. Patrick Sheridan and New Texmont Explorations Ltd. for the purchase of a 100% interest in two leased mining claims (White Star Block), east and contiguous to the Frankfield Block. The purchase price payable by Gowest for the leases was to grant J. Patrick Sheridan a sliding scale net smelter return royalty in respect of gold production from the relevant properties equal to 1.0% at gold prices less than US$950 per ounce and 1.5% at gold prices equal to or greater than US$950 per ounce (the "NSRR"). Pursuant to the purchase agreement, J. Patrick Sheridan assigned and transfer all of his right, title and interest in and to the NSRR to New Texmont. The NSRR is subject to the same terms and conditions (and form part of the same royalty interest) as previously granted by Gowest to New Texmont as set out in an Acquisition Agreement dated December 19, 2008 between the Gowest and New Texmont. The royalty was transferred to SPG Royalties under the same term on February 18, 2015.

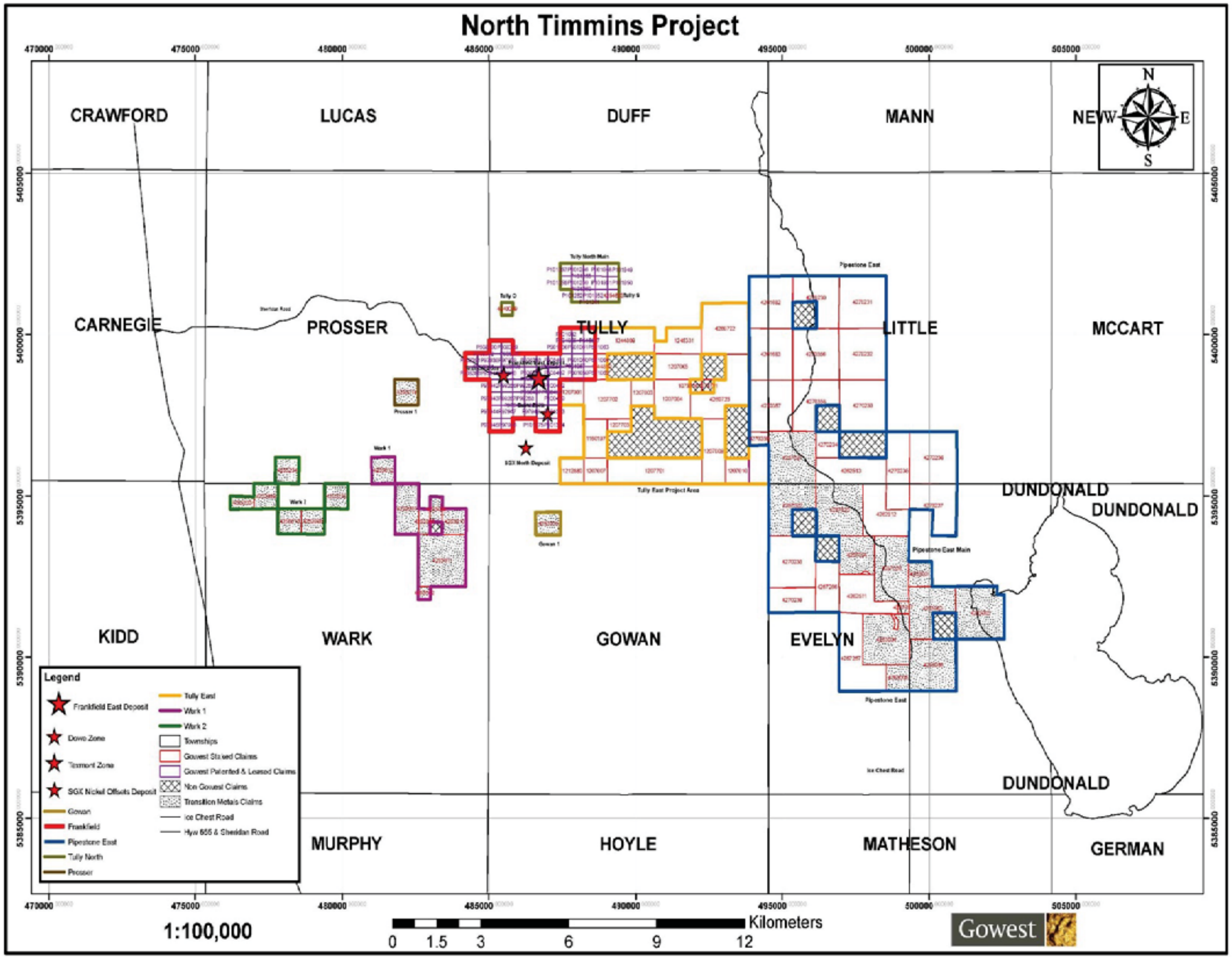
The claims that are labeled with the Transition name in Table 4.1 are held by Gowest under an option and joint venture with Transition Metals Corp. Under the terms of the agreement, dated February 10, 2011, Gowest may earn either a 60% equity interest or a 75% equity interest in the claims subject to various conditions. To earn a 60% undivided interest, Gowest must;

- Make a C$50,000 payment upon execution (Paid).
- Pay a further C$50,000 on the one year anniversary of the agreement (Paid).
- Incur exploration expenditures of C$200,000 by the 18-month anniversary of the agreement (Incurred).
- Incur aggregate exploration expenditures of C$1,000,000 by the 36-month anniversary of the agreement. This condition was amended by issuing 100,000 shares and extending an additional year until April 25, 2015. Gowest is currenty discussing an extension with Transition.
- Issue 100,000 shares upon execution of the agreement (Issued).
- Issue 300,000 shares before the 36-month anniversary of the agreement (Issued).

For Gowest to increase its interest to 75%, Gowest must advise Transition of its intention to do so within 30 days of when it has earned 60%. Further Gowest must;

- Incur additional exploration expenditures of C$2,000,000 within a 2-year period.
- Issue a further 150,000 shares of Gowest to Transition. Once this stage is reached, the partners have deemed expenditure positions and normal dilution provisions would prevail.

Figure 4.1: Property Map



4.3 Environmental and Permitting

Gowest initiated environmental studies of the Bradshaw Project at an early stage in 2010. These studies have been carried out by Golder Associates and their local partner Blue Heron Environmental.

Five years of comprehensive environmental baseline studies of the project and regional area have now been completed including:

- Air quality and noise;
- Aquatic resources (fish and benthic invertebrates) and habitat;
- Wildlife and habitat;
- Species-at-Risk;
- Overburden conditions;
- Surface water quality and flows;
- Groundwater quality and paths;
- Rock Geochemistry; and
- Archaeology Study;

Gowest reports that there are no outstanding or pending adverse environmental issues attached to the Frankfield Property. No mining or other potentially disruptive work has been carried out, on the property, beyond that described in this report.

Gowest is an active member of the local community, with an exploration office in Timmins, Ontario that offers local residents an easily accessible location to learn about Gowest and the Project. Gowest continues to engage and consult with the local communities, including First Nations and the Métis community. Through meetings, site tours and regular communications, Gowest strives to ensure engagement with all members of the local communities. Through advice from the Ontario Ministry of Northern Development and Mines Provincial Crown, Aboriginal groups identified to be consulted regarding the mine development of the project are:

- Matachewan First Nation
- Mattagami First Nation
- Taykwa Tagamou Nation
- Metis Nation of Ontario
- Timmins Metis Council
- Metis Nation of Ontario Northern Lights Metis Council

As at April 1, 2013, Exploration Plans and Permits are now required for some early exploration activities by Ontario Mining Act regulations. The requirement of a Plan or Permit is dependent on the activity being completed. Gowest holds an active Exploration Permit (PR-13-10072, expiry 28/03/2016) for any future exploration diamond drilling activities on the Bradshaw Project. Neil M. Gow has been advised by Gowest that there are no aboriginal issues that would be expected to delay the project at this time

Mr. Gow is not aware of any significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Bradshaw Project centre is located in the southwest part of Tully Township, approximately 32 km north-northeast of the City of Timmins, Ontario. Surface access to the Frankfield Block is easily gained from Timmins via Highway 655 and an all-weather gravel road that turns east off Highway 655, 33.2 km north of the intersection of Highways 101 and 665 and 11.5 km north of the Kidd Creek Mine access road. This 13.5 km long all-weather road ends at the Prosser/Tully Township line. The site of Gowest's Bradshaw Deposit (formerly known as the Frankfield East Deposit) is approximately 1.5 km further east along a drill road. Alternate access to the property is by charter helicopter service from Timmins.

The eastern portion of the Bradshaw Project area is easily accessed from Timmins via Highway 101 East and 13 kilometres north of the highway via the all-weather Ice Chest Lake gravel road. Various ATV trails provide access from this road to the Project area; however, an Argo is required to ford streams and negotiate the swampy conditions. Alternate access to the Project is by charter helicopter service from Timmins.

5.2 Climate

The climate is typical of northern boreal forest areas with the Project area experiencing four distinct seasons. There are extended periods of subzero temperatures during the winter months of November through March. Daily average winter temperature in January is -17.5° C with daily average maximum and minimums of -11° C and -23.9° C respectively and an extreme daily minimum of -44.2° C. Daily average summer temperature in July is +17.4° C with daily average maximum and minimums of +24.2° C and +10.5° C respectively and an extreme daily maximum of +38.9° C. The region has average annual precipitation of approximately 83 cm including approximately 56 cm of rain, largely during the months of April to October and up to 3.1 metres of winter snow accumulation, occurring largely between the months of November and April (Environment Canada, 2011).

Mineral exploration can be conducted year-round, however because of the swampy ground conditions on the Project, exploration activities such as geophysical surveys and diamond drilling are more easily conducted in the winter due to better accessibility after freeze-up.

5.3 Local Resources and Infrastructure

All-weather gravel road access is currently available to the Prosser/Tully Township line in the north-western part of the Project. Access to the Bradshaw Deposit could be achieved by constructing approximately 1.0 km to 1.5 km of new gravel road to connect with the existing gravel road network. Numerous drill trails crosscut the Project area. Despite wet and swampy ground conditions common throughout the Project area, the drill trails can be accessed by all-terrain vehicles and industrial equipment such as dozers, skidders and muskeg tractors during summer months. Access is easier during the winter months when the ground is frozen. 115 kV and 500 kV electric transmission lines paralleling Highway 655 are located approximately 10 km and 13.5 km west of the property respectively. The West Buskegau River, located on the Frankfield claim block, offers an abundant source of process water. Large quantities of aggregate resources are located adjacent to Highway 655, approximately 15 km west of the Frankfield claim block.

Gowest maintains a secure and well-equipped combined field office and core logging/sampling facility at 115 Jubilee Avenue East, Timmins.

The City of Timmins is the nearest source of mining-related commercial services and an abundant pool of managerial and skilled labour. Timmins is serviced by modern telecommunications, commercial airlines, rail service and truck transportation.

Gowest holds sufficient surface rights necessary for potential future mining operations including tailings storage areas, waste disposal areas and a processing plant.

5.4 Physiography

Regional-scale poorly drained swamp dominates the Project area. The area topography is flat with an elevation of approximately 295 metres above sea level. Relief is only a few meters with drier clay ridges rising above open and forested swampy areas. All streams and rivers in the area are part of the Arctic watershed. The West Buskegau River, although a potential source of abundant water for the Project, provides little drainage for the low-lying terrain. Drainage patterns are poorly developed due to the low topographic relief and to the extensive clay cover immediately below the vegetation layer. Many of the diamond drill holes form natural wells. Overburden is generally deep in the region, with depths up to 65 metres. However, overburden in the area of Bradshaw Deposit is generally shallow, ranging from 2 to 15 metres thick. Isolated rock exposures are present in the vicinity of the Sheridan Deposit (formerly known as the Texmont Deposit) and approximately 300 metres northeast of the Bradshaw Deposit; the latter outcrop area may be an ideal location for establishment of an underground ramp.

Vegetation consists of poorly developed black spruce, patches of alders and low shrubs. The immediate vicinity of the deposit has been partially cleared of trees, due to diamond drilling campaigns over the years.

6.0 HISTORY

Earlier NI 43-101 reports have dealt with the Frankfield claim block in isolation. This current report discusses the Frankfield Property and claims that are contiguous with Frankfield or closely related and which make up the Bradshaw Project. In the interests of simplicity, the claims have been divided up into township claim groupings/properties (Figure 4.1).

6.1 Frankfield Block

The following description of exploration history is adapted and updated from Harron (2006).

Following the discovery of the nearby Kidd Creek Mine in 1964, exploration activity intensified in Tully and other surrounding townships. In 1964, Texasgulf Sulphur Co. Ltd. completed one diamond drill hole (“DDH”) in the S1/2 Lot 10, Concession III of Tully Township (current claim 508402) to test an airborne electromagnetic (“AEM”) conductor, which proved to be graphite.

In 1965, Patino Mining Corporation held the S1/2 of Lot 11 Concession III, Tully Township and the S1/2 of Lot 1 Concession III Prosser Township (most of the current Gowest / New Texmont block) and completed both a magnetic and electromagnetic (“EM”) surveys. The claims were allowed to lapse.

Texasgulf Sulphur Co. Ltd. in 1963-64, and Texmont Mines Ltd. (Texmont) in 1968 covered the four Prosser Township claims with magnetic and EM surveys. In 1969, Texmont completed two (2) diamond drill holes in the southeast corner of Lot 1 Concession III Prosser Township (current claim 508394) to investigate an EM conductive horizon. The causative source was graphite.

In 1968, Acme Gas and Oil Ltd. (Acme) staked eight (8) claims in the south half of Lots 10 and 11, Concession III, Tully Township (area of current claims 508395to 508402). Magnetic and vertical loop EM (“VLEM”) surveys were completed on four (4) claims in Lot 10 Concession III Tully Township (area of current claims 508399-508402). Acme optioned the 8 claims to McIntyre Mines Limited (McIntyre) in 1969. McIntyre completed three (3) diamond drill holes in the east central part of the Acme claim block to test magnetic and electromagnetic responses (area of current claims 508398 and 508400). The diamond drill holes encountered low values of Cu, Zn and Au in diorite and intermediate volcanic rocks. In 1975 Acme optioned the 8 claims to Frankfield Explorations Ltd. (“Frankfield”). The Acme claims lapsed in 1978.

In 1969, Noranda Exploration held claims covering the current White Star/Frankfield leases and completed two diamond drill holes totaling 296 metres (area of current claims 501057 and 501058). An additional two diamond drill holes (totaling 258 metres) were completed by Noranda in 1973 northeast of the earlier holes (area of current claims 515807 and 501063). No assays are reported on the Noranda drill logs and the drilling intersected intermediate to mafic volcanics and some ultramafics. The claims held by Noranda were allowed to lapse sometime after 1973 and were subsequently acquired by White Star Copper Mines Limited. White Star carried out a ground EM survey on the 12 claim block 501055-501065 and 515807 during March and April 1981. A series of four conductors in the southwest portion of the claim block were detected by the EM survey. These four conductors were tested by White Star with a single diamond drill hole each (416 metres total drilling) from 27 June to 29 July 29, 1981.

In 1978, Gold Shield Syndicate (Gold Shield) staked claims 508391-508394 being the S1/2 Lot 1, Concession III Prosser Township, and claims 508389 and 508390 being the S1/2 of N1/2 Lot 12 Concession III Tully Township, as well as claims 508395-508402 being the S1/2 of Lots 10 and 11, Concession III Tully Township. These claims cover the northern (down-dip) portion of the Bradshaw Deposit and make up a part of Gowest’s current Frankfield Property (the Gowest-New Texmont block). Gold Shield Syndicate completed ground magnetic and VLEM surveys on claims 508395-508398 (S1/2 of Lot 11 Concession III, Tully Township). The geophysical surveys utilized N-S lines 122 metres apart. The magnetic survey defined a northwest trending fault diagonally across the S1/2 of S1/2 Lot 11 Concession II. Also defined was a fault on the north flank of a magnetically positive feature interpreted as ultramafic rocks, extending N70°E from the southwest corner of S1/2 of Lot 11 Concession III. The VLEM survey defined several weak conductive features in the S1/2 of the 4 claim group. Three conductive horizons interpreted to be graphite and disseminated sulphides were located. The conductors trend N050°E in the southwest corner to N070°E in the central part and 090° in the south eastern part of the 4 claim block. The entire 122 metre width of the combined conductive horizons is interpreted as a shear zone (Bradshaw, 1978).

In June 26, 1979, Romex Resources Inc. (Romex) entered into an option/joint venture agreement with Gold Shield Syndicate to earn an interest in the 14 claims.

In 1980, Gold Shield completed magnetic and Crone “Radem” electromagnetic surveys on 10 claims (508391 to 508394 in Prosser township; 508389, 508390 and 508399 to 508402 in Tully Township). In Prosser Township, the magnetic data defined a positive magnetic feature interpreted as folded ultramafic flows. In the S1/2 Lot 10 Tully Township claims, the magnetic data defined a 60° fabric and a N-S diabase dyke. The Radem electromagnetic survey did not define any noteworthy conductive horizons due to the instruments limited penetration of the extensive clay overburden.

Six holes (1,025 metres) drilled in 1980 and 1982 (80-1 to 80-4 and 82-2 and 82-4) tested the down dip extension of the Bradshaw Main zones on the Gold Shield property (claims 508396, 508397 and 508400). In 1983, Romex Resources Inc. (Romex) earned a 17% interest in the 14 claims, pursuant to the 1979 option/joint venture agreement. Gowest subsequently acquired Gold Shield's 83% interest in the 14 claim property.

The Bradshaw Deposit is located on the north boundary of the Intex-Frankfield block, immediately adjacent and south of the Gowest block. To cover the northward down-dip extension of the Bradshaw Deposit onto Gowest-Romex ground, New Texmont Explorations Ltd., (which owned 50% of Intex Mining Company at the time) entered into an option / joint venture agreement with Gowest and Romex on October 21, 1987. Under the terms of the agreement, New Texmont could earn a 50% interest in the Gowest-Romex Property by expending $400,000 prior to June 30, 1989 (Pearson, 1989).

On March 17, 1989 Gowest purchased Romex's 17% interest in 14 claims P 508389 to 508402 (situated in Tully and Prosser townships) resulting in a 50:50 joint venture between Gowest and New Texmont.

In 1989 New Texmont and Intex Mining Co. Ltd (50% owned by New Texmont) entered into a joint venture agreement with Zenmac Zinc Ltd. (an affiliated corporation) to finance the continued drilling and underground exploration of the Bradshaw Deposit occurring on both the Gowest / New Texmont and Intex properties. Drilling by the Intex / Texmont / Zenmac joint venture in 1988 amounted to 5,350 metres at 20 sites (DDH's 88-1 to 88-19 and 88-21).

Two holes (89-GO-1 and 89-GO-3, totaling 1,216 m) were drilled in 1989 to test the Bradshaw Deposit at depth. 89-GO-3 returned an assay of 5.45 g/t Au over a core length of 22.65 metres at an approximate depth of 488 metres vertical.

In 1990, the Gowest / New Texmont joint venture completed diamond drill holes 90-GO-4 (666.6 m) and 90-GO-5 (715.4 m), to test areas approximately 61 metres east and west of previous gold intersections of 5.45 g/t Au over 22.65 metres and 4.79 g/t Au over 8.07 metres (89-GO-3). Drill hole 90-GO-4 returned gold values of 6.30 g/t Au over 4.9 metres and 3.33 g/t Au over 10.42 metres at a vertical depth of 518 metres. Drill hole 90-GO-5 returned a gold assay of 2.39 g/t Au over 11.7 metres at a vertical depth of approximately 564 metres.

In 1990, Cyprus Gold (Canada) Ltd. (Cyprus) acquired an option to earn a 70% interest in the Bradshaw Deposit from the Gowest / New Texmont joint venture, and the neighbouring Sheridan Deposit from Intex and Frankfield. The exploration program consisted of core re-logging and sampling of 15 previous drill holes (209 samples), magnetic and HLEM surveys and diamond drilling of 7 holes totaling 3,638 metres (T-91-1 to T-91-6 and T-91-9). The object of the drilling was to test the gold mineralization potential of the Bradshaw Deposit to a depth of 600 m. Drill hole T-91-6 penetrated the Main zone at approximately 600 metres and returned a value of 2.37 g/t Au over a core length of 3.0 m, indicating a significant depth potential for the Main Zone mineralization. Cyprus concluded at the time that the Bradshaw Deposit is approximately 480 metres long at the bedrock surface which diminishes with increasing depth along a steep westward plunge to about 200 metres strike length at a depth of 300 metres. Cyprus dropped the option in 1991.

In 2004, the Gowest / New Texmont joint venture proceeded with a diamond drilling program which consisted of 23 holes totaling 6,538 metres (GW04-01 to GW04-22 and GW04-25). The diamond drill program was designed to intersect the northerly dipping mineralized horizons of the Bradshaw Deposit at 50 metre intervals, both horizontally and vertically, between a depth of 100 and 300 metres. The 2004 drill program is discussed in Section 10. Two drill holes (GW04-22 and GW04-25) successfully intersected the target at about a 300 metre depth. At the end of this drill program the Main Zone gold mineralization (M1 and M2) was recognized as being 600 metre long, to a drilled depth of 300 metres, with indications that the gold mineralization continues to a depth of at least 600 metres. The steeply north dipping Main zone (-71°) appeared to have an average width of 3.7 metres in the eastern part and 8.3 m, in the western part. Assay results from mineralized zones in the hanging wall of the Main zone (Quartz Breccia zones B1 and B2) were beginning to show potentially economic mineralization, but were poorly understood.

6.2 Prosser Block

Work on the Prosser Block is included in Table 6.1. Much of the geophysical work is old and has probably been superseded by more recent Government work.

Table 6.1: Work History Prosser Property

Year	Afri File No.	Company	Work Type	Results
1966	42A14SE0106	CANICO	AEM, AMAG	Airborne survey covering Abitibi-Price claims in ten townships including Prosser Twp.
1964 - 1966	42A11NE0188	KENILWORTHMINES	DD	Ground Mag and EM followed by one drill hole (93.5 m) no assays reported.
1968	T1448	TEXMONT MINES	EM, MAG	Ground Mag and VLEM.
1970	42A14SE8398	McINTYRE PORCUPINE MINES	COMP	Geophysical and Geological Compilation
1975	42A11NE0191	GEOEX LTD	EM	Ground VLEM over 4 claims in Central Prosser Twp.

Year	Afri File No.	Company	Work Type	Results
1988	42A11NE0181	FALCONBRIDGE	EM, MAG	Ground HLEM and Mag over 4 claims in Central Prosser Twp.
1999	42A11NE0097	PENTLAND FIRTH	EM, MAG	Ground HLEM and Mag over 4 claims in Central Prosser Twp.
1999	42A11NE2006	PENTLAND FIRTH	IP	IP survey over 4 claims central Prosser. Weak conductor located.
2001	42A11NE2011	PEGG C.	GEOL,GEOC, EM	Geological mapping located intermediate to mafic volcanic outcrops in the south end. VLF survey conducted. Soil sampling (70samples) over VLF survey grid. No assay data for sampling.
2005	2000000976	PEGG C.	GEOC	Soil pH survey- 70 samples
2011		GOWEST GOLD	GEOL	Geological mapping confirmed above results.
2012		GOWEST GOLD	GEOC	Soil Gas Hydrocarbon (SGH) survey over entire block.

6.3 Tully East Claims

A summary of the previous assessment work on Tully East claim block is set out in Table 6.2. The Tully East claim block is contiguous with the Frankfield Block.

Table 6.2: Work History Tully East Property

Year	AFRI FILE No.	Company	Work Type	Results
1968	42A14SE0157	CINCINNATI PORCUPINE MINES	AEM, MAG, EM	Prior to 1968 an airborne EM survey was conducted.. Mag and VLEM over 7 claims.
1969	42A14NE023 6	CINCINNATI PORCUPINE MINES	DD	3 ddh, T-69-1 to 3 in NW Tully Twp., no assay data. Holes intersected a NW trending carbonate zone in basalts on the south side of an ultramafic body. Apparently a gold value was encountered in either hole 1 or 2.
1977	42A14SE0129	WESTERN MINES	AEM	Airborne INPUT electromagnetic survey by Questor Surveys Ltd over 6 claims west central Tully Twp., which covers the SW 1/4 of the Tully North Property.
1982	42A14SE0107	NEWMONT EXPLORATION	EM, MAG, IP	Mag, Max-Min HLEM and IP surveys over the entire Tully North Property.
1983	2451	NEWMONT EXPLORATION	DD	11 ddh MN81-1, MN81-3 to 12, in NW Tully Twp. Follow up to Cincinnati Porcupine Mines drilling. Multiple gold values with best value of 7.1 gpt Au over 1.5 meters in hole MN83-1. The gold zone is characterized by disseminated arsenopyrite-pyrite cut by ankerite veinlets and hosted in a broad zone of carbonated-silicified basalts.
1987	42A14SE0102	ESSO MINERALS	GEOL, EM	Geological mapping on present claim 4254623, no outcrop on the claim but 4 pillowed mafic volcanic outcrops just south of SW corner HLEM survey but results not reported.
1990	42A14SE0100	L.SALO	EM, MAG	Mag and VLF survey over present claim 4254623.

Year	AFRI FILE No.	Company	Work Type	Results
2011		GOWEST GOLD	AMAG,AEM	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project.
2012		GOWEST GOLD	DD	5 ddh holes for 1,172 metres in the SW (see section 10.6 of this report)
2013		GOWEST GOLD	DD	1 ddh, totaling 225 m drilled 1.7 km east of the Bradshaw Deposit (see section 10.6 of this report).

6.4 Tully North Claim Group

This claim group is 2 km northeast of the Frankfield Block and previous assessment work on the group is detailed in Table 6.3.

Table 6.3: Work History, Tully North Property

Year	Afri File No.	Company	Work Type	Results
1968	42A14SE0157	CINCINNATI PORCUPINE MINES	AEM, MAG, EM	Prior to 1968 an airborne EM survey was conducted.. Mag and VLEM over 7 claims.
1969	42A14NE0236	CINCINNATI PORCUPINE MINES	DD	3 ddh, T-69-1 to 3 in NW Tully Twp., no assay data. Holes intersected a NW trending carbonate zone in basalts on the south side of an ultramafic body. Apparently a gold value was encountered in either hole 1 or 2.
1977	42A14SE0129	WESTERN MINES	AEM	Airborne INPUT electromagnetic survey by Questor Surveys Ltd over 6 claims west central Tully Twp., which covers the SW ¼ of the Tully North Property.
1982	42A14SE0107	NEWMONT EXPLORATION	EM, MAG, IP	Mag, Max-Min HLEM and IP surveys over the entire Tully North Property.
1983	2451	NEWMONT EXPLORATION	DD	11 ddh MN81-1, MN81-3 to 12, in NW Tully Twp. Follow up to Cincinnati Porcupine Mines drilling. Multiple gold values with best value of 7.1 gpt Au over 1.5 meters in hole MN83-1. The gold zone is characterized by disseminated arsenopyrite-pyrite cut by ankerite veinlets and hosted in a broad zone of carbonated-silicified basalts.
1987	42A14SE0102	ESSO MINERALS	GEOL, EM	Geological mapping on present claim 4254623, no outcrop on the claim but 4 pillowed mafic volcanic outcrops just south of SW corner HLEM survey but results not reported.
1990	42A14SE0100	L.SALO	EM, MAG	Mag and VLF survey over present claim 4254623.
2011		GOWEST GOLD	AMAG,AEM	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project.
2013		GOWEST GOLD	DD	6 ddh totaling 2,401 m outlining the Roussain Zone (section 10.6 of this report).

6.5 Wark 1 Claim Group

This is the eastern most of two claim groups in Wark Township and is located 2.5 km southwest of the Frankfield Block. Previous assessment work is documented in Table 6.4.

Table 6.4: Work History, Wark 1 Property

Year	Afri File No.	Company	Work Type	Results
1964	42A11NW0008	NATIONAL EXPLORATION	EM, MAG. GEOL	Ground Mag and VLEM surveys and geological mapping over NW corner of property.
1964	42A11NW0002	NATIONAL EXPLORATION	DD	8 ddh, 64-1 to 8 totaling 1015 m tested 5 conductive zones. Some assaying for gold but only trace values. Drilling encountered intermediate volcanics and mafic volcanics.
1964	42A11NE0564	NORTH AMERICAN RARE METALS	EM, MAG	Ground HLEM and Mag over South half of the property.
1966	42A11NE0570	NORTH AMERICAN RARE METALS	DD	1 ddh NAR-11 (128.6 m) on the present property. No sampling reported.
1969	42A11NE0561	MESPI MINES	EM	Ground VLEM over central-north part and outlined 3 conductors.
1970	42A11NE0562	FALCONBRIDGE	EM, MAG	Ground VLEM and Mag over South half of the property.
1972	42A11NW0004	TEXAS GULF	EM, MAG	Ground HLEM and Mag over the NW corner of the property.
1975	42A11NE0566	McINTYRE PORCUPINE MINES	DD	3 ddh, 051-75-5 to 7 totaling 471 m. No assay data. One hole intersected dacitic tuffs interbedded with argillites, the other cut quartz veins in graphitic argillite above a peridotite and the third was lost in overburden.
1980	42A11NE0186	P. HUNKIN	EM, MAG	Ground VLEM and Mag over north half of the property.
1981	42A14SE0208	PLACER	AMAG	Airborne Magnetic survey conducted by Questor Surveying covering part of Prosser and Wark townships.
1983	42A11NE0553	COMSTATE RESOURCES	AEM	Airborne Mark VI INPUT survey conducted by Questor Surveying covering Prosser, Wark and Murphy townships. No INPUT anomalies detected.
1985	42A11NE0185	GOLDEN RANGE RESOURCES	GEOC	Seven hole wacker till sampling program in the NW corner of the Property. No anomalous gold or base metal values encountered. Overburden depths of 10 to 25 m.
1985	42A11NE0183	GOLDEN RANGE RESOURCES	EM, MAG	Ground VLF and Mag over north half of the property.
1990	T3386	COMINCO	EM, MAG	Mag and Max-Min II survey over 5 claims in NE Wark Twp. and 8 claims in SE Prosser Twp. 4 weak EM conductors outlined.

Year	Afri File No.	Company	Work Type	Results
1997	42A11NE0097	PENTLAND FIRTH	EM, MAG	Ground HLEM and Mag over the north half of the property. Three HLEM conductors detected.
1999	42A11NE2006	PENTLAND FIRTH	IP	IP survey over the north half of the property. Three strong IP responses 2 of which coincide with the HLEM conductors.
2001	42A11NE2011	PEGG C.	GEOL,EM	Geological mapping and VLF conducted on north-central part of existing property. No outcrops found.
2011		GOWEST GOLD	AMAG,AEM, GEOC	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project Soil Gas Hydrocarbon Survey (SGH) over entire property.

6.6 Wark 2 Claim Group

This claim group is situated 2.5 km west of the Wark 2 claim group and previous assessment work is summarized in Table 6.5.

Table 6.5: Work History, Wark 2 Property

Year	Afri File No.	Company	Work Type	Results
1964	42A11NW0527	GLENN EXPLORATIONS	EM, MAG, DD	Ground Mag and VLEM surveys over north part of the southern claim. Numerous weak EM conductors detected. Two holes G-1 to G-2 totaling 495 m drilled (no logs in file). Hole 1 tested the strongest EM conductor. Holes encountered sediments and intermediate volcanics, but EM conductors not explained. No economic mineralization reported.
1964	42A11NW0535	WINDFALL OIL and MINES LTD	DD	3 ddh, holes 5, 7 and 8 totaling 475 m on the NE claim. Holes cut ultramafic volcanics, 1 assay reported nil gold and base metals.
1964	42A11NW0536	PCE EXPLORATION LTD.	DD	2 ddh, holes P-3 and P-4 totaling 293 m on the western claim. Holes cut ultramafic volcanics and felsic to intermediate volcanics, 2 assays reported trace gold.
1969	42A11NE0561	MESPI MINES	EM	Ground VLEM over south half of the central claim. No conductors were detected.
1970	42A14SE8398	McINTYRE PORCUPINE MINES	COMP	Geophysical and Geological Compilation
1971	42A11NW8400	TEXAS GULF	EM, MAG	Ground HLEM and Mag over the north half of the central claim. One weak conductor.
1981	42A14SE0208	PLACER	AMAG	Airborne Magnetic survey conducted by Questor Surveying covering part of Prosser and Wark townships.

Year	Afri File No.	Company	Work Type	Results
1989	42A11NW0502	FALCONBRIDGE	EM, MAG	Ground HLEM and Mag over west half of the central claim. One weak EM conductor.
1993	42A11NW0072	FALCONBRIDGE	DD	1 ddh W62-01 (269 m) on the SW1/4 of the central claim. Hole intersected argillite followed by mafic breccia and then mafic flows. 4 assays reported nil gold and base metals along with 8 whole rock samples
1996	42A11NW0068	MEUNIER-PEGG	TR	Manual stripping off of overburden from two outcrop areas of mafic volcanics on the western claim.
1998	42A11NW2005	MEUNIER	TR	same as above
2001	42A11NE2011	PEGG C.	GEOL,EM	Geological mapping and VLF conducted on the north half of the central claim. No outcrops were found.
2011		GOWEST GOLD	AMAG,AEM, GEOC	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project. Soil Gas Hydrocarbon Survey (SGH) over western half.
2012		GOWEST GOLD	GEOC	Soil Gas Hydrocarbon (SGH) survey over eastern half.

6.7 Gowan Block

This is the southernmost claim block in the Bradshaw Project and is located 2.4 km south of the Frankfield Block. Previous exploration on the property is detailed in Table 6.6.

Table 6.6: Work History, Gowan Property

Year	Afri File No.	Company	Work Type	Results
1965	42A11NE0531	NEW CALUMET MINES LTD.	EM, MAG	Ground Mag and VLEM.
1982	42A11NE0508	COMINCO	RC	2 RC drill holes, GO-124 and 125 hit bedrock at 14 m and 29 m depths. No assay data. No bedrock descriptions.
1983	42A11NE0509	COMINCO	RC	6 RC drill holes, GO-133 to 139 all hit bedrock at 16 to 23.5 m depths. No assay data. Bedrock descriptions too vague to determine rock type.
2011		GOWEST GOLD	AMAG,AEM, GEOL	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project Geological mapping confirmed the 2001 mapping.
2012		GOWEST GOLD	GEOC	Soil Gas Hydrocarbon (SGH) survey over entire block.

6.8 Pipestone East Group

This claim group is comprised of 12 unpatented mineral claims under option from Transition and 24 unpatented mineral claims staked by Gowest. The group is located in southeastern Little Township and northwestern Evelyn Township. A summary of the previous assessment work on the Pipestone East claim block is set out in Table 6.7.

Table 6.7: Work History, Pipestone East Property

Year	AFRI FILE No.	Company	Work Type	Results
1964	42A11NE0929	FIDELITY MINING	EM, MAG	Ground VLEM & Mag over part of present claims 4270231 and 4270234
1964	42A10NW0008	AUGUSTUS EXPLORATION	EM, MAG	Ground Turam EM & Mag over part of present claims 4270235 and 4270237.
1964	42A11NE0550	ALDAGE MINES	EM, MAG	Ground Turam EM & Mag over part of present claims 4270236 and 4270238.
1964	42A11SE0164	HOLLINGER	EM, MAG	Ground Turam EM & Mag over part of present claim 4270230.
1964	42A11NE0551	MARCH MINMERALS	EM, MAG	Ground VLEM & Mag over part of present claims 4262511 and 4267267
1964	42A11NE0133	HOLLINGER	DD	One hole (141 m) cut intermediate volcanics on present claim 4270357. No assay results reported.
1965	42A11NE0815	MARCH MINMERALS	DD	Three holes (457 m total) were drilled testing weak EM conductors on present claim 426511. Some quartz zones intersected in andesite volcanics but no assays reported and conductors unexplained.
1965	42A10NW0009	AREA MINES	EM, MAG	Ground VLEM & Mag over part of present claims 4270239 and 4267266
1965	42A11NE0550	TREND EXPLORATION	EM, MAG	Ground VLEM & Mag over part of present claims 4253004 and 4262511
1965	42A11SE0027	SHIELD EXP & DEV	EM, MAG	Ground Turam EM & Mag over part of present claims 4270232 and 4270233.
1966	42A11NE0134	JASCO PROSPECTING	DD	One hole JL-1 (31.5 m) drilled on outcrop cut carbonate altered andesite. No assays reported. Claim 4257021.
1967	42A11NE0813	HOLLINGER	DD	One hole E-1A (140.5 m) drilled 137.3 m of overburden then hit ultramafic. One very low Ni assay reported. Claim 4257023.
1967	42A11NE0812	HOLLINGER	DD	One hole E2 (70.3 m) drilled overburden did not hit bedrock. No assays reported. Claim 4262512.
1968	42A11NE0121	NORANDA	EM, MAG	Ground VLEM & Mag over part of present claim 4270233.
1968	42A10NW0516	HOLLINGER	DD	One hole E3 (119.8 m) drilled overburden till 104 m then finished in ultramafic. Three Ni samples but no assays reported. Claim 4253006.
1968	42A11NE0931	McINTYRE PORCUPINE MINES	EM, MAG, GEOL	Ground VLEM & Mag and geological mapping over part of present claims 4257021 and 4270231. One outcrop of

Year	AFRI FILE No.	Company	Work Type	Results
				intermediate volcanics located same as the Jasco Prospecting outcrop.
1969	42A11NE0119	NORANDA	EM, MAG	Ground VLEM & Mag over part of present claim 4270231.
1970	42A10NW0695	HOLLINGER	DD	One hole E4 (103 m) hole cut dacitic tuff followed by ultramafic .No assays reported. Claim 4253003.
1970	42A15SW0169	MAGOMA MINE	DD	One hole intersected rhyolites? with graphite horizons. No assaying reported. Claim 4270357.
1972	42A11NE0918	TEXAS GULF	EM, MAG	Ground VLEM, HLEM & Mag over part of the present claim 4270231. No conductors detected.
1972	42A11NE0118	TEXAS GULF	EM, MAG	Ground VLEM, HLEM & Mag over part of the present claim 4270356. Results poor.
1973	42A14SE0402	DR DERRY	EM, MAG	Ground Turam EM & Mag over part of present claim 4270230.
1973	42A14SE0403	NORANDA	EM, MAG	Ground VLEM & Mag over part of present claim 4270230.
1973	42A11NE0116	DR DERRY	EM, MAG	Ground Turam EM & Mag over part of present claim 4270232.
1978	42A11NE0113	NORANDA	DD	2 holes TK-1-78-4 and 5 totaling 411.4 m on present claim 4270233. Holes cut intermediate volcanics and sediments. Holes sampled and no anomalous Au, Ague and Zn values.
1978	42A11NE0114	NORANDA	DD	One hole T1-78-1 (175.9 m) on present claim 4270358. Hole tested IP anomaly which was caused by pyrite and pyrrhotite disseminations to blebs in graphite zones hosted by andesite. Eight core samples taken returning very low gold values.
1978	42A15NE0015	AMOCO PETROLEUM	DD	1 ddh, 5-1 (172.8 m) drilled on claim 4257021. It cut ultramafic volcanics followed by mafic volcanics that contained graphitic sediment units. No assay data.
1978	42A11NE0534	AMOCO PETROLEUM	DD	1 ddh, T077-8-1 (153 m) drilled on claim 4270357. It cut argillite sediments with graphite horizons. No assay data.
1978	42A11NE8377	AMOCO PETROLEUM	DD	1 ddh, 5-2 (158 m) drilled on claim 4257022. It cut mafic volcanics. Two holes (18-1 & 18-2) totaling 294.7 m on claim 4270237. Holes intersected mafic volcanics, sediments and ultramafic volcanics. No assay data.
1979	42A11NE0712	ROSARIO RESOURCES	EM	HLEM, max-min 1777,444 Kz. On claims 4257021 and 4257022.
1979	42A15SW0151	NORCEN ENERGY	AMAG	Airborne Magnetic survey by Questor Surveys Ltd over several townships including the property's northern claims in Little TWP.
1980	42A14SE0122	NORCEN ENERGY	AEM	Airborne INPUT electromagnetic survey by Questor Surveys Ltd over Tully and

Year	AFRI FILE No.	Company	Work Type	Results
				Little Townships.
1980	42A11NE0111	LACANA MINING	DD	1 ddh, T80-7 (176 m) drilled on claim 4257021. It cut mafic volcanics that contained graphitic sediment units. Core and sludge sampling with low gold values.
1980	42A11NE0106	NORCEN ENERGY	GEOL	Geological mapping did not discover any out crop on part of present claim 426183.
1980	42A11NE0109	NORCEN ENERGY	DD	Three holes (525.8 m total) were drilled on present claim 4270356. Some quartz zones and graphitic argillite intersected in andesite volcanics. Little sampling with no gold detected.
1983	42A10NW0027	L JOLIN	MAG	Ground Mag over part of present claims 4270235 & 4270236
1983	42A11NE0003	COMINCO	MAG	Ground Mag over part of present claim 4270238.
1986	42A15SW8860	ANGELA DEVELOPMENTS	AEM, AMAG	Airborne survey by Ferderber Geophysics covering several townships including Evelyn & Little Townships.
1988	42A10NW0027	ALLERSTON	MAG	Ground Mag over part of present claim 4253006
1991	42A11NE0999	FALCONBRIDGE	AEM, AMAG, AVLF	Helicopter survey by Aerodat Ltd covering several townships including Evelyn & Little Townships.
1993	42A10NW0035	HUTTERI	EM, MAG	Ground HLEM & Mag over present claim 4253006.
1993	42A11NE0102	PEPLINSKI	MAG	Ground Mag over part of present claim 4270234.
1995	42A11NE0080	GAMBLE	EM,MAG	Ground VLEM & Mag over part of present claim 4270234.
1995	42A14SE0038	PHELPS DODGE	DD	One hole (240-1) of 190 m tested an HLEM conductor which turned out to be graphitic seds at 144 to 147 m downhole in mafic volcanics. 30 samples taken with best gold value 185 ppb.
1996	42A10NW0034	ARISTA RESOURCES	AMAG, AVLF	Helicopter survey by Aerodat Ltd covering part of Evelyn Township. Geological and geophysical compilation.
1997	42A10NW0040	OREZONE RESOPURCES	DD	One hole E3 (161.7 m) drilled overburden till 67 m then intersected sediments interbedded with andesite flows. Nine core samples returned nil gold values. Claim 4270236.
1998	42A11NE2001	WIN-ELDRICH MINES	MAG	Ground Mag over parts of 4 present claims Se of Lizard Lake Evelyn Twp.
2004	20001019	INCO/AURO PLATINUM	EM, MAG, GEOL	Deep 2002 OGS Megatem conductor was covered by a 6 claim unit property. Mapping indicated no outcrop. Ground Mag and HLEM surveys, no conductor detected so overburden deeper than 100 m. Claim 4257024.

Year	AFRI FILE No.	Company	Work Type	Results
2011		GOWEST GOLD	AMAG,AEM, GEOC	Helitem EM and Mag conducted by Fugro Airborne Surveys over the Timmins North Project. Soil Gas Hydrocarbon Survey (SGH) over the Transition claims.
2013		GOWEST GOLD		3 ddh, totaling 1,291 m (see Section 10.6 of the report).

It is apparent that a significant amount of work has been carried out over many years on the claim blocks (properties) that make up the Bradshaw Project. The majority of the historical core drilling outside the Frankfield Block has been shallow, above 300 metre vertical depth.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional and Local Geology

The following has been extracted from Harron (2006) and Bradshaw (2008) with minor edits.

The Gowest NTGP, situated in the Abitibi Greenstone Belt ("AGB"), is underlain by Neoarchean supracrustal rocks of the Abitibi Subprovince of the Canadian Shield (Figure 7.1). Supracrustal rocks are divided into tectonostratigraphic units called assemblages for descriptive purposes. The reader is referred to Jackson and Fyon (1991) for a full discussion of the Archean geology of the Superior Province and Ayer et al. (2002) for a more recent interpretation of the AGB geology. Gold Deposits are structurally controlled and are widely distributed within the AGB, but all of the large Deposits occur within 2 km of the Destor-Porcupine Fault Zone, the Pipestone Fault Zone and the Cadillac-Larder Lake Shear Zone. As of 1990, 70% of all gold production in Canada has come from the AGB. Gold production plus reserves for AGB Deposits (Ontario and Quebec) calculated in 1991 were estimated at about 615 million tonnes (678 million tons) grading 7.54 g/tonne (0.22 oz/ton) Au.

Two dominantly volcanic assemblages and one dominantly sedimentary assemblage underlie the Project (Figure 7.1) (Ayer and Trowell, 2001). The eastern project area is cut by the regional northwest-trending Buskegau River Fault. The Porcupine (sedimentary) assemblage (2696-2675 Ma) underlies the south and southwestern portions of the project area and unconformably overlies the Kidd-Munro (volcanic) assemblage (2719-2711 Ma). The Kidd-Munro underlies the central part of the project area and is in fault contact to the northwest with the upper Tisdale (volcanic) assemblage (2710- 2703 Ma). To the east of the Buskegau River Fault, the Kidd-Munro assemblage rocks underlie the southeast part of the project. Upper Tisdale assemblage rocks overlie the Kidd-Munro assemblage to the north, and possibly interfolded Porcupine assemblage rocks near the contact between these two tectonostratigraphic units. The project stratigraphy is interpreted to be cross cut by later north-south faults and northeast-southwest faults.

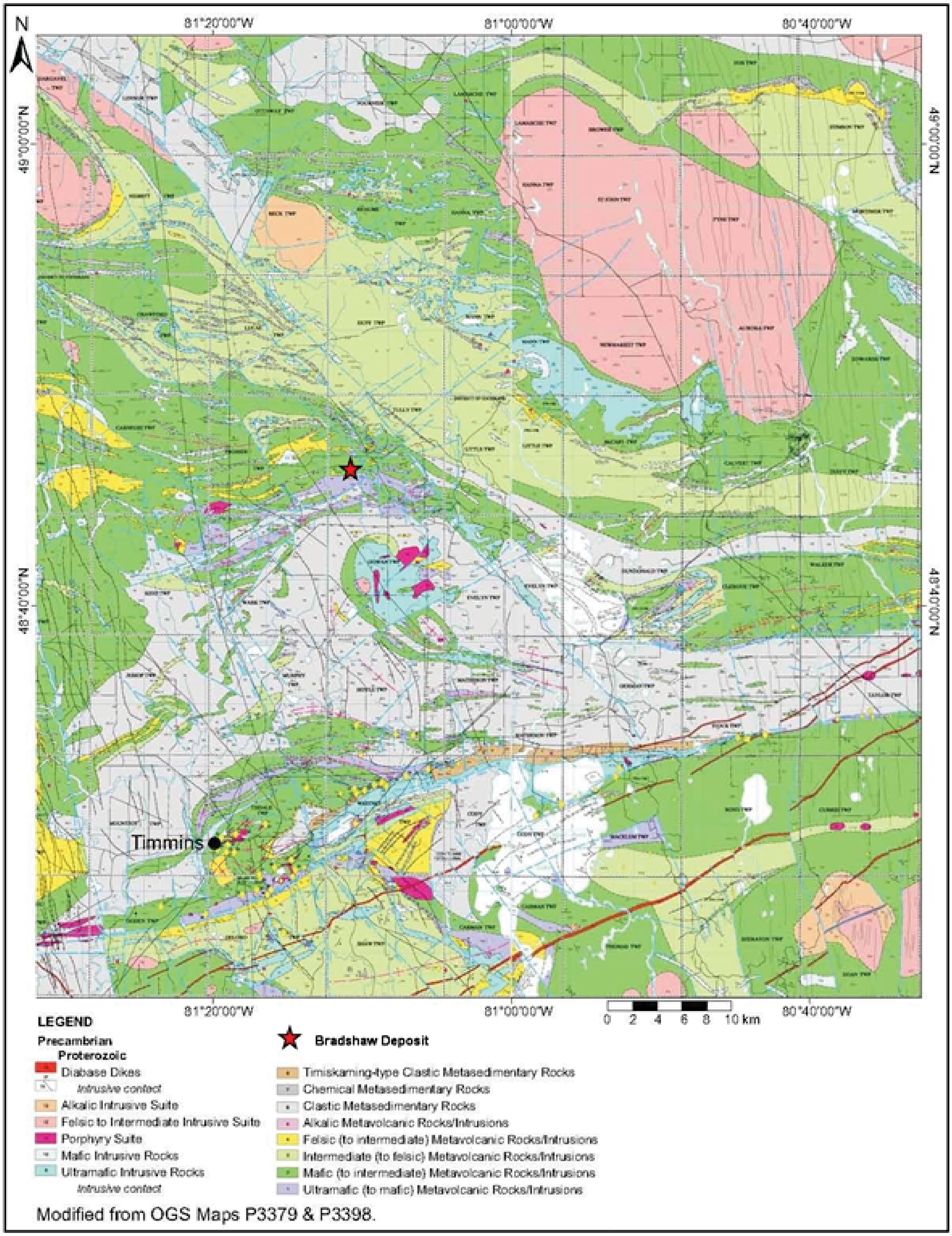
The Kidd-Munro assemblage is divisible into two distinct suites:

- A tholeiitic to komatiitic suite, which consist of komatiites, magnesium and iron-rich tholeiites; and;
- A calc-alkaline suite consisting of intermediate to felsic pyroclastic rocks, including FIIIb type rhyolites (Lesher, et al, 1986).

Rare sedimentary rocks are generally confined to narrow interflow units within the mafic volcanic rocks. Synvolcanic felsic intrusions and later diabase dykes intrude the sequence. The calc-alkaline portion of the assemblage is host to the Kidd Creek volcanogenic massive sulphide (VMS) deposit and several smaller VMS deposits located in Munro Township. The ultramafic/mafic suite is host to Gowest's Bradshaw Deposit and other gold deposits within Tully Township.

A Geological Survey of Canada ("GSC") regional airborne magnetic survey shows considerable relief within the Kidd-Munro assemblage (Dumont et al. 2002a, b). Magnetic highs appear to be coincident with unaltered ultramafic flows and magnetic lows appear to be coincident with mafic flows and altered ultramafic flows. The magnetic patterns also appear to define west verging folds, or possibly transposed stratigraphy along contact parallel faults. Airborne electromagnetic patterns appear to be following stratigraphic horizons, and drill hole data indicates that most conductive horizons are graphitic responses.

Figure 7.1: Regional Geology, North Timmins Project



The upper Tisdale (volcanic) assemblage occurs east and west of the Buskegau River Fault in the north-eastern part of Tully Township. The basal mafic / ultramafic portion of this assemblage is host to the major gold deposits of the Timmins camp, such as the Hollinger, McIntyre and Dome mines. The upper Tisdale assemblage disconformably overlies the Kidd-Munro assemblage and is comprised of intermediate and felsic, epiclastic and pyroclastic volcanic rocks of calc-alkaline affinity. The magnetic pattern over this assemblage is subdued, with low amplitude magnetic responses over stratiform gabbroic sills. Electromagnetic ("EM") responses within this assemblage are diffuse and of low conductivity. In the north-western part of Tully Township, a zone of high conductivity EM responses caused by graphite and massive pyrrhotite marks the contact between the Tisdale and Kidd-Munro assemblages.

Porcupine assemblage rocks unconformably overlie the Kidd-Munro assemblage immediately south of the Frankfield Block. The sedimentary rocks are composed predominantly of fine-grained turbiditic sedimentary rocks with minor graphitic argillite and conglomerate horizons. A detrital zircon U/Pb age of 2698 Ma (Heather et al., 1995) for similar sediments at the Kidd Creek Mine defines a maximum age of the assemblage. Porcupine assemblage rocks are also thought to occur east of the Buskegau River Fault in the east central part of the township (Berger, 2000). The magnetic pattern associated with this assemblage is subdued with stratiform electromagnetic responses.

Structural features of the bedrock are mainly interpreted from airborne magnetic surveys. Stratigraphic units as represented by their magnetic signatures generally trend east-northeast within the Kidd-Munro assemblage. This trend is also characterized by a well-developed penetrative foliation. Fold axes also appear to trend east-northeast as noted by reversals in younging directions determined from flow features. Stratigraphy parallel shear zones, such as at the Bradshaw Deposit are developed at some lithological contacts. Extensional lineations developed in the shear zones are moderately northeast plunging, a direction that is similar to lineations observed in the Timmins area (Pyke, 1982) and Hoyle Pond gold mines geology (Berger, 2000). This observation implies a similar and contemporaneous geodynamic process and possibly a similar metallogenic connotation, suggesting an untested gold potential along these structures in Tully Township.

Within the upper Tisdale assemblage, magnetic patterns indicate northwest-trending lithologies cut by east-northeast-trending late faults. Stratigraphic facings indicate younging directions towards the northeast within this assemblage (Berger, 2000). The distribution of EM conductors in the north-western part of Tully Township suggests large amplitude northwest-trending folds.

7.2 Property Geology

The following geology of the Frankfield Block has been extracted from Harron (2006) and Bradshaw (2008) with minor edits.

Holocene organic deposits of peat and black muck cover much of the map area. Underlying the organic deposits are extensive Quaternary glaciolacustrine deep water varved silts and clays of the Barlow-Ojibway Formation up to several meters thick overlying Matheson Till.

The bedrock geology of the Project is mainly derived from drill core observations and geophysical interpretations due to the extensive overburden and swamp lands characteristic of the region. The property is underlain by tholeiitic basalt flows and komatiitic basalt to peridotite flows of the Kidd-Munro assemblage. The tholeiitic basalt flows dominate the northern half of the property and the komatiitic peridotite flows the southern half (Figure 7.2). Thin (<10 m) units of pyritic graphitic argillite interflow sediments are commonly at or close to the contacts of the komatiitic peridotite flows in the north tholeiitic volcanic sequence.

In detail, the Kidd-Munro assemblage on the property consists of magnesium-rich and iron-rich tholeiites, which range from pale green-gray to dark green in colour. Textures include massive and pillowed flows with abundant flow top breccia and occasional variolitic and spherulitic horizons. Drilling has intersected thin (5-30 m) komatiitic peridotite flows intercalated in the north tholeiitic volcanic sequence. Thin (<10 m) units of pyritic graphitic argillite interflow sediments are commonly at or close to the contacts of the komatiitic peridotite flows in the tholeiitic volcanic sequence. Quartz-calcite veinlets cut the various units at all angles. Minor amounts of pyrite and pyrrhotite are common throughout the sequence and concentrations are slightly enhanced near pillow rims and siliceous flow top breccias. Depositional indicators demonstrate a steeply north dipping and north younging direction for the volcanic sequence. Highly altered ultramafic rocks, which are probably komatiitic flows as spinifex textured flow tops have been observed in drill core, occur in the southern and central portions of the Project. The ultramafic flows are generally altered to fine grained talc-serpentine-carbonate mineralogy.

Structural geology of the Bradshaw Project is largely unknown. Previous operators interpreted a north trending dextral fault at the western end of the Bradshaw Deposit. Berger (2000) suggested that the region (including the NTGP) is characterized by early northwest trending faults and later N70°E trending faults. The stratigraphy has been deformed by at least two periods of deformation, as is common in the AGB. However, the paucity of outcrops severely hampers the elucidation of the fold patterns on the property. Further interpretation of Gowest's detailed airborne magnetic survey and compilation with other exploration datasets may assist in determining the Project's structural geology.

7.3 Mineralization

7.3.1 General

Four gold mineralization areas presently exist on the Bradshaw Project (Figure 7.2).

7.3.2 Bradshaw Deposit

The Bradshaw Deposit comprises a geological Main Zone and several lesser Hanging Wall Zones. Gold mineralization in the Main Zone occurs primarily within a fractured and brecciated altered horizon previously interpreted as a shear zone in hanging wall basaltic flow rocks at or near the contact with steeply north-dipping (85°) footwall ultramafic rocks to the south (Figure 7.3).

The mineralization is not confined to narrow vein-like structures (as can be seen in many other deposits in the area) but rather in a more massive/tabular structure that is consistently present throughout the mineralized horizon. This characteristic is shared by the major past gold producers in the Porcupine camp including Hollinger, McIntyre and present producer Goldcorp at their Dome and Hoyle Pond deposits.

Within the geological Main Zone, higher-grade gold mineralization is localized along the footwall of the horizon, termed the MZ1 Zone (previously referred to as M1 Zone - Harron, 2006) and occasionally along the hanging wall of the horizon, termed the MZ2 Zone (previously referred to as M2 Zone - Harron, 2006). Both gold mineralized zones appear to rake steeply to the east based on current drill data. Their variation in widths may reflect tectonically controlled shoots or boudinage structures.

Figure 7.2: Property Geology, North Timmins Project

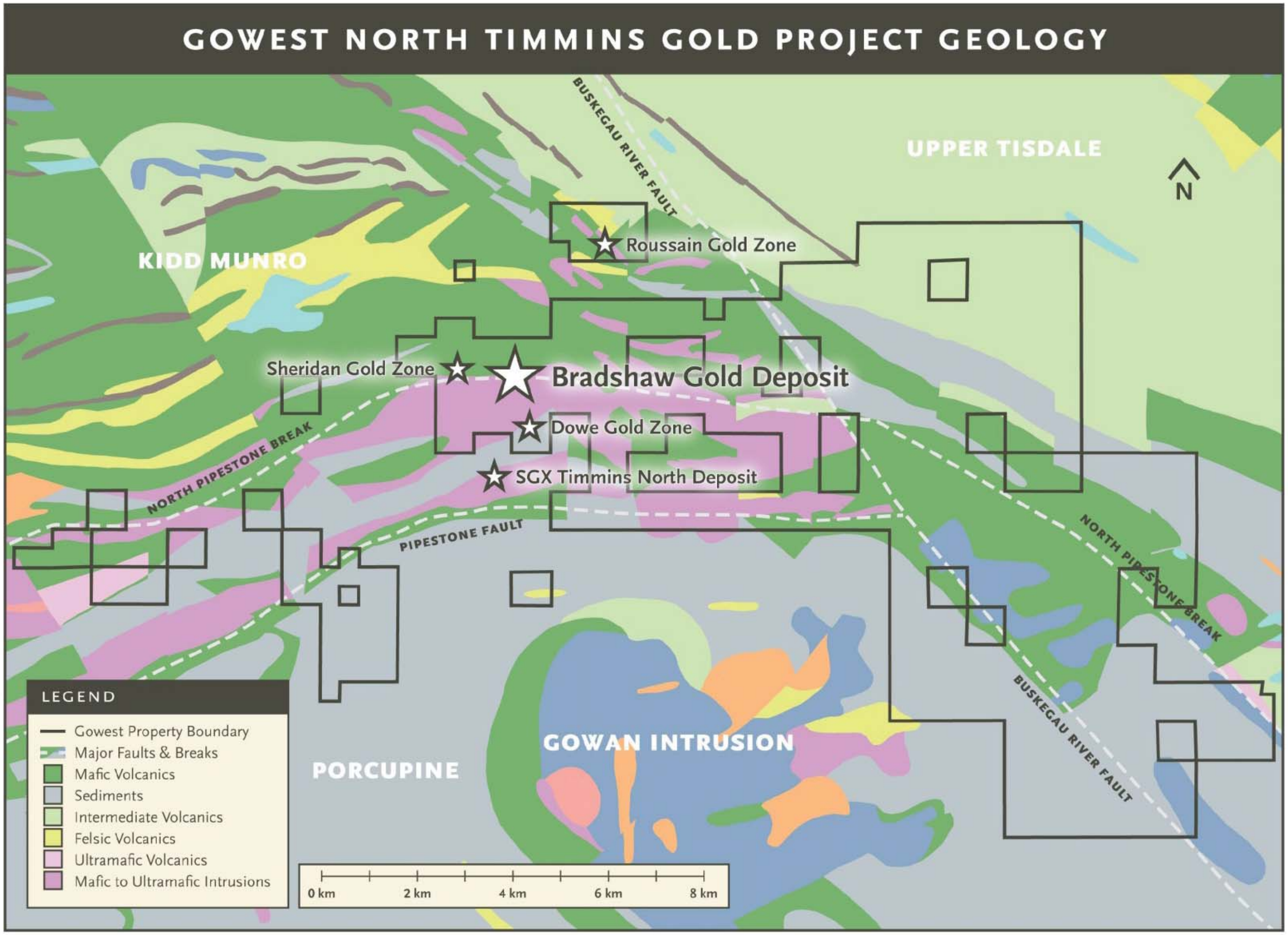
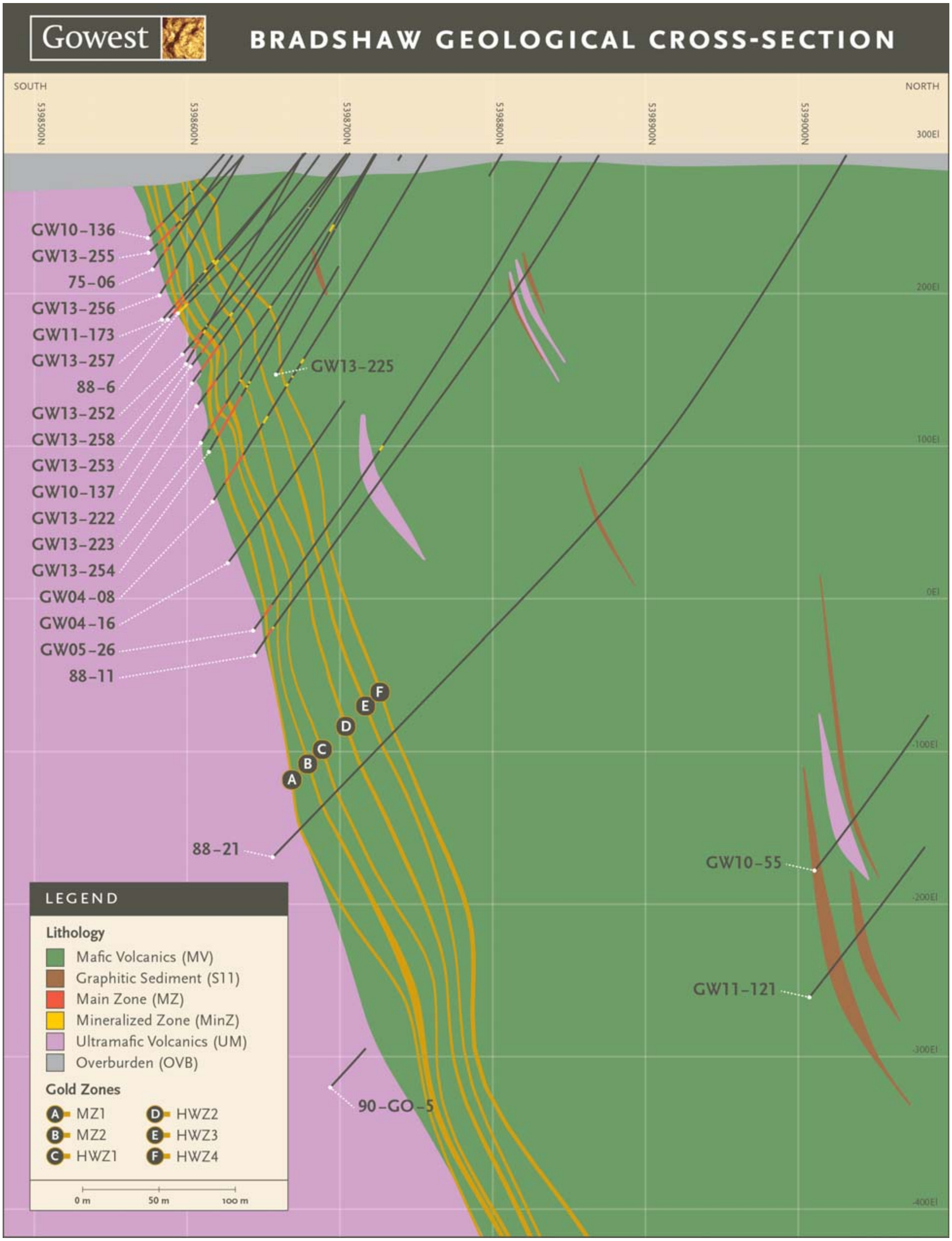


Figure 7.3: Bradshaw Deposit Geological Section (486650e)



Sporadic, anomalous to lower-grade gold mineralization is present between these subzones. Pervasive silicification, minor quartz-ankerite veining, hematite staining and presence of tourmaline generate a recognizable mauve to pinkish-grey hue for the mineralized zone. Total sulphide content of the mineralized horizon varies from 3-30% with occasional 2-5 cm wide bands of massive arsenopyrite and pyrite. Most of the sulphide component in the Main Zone is in the form of seams, bands and clots of sulphides accompanied by zones of heavy disseminations of 5-15% sulphides over 5-10 cm core lengths. The largest concentrations of arsenopyrite correspond to the highest gold concentrations. Visible gold is not a feature of this type of mineralization. Some late stage fracturing and brecciation of the mineralized horizon has caused varying amounts of sulphide remobilization (Roussain, 2004).

Similar mineralization forms multiple structures believed to be subparallel to the strike and dip of the Main Zone and are referred to as hanging wall zones as they are located immediately north of the main zone. They are highly silicified zones accompanied by intense bleaching, brecciation and quartz flooding, tourmaline, 5-10% pyrite and arsenopyrite. The overprint of silica flooding and white quartz veining makes the hanging wall zones appear different from the Main Zone but the gold is associated with the sulphide component as in the main zone. As in the Main Zone, higher concentrations of arsenopyrite give rise to higher gold values. A total of six such parallel structures (HWZ1 to HWZ6) have been identified in locations and are significant contributors to the total number of ounces of gold contained within the overall Bradshaw Deposit.

To date, the deposit has a drilled strike length in excess of 950 m, trending N070-080°E, and has been tested to a depth in excess of 1,000 m. The horizontal width of the geological Main Zone varies from 2 to 22 metres. The Bradshaw zones are from 1 to up to 15 metres in horizontal width and average 2 to 3 metres. The deposit remains open along strike and at depth.

Overburden depth along the strike length of the deposit ranges from 10 to 16 metres and averages approximately 12 metres deep.

7.3.3 Sheridan Deposit

In the northwest part of the Frankfield Block, the Sheridan Gold Deposit (formerly the Texmont Deposit) is hosted in a carbonate, hematite and sericite altered shear zone within a sequence of tholeiitic basalt flows. The shear zone strikes N086°E and dips 75° to the north. Sheridan is a sulphide mineralized zone of 3-5% disseminated pyrite and very fine microscopic arsenopyrite with quartz vein flooding. Intex reported in 1982 that the Sheridan Gold Zone contained 103,400 t averaging 7.54 g/t Au across and average width of 3 metres along a strike length of 152 metres and to a depth of 75 metres. Gowest is not treating this historic mineral resource estimate as a NI 43-101 compliant resource verified by a qualified person and the estimate should not be relied upon. Subsequent drilling by Cyprus Gold (Canada) Limited in 1991 showed that the deposit extended to at least a depth of 360 m. Gowest drilled six diamond drill holes in 2010. The two best intersections obtained were 4.1 g/t Au over a down-hole length of 13.7 metres and 4.1 g/t Au over 0.6 metre. The Gowest drilling has traced the Sheridan Zone for 250 metres and it is open along strike and at depth below 200 metres. Gowest has not prepared a mineral resource estimate for this deposit.

7.3.4 Dowe Gold Showing

The Dowe showing is located on the southeast part of the Frankfield Block (Figure 7.2). Gowest drilled a single hole GW13-242 at the showing, in 2013. The hole returned an anomalous gold value of 0.80 g/t Au over 0.7 metres in the altered mafic volcanics, 170 metres down dip of the Dowe gold zone. This zone has reported gold drill intercepts of 2.86 g/t Au over 3.38 metres, 1.8 g/t Au over 12 metres, and 1.9 g/t Au over 3.4 metres from drilling in 1997-1998 by previous owners. Mineralization is reported to occur in quartz veins within a wedge of mafic volcanic rocks enclosed in ultramafic rocks. The mafic rocks are reported to be ankeritized and slightly silicified. Visible gold has been reported, together with pyrite. No arsenopyrite has been reported. The showing is reported to have a strike length of at least 150 metres and has not been closed off.

7.3.5 Roussain Gold Showing

The showing is located 3 km northeast of the Bradshaw Gold Deposit on the North Tully claim group (Figure 7.2). Drilling by Newmont exploration in 1983 outlined a northwest trending carbonate zone in basalts on the south side of an ultramafic horizon and in association with a graphitic horizon. Gold intercepts of 7.1 g/t Au over 1.5 m, 1.65 g/t Au over 5.1 m, and 1.65 g/t Au over 1.8 metres were reported from this drilling. Gowest undertook an exploration drilling program on Roussain consisting of six holes (2,401 metres) in February-March 2013. The Gowest drilling results confirmed and exceeded the historic 1980's assays. Four gold mineralized zones were identified in hole GW13-236 with the best gold zone returning 5.01 g/t Au over 4.4 metres which included 12.00 g/t Au over 0.7 metres. The wider upper gold zone is hosted within a pyritic sedimentary unit and the three other gold zones in carbonated-silicified basalts. These three zones, which includes the 5.01 g/t Au over 4.4 metres are characterized by disseminated arsenopyrite-pyrite mineralization, which is a similar environment to that hosting the Bradshaw Gold Deposit. The Roussain gold zones are open to the southeast and at depth.

8.0 DEPOSIT TYPES

The sulphide enrichment gold deposit model best describes the mineralization of the Bradshaw Deposit.

The sulphide enrichment deposit model is characterized by a dominance of sulphide minerals over quartz veins, and is localized in shear zones adjacent to rheologically differing mafic to ultramafic volcanic rocks of tholeiitic petrochemistry. Mineralization typically comprises native gold associated with disseminated to massive arsenopyrite and vein hosted pyrite and arsenopyrite in silicified chloritic and sericitic schists, within a broad zone of potassium metasomatism and wall rock sulphidation (disseminated pyrrhotite and pyrite). Carbonatization of the wall rocks is a less conspicuous feature than silicification. Pervasive silicification and silicate alteration minerals developed within the shear zone consists of quartz, albite, chlorite, actinolite, tourmaline and amorphous carbon, suggesting a dominance of silicic and potassic alteration. Canadian examples of sulphide enrichment gold deposits include the Madsen and Starratt-Olsen deposits in the Red Lake Camp, (Durocher, 1983) and the ULU deposit in the High Lake Greenstone Belt in Nunavut. The best Ontario examples of sulphide enrichment gold zones include the gold zones of the Holloway and Holt mines about 100 km east of Timmins (Valliant and Bergen, 2008), and the flow ores of the historic giant Kerr Addison Mine about 150 km southeast of Timmins. In the Porcupine Timmins Gold Camp documented examples include the Bell Creek Mine Zone (Pressacco, 2011) and the historic Moneta Mine gold zones.

In the search for sulphide enrichment and quartz lode gold mineralization magnetic, induced polarization/resistivity (IP/RES) surveys can define favourable host environments. Alteration destroys the magnetic minerals in mafic and ultramafic rocks resulting in subdued magnetic patterns. Silica alteration results in enhanced resistivity, while the presence of arsenopyrite and other sulphide species in the quartz veins and their alteration envelopes produce a positive chargeability response. Surveys over other Canadian examples of this type of mineralization have demonstrated the utility of IP/RES and mise-à-la-masse survey methods in defining mineralization. Previous geophysical surveys on the Project have not included IP/RES surveying due to the thick clay overburden of the area, but have relied on HLEM surveys to delineate graphitic horizons in the volcanic stratigraphy (Trinder, 2011).

Typical soil geochemical surveys are not particularly effective in the Bradshaw Project area, as a result of extensive thick overburden cover (greater than 15 metres and locally up to 50 metres thick). Gowest has conducted several SGH geochemical surveys over various parts of the Project area in an effort to "see through" the deep overburden. The SGH results are being evaluated and compiled by Gowest with other exploration data sets to determine its effectiveness on the Bradshaw Project.

9.0 EXPLORATION

9.1 Airborne Geophysics

Gowest has undertaken a number of exploration campaigns almost entirely centred on the Frankfield claim block in the past. In 2009, Fugro Airborne Surveys Corp. conducted helicopter-borne DIGHEM V electromagnetic/resistivity/magnetic survey. A total of 438 line-km were flown. The details and results of the survey have been discussed in previous reports, including Ling (2012). The survey was helpful in that it appears to show that mineralization is associated with bedrock features.

Once the Transition claims were optioned in early 2011, Gowest Gold contracted Fugro Airborne Surveys to conduct a HELITEM electromagnetic and magnetic airborne geophysical survey over the North Timmins Project. It was flown between June 26th and July 9th, 2011 and amounted to 1,822.3 line km. The airborne geophysical survey was carried out to map the geology and structure of the area. Data was acquired using a HELITEM electromagnetic system, supplemented by a high-sensitivity cesium magnetometer. A GPS electronic navigation system ensured accurate positioning of the geophysical data with respect to the base map coordinates. The geophysical data obtained by Fugro was processed and interpreted for Gowest by Mark Shore a consulting geophysicist. A series of geophysical maps were produced of the survey area. These updated geophysical maps along with Ontario geological and drill hole data were utilized by Dr. Philips Thurston (Laurentian University) to produce an updated geological/structural base for the project area.

As most of the exploration has concentrated on testing the Bradshaw Deposit, full use has not been made of the airborne geophysical survey results. As Gowest is able to develop exploration campaigns to test claims away from the Frankfield Block, the results of the airborne survey may be more useful.

9.2 Soil Gas Hydrocarbon (SGH) Surveys

Gowest undertook a SGH geochemical survey in 2009 on the Frankfield Block. The technique is marketed by Activation Laboratories (ActLabs) of Ancaster, ON. The results of the survey have been discussed in some detail previously in Chapter 6, and specifically in the following items within this chapter. The results of the survey are not particularly useful by themselves and further follow-up work is required. As with the airborne magnetic survey, little follow-up has occurred because all of the Gowest effort has gone into the drill testing of the Bradshaw Deposit.

In late 2011, an SGH survey was conducted, on the Wark 1 Property, to evaluate its gold mineralization potential. A total of 680 soil samples were collected. The interpretation of the SGH survey results by Dale Sutherland of ActLabs outlined four REDOX cells have weak to moderate potential to be a gold mineralization target. An SGH survey was conducted, on the Wark 2 Property from October 7 to 13, 2011. Soil samples were collected from 130 sites on the western half of the Wark 2 Property. The interpretation of the SGH survey results outlined a strong 1,500 metres long oval REDOX cell trending east-west in the survey area. This REDOX cell was judged by Dale Sutherland of ActLabs to have a strong potential to be a base metal mineralization target. The details and results of these SGH surveys were filed for assessment with the Ontario Ministry of Northern Development Mines (MNDM) in 2012.

A third larger SGH survey consisting of 2,320 samples was carried out on the Transition claims of the Pipestone East Property, from July to September 2011. The interpretation of the SGH survey results by Dale Sutherland of ActLabs outlined a well-defined gold halo anomaly in the central portion of the northern half of the survey area. This oval REDOX cell is very large 2.5 x 4 km in size. A more intense nested halo REDOX cell (800 metres x 1,000 metres) occurs in the eastern central portion of the larger cell and was interpreted to be a strong gold mineralization target. The Pipestone East SGH survey was filed for assessment with the Ontario MNDM in 2013.

In the fall of 2012, geochemical soil gas hydrocarbon (SGH) surveys were conducted, on the Gowan, Prosser and Wark 2 blocks to evaluate their gold mineralization potential. A total of 50 samples were collected on each of the Gowan and Prosser surveys. The interpretation of the Gowan SGH survey results by Dale Sutherland of ActLabs outlined a sharp 250 metres long oval REDOX cell in the south central portion of the survey area. This REDOX cell was interpreted to have a strong potential to be a gold mineralization target. Dale Sutherland's interpretation of the Prosser SGH survey results outlined a moderate to strong 350 metres long oval REDOX cell in the southeast central portion of the survey area and it was judged to be a good gold mineralization target.

The third 2012 SGH survey consisted of 304 Soil samples being collected on the northern claim block and eastern half of the Wark 2 Property. The interpretation of the SGH survey results by Dale Sutherland of ActLabs outlined two REDOX cells. The first is a strong oval REDOX cell with an east-west strike length of approximately 750 metres and a width of 400 m. It is interpreted to have good gold mineralization potential and is located in the south-eastern part of the Wark 2 Property. The second is a strong oval REDOX cell, 450 metres long and 200 metres wide. This east-west smaller oval REDOX cell occurs in the eastern part of the Wark 2 Property. The results of the three 2012 SGH surveys were filed for assessment with the Ontario MNDM in 2013.

Gowest is evaluating the results of the SGH surveys on the various properties in conjunction with the 2011 airborne geophysical survey to develop future drill targets.

9.3 Geology Surveys

Gowest Gold conducted geological mapping on the Prosser and Gowan claim blocks in 2011. This was carried out to satisfy assessment work requirements and filed with the Ontario MNDM on November 15, 2011. No bedrock exposure was located on the Gowan block and a single mafic volcanic exposure was found along the northern claim boundary of the Prosser block. Rock samples for gold analysis were not collected during the geological surveys.

10.0 Drilling

10.1 General

Drilling of the Frankfield Property, and in particular the Bradshaw Deposit, has continued for many years. Details of the pre-2004 drilling are sketchy and it is likely that sampling and quality control/quality assurance were not up to current standards.

10.2 Diamond Drilling 2004 to 2011

There has been a significant amount of NQ diamond drilling since 2004. Drilling in the period 2004 to 2011 is summarized in Table 10.1 and details of the various drilling programs are summarized by Trinder (2011).

Table 10.1: Diamond Drilling 2004 To 2011 Frankfield Property – Gowest

Program	Drill Hole Series	Total Drill Holes	Total Meters
2004	GW04-01 to GW04-22, GW04-25	25	6,538
2005	GW05-01 to GW05-03, GW05-23 to GW05-24, GW05-26 to GW05-30	10	3,989
2006	GW05-31 GW06-32 to GW06-38	8	1,407
2008	GW08-39 to GW08-44	6	1,275
2010	GW10-45 to GWH10-120, GW10-60WA, GW10-60WB, GW10-122 to GW10-138, GW10-140 to GW10-146	102	30,621
2011*	GW11-121, GW11-139, GW11-147 to GW11-162	18	8,586
Total		**169**	**52,416**

Note: As at April 24, 2011 (Trinder, 2011)

During the 2004 to 2008 drilling campaigns the drill hole collars were surveyed by Talbot Surveying of Timmins, Ontario with a Real-Time GPS.

10.3 Drilling 2011 to 2012

Gowest continued drilling on the Frankfield Block in 2012-2013 and details of the NQ drilling are summarized by Gow (2012). A further 51 holes were drilled for an aggregate depth of 16,148 metres and details are set out in Table 10.2.

Table 10.2: Diamond Drilling 2011 To 2012 Frankfield Property – Gowest

Program	Drill Hole Series	Total Drill Holes	Total Metres
2011	GW11-163 to GW11-199	37	9,743
2012	GW12-200 to GW12-203, GW12-207, GW12-208, GW12-211, GW12-215 to GW12-221	14	6,405
Total		**51**	**16,148**

All the drilling on the Frankfield Block since 2004 drilling has been carried out by Norex Drilling Limited (Norex) of Porcupine, ON with the exception of five deep holes (5,299 metres) carried out by Bradley Brothers of Timmins, ON. Both are reputable drilling contractors with good reliable records.

10.4 Drilling 2013 to 2014

An infill diamond drilling program was conducted in 2013 at the Bradshaw Deposit. It consisted of a further 47 holes totaling 8,514 m. The objective of the infill drilling program was to select two or three areas with good gold tenor and continuity in preparation for future underground test mining (bulk sampling). The central shallow (to a maximum depth of 200 m) portion of the Bradshaw Deposit was targeted. A small diamond drilling program consisting of six holes totaling 2,528 metres was completed from July 24 to September 8, 2014. Its objective was to extend the limits of indicated resources in select areas.

Details of the 2013-2014 drilling and the most recent drilling campaigns are set out in Table 10.3.

Table 10.3: Frankfield Block Drill Hole Data 2013-2014

Hole ID	Northing	Easting	Elevation	Azimuth (°)	Dip (°)	Length (m)	Core Size
GW13-222	5398725	486660	291.80	183	-56	203	NQ
GW13-223	5398725	486660	291.80	183	-60	224	NQ
GW13-224	5398740	486640	291.00	183	-57	230	NQ
GW13-225	5398740	486640	291.00	183	-61	257	NQ
GW13-226	5398725	486620	291.90	183	-57	230	NQ
GW13-227	5398725	486620	291.60	183	-61	239	NQ
GW13-228	5398740	486680	291.30	183	-57	221	NQ

Hole ID	Northing	Easting	Elevation	Azimuth (°)	Dip (°)	Length (m)	Core Size
GW13-229	5398740	486680	291.50	183	-62	242	NQ
GW13-230	5398742	486720	291.00	183	-57	220	NQ
GW13-231	5398742	486720	290.90	183	-62	236	NQ
GW13-232	5398610	486580	291.60	180	-50	80	NQ
GW13-233	5398630	486580	291.10	180	-50	98	NQ
GW13-234	5398630	486580	291.10	180	-60	112	NQ
GW13-242	5397655	487015	295.60	180	-63	500	NQ
GW13-243	5397600	485400	292.80	180	-55	257	NQ
GW13-244A	5398750	486700	291.40	180	-75	74	NQ
GW13-244	5398750	486700	291.40	180	-75	371	NQ
GW13-245	5398618	486700	291.70	180	-75	119	NQ
GW13-246	5398683	486700	291.00	183	-52	140	NQ
GW13-247	5398708	486700	291.50	183	-57	170	NQ
GW13-248	5398708	486700	291.10	183	-65	212	NQ
GW13-249	5398708	486710	291.80	180	-52	170	NQ
GW13-250	5398643	486680	291.40	180	-48	92	NQ
GW13-251	5398643	486680	291.40	180	-60	113	NQ
GW13-252	5398705	486660	292.00	180	-51	170	NQ
GW13-253	5398705	486660	292.00	180	-56	173	NQ
GW13-254	5398725	486660	291.70	183	-59	221	NQ
GW13-255	5398636	486640	290.90	180	-48	89	NQ
GW13-256	5398636	486640	290.90	180	-60	107	NQ
GW13-257	5398675	486640	291.50	183	-52	140	NQ
GW13-258	5398675	486640	292.10	183	-62	158	NQ
GW13-259	5398628	486600	291.40	180	-48	95	NQ
GW13-260	5398628	486600	291.40	180	-60	109	NQ
GW13-261	5398660	486600	291.80	183	-59	143	NQ
GW13-262	5398618	486560	291.60	180	-45	86	NQ
GW13-263	5398618	486560	291.60	180	-53	89	NQ
GW13-264	5398656	486560	291.70	180	-56	152	NQ
GW13-265	5398713	486610	290.30	183	-56	206	NQ
GW13-266	5398722	486760	291.70	183	-57	191	NQ
GW13-267	5398758	486760	290.40	183	-57	233	NQ
GW13-268	5398672	486780	290.20	180	-50	110	NQ
GW13-269	5398723	486780	291.90	183	-55	179	NQ
GW13-270	5398775	486780	291.70	183	-57	251	NQ
GW13-271	5398657	486800	290.40	180	-45	92	NQ
GW13-272	5398657	486800	290.40	180	-60	101	NQ
GW13-273	5398718	486800	291.90	183	-55	167	NQ

Hole ID	Northing	Easting	Elevation	Azimuth (°)	Dip (°)	Length (m)	Core Size
GW13-274	5398768	486800	291.80	180	-52	212	NQ
GW13-275	5398768	486800	291.80	183	-57	230	NQ
2013 Total						**8,514**	
GW14-276A	5398768	486500	291.00	180	-61	113	NQ
GW14-276	5398936	486502	290.50	180	-62	540	NQ
GW14-277	5398792	486496	291.10	183	-61	378	NQ
GW14-278	5398948	486608	291.60	180	-70	579	NQ
GW14-279	5398934	486949	292.50	180	-66	447	NQ
GW14-280	5398710	486335	292.10	180	-61	360	NQ
GW14-281	5398666	486450	292.00	180	-61	224	NQ
2014 Total						**2,528**	

The above 2013-2014 drilling was carried out by Norex Drilling limited (Norex) of Porcupine, Ontario.

Since 2010, drill hole collars were positioned by Gowest personnel with a hand held GPS unit. All diamond drill holes were aligned by drilling crews employing an Azimuth Pointing System (APS) rented from Reflex instruments of Timmins Ontario. The Azimuth Pointing System (APS) is a GPS based compass that provides a True North Azimuth measurement and position. Since the APS is not using the earth's magnetic field to determine the azimuth, it is not affected by ferrous anomalies (metal) from the ground or surrounding structures. The APS uses two antennas to calculate an azimuth solution. The APS surveys the drill hole collar coordinates and elevation in UTM coordinates (NAD83) utilizing total station GPS instrumentation. This data was recorded and subsequently inputted by Gowest personnel into a Surpac computer database.

As a verification of the collar co-ordinates, Gowest resurveyed approximately 80% of the holes since 2010 using the APS. Three survey bars were installed on bedrock at the Bradshaw Deposit drilling area in October 2014. A further check of ten holes was then conducted by Talbot Surveying of Timmins, Ontario with a Real-Time GPS from the survey bars. The Talbot surveyed drill hole collar co-ordinates were within ± 2 metres of the Gowest APS surveyed collar co-ordinates.

During drilling, the contractor conducted down hole surveying utilizing a Reflex EZ-Shot®, an electronic single shot instrument. It accurately measures six parameters in one single shot; azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength and temperature. Single shot tests were taken 15 metres or so below the casing and every 50 metres down the drill hole. Casing was left in each of the holes and the stand pipes were capped.

Industry standard core sampling protocols are used by Gowest on all drill holes. These protocols are documented in hard copy Gowest sampling procedures, which are described in this section.

At the drill site, the drilling contractor places drill core into wooden tray boxes along with 'marker blocks" to indicate measured distances down the drill hole from the collar. During drilling programs, drill core is collected by Gowest technicians at the drill sites or the drill access trail every drilling day and moved to a secure logging facility. The secure logging facility is located at 115 Jubilee Avenue East Timmins, Ontario.

At the logging facility, the length of drill core recovered was compared to the position of depth markers in the core boxes by a technician in order to check for misplaced markers and to calculate the amount of core loss, if any. Prior to lithological logging and sampling, a Gowest geo-technician photographs the core, cleans the core if necessary, completes a geotechnical log of core recovery, RQD and fracture analysis measurements, and records magnetic susceptibility data.. The core is then logged and sampled by qualified geologists. Geological descriptions of the core and sampling intervals with corresponding identifier numbers were entered onto a "diamond drill log record" captured on a laptop computer. Sampling of the core was based on visual observations of sulphide mineralization and samples were collected within lithologically homogeneous intervals with due regard for varying mineralogy and textures. Sample intervals did not cross geological boundaries. Generally, the sample length within mineralized zones was on the order of 0.5 to 1.0 metre or less. The NQ core selected for sampling was split in half by a hydraulic splitter and a half bagged with the first part of a three-part assay tag bearing a unique identifier number. The other half of the core was stored at the logging facility with the second part of the three part assay tag bearing an identical unique identifier number placed in the core box at the beginning of the sample interval. Records of the sampled intervals and sample numbers are recorded in the computerized drill logs, and the third part of the assay tag is filed.

The spilt drill core is securely stored at the Norex Drilling office/core storage facility, 7210 Highway 101 East in Timmins and the whole core is stored outside at Rob Roy Contracting, 6033 King Street in Timmins.

The work was completed by experienced personnel with a history of work in the Timmins camp. In the opinion of Mr. Gow, Gowest personnel used industry best practices in the collection, handling and management of drill core assay samples. There is no evidence that the sampling approach and methodology used by Gowest introduces any sampling bias or contamination.

10.5 Drilling Outside the Frankfield Block

Gowest conducted exploration drilling outside of the Frankfield Block in 2012-2013 and this drilling is summarized in Table 10.4. On the southeast portion of the Tully East Property, five holes (GW 12-204 to 209) were drilled totaling 1,172 m. They were drilled from February 7 to March 8, 2012. These holes tested airborne electromagnetic conductors at or near lithological contacts thought to be similar settings as the Bradshaw Deposit. Although the holes returned no significant gold values, they provided valuable information on the geology of the areas tested.

A three hole diamond drilling program totaling 1,291 metres was conducted on the Pipestone East Property (Transition portion). The three holes were labelled as GW12-212/213/214 and were drilled from March 7 to April 20, 2012. Drill hole GW12-212 targeted an EM conductor proximal to Cross Lake Fault, Hole GW12-213 targeted the north portion of a large SGH gold anomaly coinciding with a magnetic low-magnetic high contact and hole GW12-214 targeted central portion of a large SGH gold anomaly and a weak EM conductor. The holes returned no significant gold values, but bole GW12-213 intersected strongly carbonate-sericite altered volcanic rocks about a quartz vein zone which is encouraging for gold mineralization.

A 2013 winter exploration drilling program of six holes (GW13-235 to -240) totaling 2,401 metres was conducted on the Tully North claim block. The program was geared to investigate historic reports of several gold values from drilling in the early 1980s, which included a reported intersection of 7.1 g/t Au over 1.5 m. The drill results confirmed and exceeded the historic 1980s assays. Four gold mineralized zones were identified in hole GW13-236, with the best gold zone returning 5.01 g/t Au over 4.4 m, including 12.00 g/t Au over 0.7 m. The wider upper gold zone is hosted within a pyritic sedimentary unit and the three other gold zones in carbonated-silicified basalts. These latter three zones, which includes the 5.01 g/t Au intercept over 4.4 metres are characterized by disseminated arsenopyrite-pyrite mineralization, which is a similar environment to that hosting the Bradshaw Deposit. Drill hole GW13-238, located 250 metres southeast and along strike of GW13-236, intersected several gold values that are interpreted as extensions of the gold zones in GW13-236. Two of the gold zones are open to the southeast and all the zones are open at depth below hole GW13-236, which penetrated the lowest zone at an estimated vertical depth of approximately 240 m.

Hole GW13-241, a shallow hole (225 metres long) drilled 1.7 km east of the Bradshaw Deposit on the Tully East Property, intersected a mafic volcanic-ultramafic volcanic contact. This appears to be the eastern extension of the deposit stratigraphy. The hole intersected anomalous arsenic values at the contact, but no significant gold values.

Table 10.4: Drill Hole Data Tully East, Pipestone East and Tully North Properties

Hole ID	Northing	Easting	Elevation	Azimuth (°)	Dip (°)	Length (m)	Core Size
GW12-204	5395725	493250	295	180	-55	263	NQ
GW12-205	5395850	492249.2	302.5	200	-55	269	NQ
GW12-206	5396450	492800	295	180	-60	257	NQ
GW12-209	5396575	492800	295	180	-55	81	NQ
GW12-210	5396575	492800	295	180	-55	302	NQ
GW12-212	5395672	494797.4	306.6	225	-55	530	NQ
GW12-213	5394748	497001	340.6	180	-57	372	NQ
GW12-214	5394340	497001.3	329.7	180	-57	450	NQ
GW13-235	5401368	488540	290	225	-52	302	NQ
GW13-236	5401410	488583	290	225	-62	500	NQ
GW13-237	5401553	488445	287	225	-55	401	NQ
GW13-238	5401247	488754	290	225	-60	347	NQ
GW13-239	5401915	488475	290	225	-50	501	NQ
GW13-240	5401720	488285	290	225	-50	350	NQ
GW13-241	5398925	488850	291.9	210	-50	225	NQ
Total						5,407	

Core handling procedures for the drilling outside the Frankfield Block was the same as for the Frankfield Block discussed above.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation Prior to 2010

The following has been extracted from Trinder (2011).

Security of samples prior to dispatch to the analytical laboratory was maintained by limiting access of un-authorized persons to the secure core handling facility. Detailed records of sample numbers and sample descriptions provided integrity to the sampling process. Labelled samples packed in sealed bags robust enough to survive the journey to the assay laboratory also provided sample integrity. The assay laboratory completed sample preparation operations at their locations, and employed bar coding and scanning technologies that provided complete chain of custody records for every sample.

The security and integrity of the samples submitted for analyses was un-compromised, given the secure (fenced) core handling location, adequate record keeping, prompt expediting of samples, and the analytical laboratories' chain of custody procedures.

Assaying of the samples from the 2004 to 2008 drill programs was completed by Swastika Laboratories Limited (Swastika), P.O. Box 10, 1 Cameron St., Swastika, Ontario, P0K 1T0. Swastika participates in the Proficiency Testing Program for Mineral Analysis Laboratories, a testing program conducted bi-annually by the Standards Council of Canada. Swastika is the holder of a Certificate of Laboratory Proficiency. Sample preparation follows industry best practices and procedures. The analytical methods used are routine and provide robust data associated with a high degree of analytical precision.

Sample preparation at Swastika starts comprised drying of the samples and crushing to ½ inch in a jaw crusher and then to -10 mesh in a roller crusher. The sample was split with a Jones riffle, and 350 g of material taken for analysis; the remainder was placed in a numbered plastic bag and stored. The 350 g sample was then pulverized (85-95% passing minus 150 mesh) and homogenized, and was then ready for assay. Compressed air was used to clean the equipment between samples, and the roller crusher is also cleaned with a wire brush. Barren material was crushed between sample batches. All Gowest samples were analysed for Au using fire assay/atomic absorption (FA/AA) techniques on 30 gram samples. Samples that returned Au values greater or equal to 10 g/t Au were re-assayed by FA/gravimetric methods using a 30 g sample.

In addition to standards submitted by Gowest, Swastika conducted check assays on 10% of the samples to monitor assay repeatability, and analysed a second pulp for samples that return high assays. They also analysed their own standards and blanks with every batch of samples. Swastika's employees are independent from Gowest and Gowest personnel were not involved in sample preparation and analysis.

The security, sample collection, preparation and analytical procedures undertaken on the Frankfield Gold project during the 2004 to 2008 drill programs is considered to conform to industry standards.

11.2 Gowest 2010 To 2013 Sample Preparation, Analysis and Security

The following is a description sample preparation, analyses and security protocols and procedures utilized by Gowest for the 2010 to 2013 drill programs, updated and as previously discussed by Trinder, (2011).

Security of samples prior to dispatch to the analytical laboratory was maintained by limiting access of un-authorized persons to the secure core handling facility. Detailed records of sample numbers and sample descriptions provided integrity to the sampling process. Labelled samples packed in sealed bags robust enough to survive the journey to the assay laboratory also provided sample integrity. The assay laboratory completed sample preparation operations at their locations, and employed bar coding and scanning technologies that provided complete chain of custody records for every sample.

It is considered that the security and integrity of the samples submitted for analyses was un-compromised, given the secure core handling and storage locations, adequate record keeping, prompt expediting of samples, and the analytical laboratories' chain of custody procedures.

Samples are delivered to ALS Minerals (ALS) Timmins branch laboratory, 2090 Riverside Drive, Unit 10, Timmins, Ontario. Samples were prepared at the Timmins facility and sample pulps were forwarded to the ALS Mineral Laboratory in North Vancouver, British Columbia for analysis. The Timmins branch laboratory is individually certified to standards within ISO 9001:2008. The North Vancouver analytical facility is individually certified to standards within ISO 9001:2008 and has received accreditation to ISO/IEC 17025:2005 from the Standards Council of Canada (SCC) for methods including: Fire Assay Au by Atomic Absorption (AA); Fire Assay Au and Ag by Gravimetric finish; Aqua Regia Ag, Cu, Pb, Zn and Mo by AA and Aqua Regia Multi-element by ICP and MS. Sample preparation follows industry best practices and procedures. The analytical methods used are routine and provide robust data associated with a high degree of analytical precision.

At the Timmins facility, the sample was logged in the tracking system, weighed, dried and finely crushed to better than 70 % passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of up to 1000 g was taken using a riffle splitter and pulverized to better than 85 % passing a 75 micron (Tyler 200 mesh) screen. Compressed air was used to clean the equipment between samples. Barren material was crushed between sample batches. ALS then forwarded a split of the sample pulp to the North Vancouver Mineral Laboratory for analysis.

Gowest requested the following analyses on all drill core samples in the period 2010 to 2012:

- Gold Fire Assay - AAS Finish (ALS Code Au-AA23)
  - A 30 gram prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, is quartered with 6 mg of gold-free silver and then cupelled to yield a precious metal bead.
  - The bead is digested in 0.5 mL dilute nitric acid in the microwave oven, 0.5 mL concentrated hydrochloric acid is then added and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 4 mL with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.
  - Lower detection limit: 0.005 ppm; Upper detection limit: 10 ppm (10 g/t Au)
- Multi-Element ICP-AES Analysis (ALS Code ME-ICP41)
  - A prepared sample is digested with Aqua Regia in a graphite heating block. After cooling, the resulting solution is diluted to 12.5 mL with deionized water, mixed and analyzed by inductively coupled plasma-atomic emission spectrometry. The analytical results are corrected for inter-element spectral interferences.
  - Partial leach.
- Bulk Sample Density (ALS Code OA-GRA08)
  - The core section (up to 6 kg) is weighed dry. The sample is then weighed while it is suspended in water. The specific gravity is calculated from the following equation:

    SG= Weight in air (g) / (Weight in air (g) - Weight in Water (g))
  - Conducted on all samples up to hole GW12-213.

Over-limit results (gold, arsenic and sulphur) are analysed by the following methods:

- Gold Fire Assay - Gravimetric Finish (ALS Code Au-GRA21)
  - A 30 gram prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents in order to produce a lead button. The lead button containing the precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold.
  - Lower detection limit: 0.05 ppm; Upper detection limit: 1000 ppm
- Ore-Grade Multi-Element ICP-AES Analysis (ALS Code ME-OG46)
  - A prepared sample is digested in 75% aqua regia for 120 minutes. After cooling, the resulting solution is diluted to volume (100 mL) with de-ionized water, mixed and then analyzed by inductively coupled plasma - atomic emission spectrometry or by atomic absorption spectrometry.
- Total Sulphur - Leco Analysis (ALS Code S-IR08)
  - The sample is analyzed for Total Sulphur using a Leco sulphur analyzer. The sample (0.01 to 0.1 g) is heated to approximately 1350 °C in an induction furnace while passing a stream of oxygen through the sample. Sulphur dioxide released from the sample is measured by an IR detection system and the Total Sulphur result is provided.
  - Lower detection limit: 0.01%; Upper detection limit: 50%.

Gowest had ALS Minerals carry out all the above analyses with the exception of Bulk Sample Density on all drill core samples during the 2013 drilling campaign.

In addition to routine screen tests, sample preparation quality was monitored internally at ALS Minerals through the insertion of sample preparation duplicates. For every 50 samples prepared, an additional split was taken from the coarse crushed material to create a pulverizing duplicate. The additional split was processed and analyzed in a similar manner to the other samples in the submission.

Internal quality control samples including certified reference materials, blanks, and duplicates were inserted within each analytical run. The blank was inserted at the beginning, standards were inserted at random intervals, and duplicates were analyzed at the end of the batch. The minimum number of quality control samples required to be inserted were based on the rack size specific to the method.

All ALS Minerals analytical facilities in North America participate in round robin and external proficiency tests for the analytical procedures routinely done at each laboratory. The laboratories also routinely participate in proficiency tests organized by the Canadian Certified Reference Materials Projects, Geostats and a number of independent studies organized by consultants for specific clients.

11.3 Gowest 2014 Sample Preparation, Analysis and Security

The following is a description sample preparation, analyses and security protocols and procedures utilized by Gowest for the 2014 drill program. Security protocols and procedures were the same as discussed in the section above.

Split core samples were delivered directly by Gowest personnel to Activation Laboratories Ltd (ActLabs) Timmins branch laboratory, 1752 Riverside Drive, Timmins, Ontario. Samples were prepared and analyzed at the Timmins facility. The Timmins branch laboratory is individually certified to standards within ISO 9001:2008. Sample preparation follows industry best practices and procedures. The analytical methods used are routine and provide robust data associated with a high degree of analytical precision.

Upon the samples arriving at the ActLabs facility, they are examined for integrity, each sample is logged in the tracking system, weighed, and dried. The entire sample is crushed up to 90% passing a nominal minus 10 mesh (1.7 mm), mechanically riffle split to obtain a representative sample(250 g) and then pulverized (mild steel) to at least 95% minus 150 mesh (105 microns). Quality of crushing and pulverization is routinely checked as part of our quality assurance program.

Gowest requested the following analyses on all drill core samples in 2014: Gold Fire Assay - AAS Finish (ActLabs Code 1A2), Multi-Element ICP-AES Analysis (ActLabs Code 1E2) with all pulp samples having gold values greater than 10 ppm being re-assayed by Gold Fire Assay - Gravimetric Finish (ActLabs Code 1A3). The descriptions of these analyses are similar to the ALS analysis in the above section.

ALS and ActLabs employees are independent from Gowest and Gowest personnel are not involved in sample preparation and analysis.

The security, sample collection, preparation and analytical procedures undertaken on the Bradshaw Project during the 2010 to 2014 drill programs are considered by Montgomery to conform to industry standards. No drilling, sampling or recovery factors have been identified that could result in sampling bias or otherwise materially impact the accuracy and reliability of the assays and, hence, the resource database.

11.4 Gowest Quality Control 2010 to 2012

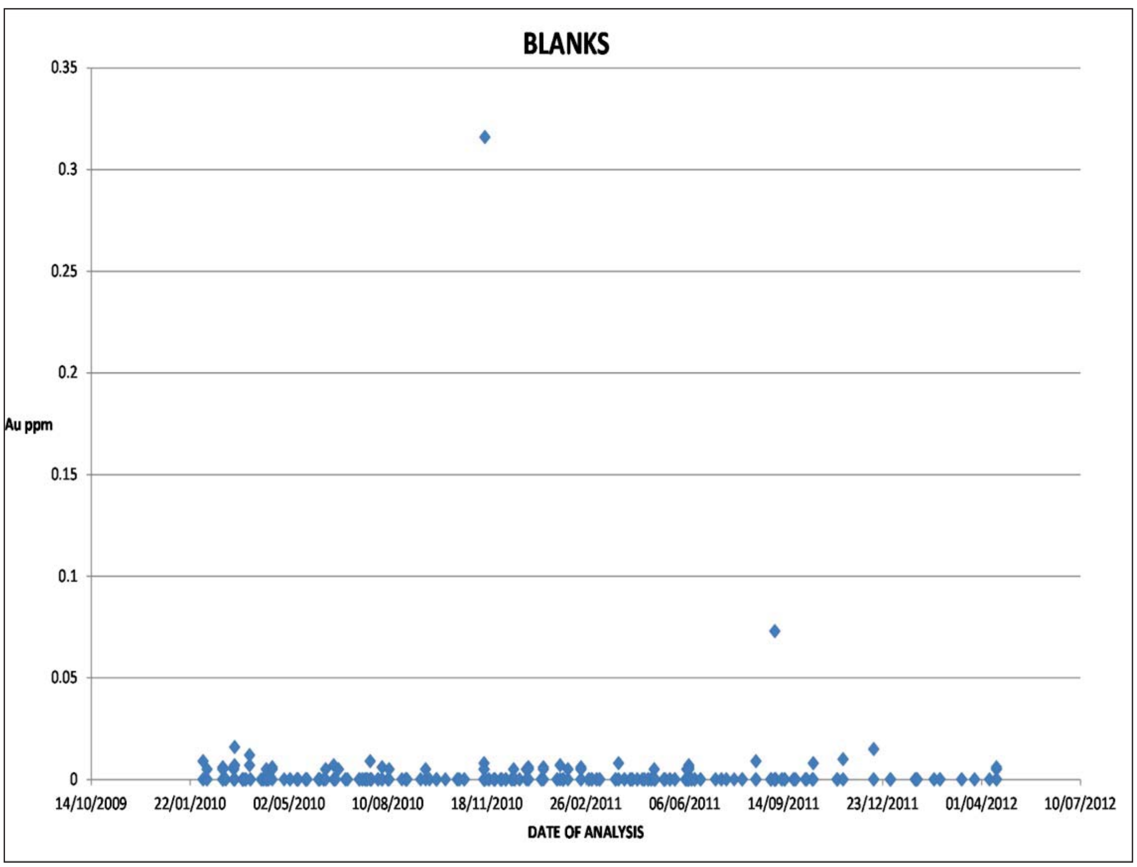
The monitoring and assessment of QA/QC data attempts to provide adequate confidence that sample and assay data obtained from these laboratories can be used for resource estimation. Gowest has implemented formal analytical quality control measures since 2004. Details of the 2004-2008 QA/QC sampling protocol are summarized in Trinder (2011).

11.4.1 Blanks

Gowest inserted a blank into the sample stream at a rate of about 1 in 20 samples. A total of 460 blank samples were inserted during the 2010 to 2012 drilling campaign which represents about 3.9% of the sample database for this period. The blank material used was pre-pulverized silica flour.

Of the 460 blank samples analysed from 2010 to 2012 at the ALS laboratory, 95.00% correctly identified the blank sample as having a gold content below or at the lower limit of detection (0.005 ppm Au). An additional 4.35 % of the analyses identified the blank sample as containing less than or equal to 0.015 ppm Au. Only three of the blank material values failed (>0.015 ppm Au, three times detection limit) which represents about 0.65 % of all the blank samples submitted for this period (January 2010 to June 2012). Although any failure during a QA/QC program should be investigated, the sizes of the failures were not enough to be of a serious concern for the purposes of this report.

Figure 11.1: Blanks



The results imply that the lab has minimal cross sample contamination, or analytical error in the assaying of blank material.

11.4.2 Standards

A certified standard was inserted into the sample stream at a rate of about 1 in 20 samples. A total of 472 standard samples were inserted during the drilling campaign which represents about 4 % of the sample database for this period. Three certified standards are currently used by Gowest for the QA/QC assessment of the ALS laboratory (see Table 11.1). Standard OREAS-19A has the largest number of assays and was used throughout the 2010-2012 drilling campaign while standards OREAS-16A and OREAS-16B were introduced in early 2011. All three standards were obtained from Ore Research and Exploration Pty Ltd (ORE) of Australia through Analytical Solutions Ltd. of Toronto Ontario.

They range in certified mean grade from 1.81 to 5.49 g/t Au and represent well the gold grades of the Bradshaw Deposit. Control plots for the assaying of each standard by ALS Laboratory are presented in Figure 11.2 to Figure 11.4.

Table 11.1: Standards 2010 To 2014 North Timmins Project Drilling

Standard	No. of Analyses	Certified Grade	St. Dev	+ 3 St.Dev	-3 St. Dev
OREAS 16A	141	1.81	0.18	1.63	1.99
OREAS 16B	145	2.21	0.07	2.00	2.42
OREAS 19A	489	5.49	0.10	5.19	5.79

The low-grade OREAS16A has an accepted value of 1.81 g/t Au with a between labs 99th confidence of 0.18 g/t Au. The mean grade of the QA/QC samples submitted was 1.81 g/t Au, equal to the accepted certified value and within the confidence level set for between labs. There were no failures within the QA/QC sample suite submitted (Figure 11.2). Overall, there is some variance in the sample results throughout the 2010-2012 drill campaign, but there is no drift evident.

The medium-grade OREAS16B has an accepted value of 2.21 g/t Au with a between lab's 99th confidence of 0.07 g/t Au. The mean grade of the QA/QC samples submitted was 2.20 g/t Au, very slightly below the accepted value and within the confidence level set for between labs. There were no failures within the QA/QC sample suite submitted (Figure 11.3). Overall, there is some variance in the sample results throughout the campaign, but there is no drift evident.

Standard OREAS-19A has the largest number of assays and was used exclusively for the quality control of drill holes GW10-45 to GW10-110. The higher-grade OREAS19A has an accepted value of 5.49 g/t Au with a between labs 99th confidence of 0.10 g/t Au. The mean grade of the QA/QC samples submitted was 5.46 g/t Au, very slightly below the accepted value and within the confidence level set for between labs (Gow, 2012).

While there are some occasions in which Standard OREAS-19A assays were beyond -3SD, 93% of samples are within 3SD of the certified standard grade of 5.49 ppm Au. Of the 23 samples outside of 3SD, 19 are below -3 SD indicating that there is potential for underreporting of gold grades (Figure 11.4). Sample batches associated with failed standards were re-assayed as part of the ongoing QA/QC practice of checking the standard assay values against their expected values as sample data is received from the lab. Eleven batches were flagged and re-assayed, and the re-assay results showed good repeatability with the first results in all cases, so no further action was required.

Figure 11.2: Standard 16A

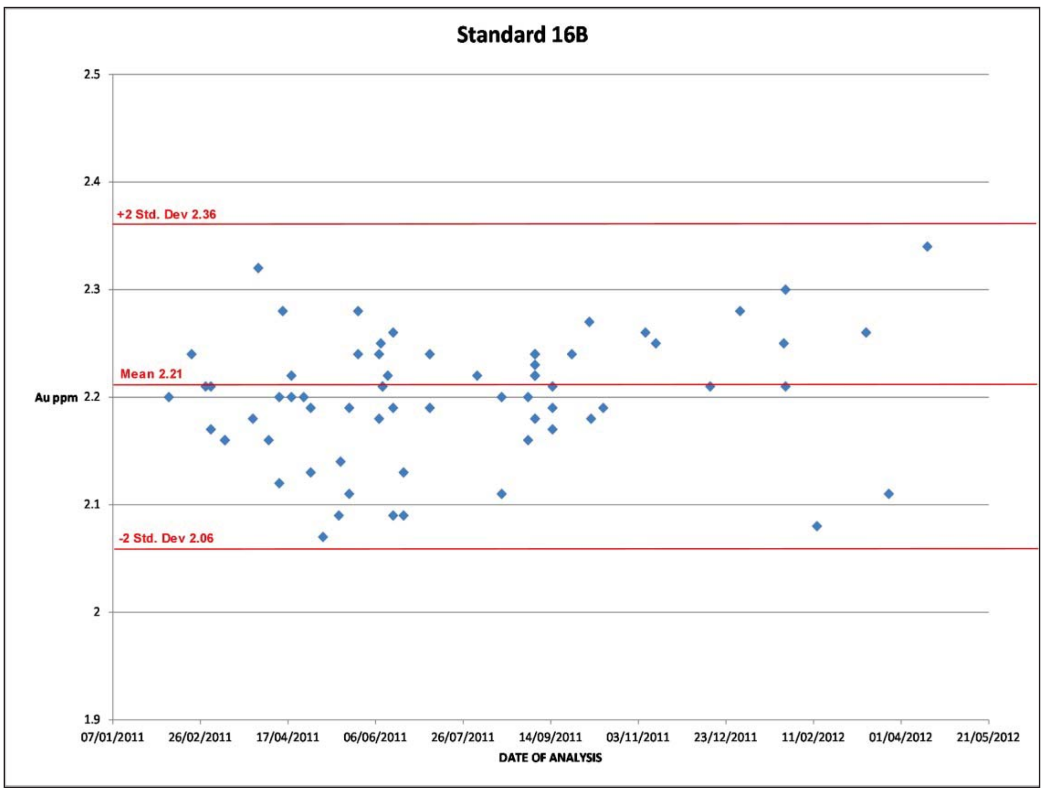


Figure 11.3: Standard 16B

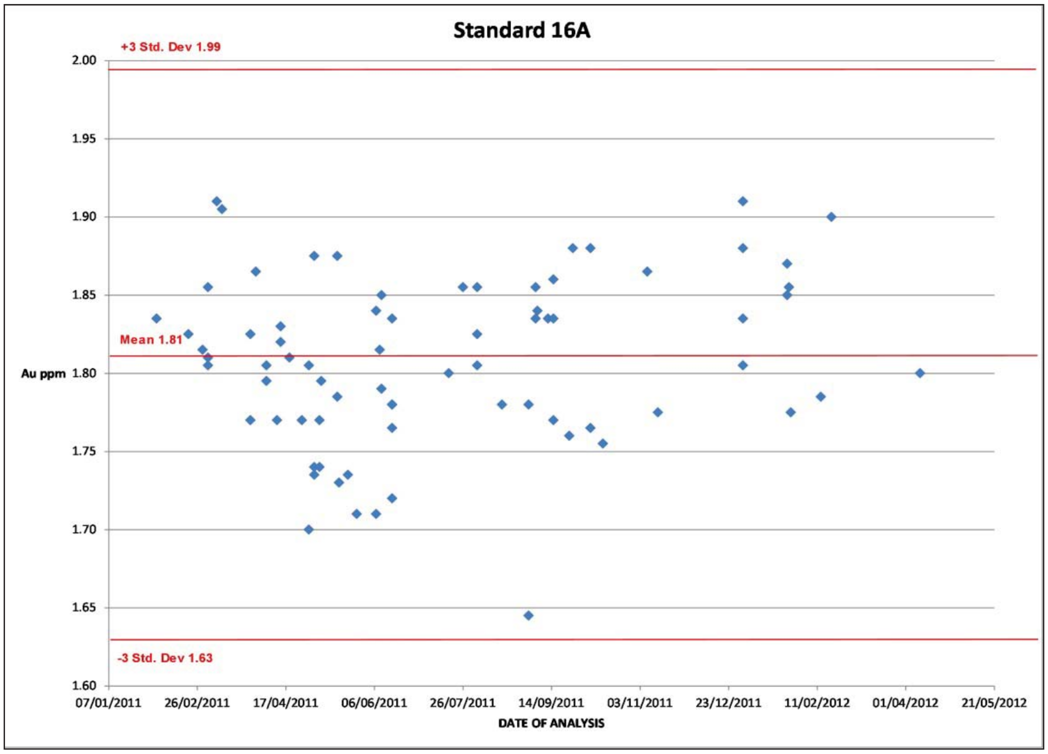
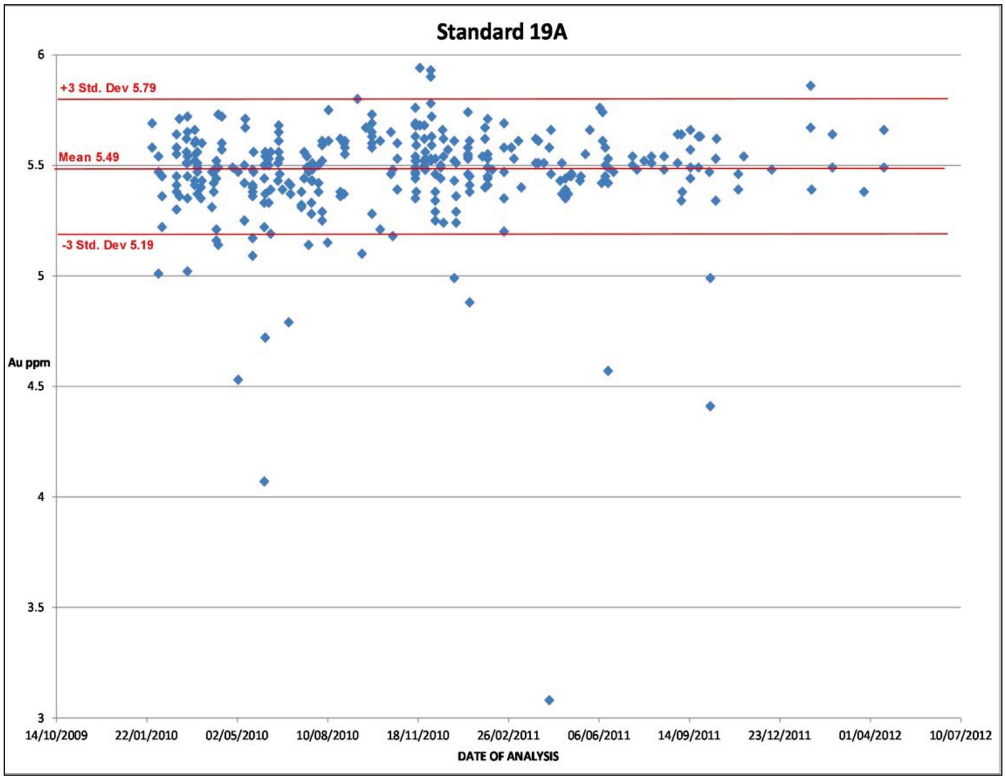


Figure 11.4: Standard 19A



11.4.3 Duplicates

Core Duplicates

In 2010, eight quarter core duplicates were taken from holes GW10-113, GW10-114, GW10-119 and GW10-125. The core duplicates show good repeatability, 75% of samples have a HARD value less than 20% of the sample mean (5). The repeatability of field duplicates is indicative of a low nugget effect (the inherent variability of gold content in samples from the same piece of core) and demonstrates acceptable levels of assay lab precision.

There are an insufficient number of samples to determine the precision of all ALS analyses or the natural variability of gold in core samples (Trinder, 2011).

A further eight core samples were taken by quartering as a test of the previous sampling and assay by Mr. Neil Gow in March 2011. These samples were selected to test a range of values with a concentration on values close to the likely cut-off grade of the mineral resource to give some indication of reliability at this important level. Table 11.2 shows the sample data and results.

Table 11.2: Core Duplicate Results 2011 Bradshaw Deposit

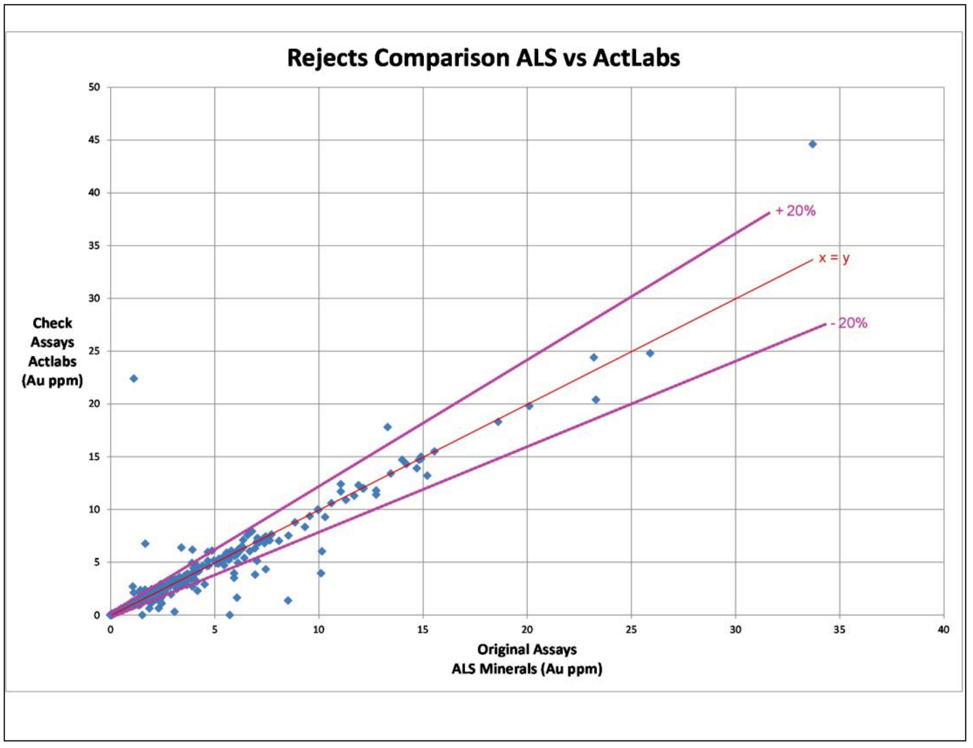
Sample ID	Hole ID	From (m)	To (m)	Length (m)	Original Value g/t Au	Check Value g/t Au
503109	GW10-113	38.0	38.5	0.5	4.42	3.8
503110	GW10-113	39.1	39.7	0.6	2.29	1.33
503111	GW10-114	60.8	61.8	1.0	3.62	3.99
503112	GW10-114	61.8	62.6	0.8	1.885	4.03
503113	GW10-114	70.7	71.4	0.7	6.49	6.59
503114	GW10-119	125.9	126.3	0.4	2.21	1.78
503115	GW10-125	38.9	39.3	0.4	1.755	1.72
503116	GW10-125	36.4	37.0	0.6	2.53	1.8

These results are interpreted to indicate reasonably good correlation for the number of samples collected. It is noticeable that there is good correlation for the samples close to 2 g/t Au. The mineralization of the Gowest deposit is fine grained and does not lend itself to sample selectivity. Further, there is no significant nugget effect (Gow, 2012).

REJECT DUPLICATES

Gowest has conducted reject duplicate sampling of approximately 5% of coarse crush rejects of samples from previously sampled holes GW10-45 to GW10-163. A total of 374 samples were sent to ActLabs in Timmins for gold analysis. The comparison of the original ALS to the check ActLabs gold values for the samples are displayed in Figure 11.5.

Figure 11.5: Rejects Comparison ALS versus Actlabs

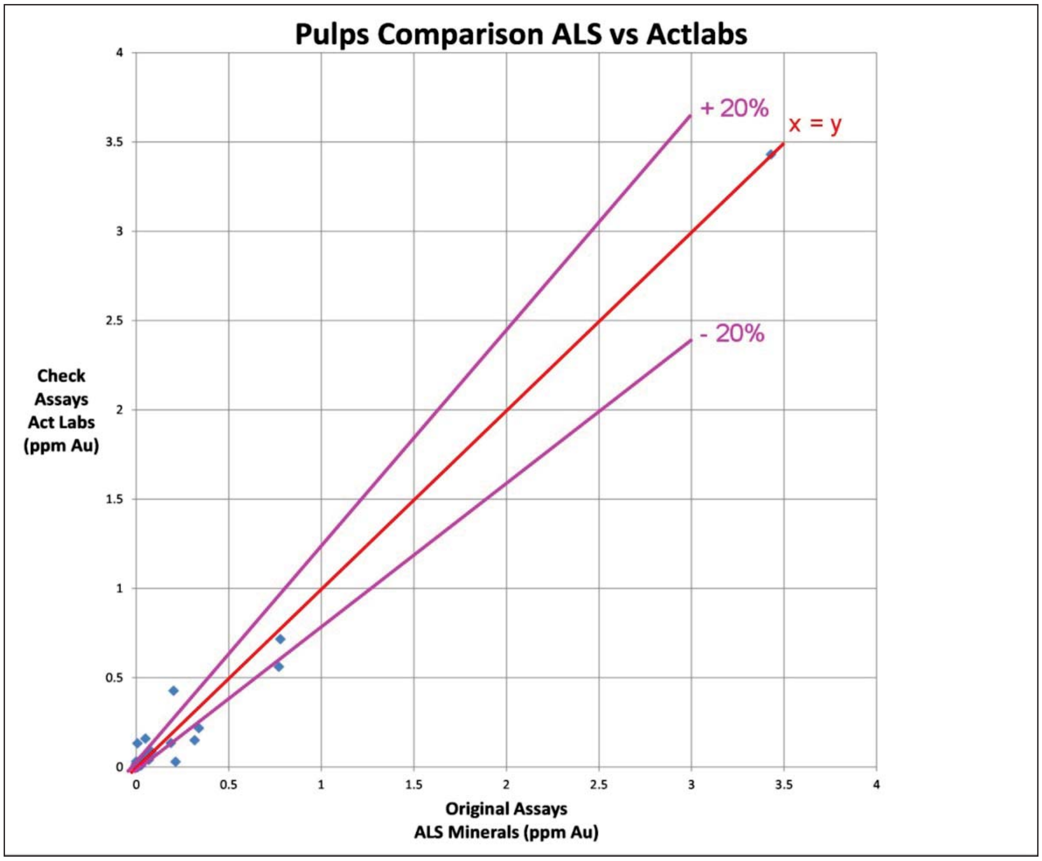


The reject duplicates showed very good repeatability with 93% of the samples being within ± 20% of the sample mean. The precision of the ALS analyses for holes GW10-45 to GW10-163 is very good when one takes into account the inherent natural variability of gold in rock or core samples.

PULP DUPLICATES

From hole GW10-164 onwards Gowest established a protocol of having ALS Labs forward a cut of the master pulp to ActLabs for pulp duplicate (check) analysis, at a rate of about 1 in 25 samples. A total of 72 pulp samples were analyzed at ActLabs as of November 2012 (Gow, 2012). Based on the results of these samples the precision of the ALS analyses is good (see Figure 11.6).

Figure 11.6: Pulps Comparison ALS versus Actlabs



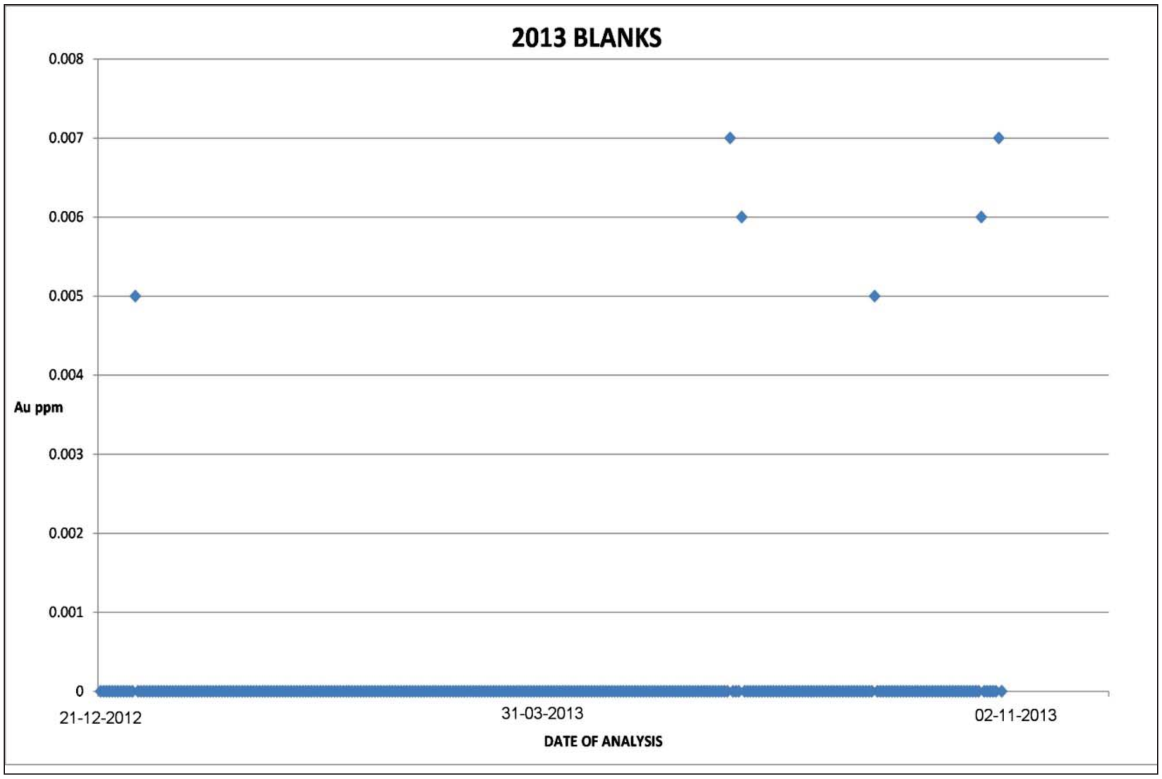
11.5 Gowest Quality Control 2013

11.5.1 Blanks

Gowest inserted a blank into the sample stream at a rate of about 1 in 20 samples. A total of 312 blank samples were inserted during the 2013 drilling campaign which represents about 5 % of the sample database for this period. The blank material used was pre-pulverized silica flour.

Of the 312 blank samples analysed in from 2012 to 2013 at the ALS laboratory, 98.1% correctly identified the blank sample as having a gold content below or at the lower limit of detection (0.005 ppm Au). An additional 1.9 % of the analyses identified the blank sample as containing less than or equal to 0.015 ppm Au. Gowest employed a policy that if any blank yields a gold value above 0.015 ppm Au, the batch of sample containing the blank should be re-assayed. No blank material values failed (>0.015 ppm Au, three times detection limit).

Figure 11.7: 2013 Blanks



The results imply that the lab has minimal cross sample contamination, or analytical error in the assaying of blank material.

11.5.2 Standards

A certified standard was inserted into the sample stream at a rate of about 1 in 20 samples. A total of 282 standard samples were inserted during the drilling campaign which represents about 4.5 % of the sample database for this period. Three certified standards are currently used by Gowest for the QA/QC assessment of the ALS laboratory (see Table 11-1). All three standards were obtained from Ore Research and Exploration Pty Ltd (ORE) of Australia through Analytical Solutions Ltd. of Toronto Ontario.

They range in certified mean grade from 1.81 to 5.49 g/t Au and represent well the gold grades of the Bradshaw Deposit. Control plots for the assaying of each standard by ALS Laboratory are presented in Figure 11.8 to Figure 11.11.

Figure 11.8: 2013 Standard 16A

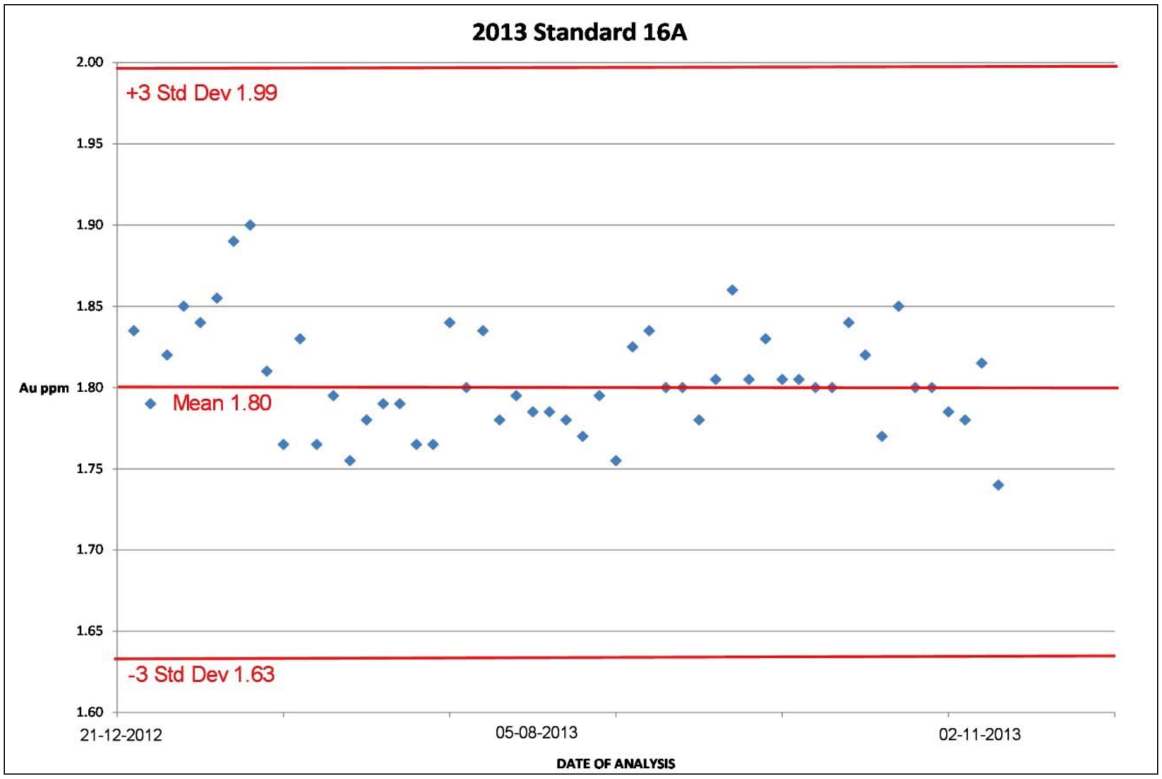


Figure 11.9: 2013 Standard 16B

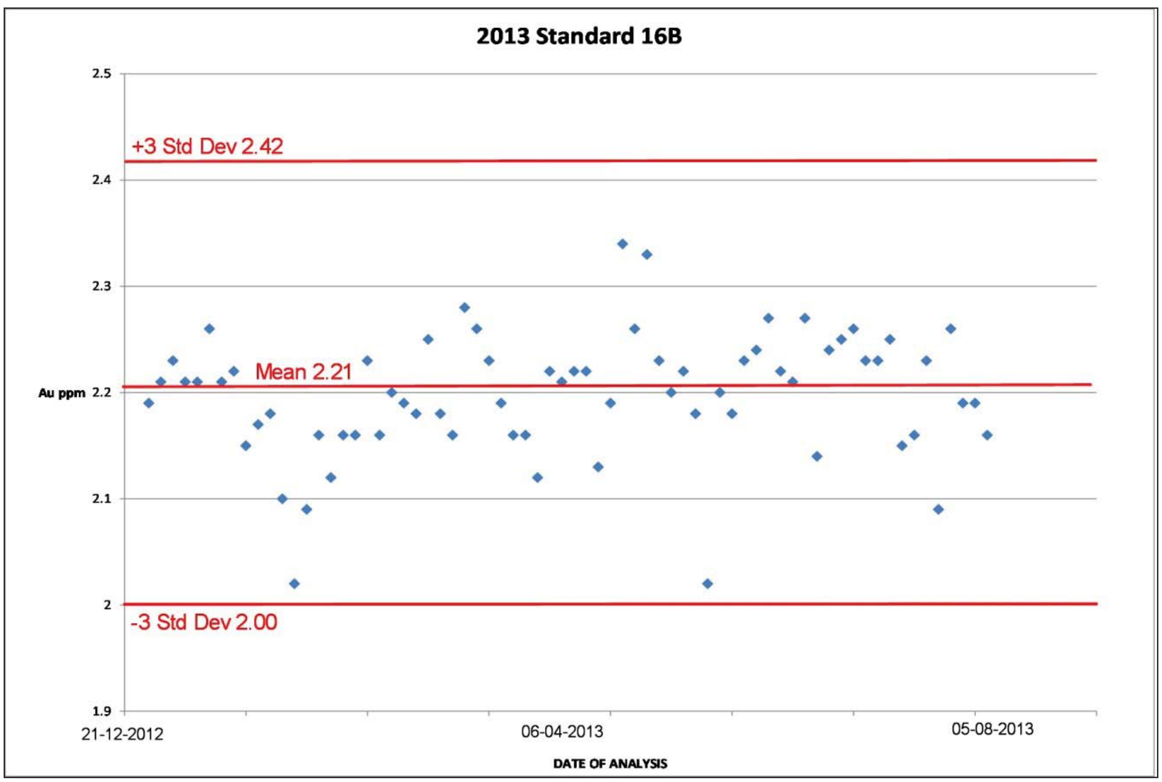
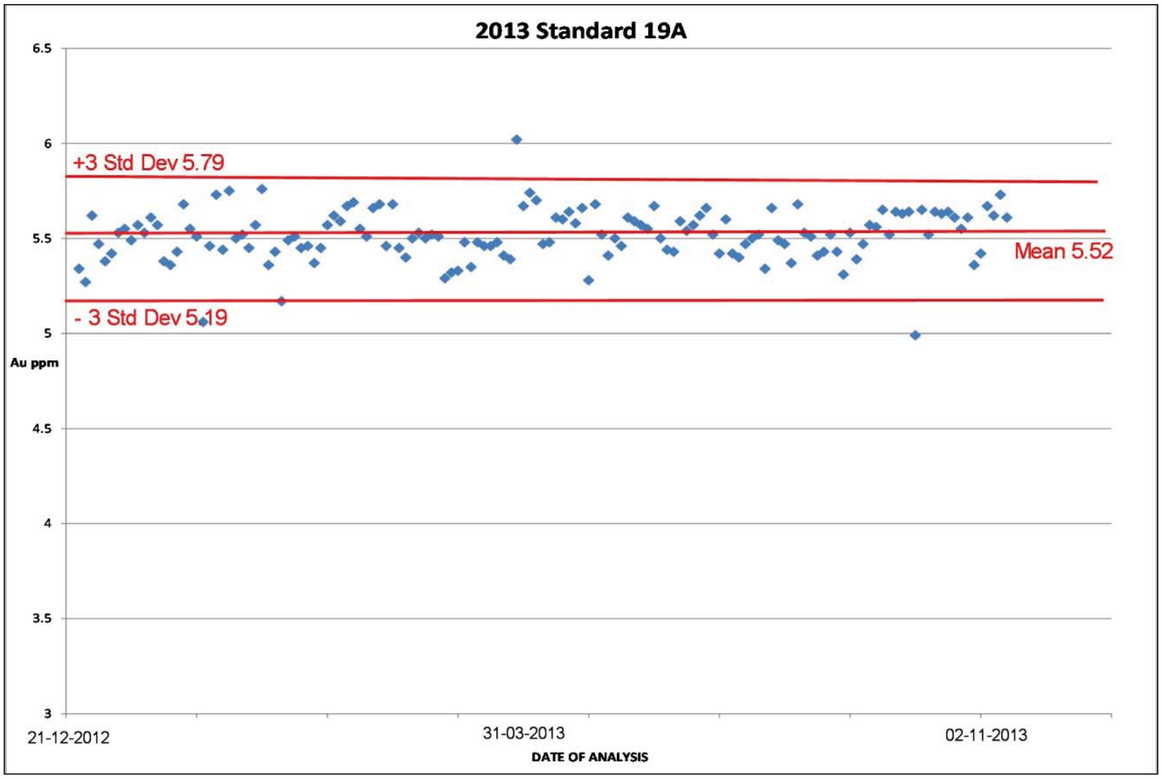


Figure 11.10: 2013 Standard 19A



The low-grade OREAS16A has an accepted value of 1.81 g/t Au with a between lab's 99th confidence of 0.18 g/t Au. The mean grade of the QA/QC samples submitted was 1.80 g/t Au, very slightly below the accepted certified value and within the confidence level set for between labs. There were no failures within the QA/QC sample suite submitted (Figure 11.8). Overall, there is some variance in the sample results throughout the 2013 drill campaign, but there is no drift evident.

The medium-grade OREAS16B has an accepted value of 2.21 g/t Au with a between lab's 99th confidence of 0.07 g/t Au. The mean grade of the QA/QC samples submitted was 2.21 g/t Au, exactly the certified accepted value and within the confidence level set for between labs. There were no failures within the QA/QC sample suite submitted (Figure 11.9). Overall, there is some variance in the sample results throughout the campaign, but there is no drift evident.

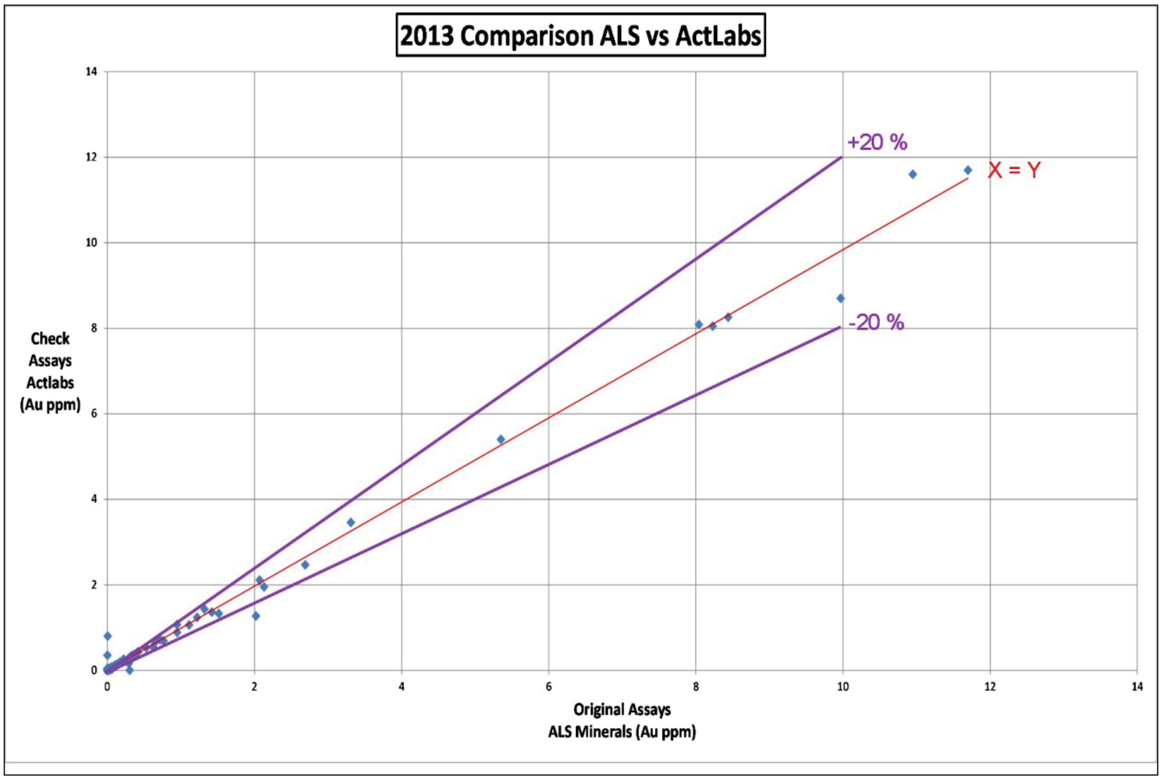
The higher grade Standard OREAS-19A has the largest number of assays and was used more in order to monitor higher grade assay results received from the laboratory. It has an accepted value of 5.49 g/t Au with a between lab's 99th confidence of 0.10 g/t Au. The mean grade of the QA/QC samples submitted was 5.52 g/t Au, slightly above the accepted value and within the confidence level set for between labs (Figure 11.10).

Standard 19A had 4 failures out of 152 analyses (2.6%) which is not a major concern as all other standards passed in their sample shipment. Two of the failures were investigated by having ALS repeat the gold analysis on the sample pulps. The failed standard passed on re-analysis and the repeatability of the other assays in the batch is satisfactory.

11.5.3 Pulp Duplicates

Gowest established a protocol of having ALS Labs forward a cut of the master pulp to ActLabs for pulp duplicate (check) analysis, at a rate of about 1 in 25 samples during the 2013 drilling program.

Figure 11.11: 2013 Pulps Comparison ALS versus Actlabs



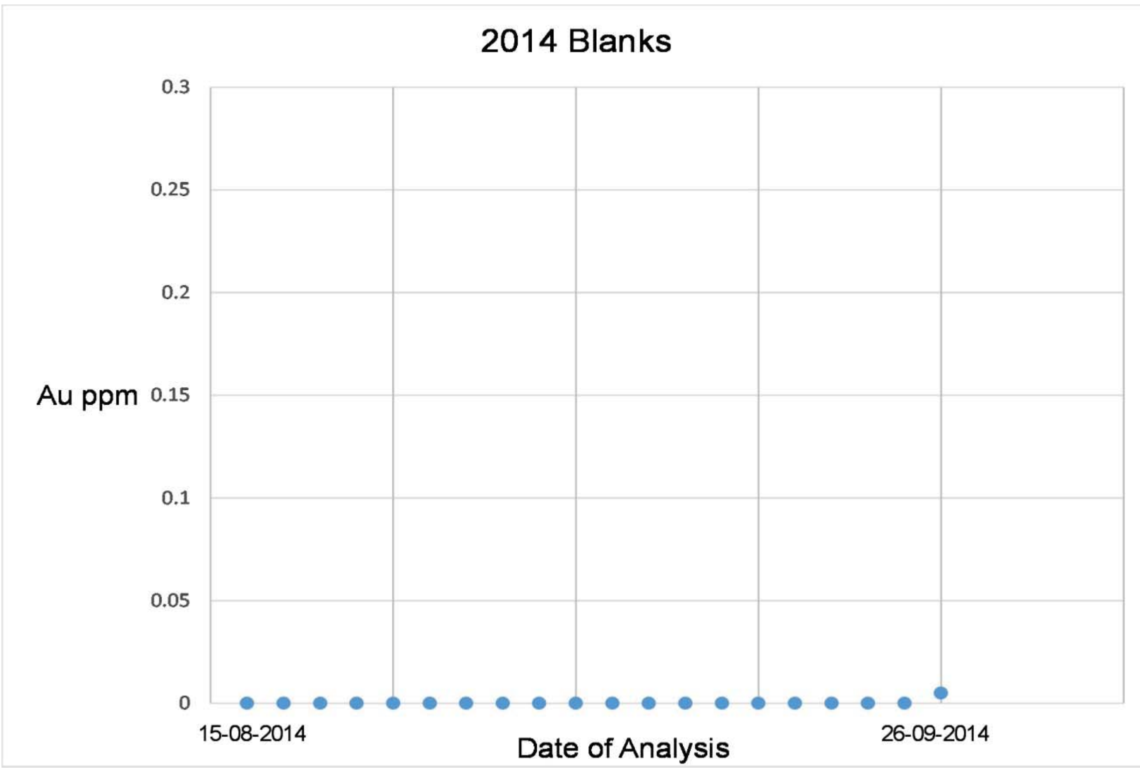
The duplicates showed an excellent repeatability with all the samples being within ± 20% of the sample mean. The precision of the ActLabs gold analyses was very good for the 2014 drilling campaign.

11.6 Gowest Quality Control 2014

11.6.1 Blanks

Gowest inserted a blank into the sample stream at a rate of about 1 in 20 samples. A total of 20 blank samples were inserted during the 2014 drilling campaign which represents about 4.7 % of the sample database for this period. The blank material used was pre-pulverized silica flour.

Figure 11.12: 2014 Blanks



Of the 20 blank samples analyzed at the ActLabs Laboratory, all correctly identified the blank sample as having a gold content below or at the lower limit of detection (0.005 ppm Au). Gowest employed a policy that if any blank yields a gold value above 0.015 ppm Au, the batch of sample containing the blank should be re-assayed. No blank material values failed (>0.015 ppm Au, three times detection limit).

The results imply that the lab has minimal cross sample contamination, or analytical error in the assaying of blank material. The author recommends that Gowest continue to use and monitor blank samples and flag any serious concerns with the laboratory staff, as soon as a failure is observed.

11.6.2 Standards

A certified standard was inserted into the sample stream at a rate of about 1 in 20 samples. A total of 21 standard samples were inserted during the drilling campaign which represents about 4.9 % of the sample database for this period. Three certified standards are currently used by Gowest for the QA/QC assessment of the ALS laboratory (see Table 11-1). All three standards were obtained from Ore Research and Exploration Pty Ltd (ORE) of Australia through Analytical Solutions Ltd. of Toronto Ontario.

They range in certified mean grade from 1.81 to 5.49 g/t Au and represent well the gold grades of the Bradshaw Deposit. Control plots for the assaying of each standard by ALS Laboratory are presented in Figure 11.13 to Figure 11.15.

Figure 11.13: 2014 Standard 16A

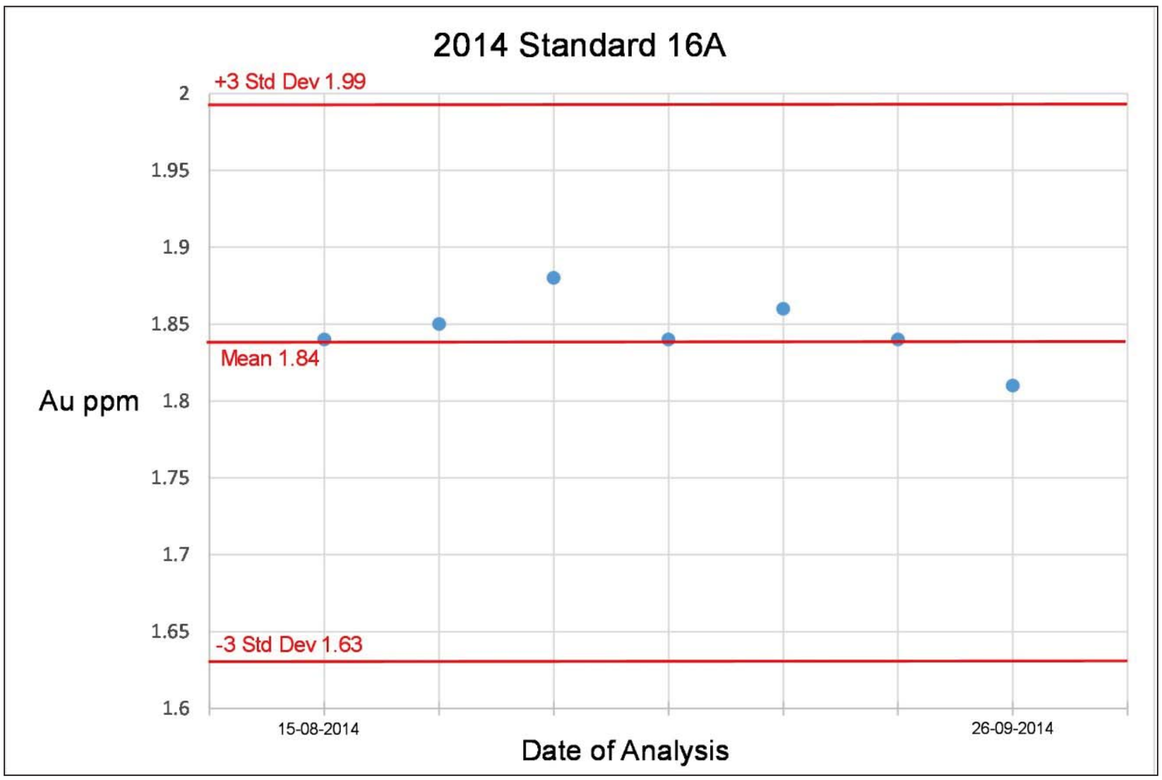


Figure 11.14: 2014 Standard 16B

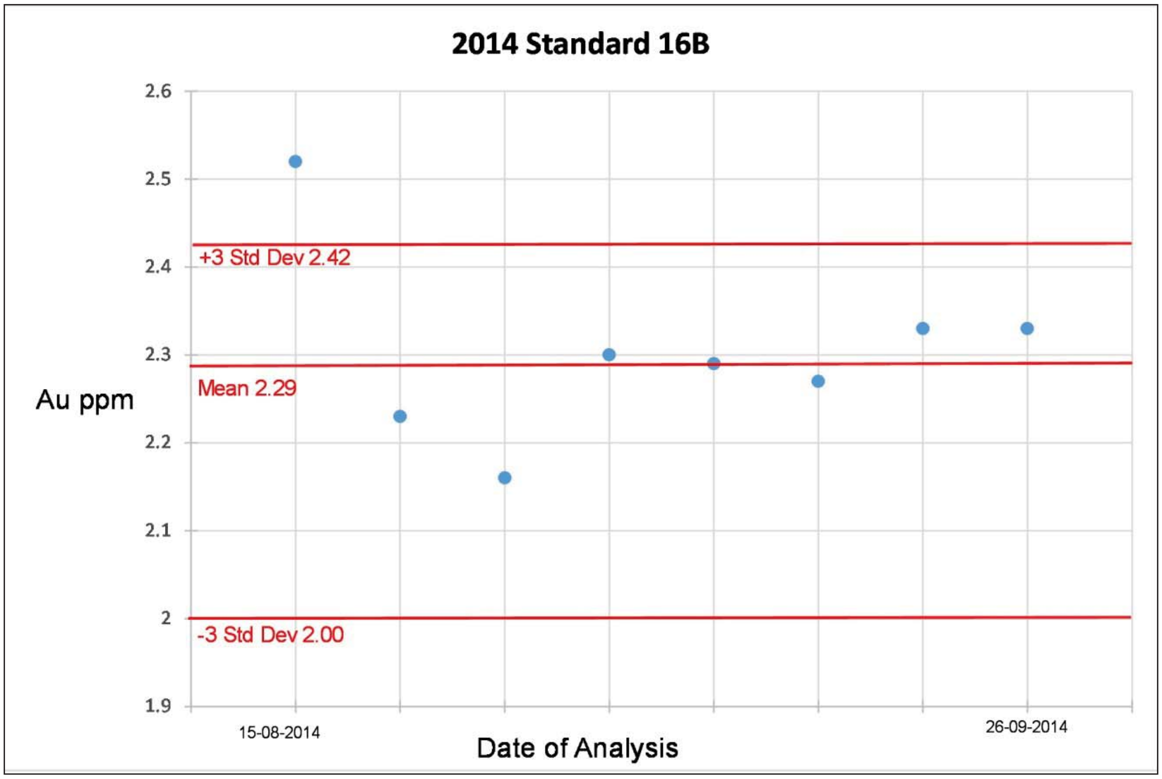
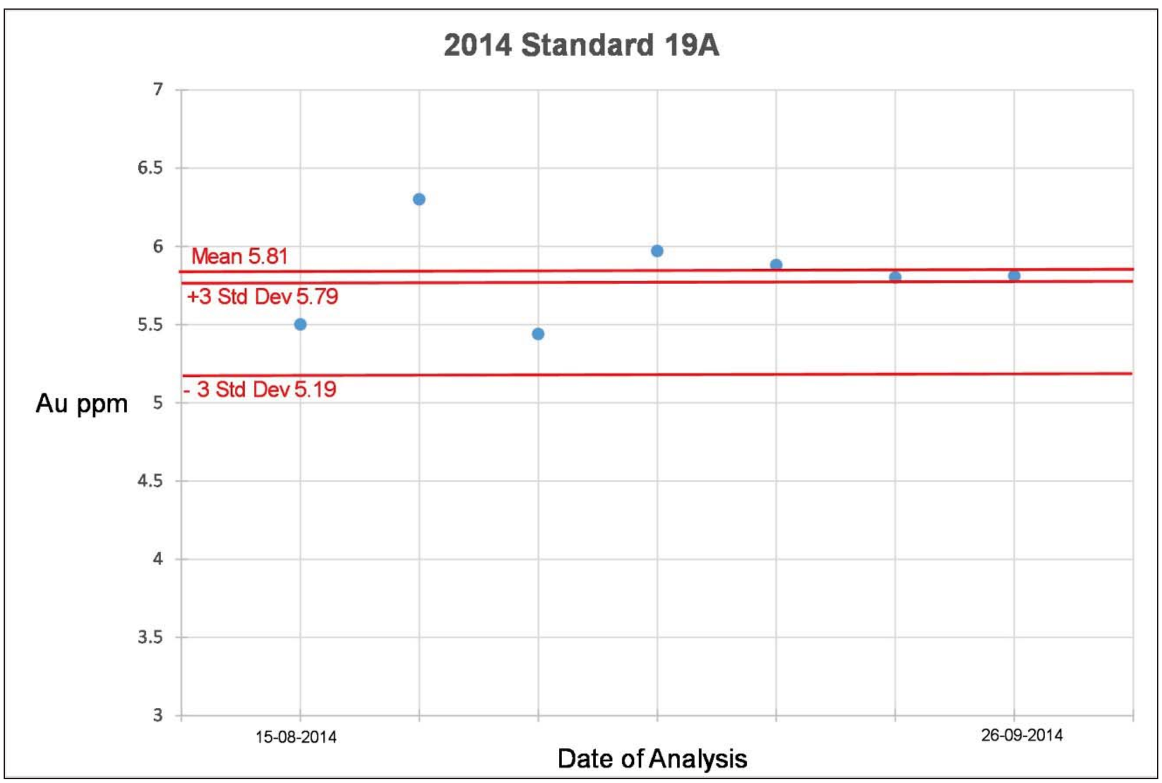


Figure 11.15: 2014 Standard 19A



The low-grade OREAS16A has an accepted value of 1.81 g/t Au with a between lab's 99th confidence of 0.18 g/t Au. The mean grade of the QA/QC samples submitted was 1.84 g/t Au, very slightly above the accepted certified value and within the confidence level set for between labs. There were no failures within the QA/QC sample suite submitted (Figure 11.13). Overall there is some variance in the sample results throughout the 2014 drill campaign, but there is no drift evident.

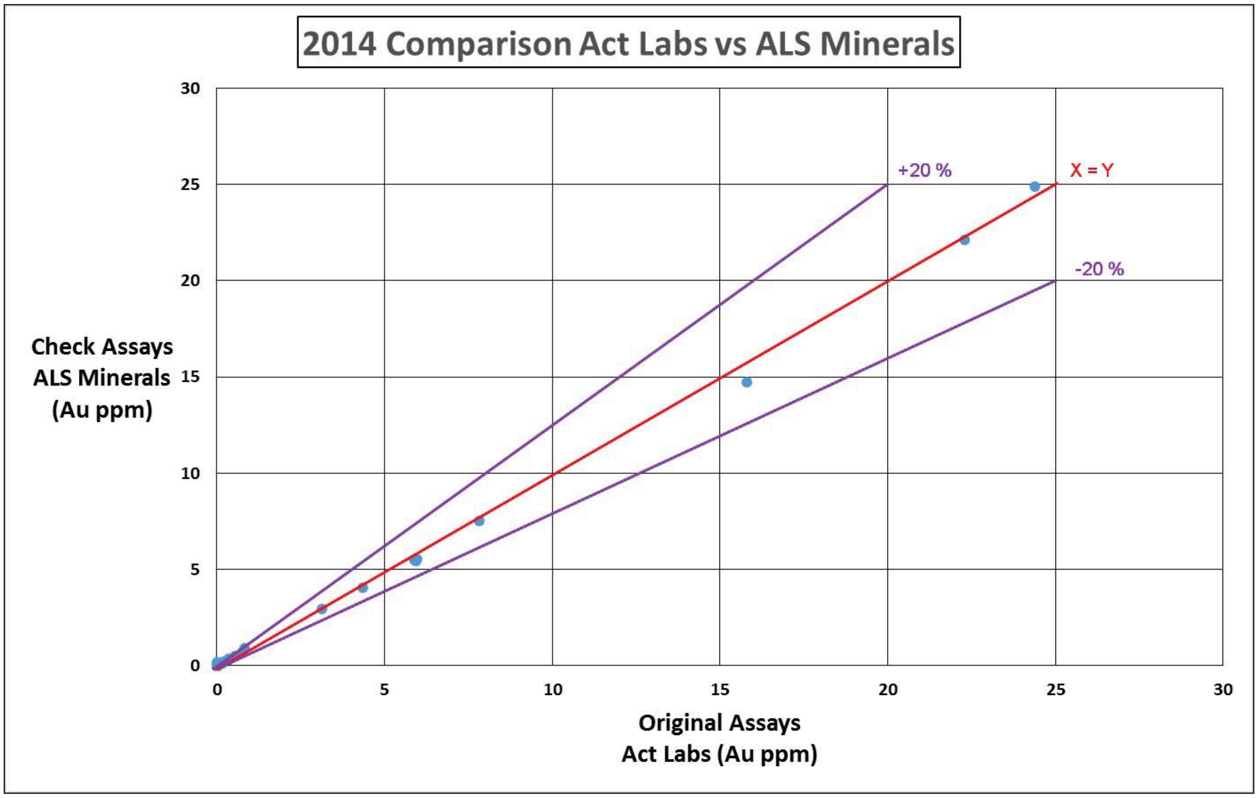
The medium-grade OREAS16B has an accepted value of 2.21 g/t Au with a between lab's 99th confidence of 0.07 g/t Au. The mean grade of the QA/QC samples submitted was 2.29 g/t Au, above the accepted value and within the confidence level set for between labs. There was one failure within the QA/QC sample suite submitted (Figure 11.14). This coincided with a failure of the next standard OREAS-19A in the sample shipment. ActLabs was notified and the batch containing the failed standards was re-analysed. The two standards and two blanks in the re-analysis passed. The first ActLabs assays were replaced in the database by the new gold values obtained by the re-assay.

The higher grade Standard OREAS-19A has an accepted value of 5.49 g/t Au with a between lab's 99th confidence of 0.10 g/t Au. The mean grade of the QA/QC samples submitted was 5.81 g/t Au, above the accepted value and slightly above the confidence level set for between labs. It was noticed that Standard 19A after the first two shipments returned assay values just above the acceptable range of ± 3 standard deviations (Figure 11.15). However, the Standard 16A and 16B results in the same batch of assays passed within the acceptable range of ± 3 standard deviations. As a result of the lower number of blank and standard assays in the smaller 2014 drilling program, it is difficult properly assess the results. Kevin Montgomery (P. Geo) of Gowest Gold has recommended internally that Gowest more closely monitor any new laboratory when utilizing it as the primary laboratory in the future.

11.6.3 Pulp Duplicates

Gowest continued its pulp duplicate protocol in 2014, at a rate of about 1 in 25 samples. In addition, extra samples that comprise mineralized zones in batches that had Standard OREAS-19A were re-analyzed. A total of 35 sample pulp splits from the 2014 assaying were sent to ALS Minerals for the check analysis, along with blanks and new standards. The results of the ALS check analysis compared well to the original ActLabs assay results (Figure 11.16).

Figure 11.16: 2014 Pulps Comparison Actlabs versus ALS



The duplicates showed an excellent repeatability with all the samples being within ± 20% of the sample mean. The precision of the ActLabs gold analyses was very good for the 2014 drilling campaign.

11.7 Quality Assurance and Quality Control Conclusions

The QA/QC program at the North Timmins Project has allowed a broad assessment of analytical accuracy and precision since 2004.

A total of 863 blanks which amounts to 4.4 % of drill core samples (19,655) and 851 certified standards which amounts to 4.3 % of drill core samples have been assessed. This corresponds to the analysis of drill holes GW04-03 through GW14-281. There is no QA/QC data available for assays obtained prior to 2004 and when those assays are included the overall percentage of QA/QC standard samples drops to 3.7 %.

Positive QA/QC results obtained between 2004 and 2008 at Swastika Laboratories, support previous assertions that despite the greater degree of uncertainty in Swastika assays obtained prior to 2004 due to the lack of QA/QC programs, they are suitable for use in resource estimation (Trinder, 2011).

Standards OREAS-16A and OREAS-16B performed well however, OREAS-19A standard results from the ALS laboratory indicate a slight occasional bias towards under-reporting of gold grades from 2010 to 2012. Under-reported gold grades often occur as outliers associated with troughs in cyclical trends indicative of instrumental analytical drift. It is best practice to alert the assay laboratory when successive standard analyses are beyond 3 standard deviations from the standard value. If necessary, the batch should be reanalysed (Gow, 2012). The situation of successive standards beyond 3 standard deviations did not occur during the 2010 to 2012 Gowest drilling program. The results of the gold analysis of the standards and blanks in 2013 are acceptable for use in a resource estimation.

The performance of the standards and blanks in 2014 analysis is acceptable. The gold analysis of the blanks did not show any evidence of sample contamination. The certified standard charts do not show any evidence of significant analytical bias.

The author recommends that Gowest maintain their thorough duplicate sample program involving the submission of core, reject and pulp duplicates to its primary assay laboratory and check pulp duplicates to a secondary laboratory. The number of QA/QC core duplicate analyses is low and there is no QA/QC for pre-2004 exploration.

It is considered that QA/QC results provide sufficient confidence in assay values for use in the estimation of CIM compliant inferred and indicated resources.

12.0 DATA VERIFICATION

12.1 Site Visits

Mr. Gow visited the Frankfield Property August 28, 2015 as part of a data verification study. Collars of a number of holes were visited. The collars are well marked.

Mr. Gow visited the Frankfield Property and the offices of Gowest in Timmins March 11, 2011 for the purposes of completing due diligence work on the property. The offices of Gowest were visited and the core logs from a number of holes were examined. Logging and sampling were found to be reliable and the drill logs were accurately maintained.

The purpose of the 2015 visit was for due diligence purposes for work completed since the previous visit. Diamond drill core of a number of the newer holes was laid out and the core and appropriate logs were examined. The most recent logging was carried out either by Mr. Montgomery P. Geo or under his supervision. Logging and sampling procedures are considered to be up to industry standards and considered appropriate by Mr. Gow.

A group of pulp samples were sent for re-assay. Gow considers that this is appropriate, to test some assay variability in some of the most recent work. The assaying procedures, protocols and QA/QC work are described above. The re-assay of the pulps was completed at the ALS Laboratory in Val d'Or, PQ. The results of the re-assay values are tabulated in Table 12.1. Mr. Gow considers that the correlation between the original values and the check values is excellent.

Table 12.1: Pulp Re-assay Comparison for the Most Recent Drilling

Sample No.	Hole ID	From (m)	To (m)	Interval (m)	Original (ppm Au)	Pulp Check (ppm Au)
L883216	GW13-222	186.8	187.8	1.00	4.27	4.03
Q622991	GW13-247	152.8	153.4	0.60	6.86	6.93
Q623634	GW13-254	196.0	196.9	0.9	8.86	8.55
Q618161	GW13-262	58.4	59.0	0.6	7.89	7.64
Q618754	GW13-269	106.6	107.4	0.8	3.38	3.52
Q619184	GW13-273	144.2	145.1	0.9	9.75	9.77
Q619422	GW14-276	459.5	460	0.5	6.1	6.07
Q620024	GW14-279	201	202	1.0	3.64	3.54
Q620189	GW14-281	196.3	197	0.7	9.42	9.42
Q619500	GW14-277	354.5	355.4	0.9	4.79	4.43

12.2 Verification of Resource Database

Drill data validation was carried out in both the Timmins and Toronto offices of Gowest. This work included detailed examination of the QA/QC results, and where appropriate, further analyses. Exploration results were entered into the database in either Timmins or Toronto, for different drill campaigns. Rigorous checking of the data after entry was completed.

It is the author's opinion that the Gowest drill hole database is suitable for use in the estimation of a Mineral Resource.

13.0 Mineral Processing and Metallurgical Testing

13.1 General

Sections 13.2 to 13.3.6 in this Section are taken from the report of Ling and Trinder, (2012).

Section 13.3.7 Ore Sorting, is added to this section to provide test information and results.

13.2 2008 Testwork

Bradshaw (2008) reported on preliminary metallurgical tests undertaken by Gowest at SGS Lakefield Research Limited (SGS) in 2008.

A gold deportment study was completed to examine the distribution of the gold in the Bradshaw mineralized samples. The study consisted of a microscopic evaluation for visible gold and secondary ion mass spectroscopy (SIMS) for the quantification of submicron gold in sulphide particles. The study concluded:

- Approximately 4% of the gold in the minerallized samples occurred as visible gold with an average size of 13 microns.
- 96% of the gold in the sample exists as submicroscopic gold within the mineralization particles.
- The main submicroscopic gold carrier in the mineralization is arsenopyrite.
- The mineralization's pyrite content is only a minor carrier of submicroscopic gold.
- Gold content in the arsenopyrite grains ranged from 130 to +200 ppm while that in the pyrite grains was generally in the order of a few ppm or less.

Direct cyanidation test-work was completed at SGS in 2008. The goal of this testwork was to determine the response of the Bradshaw mineralization to direct cyanidation. The test-work program consisted of grinding the sample to approximately 80% minus 45 microns. A series of 6 cyanidation tests were then performed at 48, 72 and 96 hours with carbon additions of 0 and 15 g/L for each leach period.

The cyanidation tests with no carbon addition resulted in a gold recovery to solution of 5%. With carbon addition, the gold recoveries ranged from 6 to 9%. Cyanide consumptions for all the tests were reasonable and ranged from 1-2.4 kg/t. Lime consumptions ranged from 0.7-1.2 kg/t.

It was concluded from the direct cyanidation test-work that the gold in the Bradshaw mineralization is refractory and is contained within the mineralization's sulphide content. It appears from the test-work that the carbonaceous content (organic and graphitic) in the mineralization is low and unlikely to present problems for processing via cyanidation.

Preliminary flotation studies on the Bradshaw composite sample were performed at SGS in 2008 to examine gold recoveries in the concentrate produced. Test-work consisted of a rougher flotation step followed by a two-stage cleaner flotation. Gold recovery during rougher flotation was 90% into a concentrate that consisted of 21% of the original sample mass. The recovery curve at this point remained quite steep and therefore it is expected that recovery improvements could be achieved by increasing mass recovery. Gold and arsenic recoveries in the concentrate were almost identical due to the fact that the vast majority of the gold in the Bradshaw mineralization is submicroscopic and contained within arsenopyrite. Cleaner flotation test-work produced a final high grade sulphur/iron/arsenic concentrate assaying 34% iron, 32% sulphur, 16% arsenic and 44 g/t Au gold. Overall gold recovery to the cleaner concentrate was 78%.

Due to the refractory nature of the Bradshaw mineralization, some preliminary pressure oxidation test-work was performed at SGS in 2008. The test-work consisted of grinding the composite sample to a size of 80% passing 50 microns followed by a rougher flotation stage to produce a gold-bearing concentrate for pressure oxidation. Pressure oxidation was performed in an agitated batch reactor with oxygen injection for sulphide decomposition. The reactions were allowed to occur for a period of two hours at a temperature of 200°C and an operating pressure of 310 psig (75 psig of oxygen over pressure). Following pressure oxidation, the residue solids were filtered from the slurry and subjected to 48 hours of conventional cyanidation with the addition of 10 g/L of carbon to the slurry. The overall gold recovery to solution during cyanidation of the pressure oxidation residues was 98%. Silver recovery was lower at 41%. Cyanide consumption was reasonable at 1.1 kg/t of solids. As a result of the promising initial results additional test-work was recommended to better define the optimal parameters for pressure oxidation of the Bradshaw mineralization.

13.3 2010 to 2012 Metallurgical Testwork

A series of metallurgical test-work has been completed for the Bradshaw Deposit. The information presented in this section is based primarily on test-work performed by SGS Canada (Lakefield), Ontario and follows industry accepted standard practices.

Additional testwork performed as part of the ongoing metallurgical evaluation of the Bradshaw Deposit also involved:

- Golder Associates Ltd., ON
- McGill University, QC
- Tomra, Wedel, Germany

Metallurgical Sample Preparation

In 2010 Gowest drilled a series of HQ size drill holes for the purpose of generating a composite sample for metallurgical testing. The holes were located adjacent to two existing exploration holes - GW06-33/38 - which were located approximately 100m apart and intersected significant intervals of typical Main Zone style mineralisation.

Preparation of the drill core for metallurgical test-work was performed in Timmins. The Main Zone core sections were removed in 1 metre intervals and individually bagged for shipping. A total of 181 sample bags were delivered to SGS Canada in large crates. Samples were received at SGS approximately 8 kg each representing ~1 metre of drill core (HQ). The samples were individually inventoried and weighed then crushed to 100% passing ¼ inch. A single 500 g sample was riffled from 30 randomly selected intervals to be reserved for comminution tests (crusher work index (CWI), ball mill work index (BWI), and abrasion index (Ai) tests). The remaining material was crushed to 100% passing 10 mesh. A 250 g aliquot of each sample was riffled out and pulverized. From the 181 pulverized aliquots, a sample of each was submitted for gold analysis and ICP scan.

A 250 kg master composite sample (MC1) was prepared by combining the 32 individual intervals with head assays in excess of 2 g/t Au. The head grade of the MC1 composite was 5.95 g/t Au. A detailed analysis is included as Table 13.1. The MC1 composite was used in flotation, pressure oxidation and bacterial oxidation test-work as described in the remainder of this section.

Table 13.1: Analytical Scan of MC1 Composite

Element	Assays
Au g/t	5.95*
S %	3.43
$S^{=}$ %	3.30
Semi Quantitative ICP Scan	
Ag g/t	< 2.00
Al g/t	53000
As g/t	2800
Ba g/t	32.00
Be g/t	< 0.50
Bi g/t	< 20
Ca g/t	39000
Cd g/t	< 10
Co g/t	45.0
Cr g/t	56.0
Cu g/t	75.0
Fe g/t	93000
K g/t	4400
Li g/t	< 5

Element	Assays
Semi Quantitative ICP Scan	
Mg g/t	14000
Mn g/t	2100
Mo g/t	< 5
Na g/t	33000
Ni g/t	28.0
P g/t	420
Pb g/t	< 40
Sb g/t	35.0
Se g/t	< 30
Sn g/t	< 20
Sr g/t	59.0
Ti g/t	8300
Tl g/t	< 30
U g/t	< 20
V g/t	290
Y g/t	25
Zn g/t	98

average of 5.99 g/tonne and 5.90 g/tonne

Subsequent to the preparation of composite MC1 a second series of HQ drill holes was completed to provide additional metallurgical test-work feed material (Master Composite 2 - MC2). These holes were drilled in the same general locations and prepared using the same procedures outlined for composite MC1. A comparison of the head assays for the two composites is presented in Table 13.2.

Table 13.2: Head Analyses of Master Composites

Element		Master Comp 1	Master Comp 2
Gold	g/t Au	5.95	6.75
Sulphur	% S	3.43	2.79
Sulphide Sulphur	% $S^{=}$	3.30	NA
Arsenic	% As	2.19	1.81
Iron	% Fe	9.3	NA

Sample MC2 was divided and utilized primarily to prepare larger representative sulphide concentrate samples for analysis by third party groups interested in processing the Frankfield East flotation concentrate. These samples were also utilized for filtration and thickening data as well as pressure oxidation optimization studies that are currently underway at SGS Canada.

13.3.1 Mineralogy Studies

Early on in the recent exploration activities (subsequent to 2008) it was determined that the mineralogy at Bradshaw was unlike many of the more "conventional" gold deposits in the Timmins area. Although highly silicified, the mineralized zones are largely absent of large structures of white quartz and visible gold. Instead, the brecciated and altered host rock is filled with fine sulphides that comprise anywhere from a few percent to in excess of 30% of the overall rock matrix. Historically, the zones were further subdivided into "main zone" material located close to the contact between the mafic and ultramafic rock units and a series of sub-parallel "hanging wall" zones that were more distal to the contact and somewhat different visually with more apparent bleaching and quartz veining.

Prior to the initiation of the 2010/11 metallurgical test-work program a series of rock samples from different zones within the deposit were subjected to a program of QEMSCAN™ and XRD analysis (SGS Lakefield) to identify the type and nature of the mineral species present in the deposit. The results of this program provided insights into the physical characteristics of the deposit. This included:

Arsenopyrite and pyrite were the primary carriers of gold with the fine gold grains (submicron to 10 microns in size) being largely attached to or locked within the sulphides.

Sulphide minerals were comprised almost exclusively of pyrite and arsenopyrite with variations in the ratio of these species in the different mineral zones.

Sulphide grain sizes were very similar in the different ore zones (main versus hanging wall) with >80% liberation at a particle size of 20-30 microns.

The non-sulphide minerals in the different ore zones were relatively similar with the exception of a quantity of micas/clays in the hanging wall areas that was largely absent in the main zone.

Overall, the QEMSCAN™ and XRD data confirmed that differences between the mineralization present in the historically identified main and hanging wall zones were in fact minimal and both areas should respond similarly to metallurgical treatments

13.3.2 Flotation

Gowest initiated an extensive program of flotation test-work at SGS in 2010/11. The program was divided according to two general methodologies. First, a bulk concentrate was produced containing both the pyrite and arsenopyrite. After this work was completed a second part of the program examined the production of separate pyrite and arsenopyrite concentrates. In both cases, cleaning stages were utilised to upgrade the initial rougher concentrate. Following the completion of the single stage batch tests a program of locked-cycle tests was completed to simulate the operation of the flowsheets with recycling of the intermediate products.

Bulk Sulphide Flotation

Bulk sulphide flotation resulted in high gold recoveries. With staged additions of sodium hydrosulphide and potassium amyl xanthate, 96% of the gold was recovered to a concentrate containing 25% of the feed mass and assaying 21 g/t Au, 12% S and 7.4% As. It was possible to lower the mass recovery of the concentrate material to less than 16% by adding sulphide cleaners after the rougher circuit. The batch results are summarized in Table 13.3.

Table 13.3: Bulk Cleaner Flotation Test Results

Product	Wt	Assays, g/t, %			% Distribution		
	%	Au	S	As	Au	S	As
2nd Cl Concentrate	15.5	36.1	21.0	12.7	93.2	93.1	92.2
1st Cl + Cl Scav Conc	23.7	24.5	14.3	8.64	96.4	96.6	95.8
Rougher Concentrate	33.7	17.4	10.1	6.15	97.7	97.5	97.2
Rougher Tailing	66.3	0.21	0.13	0.091	2.3	2.5	2.8
Head (calc)	100.0	6.00	3.49	2.13	100.0	100.0	100.0

**Test F16 - SGS Project 12416-001 Final Report (June 29, 2011)*

The results of the bulk concentrate batch test-work completed to date indicate the potential for recovery of up to 98% of the gold during rougher flotation into a concentrate that represents approximately 25% of the initial ore mass (assuming recycle of middlings). Cleaner flotation is able to upgrade this concentrate resulting in a product with a final mass representing 15-20% of the initial ore mass and a gold grade of 30-35 g/t.

Selective Arsenopyrite-Pyrite Flotation

Bulk sulphide flotation was able to recover the gold into a concentrate with high gold recoveries for further processing. In order to reduce the amount of material being shipped and/or processed and therefore to reduce overall processing costs, selective arsenopyrite-pyrite flotation was investigated. The opportunity for a selective flotation process results from the strong association of gold and arsenopyrite in the Bradshaw Deposit.

For the production of separate pyrite and arsenopyrite concentrates two different general flowsheets were examined:

- Bulk flotation followed by separation of the pyrite/arsenopyrite; and
- Sequential flotation.

Although both arrangement were promising it was determined that the sequential flotation process offered advantages in maintaining high gold recoveries while also providing better concentrate upgrading (higher gold grade in final concentrates). The sequential flotation flowsheet was optimised using single stage tests and then simulated with a final locked-cycle program.

In the sequential arsenopyrite-pyrite process the ore was ground with lime and conditioned at pH 11 in order to depress pyrite flotation. Stage additions of CMC for gangue depression, copper sulphate for arsenopyrite activation and a thionocarbamate as a collector were made to selectively recover an arsenopyrite rougher concentrate containing 92% of the gold, 18% of the pyrite and 90% of the arsenopyrite. The results of the best batch test are shown in Table 13.4. Subsequently, three locked cycle tests (LCT) were conducted to investigate the effect of recirculating middling streams on the sequential arsenopyrite-pyrite flotation process. The test LCT3 flowsheet is shown in Figure 13.1 Flowsheet utilized for locked-cycle flotation test-work (LCT3). The projected results from these cycle tests are presented in Table 13.5.

Overall, the locked cycle test-work program was able to recover 92-93% of the gold into an arsenopyrite cleaner concentrate with 6-7% of the original ore mass. The grade of this concentrate was +90 g/t Au. The final pyrite concentrate contains ~1.5% arsenic with a mass recovery of approximately 5%. The effectiveness of the selective flotation process at separating and concentrating the sulphide minerals is apparent when examining the final concentrates. The combined arsenopyrite + pyrite concentrate has the same overall gold recovery that was achieved in the prior bulk flotation test-work with only half of the concentrate mass. Further optimisation work is underway to determine the best conditions for maximizing gold grades and recoveries.

Table 13.4: Results of Batch Sequential Arsenopyrite-Pyrite Flotation Test

Product	Wt %	Assays, g/t, %					Distribution, %				
		Au	S	As	Py*	Aspy*	Au	S	As	Py*	Aspy*
Aspy 2ndCl Conc	4.1	95.3	20.1	33.4	10.9	72.6	63.5	24.2	61.1	9.7	61.1
Aspy Ro Conc	10.8	52.3	12.0	18.7	7.5	40.6	92.2	38.3	90.3	17.9	90.3
Py Ro Conc 1	5.7	3.77	32.7	1.61	59.9	3.5	3.5	55.4	4.1	75.6	4.1
Py Ro Conc 1+2	8.0	3.72	24.7	1.63	44.8	3.5	4.9	58.4	5.9	78.9	5.9
Aspy + Py Ro Conc	18.8	31.6	17.4	11.4	23.4	24.8	97.1	96.6	96.2	96.8	96.2
Rougher Tailing	81.2	0.22	0.14	0.11	0.2	0.2	2.9	3.4	3.8	3.2	3.8
Head (calc)	100.0	6.11	3.38	2.23	4.5	4.8	100.0	100.0	100.0	100.0	100.0

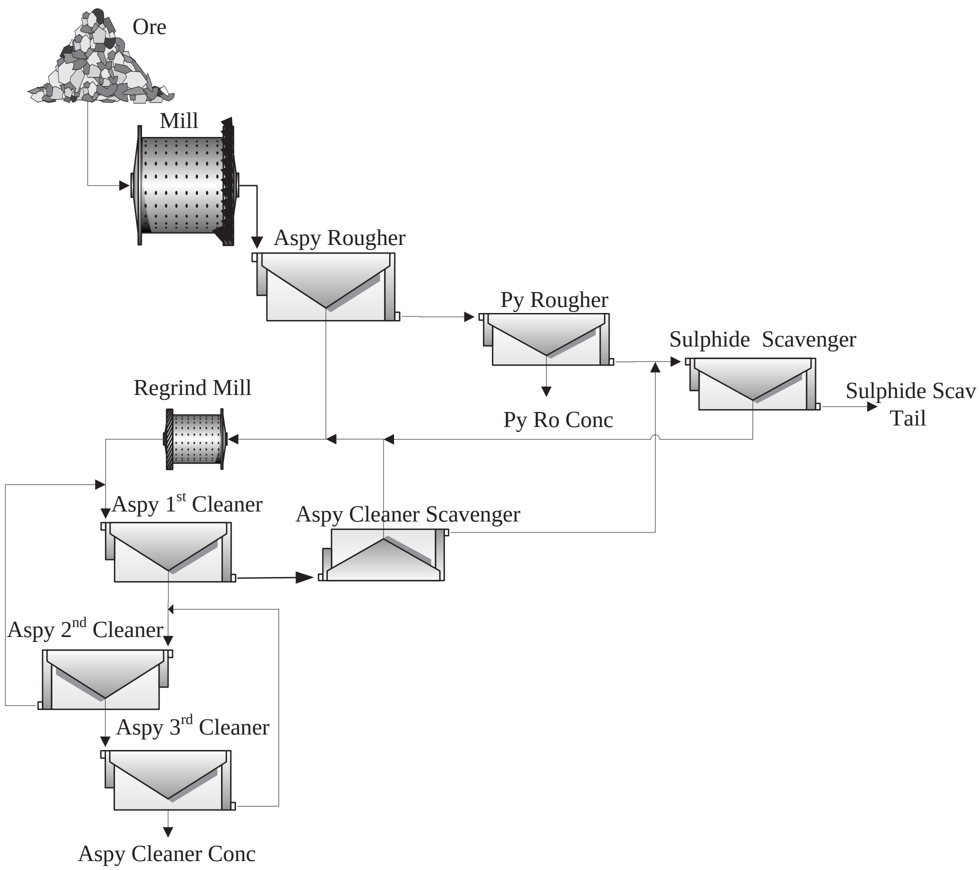
**Test F17, SGS Project 12416-001 Final Report (June 29, 2011). Calculation based on the assumption that all arsenic was present as arsenopyrite and the remaining sulphur was present as pyrite*

Table 13.5: Projected Results from Locked Cycle Tests

Test No.	Product	Wt %	Assays, g/t, %					Distribution, %				
			Au	S	As	Py*	Aspy*	Au	S	As	Py*	Aspy*
LCT3	Aspy 3rd Cl Conc	6.4	93.7	16.2	24.3	10.8	52.9	92.7	40.3	90.2	20.2	90.2
	Py Ro Conc	4.8	4.64	28.6	1.54	52.3	3.4	3.5	54.1	4.3	74.3	4.3
	Rougher Tailing	88.8	0.27	0.16	0.11	0.2	0.2	3.8	5.5	5.5	5.5	5.5
	Head (calc)	100.0	6.44	2.56	1.72	3.4	3.7	100.0	100.0	100.0	100.0	100.0

**calculation based on the assumption that all arsenic was present as arsenopyrite and the remaining sulphur was present as pyrite*

Figure 13.1: Flowsheet Utilized for Locked-cycle Flotation Test-Work (LCT3)



13.3.3 Pressure Oxidation

Pressure oxidation test-work (POX) was first commissioned by Gowest and conducted at SGS in 2010 and more tests were completed in 2011. A bulk arsenopyrite/pyrite flotation concentrate was produced and subjected to pressure oxidation in a batch reactor under a range of conditions. Four POX tests were carried out to evaluate different test conditions, namely, retention time, acid concentration and partial acid recycling. The standard POX conditions are given below:

Agitation Speed = 720 rpm
Reaction Temperature: = 200°C
Oxygen Flow = ~250 mL /min
Total Pressure: = 315 psi

Table 13.6 presented test results of test performed to date.

Table 13.6: Results of Pressure Oxidation Tests (2011)

POX Test No	Pre-Acidulation		POX Retention Time	POX Final Filtrate							POX Final Residue			
	H_2SO_4	pH		Redox pot	Fe_T	Fe^{2+}	As	Acid	Molar ratio		Solid Mass change	S_T	S^{2-}	**S^{2-} Oxidation**
	(kg/t)		(min)	(mV Ag/AgCl)	(mg/L)	(mg/L)	(mg/L)	(g/L H_2SO_4)	Fe^{3+}/Fe^{2+}	Fe/As	(%)	(%)	(%)	(%)
POX 1	85.6	1.8	60	505	17700	911	1930	39	18.4	12.3	-17.9	3.11	0.14	**99.0**
POX 2	85.6	1.8	120	575	13600	128	1820	40	105.3	10.0	-14.4	4.00	0.17	**98.8**
POX 3	57.0	3.5	90	520	13000	191	1530	38	67.1	11.4	-13.9	3.11	0.64	**95.4**
POX 4#	82.5	2.2	60	521	19700	202	2640	40	96.5	10.0	-15.1	3.81	0.17	**98.8**

POX 4: the acid solution was made up of recycled acid solution from POX 1 and2 at 39.5 g/L H_2SO_4 and concentrated H_2SO_4 and demineralised water.

Following pressure oxidation, the residue solids were filtered from the slurry, neutralised and subjected to conventional cyanidation for gold recovery.

Cyanidation tests were performed on flotation concentrate directly and on concentrate after POX. The leach results are presented in Table 13.7 Direct Cyanidation of Flotation Concentrate and Table 13.8 Cyanidation of Pressure Oxidation Residue. The leach retention time used was 48 hours for direct cyanidation and 24 hours for POX residue cyanidation.

Table 13.7: Direct Cyanidation of Flotation Concentrate

Test No	Lead Nitrate (kg/t)		Residue Assays (g/t)		**Extraction (%)**		Calc Head Assays (g/t)		Reagent Consumption (kg/t CIL feed)	
	Preaeration	Cyanidation	Au	Ag	**Au**	**Ag**	Au	Ag	NaCN	CaO
CIL-1	0.0	0.0	17.7	0.8	**7.4**	**69.3**	19.1	2.6	5.42	2.23
CIL-2	5.5	0.5	21.4	1.0	**8.7**	**46.4**	21.4	1.9	0.62	0.69

Table 13.8: Cyanidation of Pressure Oxidation Residue

CIL Test No	POX Test No	Residue Assays (g/t)		**Extraction (%)**		Calc Head Assays (g/t)		**Reagent Consumption (kg/t CIL feed)**	
		Au	Ag	**Au**	**Ag**	Au	Ag	**NaCN**	**CaO**
CIL-3	POX 1	0.86	0.9	**96.5**	**60**	24.8	2.2	**0.51**	**2.03**
CIL-4	POX 2	0.51	1.4	**97.8**	**28**	22.8	1.9	**0.71**	**7.93**
CIL-5	POX 3	0.66	1.6	**97.2**	**35**	23.3	2.5	**0.43**	**2.67**
CIL-6	POX 4	0.73	2.2	**96.8**	**22**	22.9	2.8	**0.69**	**4.84**

Processing the Bradshow Deposit mineralization via flotation followed by pressure oxidation and cyanidation produced overall gold extractions of 94-95% (total of flotation/POX/cyanidation). Other highlights include:

- Up to 99% oxidation of the sulphide minerals in the concentrate at 200 °C with a reaction time of 60 minutes.
- High iron/arsenic ratios in the POX discharge solutions (good for production of stable arsenic precipitate).
- 97-98% gold extraction from the neutralized POX discharge solids with 24 hours of cyanidation.
- Low reagent consumptions for the cyanidation of the oxidized concentrates.

Further optimisation studies are currently underway to optimise conditions including oxygen partial pressure and retention time.

13.3.4 Bacterial Oxidation

Bacterial oxidation of the Bradshaw flotation concentrates was also examined as an alternative to pressure oxidation. Test-work was initiated in 2010 under the supervision of Goldfields (BIOXTM process) and performed by SGS (Booysens,South Africa). A bulk arsenopyrite/pyrite flotation concentrate was used for the test-work which was completed in stirred reactors that contained bacteria that attack the sulphide minerals in the concentrate. Following oxidation the solids were filtered from the slurry, neutralised and subjected to cyanidation for gold recovery. A summary of the results is shown in Table 13.9. Highlights include:

- Sulphide oxidation levels of +96% (100% arsenic solubilisation) after five days of bacterial oxidation and 98-99% after 10-15 days of treatment,
- Gold dissolution of 95-96% from the oxidized solids,
- Reasonable reagent consumptions were achieved.

Table 13.9: Frankfield East Bacterial Leach Results

BIOX®				Gold Dissolution (%)	
Treatment Period (days)	% Solids	Acid Consumption (kg/t)	Sulphide Oxidation (%)	Direct Cyanidation	After BIOX® Treatment
			0	3.52	
10	20	25.6	98.4		94.8
15	20	24.1	99.1		95.9
10	25	4.7	96.5		94.2
15	25	-24.0	98.1		96.0

A stage batch neutralisation test performed on the BIOX effluent indicated that a stable ferric-arsenate precipitate can be produced using limestone and lime. The test-work results confirmed that the arsenic content in the neutralised effluents (<0.24 ppm) conforms to the US-EPA standards (EPA standard below 0.5ppm). The precipitates can therefore be considered stable for disposal to a tailings dam.

13.3.5 Flotation Tailings

Golder Associates Ltd. was retained to evaluate the geochemical characteristics of combined rougher / flotation tailings samples (which is the combined tailings from the Bradshaw Project). Metallurgical testing of the flotation tailings was carried out at SGS Lakefield, Ontario. Bulk rougher and bulk cleaner flotation tailings produced during the flotation were blended in the relative proportions that would be produced during processing and then were used for geochemical testing.

Tests carried out as part of the geochemical characterization program include:

- Elemental chemical composition
- Acid Base Accounting (ABA)
- Net Acid Generation (NAG) testing
- Short-term leach testing, including de-ionized (DI) water leach testing, detailed analysis of the NAG leachate;
- Decant water analysis
- Kinetic testing.

Results of the ABA and NAG testing indicate that the combined tailings sample is non-acid generating. The tailings contain relatively low sulphide concentrations, and the complete oxidation of sulphide minerals is predicted to take significantly less time than the depletion of available neutralization minerals. The neutralization potential of the tailings is high, and comprised primarily of carbonate minerals, which provide significant buffering capacity. Humidity cell testing has achieved metal concentrations decreased to stable concentrations where depletion calculations indicate that it could take several years to deplete the sulphide and/or Neutralisation Potential (NP) from the sample.

13.3.6 Ore Sorting

In addition to the above listed test work, Gowest investigated a possible route to reduce impact of underground mining dilution on costs of ore transportation and milling. In 2010 Gowest initiated a comprehensive program to evaluate whether the characteristic of the rocks from the Bradshaw gold deposit is amenable to an array of potentially suitable automated ore sorting techniques which include visible spectrum optical sorting (Optical), Dual Energy X-Ray Transmission sorting (DEXRT), conductivity/ magnetic susceptibility sorting (EM), and X-Ray Fluorescence Spectroscopy sorting (XRF-S).

In 2010 Gowest initiated a comprehensive program to evaluate whether the characteristic of the rocks from the Bradshaw gold deposit is amenable to an array of potentially suitable automated ore sorting techniques, including visible spectrum optical sorting (Optical), Dual Energy X-Ray Transmission sorting (DEXRT), conductivity/ magnetic susceptibility sorting (EM), and X-Ray Fluorescence Spectroscopy sorting (XRF-S).

Preliminary Ore Sorting Investigation - Benchtop Test

In June 2010, Gowest asked SGS, Lakefield to engage Terra Vision (later acquired by Commodas-Ultrasort of Germany in Aug. 2010) in Quebec City to perform preliminary ore sorting investigation.

The objective of the test was to take a sample from Gowest's Bradshaw project, characterise the rocks in the sample using several sorting sensors and then determine whether these characteristics can be used to sort the rocks by grade or another metallurgical property of interest. These results can then be used to determine if there is a sorting characteristic that warrants further investigation for full scale sorting.

The sensors used in this first preliminary test were:

1. DEXRT - Images acquired with a dual energy X-ray transmission (DEXRT) system.
2. EM - Conductivity and magnetic susceptibility acquired with a multi-frequency sensor.

The results of DEXRT and EM tests are presented in following two tables, Table 13.10 and Table 13.11. Each "Class" represents a group of rocks that have similar DEXRT or EM characteristics and can be separated from the others.

Care should be taken when looking at the recovery and mass distribution by class for the benchtop sorting as it is unlikely that the mass distribution by grade is representative of the resource. Gowest and SGS selected the sample to span the different types of mineralisation found in the resource, not to represent the distribution of these types of mineralisation in the resource.

The DEXRT test recovery curves show that almost all of the Au (96%) is contained in the first four of six classes, which accounts for only 69% of the mass. The DEXRT grade curve shows that if the first four classes were sorted to the sorter concentrate it would result in a grade of 2.82 g/t in 69% of the mass while the tails of such a sort would be 0.29 g/t.

There are only three classes in EM test as there was no measurable conductivity response and a magnetic susceptibility response at the lower limit of sensitivity of the sensor. It is unlikely that the three classes could be separated with a sensor based sorter.

Mass pull versus recovery and mass pull versus gold grade using DEXRT sorting are shown in Figure 13.2. The recovery curves show that almost all of the Au (96%) is contained in the first four of six classes, which accounts for only 69% of the mass. The grade curve shows that if the first four classes were sorted to the sorter concentrate it would result in a grade of 2.82 g/t in 69% of the mass while the tails of such a sort would be 0.29 g/t.

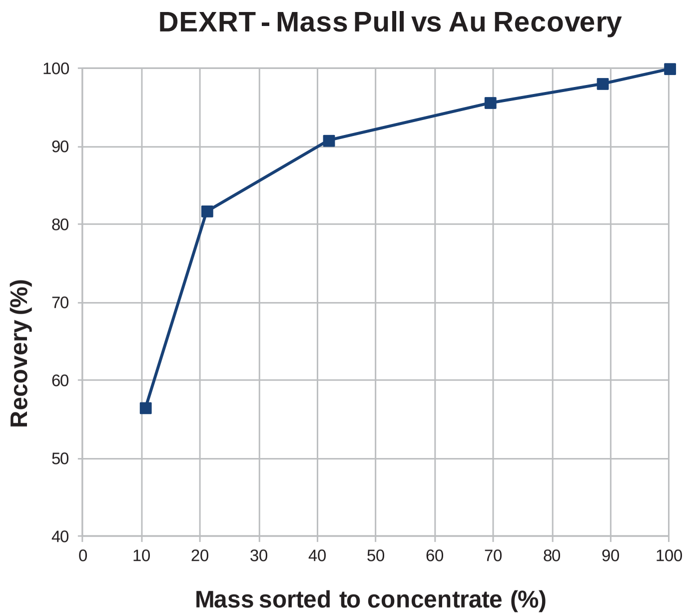
Table 13.10: DEXRT Sorting - Masses, Grades and Recoveries for Each Class

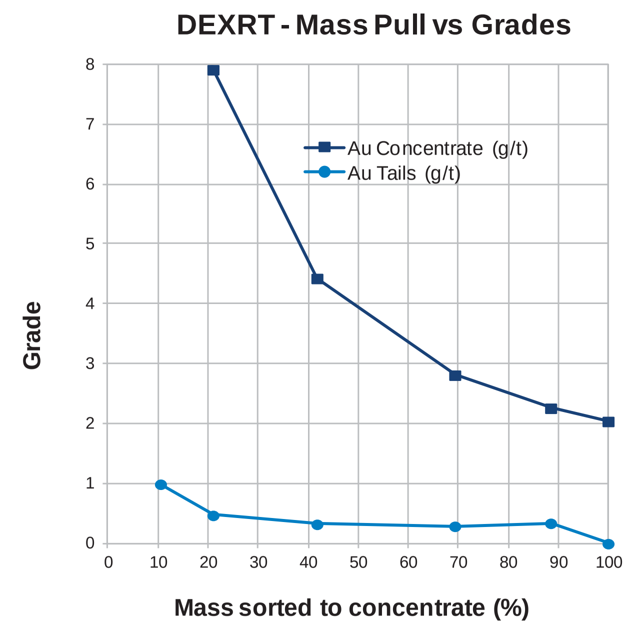
Class	Mass (g)	Au (g/t)	Mass in Class (%)	Cumulative Mass Sorted to Conc. (%)	Au Conc. (g/t)	Au Tails (g/t)	Au Distribution (g)	Au Distribution (%)	Au Cumulative Recovery (%)
1	1133.10	10.90	10.59	10.59	10.90	0.99	0.0124	56.53	56.53
2	1122.30	4.91	10.49	21.08	7.92	0.47	0.0055	25.22	81.75
3	2221.00	0.89	20.76	41.84	4.43	0.32	0.0020	9.05	90.80
4	2945.90	0.36	27.54	69.38	2.82	0.29	0.0011	4.85	95.65
5	2048.90	0.26	19.15	88.53	2.26	0.34	0.0005	2.44	98.09
6	1227.10	0.34	11.47	100.00	2.04	0.00	0.0004	1.91	100.00
Subtotal	10698.30	**2.04**	100.00				0.0218	100.00	

Table 13.11: EM Sorting - Masses, Grades and Recoveries for Each Class

Class	Mass (g)	Au (g/t)	Mass in Class (%)	Cumulative Mass Sorted to Conc. (%)	Au Conc. (g/t)	Au Tails (g/t)	Au Distribution (g)	Au Distribution (%)	Au Cumulative Recovery (%)
3	4710.50	6.13	18.01	18.01	6.13	0.44	0.0125	81.27	81.27
2	4561.00	0.47	40.33	58.34	1.58	0.36	0.0021	13.95	95.23
1	2036.70	0.36	18.01	76.36	1.36	0.00	0.0007	4.77	100.00
	11308.20	**1.36**	76.36				0.0154	100.00	

Figure 13.2: DEXRT Sorting Mass Pull versus Gold Recovery and Gold Grade





13.3.6.1 Second Bench-top Amenability Test

A second bench-top screening test was completed in September 2011 using rock samples from the Bradshaw Deposit.

The objective of this test was to take a set of specimens from the Gowest's Bradshaw project, characterize the rocks in the set using conductivity and magnetic susceptibility (EM), color (Optical), X-Ray Fluorescence Spectroscopy (XRF-S) and Dual Energy X-ray Transmission (DEXRT) features and determine whether these features can be used to sort the rocks to upgrade the gold values. These results can then be used to determine if there is a sorting characteristic that warrants further investigation for full scale sorting tests.

Gowest shipped split core rock specimens to Commodas-Ultrasort in Quebec City. One hundred rocks were chosen at random from the specimens.

Features of the 100 rocks were then acquired with the following sensors:

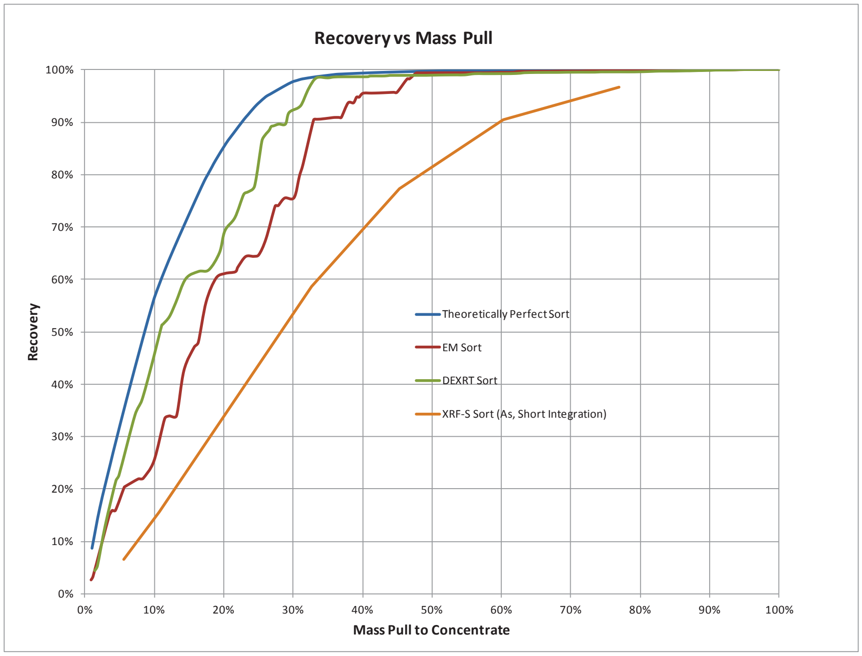
1. DEXRT - A dual energy Heimann 6040i x-ray scanner was used to acquire the x-ray transmission (DEXRT) characteristics for each rock.
2. OPTICAL - A benchtop optical sorter was used to acquire an image of each rock.
3. EM - The conductivity and magnetic susceptibility response was acquired with the GDD MPP EM2S+ probe.
4. XRF-S - The results were acquired with our benchtop test rig system configured to simulate a full scale XRF-S sorter.

The 100 rocks were assayed at ALS Global in North Vancouver, BC.

Figure 13.3 shows the theoretically perfect recovery to be 99% of the gold at a 33% mass pull to concentrate. The theoretically perfect sort curve for this sample was obtained by ordering the rocks in descending gold grade. The tests showed that DEXRT sorting appeared to have the best potential for sorting by grade as the DEXRT recovery curve approaches the theoretically perfect curve at approximately 35% mass pull to concentrate. The DEXRT sort showed a 98% gold recovery at 33% mass pull to concentrate.

The specimens are also amenable to sorting with the XRF-S sensor used in this study, although the upgrading is not as significant as with the DEXRT sensor. A mass pull of 60% to concentrate resulted in 90% recovery of the gold.

Figure 13.3: Gold Recovery versus Mass Pull Using Various Sorting Sensor

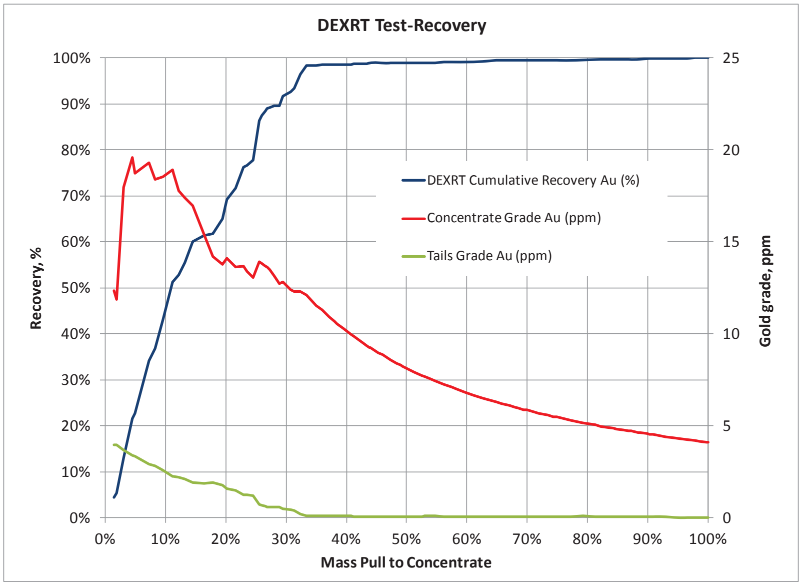


Although Figure 13.3 appears to show EM sorting with similar performance to the DEXRT sensor the sort results from this test for this EM sensor are only applicable at the laboratory scale and not useful for a full scale sorter.

Optical sorting results have not been included in the above figure as no pattern or visual characteristic could be defined that was usable by an optical sorter. The specimens were not amenable to optical sorting.

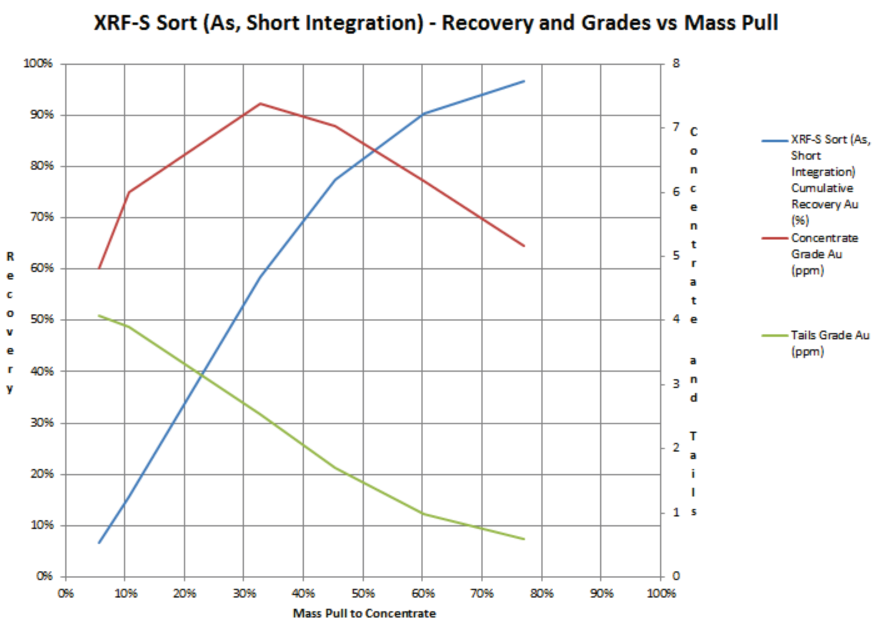
Figure 13.4 shows the result of sorting the composite (crushed to a diameter of less than 0.75 inch) consisting of a wide range of Bradshaw drill core intersections ranging in gold content from 0 g/t (waste rock) to over 10 g/t (high grade main zone) and averaging 4 g/t. Despite the relative low gold content of the composite, results from the test-work confirmed an extremely efficient separation by DEXRT test. Greater than 50% of the rock mass was rejected resulting in a final crushed rock product containing 12-15 g/t gold with only 2-3% gold losses.

Figure 13.4: DEXRT Test - Mass Pull versus Gold Recovery and Grade



For the XRF test, as shown in Figure 13.5 a mass pull of 60% to concentrate would result in a recovery of 90% of the gold. For this same mass pull the concentrate would grade 6.8 ppm gold with tails of 0.99 ppm gold. The graph shows the Au recoveries and grade, however, the sensor was not able to sort directly for gold grade.

Figure 13.5: XRF Test - Mass Pull versus Au Recovery and Grade



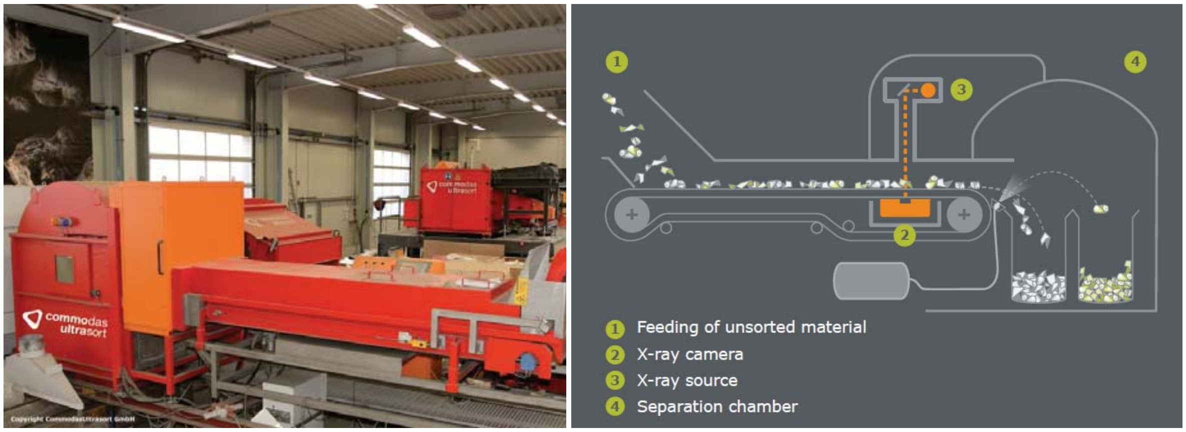
Pilot Test - 2012

A pilot test was conducted in April, 2012. One-ton bulk sample taken from the Bradshaw (previously named the Frankfield East) Deposit was provided by Gowest and sorted on automated sorters at the Tomra (previous Commodas-Ultrasort lab) in Wedel, Germany.

Test System - XRT Sorter

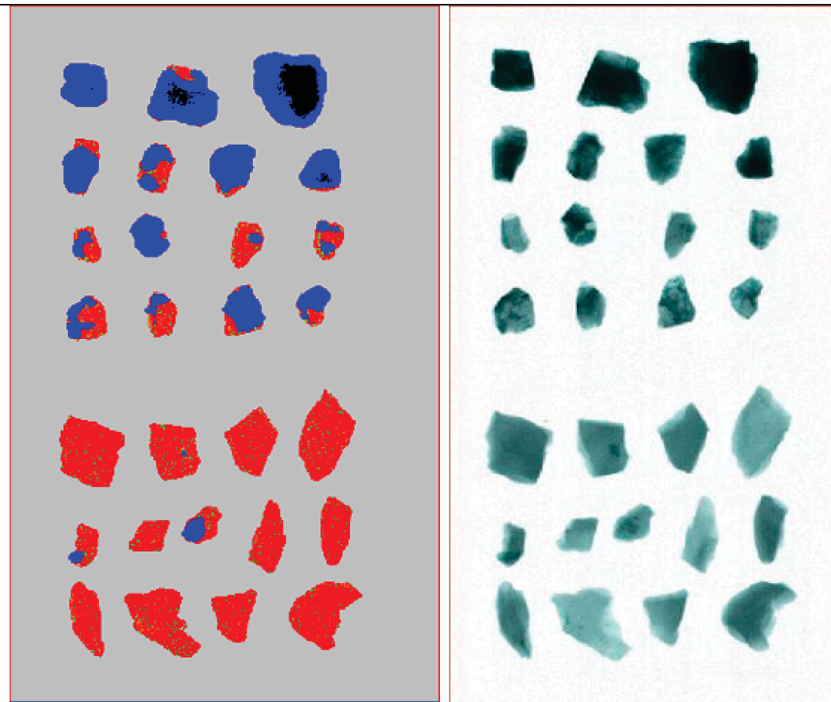
The two pictures below show a Slag Secondary XRT belt machine that was used in pilot test to scan and sort samples and a schematic of the functional principals of the XRT sorter.

Figure 13.6: XRT belt machine and schematic of the functional principals



The sorter functions by using a broad-band electrical x-ray source that is applied to the material to be sorted while it is moving along the belt. The X-ray sensor system below the material produces a digital image of the material being sorted, using two different energy bands. The X-ray attenuation through the material is different within the two bands and depends on both the thickness and the density of the material. An image transformation of the density images of the two bands then makes it possible to classify each pixel according to atomic density. Classification proceeds relative to a reference density, to which the system has been calibrated. Depending on the classification the selected particles are either "ejects", diverted upwards by air jets (Material Stream A) or "accepts" in the other stream (Material Stream B). It is important to note that "eject" refers to the material that the system has been configured to blow out of the material stream; this can be either the waste or the product. Figure 13.7 shows an example of a XRT image and the transformed image used to determine whether a particle is an "eject" or "accept". See Appendix A – Sorter for test results and further descriptions about the sorter.

Figure 13.7: Classified XRT image (left) and original raw XRT image (right)



Example of high grade particles.

Particles with visible Pyrite inclusions.

Examples of low grade particles.

Particles with no visible inclusions.

Test Samples and Procedure

Gowest shipped four barrels of rock specimens to the Tomra lab in Wedel, Germany. These barrels contained specimens of four grades ranging from low to high grade (soapstone, waste, low, and high) as well a range of sizes (1/4"-5/8", 5/8"-1", 1"-1½"). Subsamples were used to create a training set. The training set was created by passing rocks in the subsamples through a benchtop XRT sorter and then classifying the rocks into four predetermined categories based in their grade.

As shown in table below each grade was assigned a label in which it was tested for each of the three size fractions.

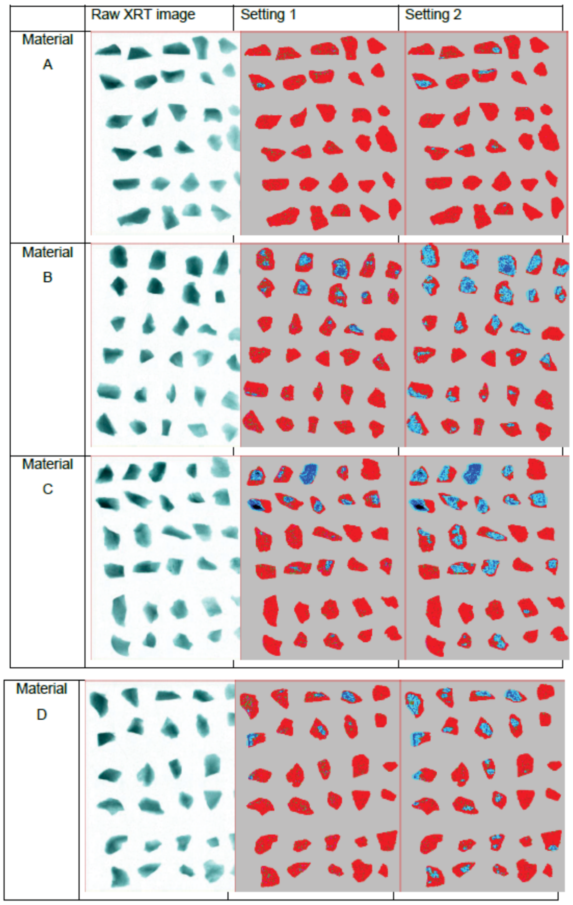
Sample	Grade
Material A	Soapstone
Material B	Waste
Material C	High Grade
Material D	Low Grade

Individual test images of the training sets were taken to enable the sorting unit to separate the material into:

- Higher grade ore as "Ejects*"
- Lower grade ore as "Accepts*"

Each material was tested twice using a different setting each time. The first setting was more selective of the high grade material that it "ejected", while the second setting was less selective and "ejected" more material. After these tests were completed, a mixed input fraction from each grain size was generated and sorted. Figure 13.8 displays images of the classes for the size range of 1"-5/8". Blue inclusions are indicative of high atomic density, "ejected" specimens.

Figure 13.8: XRT Images and Settings for all Grades, 1"-5/8"



Test Results

The pilot test used widely available production scale DEXRT ore sorting equipment and was performed under commercial operating conditions. The crushed material is transferred at high speeds along a conveyor belt in front of an x-ray sensor that analyzes the signatures of individual rocks to detect the FeAsS in the crushed ore. The sensors then trigger a series of individually controlled air jets to separate the uneconomic ore with less than approximately 0.3 g/t gold.

All of the sorted material processed in this test was sent to SGS Lakefield for assay.

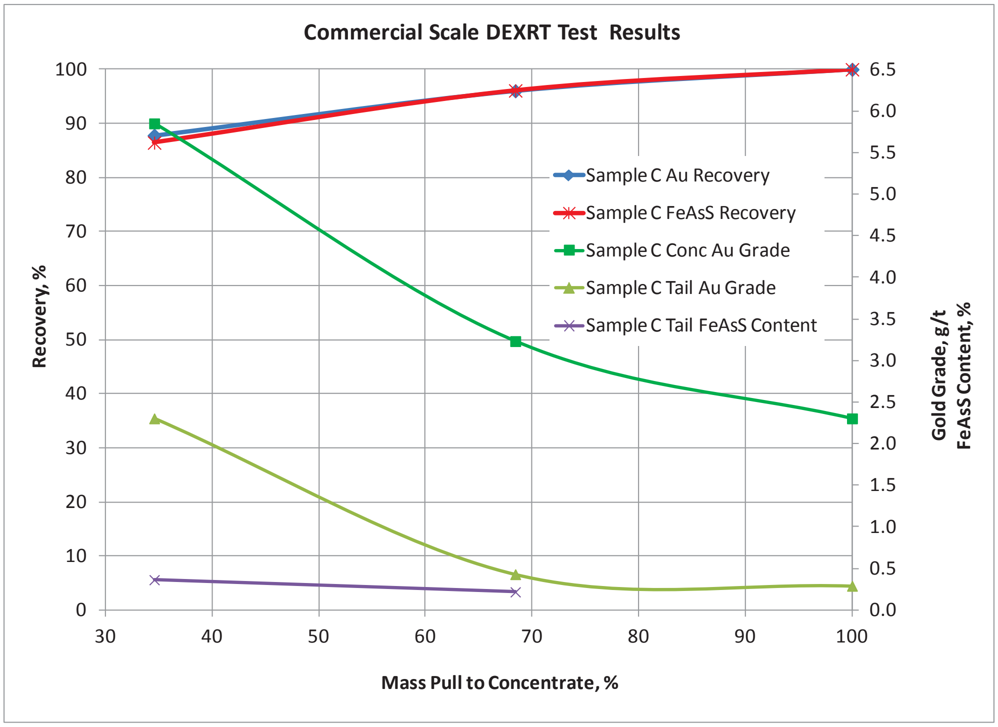
The figures in Figure 13.9 presented results for high grade (C) and low grade (D) samples, showed that the equipment used was able to detect FeAsS down to 0.1-0.2%.

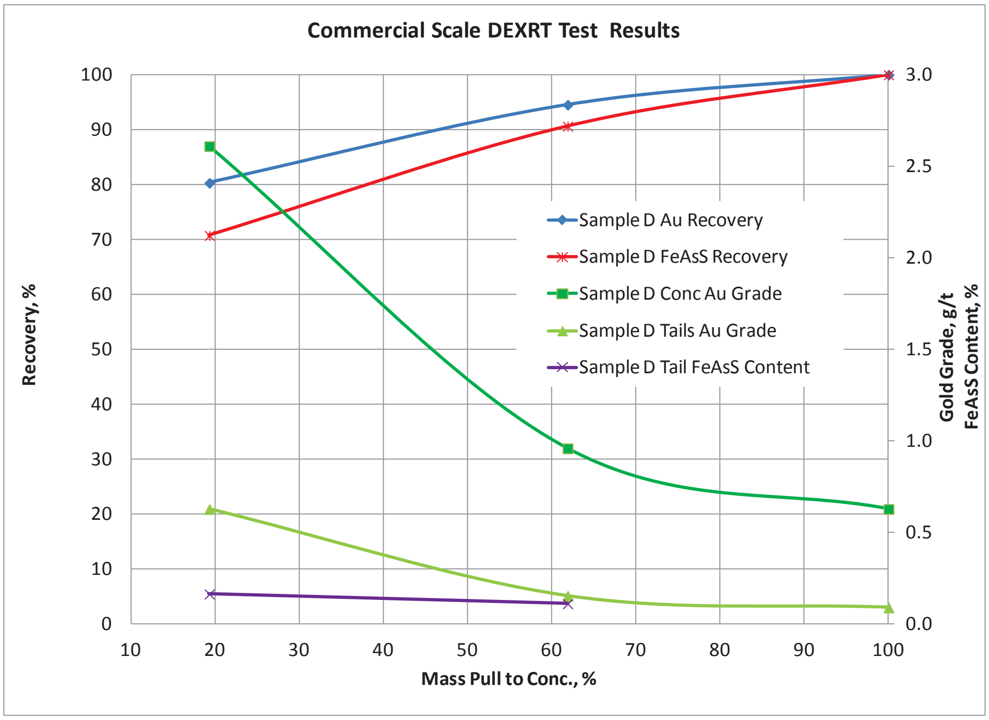
Conclusion

The tests showed that DEXRT sorting appeared to have the best potential for sorting by grade as the DEXRT recovery curve approaches the theoretically perfect curve at approximately 35% mass pull to concentrate. The specimens are also amenable to sorting with the XRF-S sensor, although the upgrading is not as significant as with the DEXRT sensor.

The bulk tests showed that XRT sorting is a very effective technology for sorting Gowest's material and it is a mature technology as compared to XRF-S sorting.

Figure 13.9: Pilot Test - Mass Pull to Concentrate versus Recovery and Grade - Sample



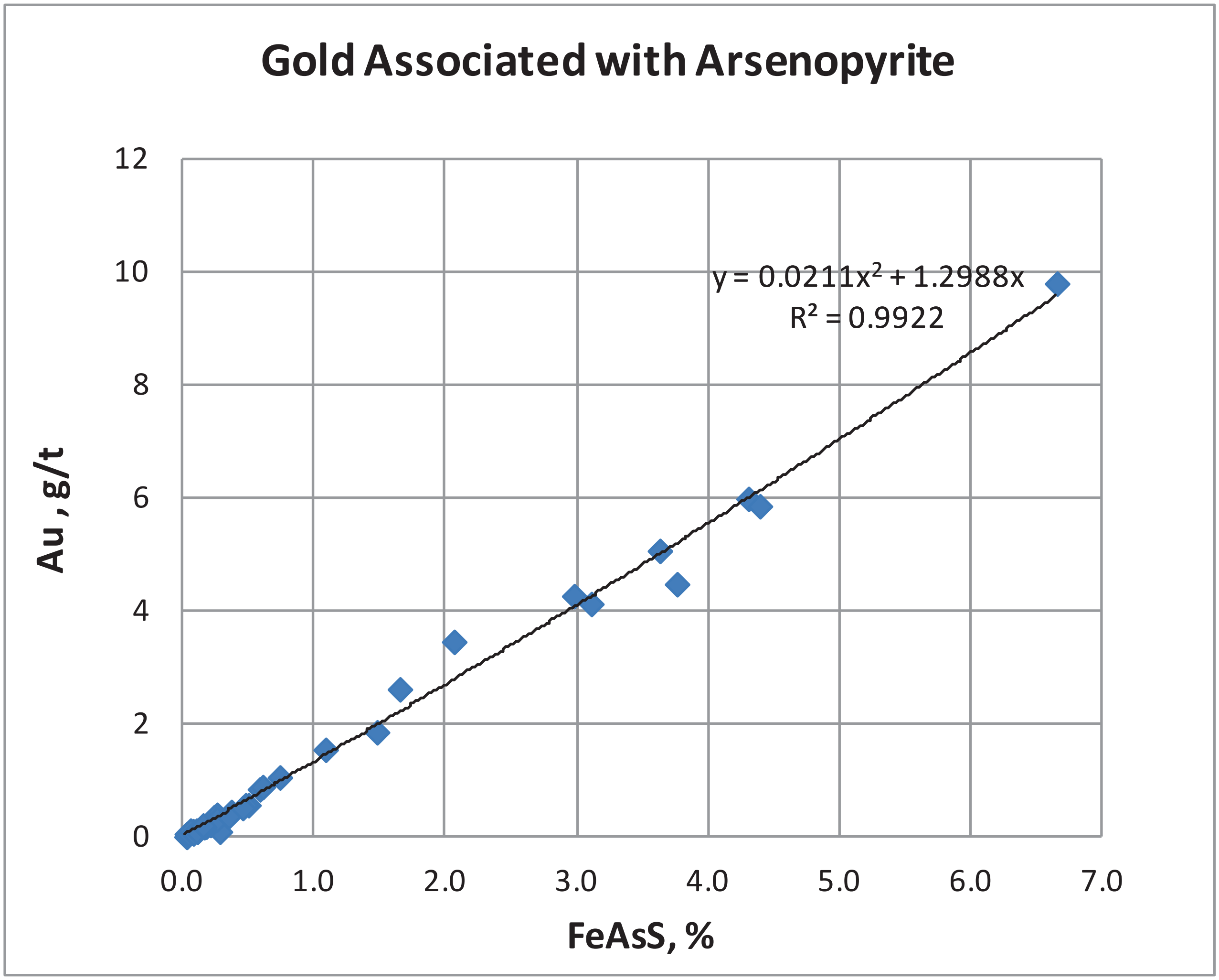


Note: Sample C = High Grade and Sample D = Low Grade

Figure 13.10 is based on 42 samples that were sorted via the DEXRT method, which shows that the gold in Bradshaw Deposit is closely associated with arsenopyrite (FeAsS), such that the higher the concentration of FeAsS, the higher the gold grade.

Details of these test programs summarized in this section can be found in various test reports as outlined in Section 27.

Figure 13.10: Bradshaw Ore Gold Associated with Arsenopyrite



14.0 Mineral Resource Estimates

14.1 Summary

An updated mineral resource estimate was prepared for the Bradshaw Deposit. The effective date of the updated Resource is January 12, 2015. The purpose of the update was to incorporate new drill hole and assay data from infill drilling completed in 2013 and 2014 since the last reported mineral resource in November 2012 (Gow et al, 2012). The new model and resource is based on diamond drilling and assay data from 322 drill holes (93,867 m) and 22,736 assay samples. Of this total, 62 drill holes (12,414 m) came after the November 2012 mineral resource. These new drill holes represent a significant increase in the density of geological and assay information available in the upper half of the deposit, which now has drill hole spacing of approximately 25 metres (previously it was approximately 50 metres drill hole spacing).

This updated mineral resource estimate is in accordance with the Mineral Resources/Reserves Classification as recommended by the CIM Committee on Mineral Resources/Reserves (CIM definitions). The estimates are set out in Table 14.1.

Table 14.1: Updated Bradshaw Mineral Resource Estimate

			3 g/t Au Cut-Off		
Category	**Depth**	**Zone**	**Tonnes**	**Au Grade**	**Ounces**
Indicated	500 m	MZ1	412,503	6.14	81,429
	500 m	MZ2	634,583	5.88	119,963
	400 m	HWZ1	345,637	6.35	70,563
	400 m	HWZ2	299,258	5.33	51,281
	400 m	HWZ3	194,029	6.93	43,230
	400 m	HWZ4	127,096	6.16	25,171
	400 m	HWZ5	53,094	8.01	13,673
	400 m	HWZ6	55,666	9.36	16,751
		Total	**2,121,866**	**6.19**	**422,059**
Inferred	below 500 m	MZ1	331,752	8.64	92,153
	below 500 m	MZ2	1,078,096	4.36	151,121
	below 400 m	HWZ1	693,934	5.16	115,119
	below 400 m	HWZ2	566,913	6.45	117,559
	below 400 m	HWZ3	443,788	11.85	169,073
	below 400 m	HWZ4	514,614	6.62	109,527
		Total	**3,629,097**	**6.47**	**754,553**

Notes

1. *CIM (Canadian Institute of Mining, Metallurgy and Petroleum) definitions were followed for Mineral resources.*
2. *Mineral Resources are estimated at a cut-off grade of 3 g/t Au.*
3. *Mineral Resources are estimated at a long-term gold price of US$1,200/oz., and a US$/C$ exchange rate of $0.80.*
4. *A minimum down-hole width of 2 m was used.*
5. *Bulk density of 2.89 g/cm^3 was used.*
6. *The Mineral Resource estimate is based on drilling up to December 2014.*

The mineral resource estimate reported here was prepared by Ms. Angela Falcon P.Geo. (No. 2325), a Gowest geologist trained in the use of Surpac 6.6 software. This work was supervised by Mr. Kevin Montgomery M.Sc. (App), P.Geo. Exploration Manager for, Gowest Gold, and verified by Mr. Neil N. Gow, the Qualified Person for this estimate.

14.2 Assumptions, Methodology and Parameters

The basis of the mineral resources estimate for the Bradshaw Deposit is discussed in this section.

- data validation and preparation
- geological interpretation and method selection
- geological modelling methodology
- analysis of top cuts
- compositing of assay intervals
- data analysis of gold assays (including statistics and probability plots)
- search parameters
- block model construction
- grade interpolation and boundary conditions
- rock density
- validation of gold grade estimates
- classification of estimates with respect to 43-101 guidelines
- resource tabulation and resource reporting

14.2.1 Data Validation and Preparation

The digital Bradshaw drill database consists of five tables stored in csv file format; Collars, Surveys, Lithology, Assays (gold in g/t), and Zone Select (a coded interval table with drill hole from-to intervals for each zone). All database units are metric and all drill hole collar locations are reported in UTM NAD83 Zone 17 coordinates. All the drill holes are considered as actual and were drilled by Gowest since 2004, with the exception of historical exploration drill holes on the Bradshaw Deposit which have been validated by Gowest. The data was loaded into Surpac 6.6 software and the Zone Select table updated manually in preparation for 3D modelling.

Table 14.2: General Properties of Bradshaw Drill Hole Database

Attribute Name	Statistical Data
Effective date of Bradshaw Resource	January 12, 2015
Ownership	Gowest Gold Ltd.
Bounding Limits (UTM Nad83 Zone 17)	Easting (X)[486100, 487250]; Northing (Y) [5398000, 5399000]; Elevation (Z)[-1000, 296]
Number of drill holes	322 drill holes, up to hole GW14-281 (279 drilled by Gowest incl. 6 failed holes*, 43 historical holes)
Number of drill holes with assays	322 drill holes
Total drilled meters	93,867 m
Number of assayed intervals	22,757
Total length assayed	22,227.56 m
Rock Density	2.89 g/cm^3
Number of rock density samples	11,258 samples for 10,625.7 m

**failed hole refers to holes which were abandoned in progress after going off target.*

Montgomery and Falcon carried out statistical and visual validation of the data (audit) prior to modelling and resource estimation.

Database auditing of assays revealed that many of the historical exploration drill holes were incompletely sampled within the mineralized zones, and that due to sampling gaps the data was not suitable for modelling or resource estimation. A total of 15 holes were removed from the database and stored in a separate file for record-keeping.

In the database there are non-assayed drill hole intervals, which correspond to barren material, for example: unaltered mafic volcanic rock, and ultramafic volcanic rock. The non-assayed intervals were replaced by one-half of the gold detection limit (0.0025 g/t Au).

The visual inspection of drill hole collars revealed that some of the collar elevations were out-of-line with the trending topographic profile on the north-south drill sections. The Bradshaw drilling area is generally flat with very little outcrop and so it is considered that any drill holes within 20-50 metres of each other should have similar collar elevations, within ± 2 metre elevation. The 32 (of the total 322) collars which were out-of-line were manually shifted up or down in elevation by a maximum of ±6 m (averaging 3 m) to bring them in line with the topographic profile trend of the majority of drill holes. Additionally, a few historical exploration holes (drilled in 1970's and 1980's) whose precise location in the field could not be verified were adjusted slightly according to the topographic profile and by lining up their major ultramafic-mafic volcanic contact to the more recent Gowest holes at the Bradshaw Deposit.

Upon completion of the audit and the minor adjustments above there is a high degree of confidence in the quality of the input data and the database is suitable for modelling and resource estimation.

14.2.2 Geological Interpretation and Method Selection

The Bradshaw Deposit consists of at least eight sub-parallel gold enriched pyrite-arsenopyrite horizons within a broader gold bearing alteration zone proximal to a major structural/lithological contact running through the Frankfield Property and beyond. This contact is a N80E trending steeply north-dipping lithological contact between ultramafic flows to the south and mafic volcanic flows to the north, and a fault typically occurs within the ultramafics near or at the contact. The altered mafic volcanics are moderately-to-strongly sheared and it is believed they are part of a deformation zone termed “the North Pipestone Break”, part of the Pipestone Fault system. This deformation zone was considered to be the primary route of the sulphidic hydrothermal fluid which emplaced the mineralization. Flow contacts are roughly parallel with the major contact and are believed to be the structural controls on the enriched gold horizons.

The broad gold bearing alteration zone is variably up to 100 metres in true width, characterized by altered mafic volcanics containing gold grades from 0.1 g/t Au up to 1 g/t Au. Within the broad alteration zone are the gold enriched pyrite-arsenopyrite zones characterized by dark silvery clots and stringers of arsenopyrite texturally ranging from super-fine to coarse needles. Gold grades inside the zones are typically greater than 1 g/t Au. No visible gold has been observed in the zones during core logging. The enriched zones are visible in the core through the presence of arsenopyrite but generally not grade-predictable, meaning that it is not possible to distinguish between, for example, 2 g/t Au and 5 g/t Au in a core sample.

The eight mineralized zones, Main Zone 1 (MZ1), Main Zone 2 (MZ2) and Hanging Wall Zones (HWZ1-6) represent the gold enriched horizons and are parallel with the major contact, and interpreted to be laterally continuous along strike and at depth.

Earlier resource estimations at Bradshaw Deposit were completed using Voronoi polygons with the nearest neighbor (NN) grade estimation method (Trinder, 2011). This method assigns an average gold grade to a polygon whose extents are limited by the spacing of drill holes, and are of a uniform width that is equal to the drill hole intersection width. The local gold grade is calculated as the average gold assay value over the intersection width and applied to the entire polygon. Each polygon is supported by a single drill hole intercept, and as many gold assays as there are within the intersection (at least two in the case of the Bradshaw Deposit).

In comparison to other methods such as the inverse distance method (INV), or kriging (KR), the NN method is known to estimate gold grade with uninfluenced samples, whereas the INV method produces a bulls-eye or target pattern, where the gold grade diminishes with distance from the sample. Kriging was considered unreliable to use for this deposit due to the unstable variograms that were produced. Kriging has the effect of suppressing the expression of higher grade intersections by averaging the grade down using many samples, producing low average grades.

Table 14.3: Comparison of Estimating Methods

Interpolation Method	Zone	Volume	Tonnes	Au g/t	Oz
Inverse Distance	MZ1	138,818	401,184	5.05	65,135
Nearest Neighbour	MZ1	142,735	412,503	6.14	81,429

Extensive review of the grade distribution in block models produced using these three methods (NN, INV and KR) has been done by Gowest using the current updated wireframe model (see Table 14.3). Considering the drill hole spacing currently averaging 25 metres in the upper half of the deposit and the disseminated style of the arsenopyrite-gold mineralization in the tabular zones, the nearest neighbor method has been selected as the best method to represent the gold distribution at Bradshaw. The testing of the gold continuity (discussed below) shows that regions in the model with high-grade gold (over 3 g/t Au) have good predictability over distances of 20-50 m. It is neither beneficial nor rational to average down the local grade of a polygon by averaging with samples further away.

The methodology for modelling the grades at Bradshaw was as follows: calculate the average grade over the mineralized interval by averaging adjacent length-weighted gold samples over the mineralized interval (defines as continuous gold mineralization over a minimum of 2 metre core length). Because these values represent the average of at least two samples there is a high level of confidence in extending the influence of that calculated grade to the margin of the polygon, constrained inside a smoothed wireframe.

Additional conservatism is built in to the model by using the wireframes as block modelling constraints, where the zones are modelled as smooth and continuous horizons rather than discrete polygons with a fixed uniform width. In this way, relatively wider and narrower intersections are smoothed and over-estimation of tonnage along polygon edges is minimized.

Also, in some areas of individual zones the deposit is as narrow as 1.5m horizontal width. It is understood that these areas will be filtered out by the MSO software parameters discussed later in this report. Only resources with amenable mining widths (greater than 2 m) and above mining cut-off grade (3 g/t) will be considered for conversion to reserves.

The Gowest drilling campaigns on the Bradshaw Deposit since 2010 included metallurgical drilling that have demonstrated the gold grade continuity in select areas of the deposit. Analysis of gold results from shallow metallurgical drilling (100 metre vertical depth) in two clusters of closely spaced drill holes was undertaken in 2012 in order to examine the grade variability over short distances. The clusters were defined as being within 20 x 20 metre blocks in longitudinal view. In the east cluster there were 11 holes, and in the west cluster there were 6 holes. The study results indicated good and predictable gold continuity in these areas. The chances of intersecting gold grades below 3 g/t Au in the MZ1 were 36% (East) and 16% (West). In the MZ2, it was 27% (East). These are good probabilities for gold mineralization and are reflective of the disseminated nature of the gold-arsenopyrite mineralization at the Bradshaw Deposit.

In addition, down dip drill testing was conducted into the MZ1 East cluster in 2012 to collect mini bulk samples (250 kg each) for ore sorter testing. Four holes were drilled targeting a section of the MZ1 zone. The holes intersected the MZ1 where predicted and returned the expected lengths of mineralization to collect large enough samples for the ore sorter testing.

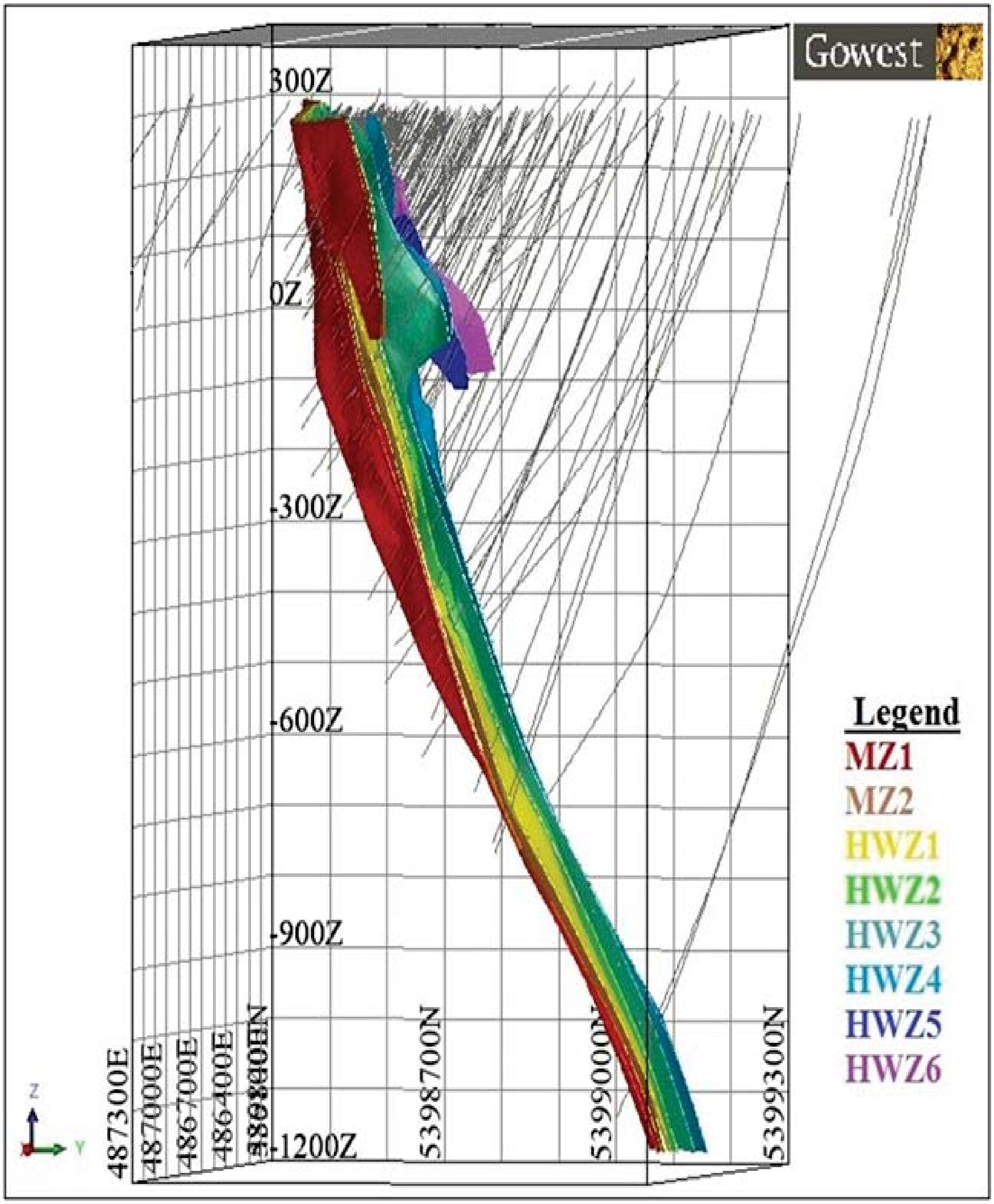
The polygonal nearest neighbor method is considered by the Gowest geological team to be the most suitable for estimating the grade of gold mineralization at the Bradshaw Deposit, and is further supported by the gold continuity studies discussed above.

14.2.3 Modelling Methodology

The assay data for each drill hole within the altered/mineralized lithology were assigned into zones based on their relative distance from the main contact and their geological association (logged by Gowest geologists as main zone or mineralized zone). For example, the two zones proximal to the contact, MZ1 and MZ2, are part of a single distinctively altered geological unit, termed the Main Zone. The degree of alteration and shearing tends to decrease with distance away from this zone and up into the hanging wall lithology, where the alteration style is more quartz/carbonate breccia. Zones occurring within this lithology are called Hanging Wall Zones, as in HWZ1-6. Criteria for identification of any zone was continuous down-hole gold mineralization over at least 2 metres of core length, traceable to similar intersections in drill holes below and above observed in 25 metre wide cross-sections. The Hanging Wall Zones were defined by following the geological continuity of the mineralization in adjacent holes and sections. Because the lateral distribution of grade continuity within the zones is variable, a minimum grade requirement was not used, and rather the mineralized interval was selected to represent the zone where a drill hole passed through.

Following tabulation of the zones, Leapfrog software was used to construct eight wireframes representing each of the mineralized zones (Figure 14.1). Zones were checked visually for snapping-selection accuracy, overlaps, and triangulation errors. A "pinch-out" option was employed to have the zone width reduced in the rare occurrence of a non-mineralized interval and at the margins of the zones where they terminated. Wireframe edges were digitized to a maximum distance of 25 metres east, west, and at depth, outside of the drill holes defining the deposit margins (and/or the end of the area tested by drilling). All zones were free from errors and validated by the software.

Figure 14.1: Bradshaw Deposit Wireframe Model (View Looking West)



The widths of the mineralized zones range from >1to 21.7 metre core length Table 14.4). A minimum core length of 2 metres for wireframes was selected in order to model the deposit for extraction using current mining methods. The average width of the resource polygons using > 3 g/t Au cut-off grade in the zone wireframes is 3.49 metre core length.

Table 14.4: Wireframe Zone Widths (Composite Lengths)

Zone	Core Length (m)		
	Minimum	Maximum	Average
MZ1	2.00	12.40	2.44
MZ2	2.00	15.20	2.71
HWZ1	2.00	8.95	2.43
HWZ2	2.00	8.00	2.25
HWZ3	2.00	21.70	2.38
HWZ4	2.00	13.00	2.22
HWZ5	2.00	6.02	2.65
HWZ6	2.00	4.00	2.25

The mineralized zone wireframes were then used as 3-D solid constraints for block modelling, whereby only blocks within the wireframes were to be interpolated with gold values.

The Leapfrog software was also used to generate an updated DTM (digital terrain model) of the overburden/bedrock interface for use as an upper constraint in the block model.

14.2.4 Metal Price

A gold price of US$1,200/oz and a $US/$CAD exchange rate of $0.80 was used for the estimates. The average gold price used for this study in Canadian dollars is $1,500.00 CDN/ounce. The lower gold price of US$1,200/oz is considered to allow a cushion in the long term price.

14.2.5 High Grade Capping

Of the 22,757 gold assay samples in the Bradshaw database, 18 samples were greater than 25 g/t Au and 10 samples were greater than 30 g/t Au (Figure 14.2). The highest gold sample to date is 40.9 g/t Au over 1 m, in drill hole GW11-198 from 132 to 133 metres down-hole depth. Basic statistics and a probability plot are shown below based on the data presented in Table 14.5. Based on this analysis, no top-cutting was necessary and the effect of top-cutting on the resource estimation is deemed negligible.

Figure 14.2: Frequency Distribution of Gold Assays In Bradshaw Database

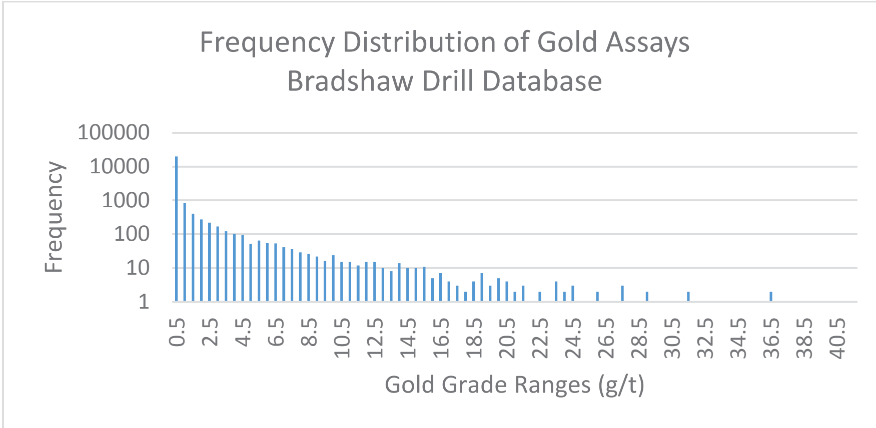


Figure 14.3: Log Probability Plot of Bradshaw Gold Assays

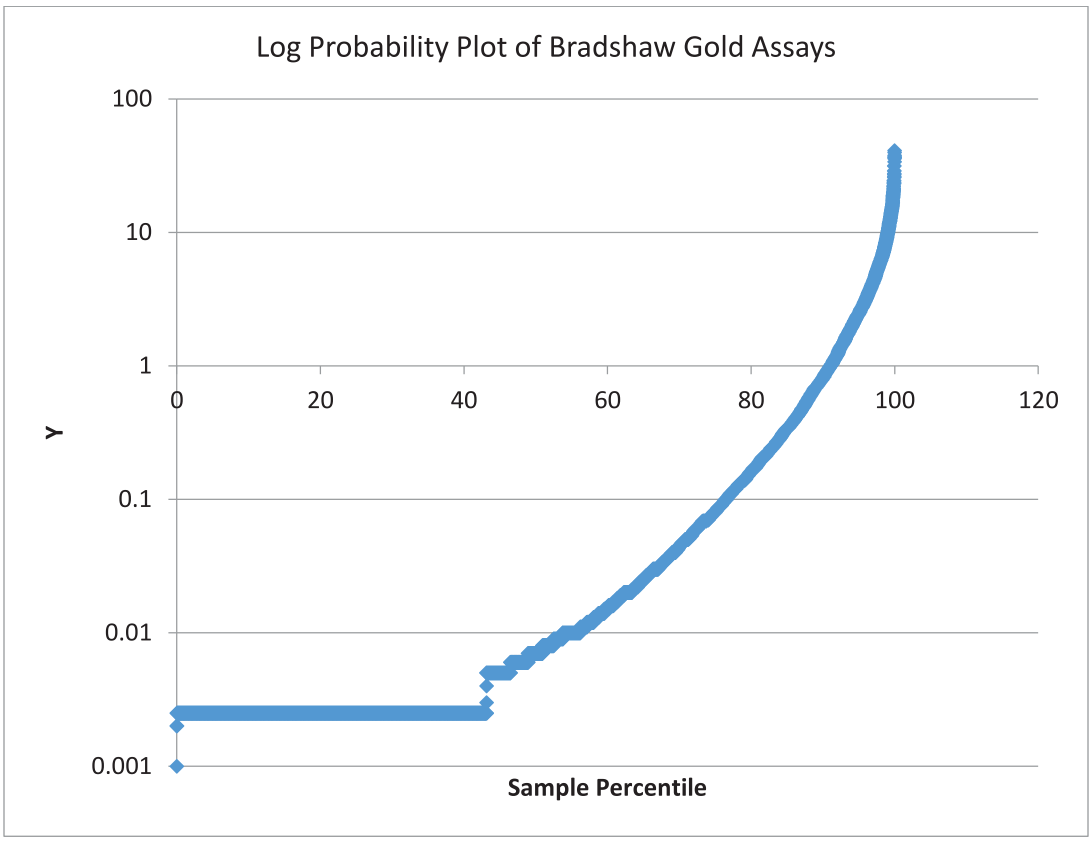


Table 14.5: Assay Statistics of The Bradshaw Drill Hole Database

Basic Assay Statistics	
Average sample length	0.98
Number of samples	22,757
Mean	0.47
Median	0.01
Maximum	40.90
Variance	3.85
Standard Deviation	1.96

14.2.6 Compositing

Wireframes of each zone were used to codify, then extract the gold assay sample data within each zone. Each of the zones represented by a wireframe then had a unique set of composites created for estimation. Each composite sample is the result of summation and averaging of the length-weighted raw gold assays.

The composites are of variable length depending on the length of the mineralized interval (or zone). The average assay sample length is 1 metre and the minimum mineralized zone intersection length is 2 m, therefore the shortest composite is 2 m. In other words, composite length is equal to mineralized zone length as shown in Table 14.4).

Basic statistics for all of the drill hole composites used in the preparation of the estimate are listed in Table 14.6.

Figure 14.4: Frequency Distribution of Gold Composites In Bradshaw Database

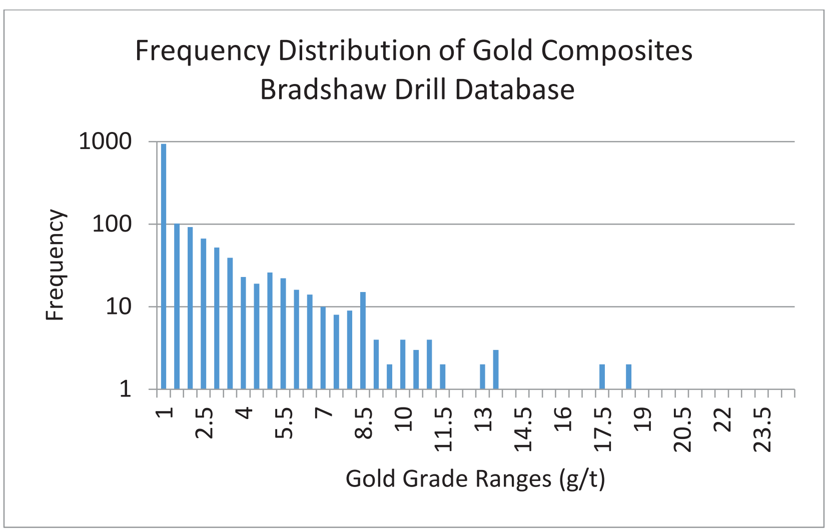


Table 14.6: Composite Statistics of the Bradshaw Drill Hole Database

Basic Composite Statistics	
Average composite length	2.42
Number of composites	1,490
Mean	1.49
Median	0.42
Maximum	22.3
Variance	6.16
Standard Deviation	2.48

14.2.7 Search Parameters

Search ellipsoids used in the sample selection for grade estimation of all zones were constructed based on the tighter infill drilling of the upper Bradshaw Deposit and the geometry of the zones. The previous resource estimation utilized a search ellipsoid with a 100 m radius (Gow, 2012). In this updated resource, an isotropic search ellipsoid with a 50 metre radius was used for each zone. Interpolations were completed with a single pass for each zone.

14.2.8 Block Model Parameters

The Bradshaw resource estimation block model consists of 4 x 4 x 2 metre blocks, with 2 m in the Y (north) direction to compensate for the relative narrow geometry of the zones. Standard sub-blocking was used down to a minimum block size of 1 x 1 x 0.5 metre (0.5 metre in the Y direction) to obtain a better fit with the wireframes. The extents of the block model are as follows (UTM NAD83, Zone 17) as shown in Table 14.7.

Table 14.7: Bradshaw Block Model Parameters

	Minimum	Maximum	Total extents	Block Size
X : Easting	486100	487252	1152 m	4 m, sub-blocks 1 m
Y : Northing	5398400	5399200	800 m	2 m, sub-blocks 0.5 m
Z : Elevation	-1200 m	300 m	1500 m	4 m, sub-blocks 1 m

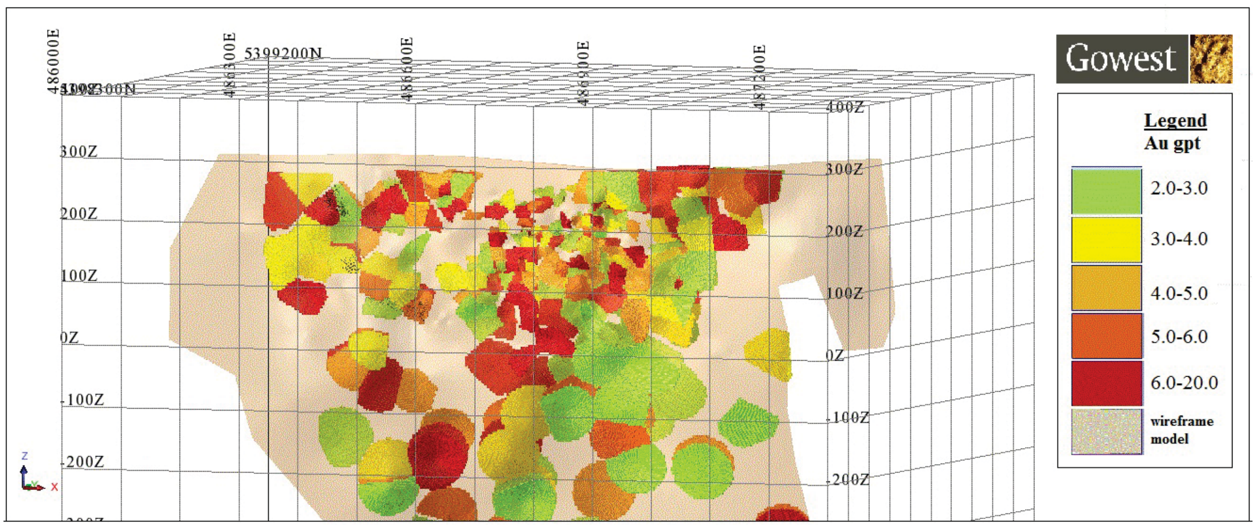
14.2.9 Grade Interpolation And Boundary Conditions

The Bradshaw block model was interpolated using Nearest Neighbor function in Surpac 6.6. The mineralized zone extents for the wireframe model of the Bradshaw indicated resources (upper 500 metres of the deposit) are shown in Figure 14.5. Each zone was interpolated individually using the composites belonging to the corresponding zone. Interpolation of grades into each zone was constrained using that zone's unique wireframe and a topographic surface (the overburden-bedrock contact) to prevent grades from populating blocks outside of any mineralized zone.

The interpolation process assigned one composite from each hole to each polygon, with each composite consisting of a minimum of 2 metres of assay-sampled material. In other words, the weighted average gold value of the entire intersection was applied to the appropriate polygon. The search distance for the interpolations was 50 m, and in general, the search did not extend past 25-35 metres before encountering an adjacent polygon for the indicated resources.

The interpolations were checked by reporting the total volume of the wireframe and reporting the total volume of blocks within each wireframe, to ensure that there were no blocks populated with gold values outside of the wireframes or in the overburden. The result of the checking confirmed that the interpolation plan was successful and no blocks outside of the wireframes or above the overburden contact were estimated with gold grades.

Figure 14.5: Bradshaw Polygonal Block Model Longitudinal View of Indicated Resources



14.2.10 Density

A specific gravity of 2.89 g/cm^3 was used to estimate the tonnage for the mineral resource estimate. This is based on the mean average bulk sample density of 6,722 gold mineralized core samples from drill holes GW10-45 to GW12-212. These readings were completed at the respective analytical laboratories using a pycnometer.

14.2.11 Dilution Gold Grade

Gowest conducted a study to determine the gold grade of rock material immediately outside the Bradshaw Deposit zone wireframes in early 2015. The study utilized all the Gowest drill hole mineralized intersections in the zone wireframes. It involved averaging the gold grade (individual assays) of two samples (averaging approximately 2 metre core length) either side of all the mineralized drill hole intersections. Several cases were examined using different cut off grades (2, 3, 4 and 5 g/t Au) and intersection lengths (2, 3, 4 and 5 m). The mean average dilution grade was determined to be 0.71 g/t Au for a cut-off grade of 3 g/t Au over 2 metre down-hole length.

14.2.12 Validation of Gold Grade Estimates

Gowest validated the Bradshaw block model resource estimate using:

- Volume checks
- Visual inspection of the model against the input composites
- Comparison of input grades with tonnage weighted output grades
- Grade trend (swath) plot
- Grade tonnage curves

Volume Check

A comparison of the total volume of the all blocks in each wireframe zone to the wireframe volume was conducted. The results indicate that the main wireframe zones were completely filled by blocks (<96 %). No over-estimation of the blocks volume occurred in any of the wireframe zones as shown in Table 14.8.

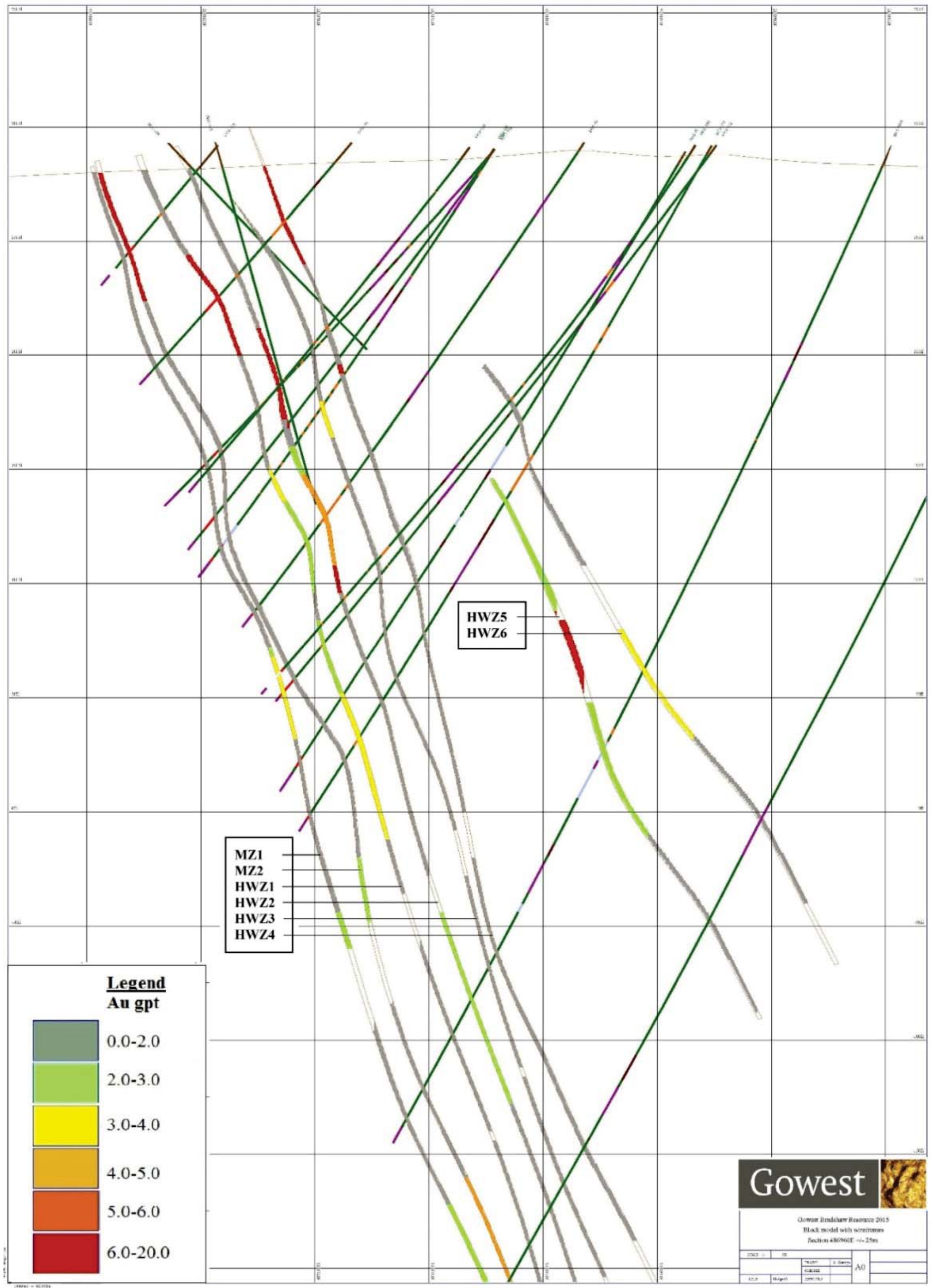
Table 14.8: Volume Comparison of Zone Wireframe and Blocks, Bradshaw Block Model

MZ1 Volume check:		**HWZ1 Volume check:**		**HWZ3 Volume check:**	
	Volume m3		Volume m3		Volume m3
Blocks	2,097,434	Blocks	1,704,242	Blocks	1,461,456
Wireframe	2,099,922	Wireframe	1,713,349	Wireframe	1,494,896
% wireframe filled	0.999	% wireframe filled	0.995	% wireframe filled	0.978
MZ2 Volume check:		**HWZ2 Volume check:**		**HWZ4 Volume check:**	
	Volume m3		Volume m3		Volume m3
Blocks	2,390,575	Blocks	1,610,442	Blocks	1,326,782
Wireframe	2,396,354	Wireframe	1,626,814	Wireframe	1,384,308
% wireframe filled	0.998	% wireframe filled	0.990	% wireframe filled	0.958
HWZ5 Volume check:		**HWZ6 Volume check:**			
	Volume m3		Volume m3		
Blocks	49,014	Blocks	42,184		
Wireframe	72,726	Wireframe	51,671		
% wireframe filled	0.67	% wireframe filled	0.82		

Visual inspection of the model against the input composites

The gold grade estimates in the block model show a good local correspondence with the drill hole input composite grades as observed in longitudinal sections and cross sections (Figure 14.6). Using cross-sections showing blocks with drill holes, a selection of local polygon grades were compared with their input composite grades and were found to be in good agreement.

Figure 14.6: Typical Bradshaw Block Model And Wireframes Section (486900e)



Grade Trend (Swath) Plots

Sectional validation graphs were created to assess the reproduction of local means and to validate the grade trends in the block model. These plots compare the mean of the estimated grades to the mean of the input grades within block model slices by easting (vertical) and by elevation (horizontal) for the portion of the deposit estimated. The graphs also show the number of input samples on the right axis, to give an indication of the support for each swath.

Validation graphs were created for each zone. All graphs are located in Appendix C, 14.1 - Geology and the plots for Main Zone 1 (MZ1) are shown in Figure 14.7 and Figure 14.8. The swath width was 25 metres as the approximate drill hole spacing and polygon size.

These plots indicate that in general there is good local reproduction of the input grades in both the horizontal and vertical directions. In some cases, the block model showed higher values than the input composites, and in other cases, the input composite values were higher than the block model values. These cases were checked, and it was observed that in swaths with less supporting data, the grade is biased by the polygons from adjacent swaths. In other words, nearby polygons are exerting influence on the average block model grades in the swath, because a portion of the polygon is inside the swath. In these cases, the portion of the polygon from the adjacent slice is affecting the average grade of that swath.

Based on the average drill hole spacing, selected estimation method and maximum grade population ranges of 50 m, all polygons are considered as reasonably representative of the grade of their corresponding intercepts.

Figure 14.7: MZ1 Vertical Swath Plot, Indicated Resources

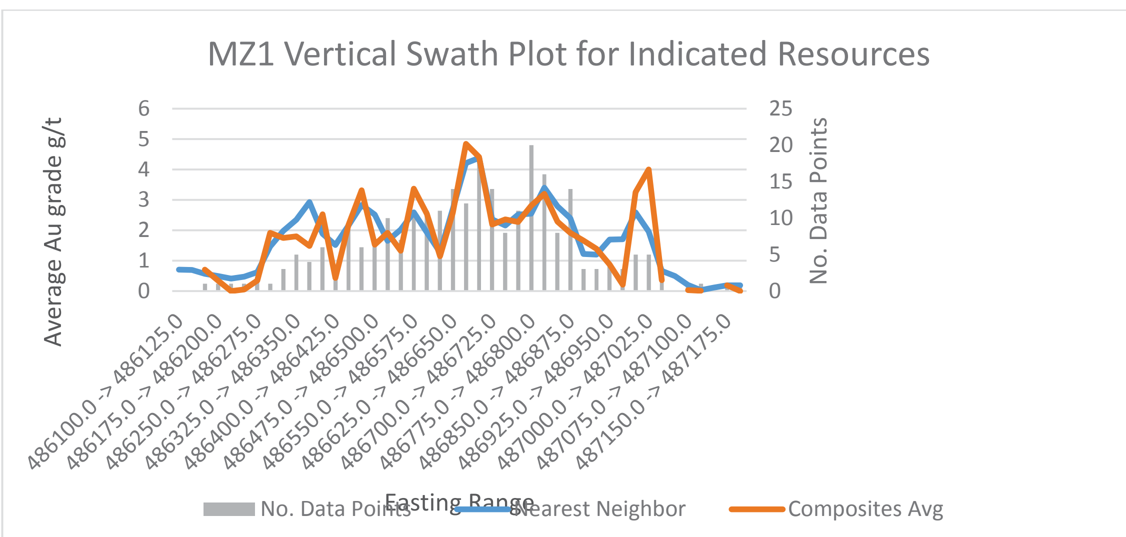
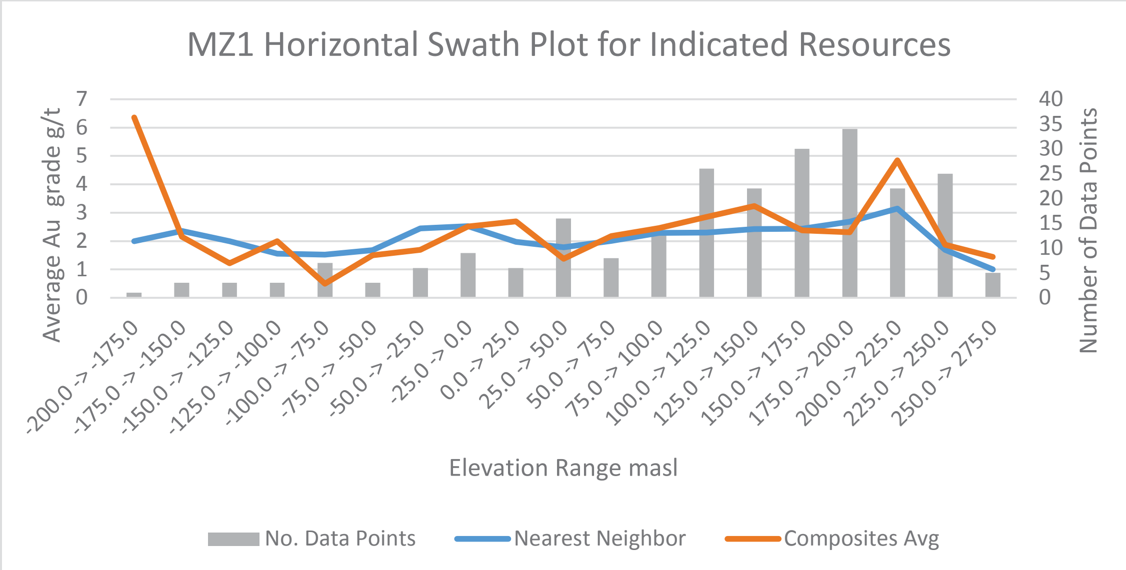


Figure 14.8: MZ1 Horizontal Swath Plot, Indicated Resources



Grade Tonnage Curves

Plots of tonnage versus the average gold grade at cutoff grades from 0 to 6 g/t Au were produced for the MZ1, MZ2 and HWZ1-4 zones of the Bradshaw Deposit. The plots show smooth curves in all cases (see Appendix C, 14-2 Grade Tonnage Curves.

14.2.13 Mineral Resource Classification

The resource classification definitions used for this estimate are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "CIM Definition Standards".

Measured Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Indicated Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical, and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Inferred Mineral Resource: that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence, and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

The classification of resources into "Indicated" and "Inferred" categories at the Bradshaw Deposit is based upon the level of confidence in the quality of the data and that the distribution of data in the deposit area reasonably represents the potential of the deposit. The recent infill drilling campaign in 2013-2014 resulting in an increase in the average drill hole spacing (25 m) provides thorough coverage of the deposit in the upper part of the deposit (a 400 to 500 metre vertical depth below surface or -100 to -200 m above sea level). Below these elevations the distribution of drill hole data points is between 50 to 150 metre spacing.

Table 14.9: Bradshaw Resource Classification Parameters

Zone	Indicated	Inferred	Vertical Metres
MZ1	Surface to -200 m ASL	Below -200 m	500 m
MZ2	Surface to -200 m	Below -200 m	500 m
HWZ1	Surface to -100 m	Below -100 m	400 m
HWZ2	Surface to -100 m	Below -100 m	400 m
HWZ3	Surface to -100 m	Below -100 m	400 m
HWZ4	Surface to -100 m	Below -100 m	400 m
HWZ5	Surface to -100 m	Below -100 m	400 m
HWZ6	Surface to -100 m	Below -100 m	400 m

14.2.14 Mineral Resource Reporting

Given all parameters listed above, Gow is of the opinion that the current Mineral Resource Estimate (Table 14.10) can be classified as Indicated and Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves. The Mineral Resources are reported in Appendix C, 14-3 at different gold cut-off grades from 2 to 4 g/t Au.

At a cut-off grade of 3.0 g/t Au , the current estimation represents 2,121,866 tonnes at 6.19 g/t Au (422,059 oz Au) in the Indicated category and 3,629,097 tonnes at 6.47 g/t Au (754,553 oz Au) in the Inferred category (Table 14.9).

Table 14.10 Updated Bradshaw Mineral Resource Estimate (as of January 12, 2015)

Category	Zone	3 g/t Au Cut Off			
		Volume	Tonnes	Au Grade g/t	Ounces
Indicated	MZ1	142,735	412,503	6.14	81,429
	MZ2	19,579	634,583	5.88	119,963
	HWZ1	119,598	345,637	6.35	70,563
	HWZ2	103,550	299,258	5.33	51,281
	HWZ3	67,138	194,029	6.93	43,230
	HWZ4	43,978	127,096	6.16	25,171
	HWZ5	18,372	53,094	8.01	13,673
	HWZ6	19,262	55,666	9.36	16,751
	Total	**734,212**	**2,121,866**	**6.19**	**422,059**
Inferred	MZ1	114,793	331,752	8.64	92,153
	MZ2	373,044	1,078,096	4.36	151,121
	HWZ1	240,116	693,934	5.16	115,119
	HWZ2	196,164	566,913	6.45	117,559
	HWZ3	153,560	443,788	11.85	169,073
	HWZ4	178,067	514,614	6.62	109,527
	Total	**1,255,744**	**3,629,097**	**6.47**	**754,553**

Notes:

1. *CIM (Canadian Institute of Mining, Metallurgy and Petroleum) definitions were followed for Mineral resources.*
2. *Mineral Resources are estimated at a cut-off grade of 3 g/t Au.*
3. *Mineral Resources are estimated at a long-term gold price of US$1,200/oz, and a US$/C$ exchange rate of $0.80.*
4. *A minimum down-hole width of 2 m was used.*
5. *Bulk density of 2.89 g/cm^3 was used.*
6. *The Mineral Resource estimate is based on drilling up to December 2014.*

15.0 MINERAL RESERVE ESTIMATES

The estimated in-situ resources at the 3.0 cut-off grade for the Bradshaw Project includes an underground resource, which is summarized in Table 15.1.

Table 15.1: Bradshaw Underground Resources

Method	Classification	Tonnes (millions)	Grade (gpt Au)	Ounces (thousands Au)
Underground	Measured (M)	0	0	0
	Indicated (I)	2.121	6.19	422
	Subtotal M and I	2.121	6.19	422
	Inferred	3.629	6.47	755
Totals	Total M+I and Inf	5.751	6.36	1,177

Based on the orientation, size, and suggested competence of the rock mass, cut and fill and underground longitudinal blasthole stoping have been selected as the most suitable mining methods. A productivity and cost analysis was performed for both mining methods and longitudinal blasthole mining method was recommended. As Bradshaw is located in Northern Ontario, personnel with mining experience are assumed to be available. The mining method will require mobile equipment commonly used in the industry.

15.1 Mining Dilution

Two sources of dilution have been considered in estimating the Bradshaw resources mined to surface.

Planned dilution includes low grade (below cut-off) material and/or waste rock that will be mined and will not be segregated from the mineralization. Sources of planned dilution include:

- Waste rock or low grade material that is drilled and blasted within the drift profile of sills.
- Waste rock or low grade material included within the drilling limits of the stope design. This includes internal waste pockets and footwall and/or hanging wall rock that has been drilled and blasted to maximize recovery and/or maintain favourable/necessary wall geometry.

Several representative stopes have been designed to determine a global average grade for planned dilution for Bradshaw. Planned dilution is directly reported for the representative stopes from the block model.

Unplanned dilution includes low grade resource, waste rock, and/or backfill from outside the planned stope limits that overbreaks or sloughs into the stope during the mucking cycle and is delivered to the mill. Unplanned dilution was estimated at 15%. This is an industry "standard" and is reasonable considering the discussions with geologists during a site visit and the visual inspection of core logs. The core indicated that the ground was of good quality in all zones except where the zones are close or in contact with the ultramafic rocks.

15.2 Mining Recovery

Two recovery factors have been considered in establishing the mined resources.

Planned recovery includes the in-situ block model resource that will be accessed, developed, and mined.

A mining recovery factor has been applied to account for material that is planned to be mined within the confines of the stope limits, but will not be recovered due to factors such as:

- Poor ground.
- Blasting difficulties (ground does not break properly and cannot be recovered).
- Geometry of the mineralization.
- Broken material that cannot be extracted (i.e. resting on the footwall, or around corners).
- Unplanned mineralized pillars left in place.

A 95% mining recovery has been used in estimating the mineable reserves because the majority of the stopes will be less than 3 metres wide, and recovery is generally increased with narrow mining stopes.

15.3 Block Model Cut-Off Grade

A 3.0 g/t Au block model cut-off grade was initially used to identify the mining areas. The following assumptions were made to establish the initial cut-off grade:

- Mine Operating Cost — $80 per tonne (ore to the portal)
- Mill Operating Cost — $40 per tonne (custom milling and transportation)
- Sustaining Capital Cost — $20 per tonne
- Total Cost — $140 per tonne

A gold price of C$1,500 per ounce ($48.23 per gram) was used which is a gold price of US$1,200.00 and a US$0.80 exchange rate.

15.4 Underground Mineable Reserve – Estimate

The mineable reserve for this deposit is based on the in-situ Indicated resources identified in the block model. There were no Measured Resources and Inferred resources were not used in any economic analysis in this report since an "Inferred Mineral Resource" has a great amount of uncertainty as to its existence and as to its feasibility. Readers are cautioned not to assume that all or any part of an "Inferred Mineral Resource" exists, or is mineable. The following methodology was used to estimate the mineable reserves mined to surface.

1. The block model was reviewed in plan and in section to identify mineral resources above the 3.0 g/t Au cut-off grade contained in a minimum 2 meter wide envelope (including dilution), and to confirm the longitudinal mining method. Some of the areas above the cut-off grade were regarded as "outliers". The cost of development to access the "outliers" was determined to be revenue generated from extraction, and the mineral resource contained in the outliers excluded.
2. Sublevels were designed at 30 metre vertical intervals, and vertical sections were cut through the model at 11 metre intervals along strike. A minimum mining width of 2 metre was applied and wireframe mining shapes were designed to encompass areas above cut-off ("outliers" excluded).
3. The in-situ tonnes and grade within each mining shape were extracted from the block model data. The mining shapes include any low grade (below 3.0 g/t) material within the shape. This material represents planned dilution.
4. The unplanned dilution was estimated at 15% based on visual inspection of core logs during a site visit to the Bradshaw project, and based on industry experience within the region.
5. The sill development was segregated from the mineral resources (in order to distribute development tonnes and grade based on the development schedule).
6. The gold is associated within a disseminated arsenopyrite-pyrite zones and the bulk of the level development between the economical stope blocks is within uneconomic zones of low grade gold bearing sulphide. This development is called "mixed" or incremental development material and is segregated from the mineral resources and not included with the ore reserve. However, material can be economically retrieved using the sorter and the revenue generated from the recovered gold ounces is added to the project to mitigate the cost of development. The development mix is considered incremental material and is of sufficient estimated grade (1.31 gpt Au) to cover the ore sorting, milling and refining costs.

7. The mining costs are applied up to the point the ore has been delivered from the underground mine to the portal. The costs for crushing, sorting, transport to the mill, milling and concentrating, smelting and refining are applied after the haulage of the ore to the portal and considered to be part of the process stream after underground mining.
8. Only Indicated resources were considered for economic extraction since there are no measured resources and inferred resources are too speculative to be considered for economic extraction.

The estimated mineable reserves and "mixed" development material included in the mining plan are summarized in Table 15.2. The mineral reserves are categorized as probable as per the CIM May 10, 2014 definition.

Mineable Reserves = 1,551,412 tonnes @ 4.92 g/tonne Au (in stopes) + 235,855 tonnes @ 4.19 g/tonne Au (sill development)

TOTAL Probable Recoverable RESERVES (factored for recovery and dilution) – 1,787,295 tonnes @ 4.82 g/tonne Au

A development schedule, production profile, and mine design have been prepared as a basis of estimate for the capital and operating costs. A life of mine cash flow analysis has been prepared. The cost estimates and cash flow analysis are included in Section 21 and 22.

Table 15.2: Estimated Mineable Reserves and "Mixed" Material by Level (g/tonne Au)

	STOPES				DEVELOPMENT Ore				"Mixed" or Incremental Development		
LEVEL	Insitu Tonnes	Recoverable Tonnes	Grade	Au ounces	Total Tonnes	Grade	Au ounces	Total Oz	Total Tonnes	Grade	Au ounces
45 (CROWN)	69,417	63,840	6.25	12,838	15,027	5.54	2,678	15,515	48,898	1.33	2,094
75 Total	143,919	138,327	5.24	23,291	23,732	3.74	2,852	26,143	61,065	1.35	2,646
105 Total	118,544	112,247	4.83	17,432	11,255	3.72	1,348	18,780	56,487	1.52	2,761
135 Total	66,471	62,575	4.29	8,632	8,628	4.36	1,209	9,841	45,388	1.25	1,819
165 Total	70,967	66,760	4.33	9,295	12,866	3.95	1,634	10,929	49,764	1.20	1,922
195 Total	102,325	95,679	4.21	12,954	13,790	4.66	2,068	15,023	54,459	1.55	2,709
225 Total	171,208	161,928	5.33	27,734	30,398	4.17	4,077	31,810	60,681	1.24	2,422
255 Total	133,302	128,134	5.33	21,954	21,202	4.33	2,950	24,904	39,741	1.12	1,435
285 Total	83,999	81,846	5.56	14,623	7,950	5.42	1,385	16,008	38,532	0.99	1,229
315 Total	50,706	48,293	4.37	6,779	8,480	4.09	1,114	7,893	39,561	1.22	1,550
345 Total	87,719	82,208	4.64	12,273	20,655	3.32	2,205	14,477	38,993	1.65	2,074
375 Total	149,432	142,000	5.12	23,366	30,389	4.09	3,993	27,358	24,544	1.51	1,195
405 Total	128,214	124,553	5.31	21,261	2,878	7.26	671	21,933	47,348	1.09	1,665
435 Total	35,733	32,729	3.61	3,803	5,915	3.07	584	4,387	18,352	1.20	710
465 Total	92,508	91,836	3.88	11,455	10,807	4.07	1,414	12,869	23,246	1.40	1,045
495 Total	118,689	118,457	4.63	17,624	11,880	4.19	1,601	19,225	19,194	1.12	691
Grand Total	**1,623,153**	**1,551,412**	**4.92**	**245,314**	**235,855**	**4.19**	**31,782**	**277,096**	**666,253**	**1.31**	**27,965**

16.0 MINING METHODS

16.1 Overview

The Bradshaw Mine (Bradshaw) design has been based on the block model "BradshawBMDec2014_Jan7.csv" supplied by Gowest on 7 January 2015. The mine design considers the resources in the measured and indicated categories between 295 metre elevation (surface) and minus 205 metre elevation (500 metres below surface). No inferred resources were considered in the mine design for this study. Engineering and cost assessment work has been completed on the measured and indicated resource material to a prefeasibility study level of detail.

The naming convention for the underground sublevels at Bradshaw is metres below surface (i.e. 125 Level is 125 metres below surface). The surface elevation is nominally 295 metres above mean sea level (i.e. 125 Level is at 170 metre elevation).

The ore zones between surface and minus -205 metre elevation are steeply dipping (60-85°) and comprised of several parallel lenses with an average width of 2-3 metres/lense to a maximum of 5 metres. These ore zones strike east-west with a current maximum strike length of 1,000 metres. The geometry of the Bradshaw Deposit zones in plan are shown in Figure 16.1, and in section in Figure 16.2. A longitudinal section of the ore zones are shown in Figure 16.3 and, Figure 16.4 presents the envisioned site plan and surface infrastructure. The existing infrastructure includes access roads. See Appendix F for a complete set of mine drawings.

Figure 16.1: 255 Level Mineralized Zone Plan with Proposed Stopes Outlines (black)

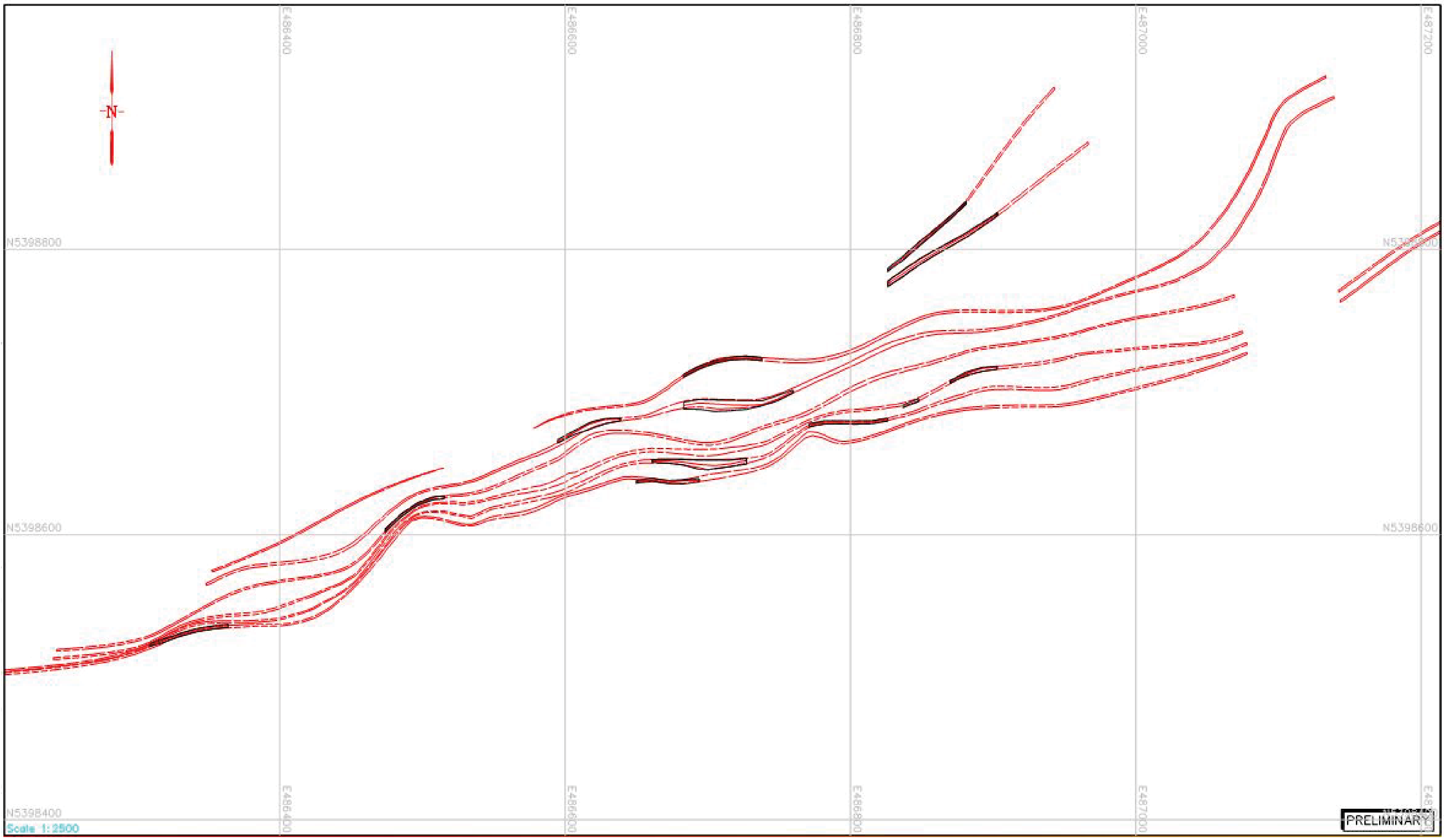


Figure 16.2: Mineralized Zone Section including Stopes (black): Section 486700 Looking East

Figure 16.3: Mineralized Zone including Stopes: Long Section Looking North (red)

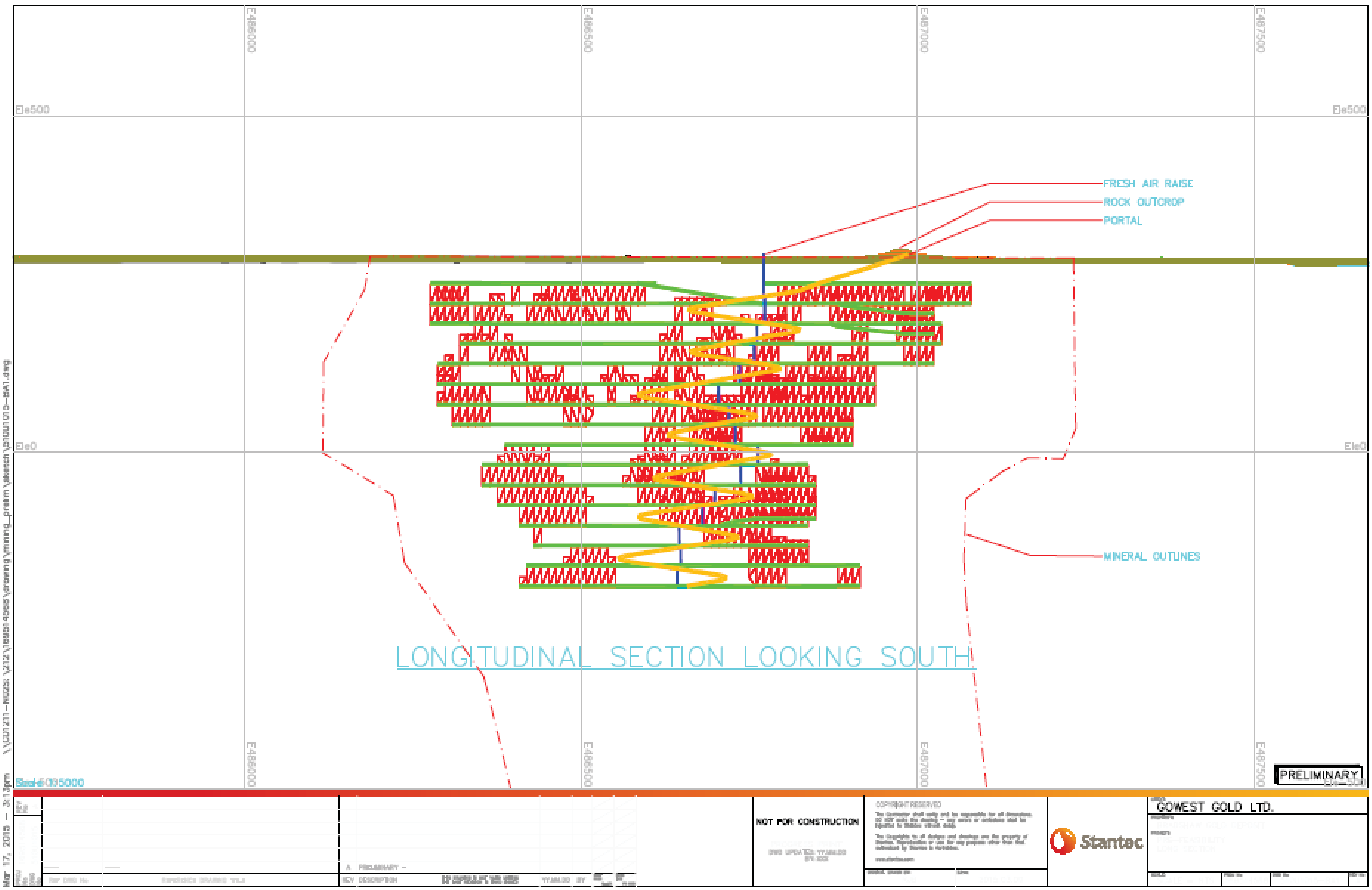
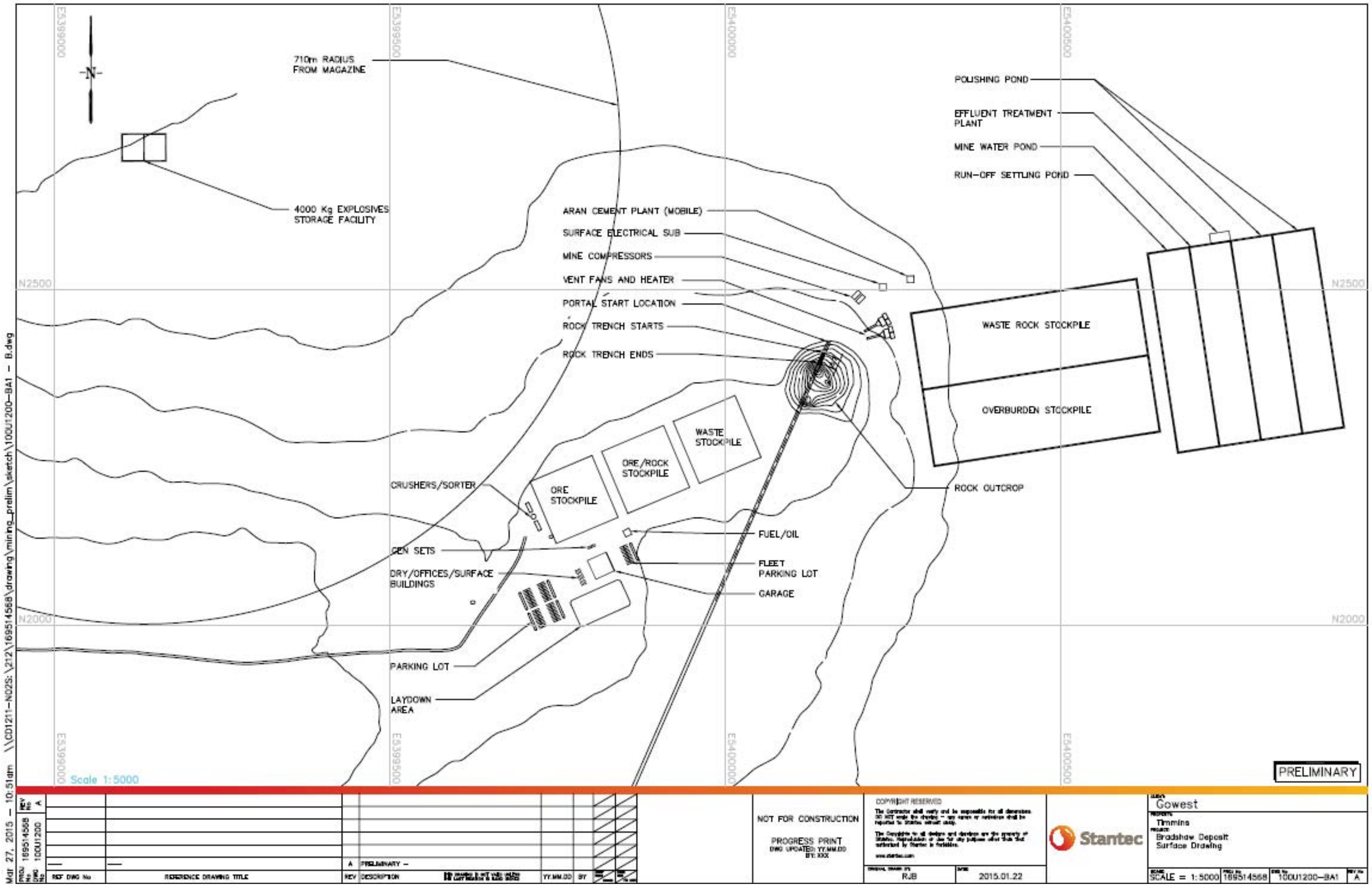


Figure 16.4: Proposed Surface Mine Infrastructure Site Plan



16.2 Underground Mining

16.2.1 Primary and Secondary Access Underground

The primary access to the underground mine will be via a single portal and main ramp from surface to the working levels. All active production levels will be accessed via the ramp (i.e. no captive levels) and personnel, materials, ore, and waste rock will be transferred via the ramp.

Secondary access/egress will be via the main return air raise. The return air raise will be constructed to draw fresh air to the ramp during the development phase and throughout the mine life, and will be equipped with a manway.

16.2.2 Stoping Methods

The Bradshaw Deposit is comprised of 8 steeply dipping (60-85 degrees) zones with a nominal thickness of 2-3 metres/zone and an overall strike length of 1,000 metres. The various ore zones are intermittent and discontinuous across the deposit, however, have continuity as sub-economic mineralized sulphide horizons that can be followed along strike and down dip. .

A trade off study on two mining methods was performed to determine which method was more suitable for this deposit. The two mining methods, mechanized cut and fill and longitudinal blast-hole, were reviewed to determine the individual mining costs and productivities for each method. The results of the study are shown in Table 16.1.

Table 16.1: Bradshaw Mining Method Costs and Productivities

Mining Method	Direct Mining Cost/Tonne	Stope Cycle (days)	TPD using 3.5 yrd LHD	TPD using 6.0 yrd LHD
Cut and Fill	$121.57	10 for 50m	130	160
Longitudinal Bulk				
20 m High	$95.58	14	140	175
30 m High	$80.67	21	150	200

For the trade-off study, tThe following mineability and dilution factors were applied to each method to determine the recovered gold ounces and revenues per tonne of mined ore. The results are shown in Table 16.2.

Table 16.2: Bradshaw Mineability, Dilution and Recoveries

Mining Method	Mineability/ Recovery	Dilution (External)	Au Oz Recovered	Value generated per tonne of mined	Revenue/Tonne (Value-Direct Mining Cost)
Cut and Fill	95%	22%	284,690	$191.96	$70.39
Longitudinal Bulk	85%	34%	268,869	$187.30	$106.63

As a result of this study, the longhole bulk mining method provided the following:

- Better revenue per tonne produced
- Lower direct mining cost
- Better productivities
- Reliable execution of the mining method

Thus, the longitudinal longhole (blasthole) stoping with unconsolidated and cemented crushed rockfill (waste rejects from the sorter) has been selected as the primary mining method. This will allow the level development to occur within the mineral wireframes, and any development connecting the various ore zones (which is below the cut-off grade) will be stockpiled as mixed development material at the entrance of the portal. This material will be later be put through the ore sorter in order to concentrate any potential gold material before sending to the mill.

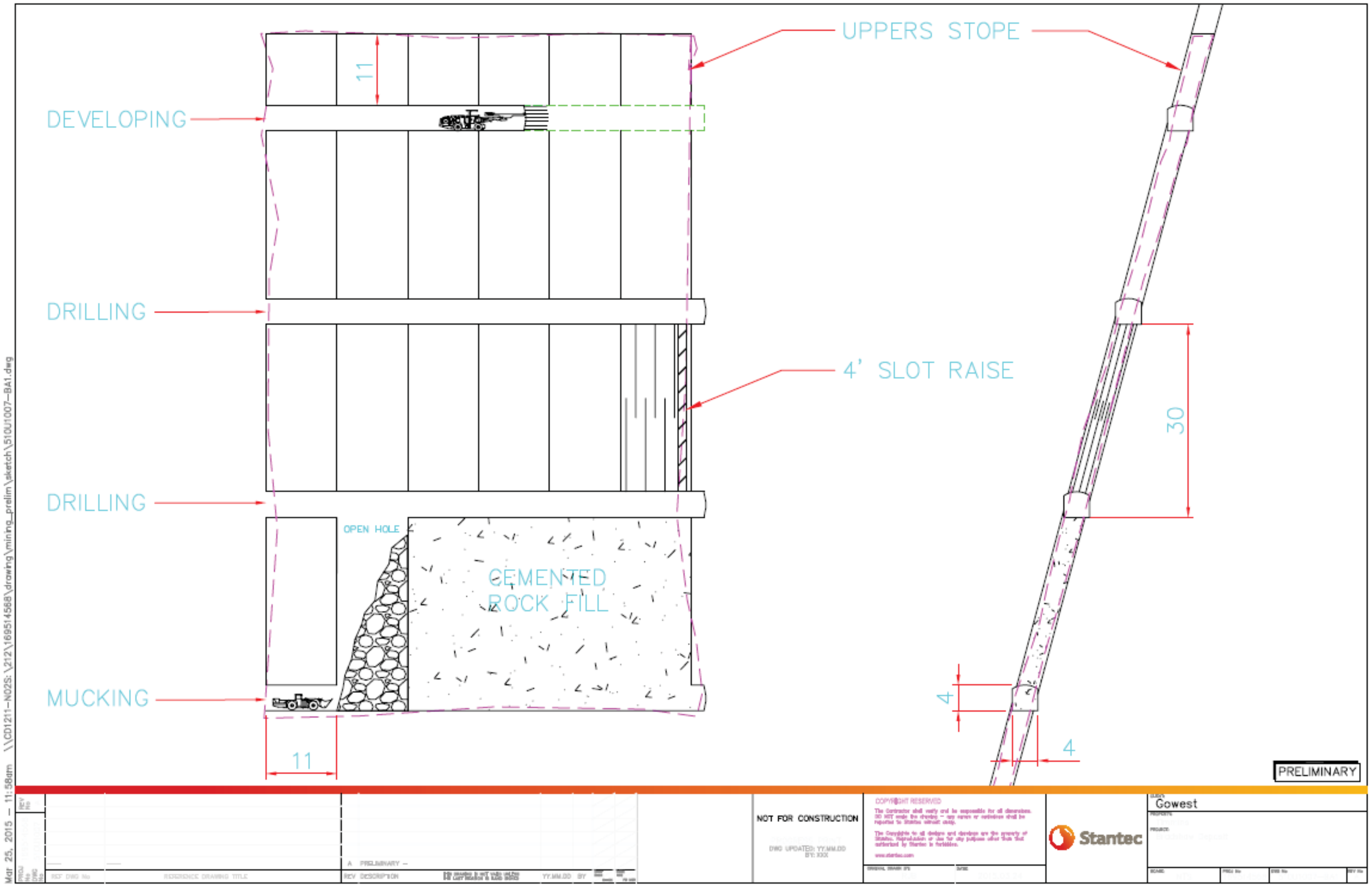
Longhole mining is a widely used and proven mining method that involves common industry equipment and labour skillsets. Longitudinal longhole stoping requires less capital and operating development than transverse. The majority of the ore zones have 2-3 metre widths with a few zones where widths ranged between 5-8 metres. In these instances, transverse mining may be considered in future studies but for the purpose of this study, longitudinal retreat was used throughout.

Sublevels have been designed at 30 metre vertical intervals (floor to floor) compared to 20 metres in order to reduce development costs. On each sublevel, the deposit will be accessed from the centre and developed east and west along strike to a minimum size of 4 metres by 4 metres (allow travel of 6 yds LHDs). In most cases, an overcut and undercut will be developed to mine the deposit, but in some instances, only an undercut will be developed, followed by longhole uppers including inverse raising. Longitudinal mining will retreat from the outer limits back to the centre access point. The typical stope length will be 11 metres but may vary depending on proximity to the ultramafics. [Stope heights are normally 30 metres with some 20 metre and 15 metre uppers.}

The deposit has been divided into mining blocks which are comprised of several stopes of various sizes. Mining will progress from the bottom of each block upwards. All stopes will be backfilled with a combination of cemented and unconsolidated waste rock. In some areas unconsolidated rockfill will be used; however, the majority will require consolidated (cemented) rockfill when mining adjacent to previously mined stopes.

The mining method is illustrated in Figure 16.5.

Figure 16.5: Longitudinal Longhole Mining Method



Stope Undercut and Overcut Development

Sill drifts (stope overcuts and undercuts) will be developed along the strike of each zone under geological control (i.e. under the direction of mine geologists) and the height and widths will generally average 4 x 4 metres. Smaller sized drifts were considered but the 4 x 4 metre size is thought to allow faster tramming and production cycling with more room to allow the vent tubing to be installed. There are some areas where the mining blocks are close together and a minimum pillar of 3 metres will be required in order to mine as separate blocks. The minimum sill width will be 3 metres.

Production Drilling

Longholes will be 64 millimetre (2.5 inch) diameter and drilled with air or electric hydraulic top hammer drills (Boart Buggy). The maximum length of each blast hole which ensure accuracy, and typically drilled to 16 metre. In order to ensure hole accuracy, longholes will be drilled from the overcut down 15 metres and from the undercut sill up 15 metres, with the ends of the boreholes overlapping by 2 metres at the midpoint of the stope. In areas where up holes are required, holes will be drilled from the undercut to the upper limit of the stope (commonly referred to as blind uppers). Other holes will be fanned up along either the hanging wall or footwall to the contours of the stope limits. The drill pattern will include a 1.8 metre ring burden with 2.2 metre spacing between ring holes. An initial slot raise will be drilled using a 4 foot raise bore hole and additional 2.5 inch holes will be drilled around the raise bore hole and blasted to create the initial void for production blasting. The estimated production drilling factor (excluding drop raises) will be 3.9 tonnes per metre of longhole. Production drills will be provided by a mining contractor who will be responsible for drilling and blasting production stopes.

To estimate the stope cycle time and productivity, an average daily production drilling rate of 180 metres has been used. Stopes will average 800 metres drilled (including additional holes around the slot raise and a contingency for re-drilling) requiring approximately 5 drilling days (including moving in and out of the workplace).

Production Blasting

Longholes will be loaded with ANFO explosives. The ANFO will be detonated with non-electric blasting caps and boosters. The powder factor will be approximately 0.78 kilograms of explosives per tonne of material. ANFO bulk bags and a mobile pneumatic unit will be used for loading. Including the slot raise, stope loading and blasting will require approximately 10 days (for a 3,300 tonne stope) and will be completed by mining personnel.

Stope Mucking

Broken muck will be extracted from stopes using 6 cubic yard (6yd) class LHDs. When the stope drawpoint brow is closed with muck, the LHD will be operated manually. When the drawpoint brow is open, the LHD will be operated via remote control with the operator located in a remote bay, a safe distance away from the brow. The LHD will tram to a remuck on the level access. Muck dumped into a remuck will subsequently be loaded with a 6 yd LHD into a 30 tonne class haul truck (30t truck) and hauled to the surface storage pads.

Material Rehandling and Underground Truck Haul

Broken muck will be rehandled from a remuck with a 6yd LHD, loaded into a 30t truck and hauled to surface or to a level where backfilling is in progress. The estimated annual productivity for truck hauling material to the storage pad is summarized in Table 16.3.

The hauling of backfill for the production stopes will be done on the return trip and has not been counted for the truck capacity. The amounts of underground backfill required are a shown in Table 16.4.

Backfill

Stopes will be backfilled with unconsolidated and cemented waste rock. Waste rock generated from development activities will be the primary source of backfill. Waste rock from underground will be loaded into trucks and hauled to the surface waste pile. If a stope is available to be filled during waste haulage, the truck will dump waste into a remuck where it will be rehandled by a 6yd LHD into the open stope. When additional waste is required for fill, the 30t trucks will backhaul from the waste pile on surface and deliver underground to a remuck near the open stope to be filled. The waste reject piles from the sorter will be utilized as much as possible since the material will be crushed and will provide good aggregate for cemented fill.

An “Aran” or equivalent cement plant will be located on surface and will initially mix cement slurry which will gravity feed a portable slurry mixing tank located close to the open stope requiring rockfill. A cement/water slurry will be transported via a series of pipelines (main ramp and levels) from the batch plant located on surface to the underground slurry tank where it will be mechanically agitated. A pipeline from the slurry tank will be directed to the open stope. The LHD operator will draw slurry from the tank while dumping a load of rockfill.

Table 16.4 summarizes the waste tonnes generated through the life of the mine, waste from the ore sorting process, the waste used for backfill and the remaining stockpile.

Stope Production Cycle

The production rate for longhole stopes has been based on an average stope size of 3,300 tonnes, with an estimated 21 day mining cycle (drill-blast-muck-backfill). The resulting production from individual stopes will average 160 tpd (not including fill cure time).

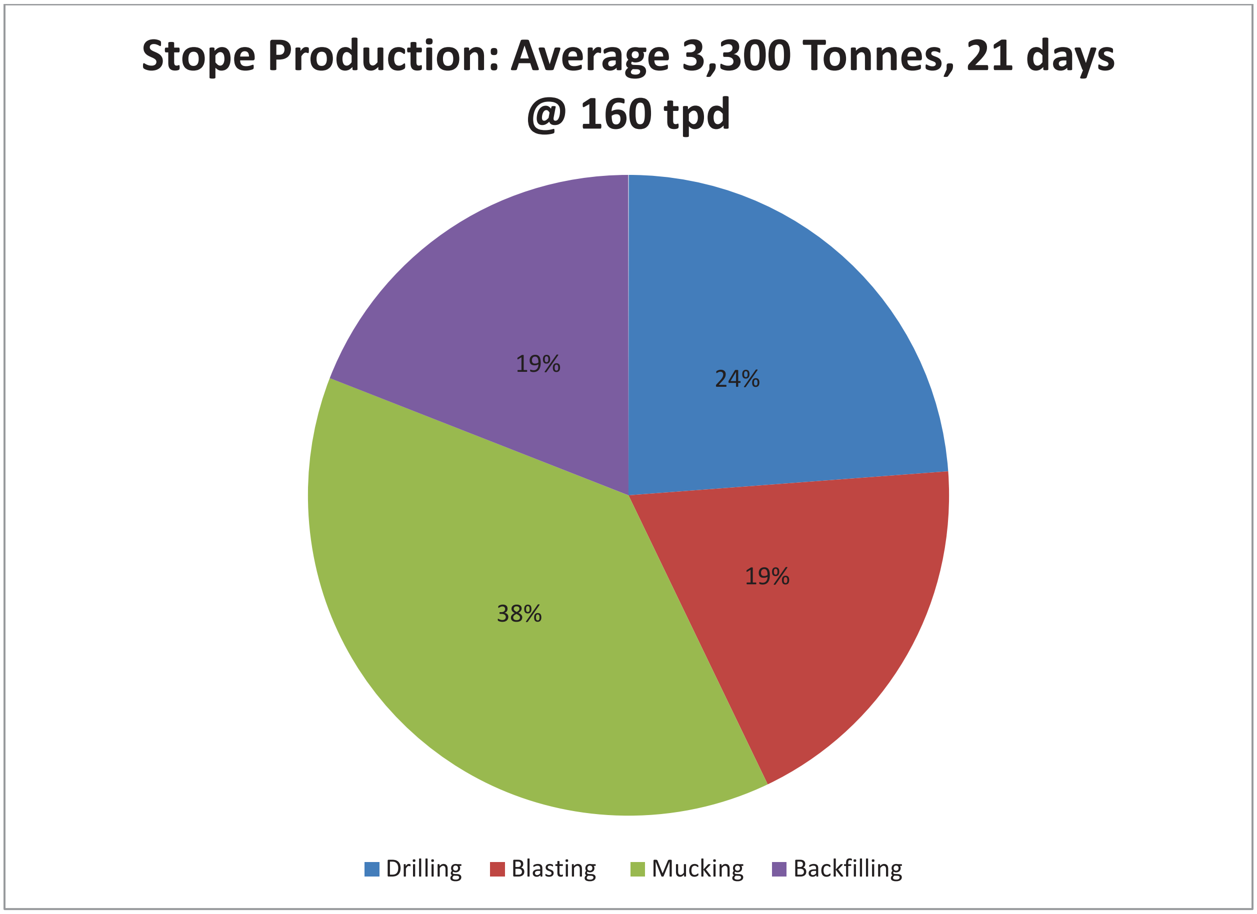
Table 16.3: Number of Underground Trucks required (Weighted Monthly Average)

Description	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Totals
Prod Tonnes	50,896	81,325	181,817	241,959	300,780	248,965	302,046	302,550	76,957	1,787,295
Dev Rock Tonnes	84,556	283,087	278,108	272,360	271,779	188,393				1,378,283
Total Annual Tonnes	135,432	364,412	459,925	514,319	572,559	437,358	302,046	302,550	76,957	3,165,578
Average Mthly Tonnes	371	998	1,260	1,409	1,569	1,198	828	829	211	964
No. of 30t Trucks Required (12450/mth max)	1	2	3	3	4	3	2	2	1	
No. of 45t Trucks Required (18450/mth max)	1	2	2	2	3	2	1	1	1	

Table 16.4: Waste Rock Requirements for Backfill

Description	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Totals
Dev Rock Tonnes	84,556	283,087	278,108	272,360	271,779	188,393				1,378,283
Rock Tonnes from Sorted Prod Ore	26,975	43,102	96,363	128,238	159,413	131,952	160,084	160,351	40,,787	947,266
Rock Tonnes from Sorted Mixed Dev	12,350	50,792	55,729	60,264	66,815	53,865				299,814
Rock Tonnes in Stockpile	123,881	376,981	430,200	460,863	498,007	374,210	160,084	160,351	40,,787	2,625,363
Tonnes of Required Backfill	19,781	24,848	72.010	94,348	125,224	98,734	151,023	151,275	38,478	775,720
Remaining Stockpile	104,100	456,233	814,423	1,180,938	1,553,721	1,829,196	1,838,258	1,847,334	1,849,643	1,849,643

Figure 16.6: Stope Production Cycle Time



During steady state production, two blocks per level will be available for mining and a minimum of four active stopes (2 levels required), with sill development from other levels, contributing to the average life of mine recovery of 675 tpd.

16.2.3 Development

There will be two development crews required throughout the project and operating period. Mining longitudinally will minimize development versus a transverse mining method.

The first development crew will complete ramp access to the first two levels (45 and 75 levles) and continue the ramp down to access the remaining levels. The second crew will complete level development to support production. The estimated development quantities are summarized in Table 16.5.

Table 16.5: Estimated Development Metres

	Capital Development				Operating Development			Total (m)
Level Access Waste (m)	FA Lateral Waste (m)	Vertical Waste (m)	Ramp (m)	Sub-Total (m)	Mix Dev Waste (m)	Ore Dev Silling (m)	Sub-Total (m)	
2,339	1,301	524	3,924	8,080	14,370	5,102	19,472	27,560
2,339	1,301	524	3,924	8,080	14,370	5,102	19,472	27,560

Ramp and Infrastructure Development

The ramp will be developed 5.0 metres wide by 5.5 metres high at a maximum gradient of 15 percent. The ramp crew will prioritize the ramp face, with development to establish the access to each sublevel and associated initial infrastructure as a secondary heading. The ramp development crew advance rate will be 5.0 metres per day including remucks. The ramp floor will include a layer of ballast material and the roadway will be maintained by a grader to minimize equipment maintenance requirements.

The main access to the levels will be developed 5.5 metres wide by 5.5 metres high up to the remuck. This will accommodate the 6yrd LHD and 30 tonne trucks on the level for loading. The sublevels and sills will be developed 4.0 metres wide by 4.0 metres high. Ancillary development such as refuge stations will be developed on the sublevel at a height of 4.0 metres. The infrastructure on sub levels will generally include:

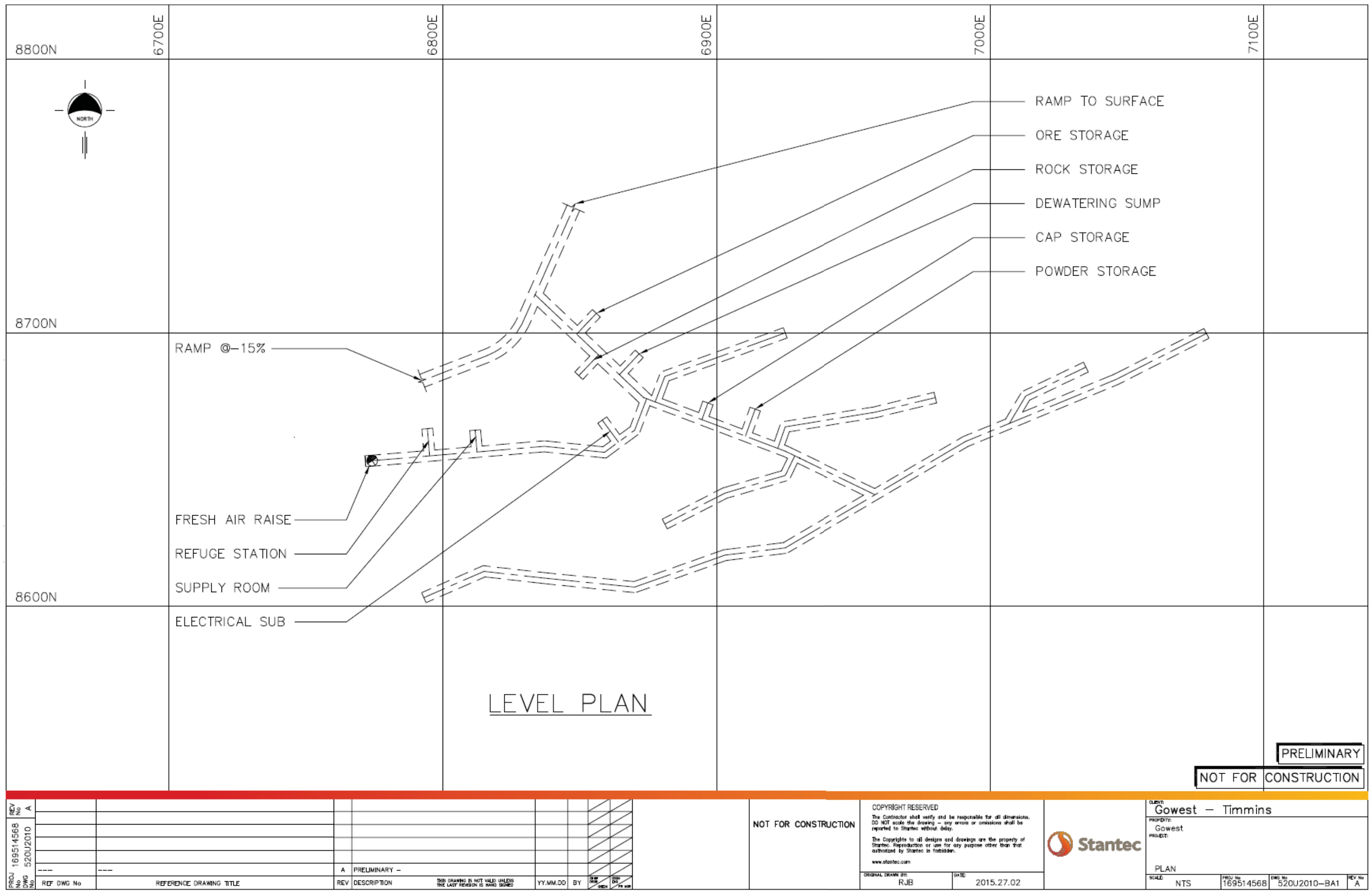
- Remucks and truck loading areas;
- Electrical substations;
- Material storage bays;
- Cap and powder mags (every 5th level);
- Sumps;
- Fresh air raise access drifts;
- Refuge stations (every 5th level);
- Main and intermediate sumps

A development crew will consist of one lead miner and two development miners. The equipment used by the development crews will include a 2-boom jumbo, 6yd LHD, and a scissor lift. The development crews will drill and blast, muck to a remuck, and install ground support using a scissor lift vehicle. The crew will install ventilation and piping services and will remuck waste into the 30 tonne haul trucks. A fourth worker (haul truck operator) will haul the waste rock to another sublevel as backfill or to the surface waste stockpile. For the purposes of this study all waste material is assumed to be non-acid generating.

Sill Development

The level development crew will develop the sill drifts. The mining crews will drill and blast, muck to a remuck, and install ground support using the scissor lift trucks. The crew will remuck broken material into a 30 tonne haul truck and a fourth worker (haul truck operator) will haul broken material to surface. Waste will be hauled to a remuck on a level where a stope is to be filled with backfill when available. Typical sublevel development is shown in Figure 16.7. Details on the mine design are found in Appendix G – Mine Design.

Figure 16.7: Typical Sublevel Development



Ground Support

Ground support will be installed in all underground excavations. Standard primary ground support in the ramp and sublevel development headings will include 1.8 metre 20 mm resin rebar (No. 6) installed at a 1.2 metre by 1.2 metre pattern with No.6 gauge welded wire mesh screen installed to within 1.5 metres from the floor along the drift walls. Where the span of the drift exceeds 7 metres (intersections, truck loading areas), 3 metre No. 7 resin rebar will be installed at a 1.7 metre by 1.7 metre pattern with No. 6 gauge screen. Shotcrete may be required for blocky ground conditions, if faults and shear zones are encountered.

Ground support in sill drifts will include 1.8 metre 20 mm resin rebar (No. 6) installed on a 1.2 metre by 1.2 metre pattern with No. 6 gauge welded wire mesh screen along the back of the drift and 1.8 metre long (FS35) friction bolts on a 1.2 metre by 1.2 metre pattern with No. 6 gauge wire welded mesh screen installed to within 1.5 metres from the floor along the walls. Access along the sill drifts will be required for the life of the level. All sill drifts will be shotcreted along the back and walls to maintain uninterrupted and safe passage during mining.

A geomechanical report outlining geotechnical assumptions, ground support and stope sizes is located in Appendix B – Rock Mechanics.

16.2.4 Underground Development Schedule

A development and production schedule has been completed for the Bradshaw Project. Underground development will begin once a decision is made to mine the bulk sample. The underground development and production schedule and annual development meters are shown in Table 16.6. Additional development schedule details have been included in the Appendix E – Level Plans.

Table 16.6: Annual Development Quantities

Description	Year 0 (m)	Year 1 (m)	Year 2 (m)	Year 3 (m)	Year 4 (m)	Year 5 (m)	Year 6 (m)	Totals
Ramp (m)	586	1,218	943	724	454			3,924
Lvl Acc (m)	228	776	605	332	398			2,339
Mixed Dev	592	2,434	2,671	2,888	3,202	2,582		14,370
Ore Dev	246	684	817	1,152	1,088	1,114		5,102
FA Dev Lat (m)	7	441	418	247	188			1,301
FA Vert (m)		165	120	104	107			496
RA Vert (m)		28						28
Totals (m)	1,659	5,746	5,574	5,447	5,437	3,696		27,560

16.2.5 Underground Production Profile

The Bradshaw project will operate two shifts per day, seven days per week. Underground crews and maintenance workers will work 10 hours per shift. Management, administration, and technical services staff will work eight hours per day from Monday to Friday. Annual production has been based on 365 days per year.

Each mining block will be extracted from the bottom upward. As the ramp is developed, access to the bottom of the mining blocks will be established. The undercut and overcut sill drifts will be developed and stopes will be extracted while the ramp development continues downward. An average of 675 tonnes per day will be achieved during the project's mine life.

Production Summary

The life of mine production profile is summarized in Table 16.7.

Table 16.7: Life of Mine Production Profile

	Production Profile LOM									Total
	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	
Dev Tonnes (t)	11,334	31,630	37,798	53,264	50,332	51,497				235,855
Dev Grade (g/t)	5.52	3.63	4.30	4.36	3.82	4.35				4.19
Stope Tonnes (t)	39,562	49,695	144,019	188,695	250,448	197,468	302,046	302,550	76,957	1,551,440
Stope Grade (g/t)	5.78	5.49	4.56	5.31	4.64	4.48	4.95	4.88	5.89	4.92
Stope and Dev Prod (t)	50,896	81,325	181,817	24,1959	300,780	248,965	302,046	302,550	76,957	1,787,295
Stope and Dev Grade (g/t)	5.72	4.77	4.50	5.10	4.50	4.45	4.95	4.88	5.89	4.82
Incremental Dev (t)	27,444	112,871	123,841	133,920	148,877	119,700	0	0	0	666,253
Incremental Dev Grade (g/t)	1.31	1.31	1.31	1.31	1.31	1.31	1.31	0	0	1.31
Gold from Dev and Stope Production (Au Oz)	9,366	12,467	26,327	39666	43,520	356,48	48,036	47,492	14,579	277,101
Gold from Incremental Dev (Au Oz)	1,152	4,736	5,197	5619	6,230	5,023	0	0	0	27,957
Total Au Oz	10,518	17,203	31,523	45286	49,751	40,671	48036	47,492	14,579	305,058
Avrg Stope and Dev Production (tpd))	139	223	498	663	824	682	828	829	855	675

16.2.6 Production Equipment

The underground mobile equipment required for production, development, and support services (excluding spares) is summarized in Table 16.8.

Table 16.8: Mobile Equipment

Equipment Type	No. Units
Development	
2-Boom Jumbo	2
LHD – 6 yd	2
Scissor Truck	3
Production/Backfill	
LHD – 6 yd	4
Longhole Drill	2
Underground Truck Haulage	
Haul Truck – 30 t	4
Services/Construction	
Personnel Carrier	4
Flat Deck Boom Truck	2
Backhoe/Forklift	2
Total	**25**

16.2.7 Ventilation

The overall ventilation system design will be a negative or “pull” system providing 189 m^3/s (400,000 cfm) to the underground workings via the main decline connected to the mining horizons. All of the exhaust air will return to surface through a 3.7 metre (12 ft.) x 3.7 metre (12 ft.) alimak Return Air Raise (RAR). This raise will be outfitted with a man-way for egress. All primary fans will be located on surface and be controlled by variable frequency drives (VFD) to mesh the production rate with the air volume requirements, thereby optimizing energy (power and propane) usage. Details on the ventilation system design are found in Appendix D – Ventilation.

Fresh air will be supplied to the extents of the mining levels using auxiliary ventilation from the main ramp. Exhaust air from the mining levels will return on the levels through regulators into the exhaust raise system to surface. The main ventilation systems consist of the following installations;

- Main fresh air supply fans and propane direct fired heaters connected to the portal 3 metre (10 ft.) x 3 metre (10 ft.) raise.
- Main return air fans connected to a 3.7 metre (14 ft.) x 3.7 metre (14 ft.) raise, which is outfitted with a man-way.

Installation of VFD's for all primary fans will allow flexibility in the air volume capacity to meet the regulatory and production requirements. The overall ventilation system is illustrated in Figure 16.8.

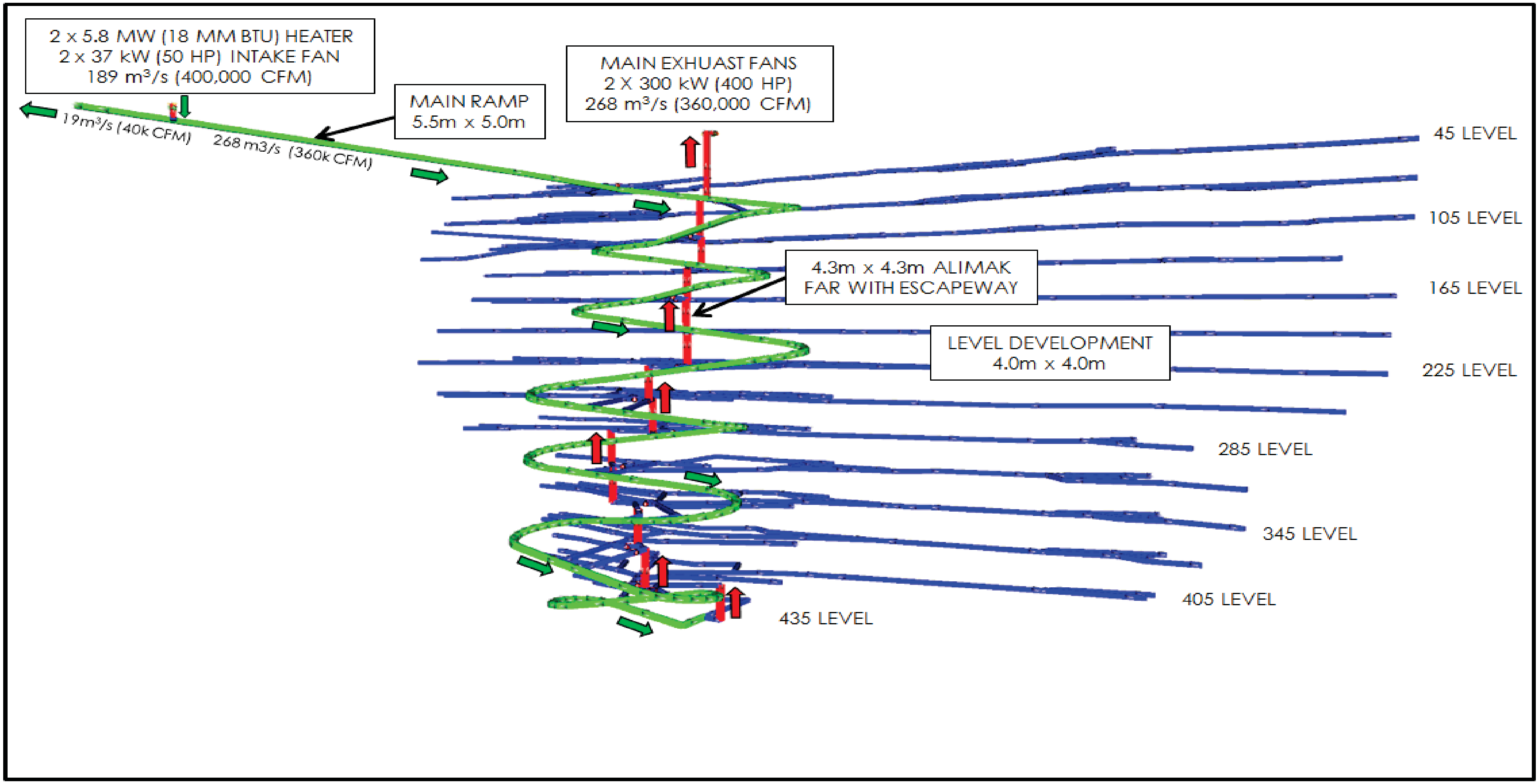
The region surrounding Bradshaw has a continental climate that is characterized by cold dry winters and relatively warm dry summers. Daily temperatures range from +25°C in summer to -22°C in the winter. Temperatures as low as -45°C and as high as +35°C have been recorded. A direct fired propane mine air heating system will be installed to maintain approximately +3°C mine intake air during winter months. A mine air cooling system will not be required.

The main fan operating duty points are listed in Table 16.9. To obtain actual operating performance data, an Alphair ventilation fan was assumed.

Table 16.9: Main Fan Duty Points

System	Number of Fans	Operating Duty Point	Connected Fan kW	Mine Air Heater
Main FAR Intake	2	189 m³/s (400 Kcfm) at 124.5 Pa (0.5" SP)	2 x 37 kW (50 HP)	2 x 5.9 MW
Main RAR Intake	2	85 m³/s (180 Kcfm) at 2.6 kPa (10.4" SP)	2 x 300 kW (400 HP)	

Figure 16.8: Ventilation System Schematic



Mine Air Volume Estimate

The Ontario Occupational Health and Safety Act and Regulations for Mines, and Mining Plants Section 183.1 states that "The flow of air must be at least 0.06 cubic metres per second for each kilowatt of power of the diesel-powered equipment operating in the workplace".

A utilization factor was applied consisting of a conservative 80% or 70% for all haulage equipment and 50% or 25% for all other equipment. The estimated diesel equipment fleet is listed in Table 16.10.

Table 16.10: Diesel Equipment Fleet and Air Volume Requirements

Equipment Type	No. Units	Engine		Utilization (%)	CMS Required (0.06 CMS/kW)
		Hp	kW		
Development					
2-Boom Jumbo	2	116	86.5	25%	2.7
Scissor Truck	3	116	86.5	25%	4.1
LHD - 6yd	2	279	208	70%	20.4
Production/Backfill					
LHD - 6yd	4	279	208	70%	34.9
Longhole Drill	2	na	na	na	na
Underground Truck Haulage					
Haul Truck - 30t	4	400	298	80%	60.4
Services / Construction					
Personnel Carrier	4	134	100	50%	12.6
Flat Deck Boom Truck	2	116	86.5	25%	2.7
Backhoe/Forklift	2	99	74	25%	2.3
Total	**35**				**140.0**
		Leakage Allow. (PFS study and leakage)		34%	50.0
		Total CMS			**190.0**
		Total CFM			**400,000**

The "leakage allowance" contingency includes consideration for the level of study and potential variation in the make/model of the final selection of diesel equipment.

Ventilation Design Parameters

The primary ventilation design parameters used in the design of the ventilation system are listed in Table 16.11.

Table 16.11: Ventilation Parameters and Design Values

Parameter	Value	Unit
Average ore production rate	675	tpd
Ventilation system	Pull	
Ventilation capacity (max)	189 (400,000)	m^3/s (cfm)
FAR air velocity (max)	11 (2,200)	m/s (fpm)
RAR air velocity (critical range)	7-12 (1400-2400)	m/s (fpm)
Power Cost	$0.08	Per kWh
Main decline: 5.0 m x 5.2 m	< 6 (<1200)	m/s (fpm)
Supported Alimak Raise (K-value)	0.0.013	kg/m^3
Haulage ramp resistance (K-value; blast tunnel –dykes)	0.012	kg/m^3
4.7 m x 4.7 m Alimak raise with manway (RAR)	0.02	kg/m^3
Silica, crystalline (TWA)	0.025	mg/m^3
DPM – TC	0.4	mg/m^3
CO (TWA)	25	ppm
CO_2 (TWA)	5000	ppm
NO (TWA)	25	ppm
NO_2 (TWA)	0.2	ppm
SO_2 (TWA)	2	ppm
Heat stress limit (wet bulb globe temperature - moderate, acclimatized, 75% work; 25% rest)	28.5	°C
Noise continuous (8 hours)	85	dBA
Noise intermittent (15 minutes)	100	dBA

The design values are based on the "Threshold Limit Values for Chemical Substances and Physical Agents" as published by the ACGIH and referenced in section 283 of the OHSA and Regulations for Mines and Mining Plants.

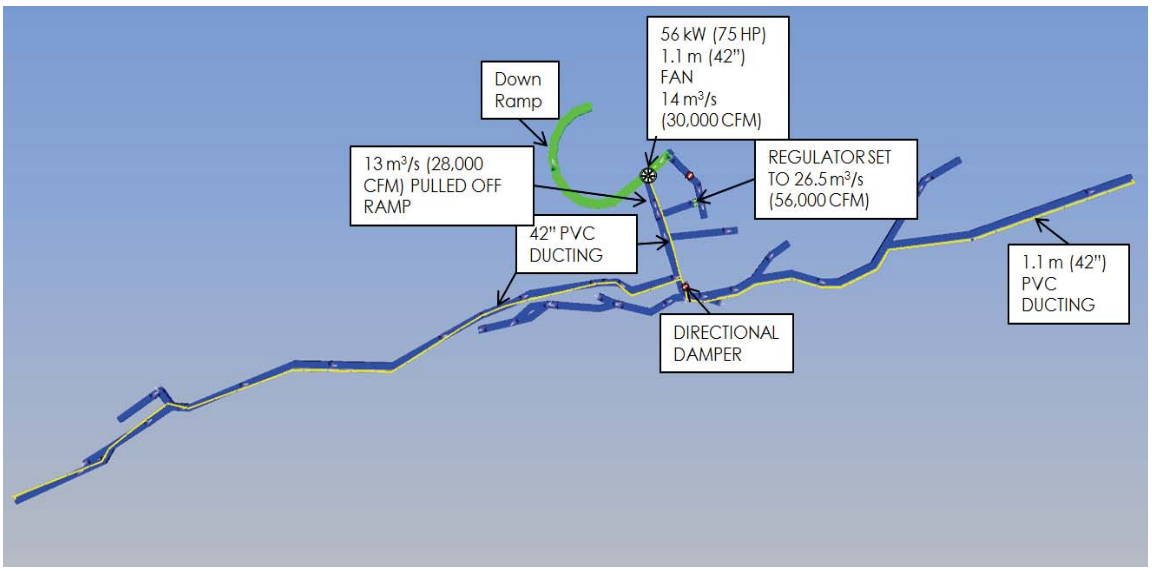
Typical Level Layout

The ventilation circuit on most operating levels will consist of a single 42 inch 56 kW (75 HP) fan, hung in the ramp, attached to 1.1 metres (42 inch) PVC ducting providing ventilation to the extremities of the orebody. The system will provide for the operation of an LHD and secondary equipment, which require a total of 18.8 m^3/s (32,000 cfm). The trucks will not be accessing the individual levels but will be loaded in the ramp or along the level access near the remuck. If development distance exceeds 300 metres (1000 ft.) a second fan will be required to be installed in series. The RAR regulator will be adjusted to allow for a total of 26.5 m^3/s (56,000 cfm) exiting the level. This allows for 13.5 m^3/s (28,000 cfm) to be pulled off the ramp for passage of the LHD to the ramp for truck loading. A typical auxiliary ventilation fan including ventilation ducting is shown in Table 16.12. A typical production level ventilation circuit is shown in Figure 16.9.

Table 16.12: Level Auxiliary Fans Installations

Duct	Fan Type	Total Number of Fans	Connected Power	Remarks
Single 1.067 m Diameter ducts (42" diameter)	Alphair 4200VAX2700, 1780 rpm. 18.8 m^3/s (37.5 Kcfm) at 1.5 kPa TP c/w 56 kW (75 HP) motor	16+	56 kW (75 HP)	Single fan every 300 metres per active sublevel required, and preferred to keep two spares on site

Figure 16.9: Typical Level Ventilation Circuit



Main Ramp Development

The 5.5 metre (18 ft.) wide x 5.0 metre (16.5 ft.) high main decline will be driven at approximately -15% from the portal ultimately to the 495 Level.

With operations in a single face heading, the maximum air volume required is for the simultaneous operation of the LHD and haulage truck (plus allowances for auxiliary equipment) should be 37 m^3/s (78,000 cfm). This will be divided equally with 18.5 m^3/s (39,000 cfm) being delivered via a two duct system to the face.

The first exhaust air connection from the main ramp will be the 45 Level, which is approximately 500 metres (1500 ft.) from the portal. An additional 100 metres (300 ft.) of duct allowance should be allowed to account for the extra time required to complete the first leg of the exhaust raise to surface.

The exhaust raise system will be divided into a series of raises in order to minimize the auxiliary ventilation system required to drive the ramp. Each section of raise will be approximately 30 metres (100 ft.) in length. The auxiliary ventilation system should be designed for a total length of 300 metres, which includes equivalent duct lengths for going around corners in the ramp and an additional 100 metres of ducting before each leg of the exhaust raise is completed.

The most practical auxiliary tubing installation consists of 2 x 1.1 metre (42 inch) diameter PVC ventilation duct each connected to a 75 kW (100 HP) fan. For each 300 metres of new development an additional 75 kW (100 HP) fan will be added to the duct line. Each duct will handle 18.5 m^3/s (39,000 cfm). Each installation should consist of a 75 kW (100 HP) fan, inlet silencer, and inlet bell with screen.

The 5.5 metre wide x 5.5 metre high main ramp will be driven at a maximum gradient of -15 percent. The longest section of the ramp to ventilate before the first fresh air connection from the portal is approximately 610 metres. The auxiliary ventilation system is designed for an equivalent length of 640 metres (2,100 feet), which includes allowances for corners in the ramp.

- LHD: 250 kW (335 HP) requiring 10.5 m^3/s (33,500 cfm).
- 50 t haulage truck: 485 KW (650 HP) requiring 23.3 m^3/s (65,000 cfm).
- Two boom jumbo: 60 kW (80 HP) requiring 3.6 m^3/s (8,000 cfm).

The total volume of air required in the ramp has been designed to accommodate an LHD and a 50t truck. During ramp development a total of 47 m^3/s (100,000 cfm) will be supplied via twin 1.22 metre (48 inch) metal ducting to the face.

During ramp development, two 1.22 metre (48 inch) diameter steel ventilation lines connected to 75 kW (100hp) fans will be installed. Each duct will provide 23.5 m^3/s (50,000 cfm).

16.2.8 Underground Mining Personnel

The personnel required on site include owners and contractor’s management, technical services (engineering and geology), administration, maintenance, supervisory, and development/production personnel. The estimated annual personnel required on site are summarized in Table 16.13.

Table 16.13: Personnel on Site (Life of Mine)

Gowest Gold

Bradshaw Gold Deposit

Manning Table People at Site

Description	Rate	Unit	Hours	Year 1 Bulk Sample	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Owner Staff												
Mine Manager	$79.00	$/Hour	10	1	1	1	1	1	1	1	1	1
Mine Engineer Planner	$65.00	$/Hour	10	1	1	1	1	1	1	1	1	1
Mine Geologist/Sampler	$65.00	$/Hour	10	1	2	2	2	2	2	2	2	2
Security/First Aid	$45.00	$/Hour	10	2	2	2	2	2	2	2	2	2
Total Owner Staff	**$3,640**	**$/Day**		**5**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
Surface Contractor Direct Labour												
Surface Material Handling												
Loader Operator	$72.49	$/Hour	10	2	2	2	2	2	2	2	2	2
Backhoe Operator	$72.49	$/Hour	10	1	1	1	1	1	1	1	1	1
Sizer Operator	$72.49	$/Hour	10	1	1	1	1	1	1	1	1	1
Crusher Operator	$72.49	$/Hour	10	2	2	2	2	2	2	2	2	2
Surface Material Handling				**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
Surface Contractor Direct Labour				**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
Contractor Underground Direct Labour												
Development												
Ramp Development												
Development - Leader Jumbo Miner	$85.02	$/Hour	10	2	2	2	2	2	2	2		
Development - Maclean Rockbolter Miner	$83.65	$/Hour	10	2	2	2	2	2	2	2		
Development - Contractor Development Miner	$80.27	$/Hour	10	4	4	4	4	4	4	4		
Ramp Development				**8**	**8**	**8**	**8**	**8**	**8**	**8**	**0**	**0**
Level Development												
Development - Leader Jumbo Miner	$85.02	$/Hour	10	2	2	2	2	2	2	2		
Development - Maclean Rockbolter Miner	$83.65	$/Hour	10	2	2	2	2	2	2	2		
Development - Contractor Development Miner	$83.65	$/Hour	10	2	2	2	2	2	2	2		
Level Development				**6**	**6**	**6**	**6**	**6**	**6**	**6**	**0**	**0**
Truck Haulage												
Truck Operator	$58.30	$/Hour	10	2	4	4	6	6	6	6	6	6
Truck Haulage				**2**	**4**	**4**	**6**	**6**	**6**	**6**	**6**	**6**
LHD Haulage												
LHD Operator	$72.00	$/Hour	10	2	2	2	2	2	2	2	2	2
LHD Operator				**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Longhole Drilling/Borehole Drilling												
Longhole Driller	$83.65	$/Hour	10	1	1	1	2	2	2	2	2	2
Longhole Drilling/Borehole Drilling				**1**	**1**	**1**	**2**	**2**	**2**	**2**	**2**	**2**
Longhole Blasting												
Longhole Blaster	$83.65	$/Hour	10	2	2	2	2	2	2	2	2	2
LHD Operator				**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Alimak Raising/Escapeway Manway												
Alimak Raise Miner	$93.24	$/Hour	10	0	2	2	2	2	2	2	2	2
Alimak Raising/Escapeway Manway				**0**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Contractor Underground Direct Labour				**21**	**25**	**25**	**28**	**28**	**28**	**28**	**14**	**14**
Contractor Indirects Labour												
Management												
Superintendant	$100	$/Hour	10	1	1	1	1	1	1	1	1	1
Underground Supervisor	$101	$/Hour	10	2	2	2	2	2	2	2	2	2
Safety Supervisor	$77.28	$/Hour	10	1	1	1	1	1	1	1	1	1
Clerk	$48.80	$/Hour	10	1	1	1	1	1	1	1	1	1
Subtotal Management	**$3,645.8**	**$/Day**		**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
Engineering												
Surveyor	$77	$/Hour	10	2	2	2	2	2	2	2	2	2
Subtotal Engineering	**$1,104**	**$/Day**		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Mechanical												
Mechanic Lead Mobile Equipment	$83.90	**$/Hour**		1	2	2	2	2	2	2	2	2
Mechanic Mobile Equipment	$60.61	**$/Hour**	10	2	2	2	2	2	2	2	2	2
Subtotal Mechanical	**$2,890**	**$/Day**		**3**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Electrical												
Electrician Lead	$83.90	**$/Hour**		1	2	2	2	2	2	2	2	2
Electrician	$60.61	**$/Hour**	10	2	2	2	2	2	2	2	2	2
Subtotal Electrical	**$2,890**	**$/Day**		**3**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Office/Wash Trailer Cleaning												
Dryman	$48.81	$/Hour	10	1	1	1	1	1	1	1	1	1
Subtotal Office/Wash Trailer Cleaning	**$976.2**	**$/Day**		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Contractor Indirects Labour	**$11,506**			**14**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**
Total Mining Contractor Labour				**35**	**41**	**41**	**44**	**44**	**44**	**44**	**30**	**30**
Grand Total Labour				**46**	**53**	**53**	**56**	**56**	**56**	**56**	**42**	**42**

17.0 RECOVERY METHODS

17.1 Crushing, Screening and Ore Sorting

A portable crushing, screening and ore sorting plant with daily production capacity of approximate 800-900 tonnes is considered. Appendix A – Sorter has details on the set-up and test results of the sorter. It is ample for the demands of the 675 tonnes/day mining rate, which will process mineralized feed that is trucked up the underground ramp of the mine and dumped at the stockpiles. The circuit will be operated at the mine site via an independent contractor. Crushing operations are currently envisioned as a two stage circuit, primary jaw and secondary cone, with ore sorting equipment between stages. The portable equipment, including transfer belt conveyors, are independently powered via diesel powered drives.

ROM ore with average size of 400 mm, max. 600 mm, is loaded with a front-end loader from mine site stockpiles to the crushing circuit. There will be two stockpiles, the majority being the Run of Mine Ore stockpile that is retrieved from the ore reserve base using blast-hole stopes, and the mixed stockpile that has to be trucked up the ramp through the portal on surface. This material consists of mineralized development muck that due to dilution is below the cut-off grade of 3 g/tonne and ordinarily discarded as waste, but contains mineralized material that can be economically retrieved by the sorter. The material to be economic must contain enough recoverable gold to pay for crushing and sorting, transportation to the mill, milling and smelting at the prevailing gold price received. Ore is processed to produce a final crushed product of minus 50 mm material prior to be transported to a toll milling facility. Primary crushed material is fed to a primary screen, which is a double deck vibrating screen with screen mesh sizes of 25, and 75 mm, respectively. Fine material passing the bottom 25 mm deck will be sent to a single deck screen with 10 mm screen panels. Two dual energy X-ray transmission (DEXRT) sorters are considered. Material of 10-25 mm from the single deck screen is fed to a fine material sorter, while material of 25-75 mm is collected and fed to a coarse material sorting machine. Oversize of +75 mm material from the primary screen top deck is fed to a cone crusher that reduces the ore size to 80% passing 50 mm. Cone crusher discharge is sent back to the primary screen.

The -10 mm fine material, as well as concentrate from two sorters are collected and will be sent to the mill for processing. Barren material from the sorters will be stockpiled as waste and will be used as underground backfill aggregate for the mined out stopes.

The crushed ore material is then transported by trucks to the ore processing facility for gold recovery operations. The ROM and "Mixed" stockpiles at the mine site are sized sufficiently to allow for normal crushing operation shutdown periods and short term production interruptions.

Based on pilot ore sorting test result, it is estimated that the mass recovery to sorting concentrate is about 53% with 98% of gold recovery. At ROM ore gold grade coming out of the portal averaging 4.82 g/t, the gold grade of the material shipped to the processing plant is expected to be as high as 9 g/t.

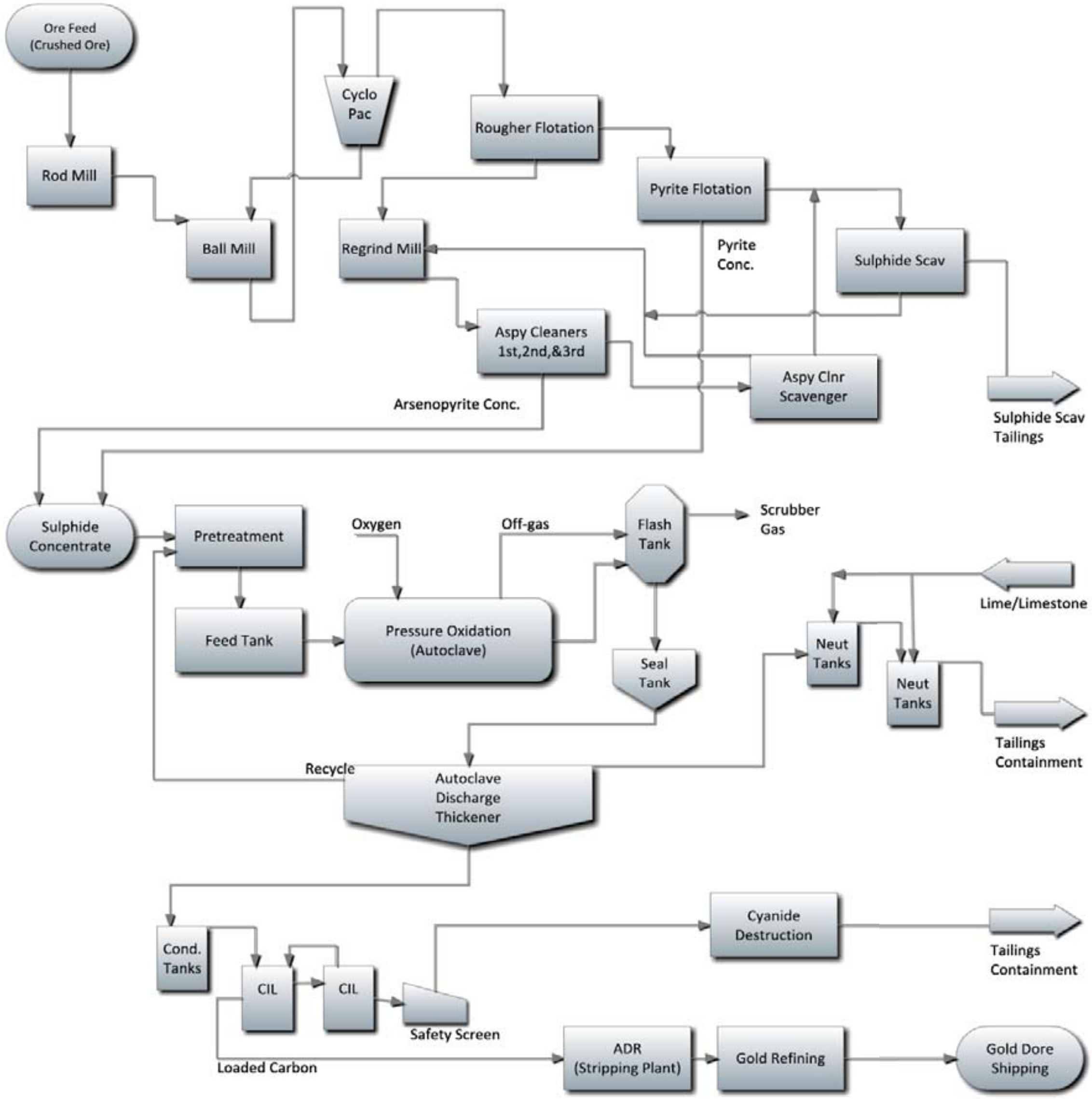
17.2 Processing Plant

A standard flow-sheet for the processing of Bradshaw ore is depicted in Figure 17.1. It is assumed that the processing plant will operate on a 24 hour per day basis with an overall availability of 93%.

The flow-sheet design incorporates the following general stages:

- Comminution (grinding and regrinding)
- Sulphide flotation
- Pressure oxidation
- Cyanidation and gold refining
- Cyanide destruction
- Tailings disposal

Figure 17.1: Block Diagram of Standard Bradshaw Project Process Flowsheet



17.2.1 Comminution

Crushed ore is hauled from mine site to mill by highway trucks of nominal capacities of approximately 34 tonnes each and transferred to a crushed ore storage bin at the plant site. The grinding circuit consists of a ball mill circuit. Ore is ground to a size of approximately 80% passing 75 microns prior to being sent to the flotation circuit.

A regrind mill circuit has been included to reduce the rougher flotation concentrate particle size to approximately 25 microns (P80) in order to improve the sulphide cleaning efficiency.

17.2.2 Sulphide Flotation

The flotation circuit consists of rougher and cleaner stages for separating the arsenopyrite and pyrite concentrates. The flotation circuit shown in Figure 17.1 represents the bench scale Locked Cycle Test (LCT) flow-sheet used in the metallurgical program (see Section 13). The rougher flotation is completed with a feed particle size of 75 microns (F80). The arsenopyrite rougher concentrate is then reground to 25 microns (P80) for final cleaning operations. The rougher tailings are fed to pyrite flotation stage for pyrite recovery, if required.

The sequential flotation circuit without applying ore-sorting generates two concentrate products:

- arsenopyrite concentrate (6.4%* mass recovery / 93% gold recovery)
- pyrite concentrate (4.8% mass recovery / 3% gold recovery)

*Note: With ore-sorting the mass pull for the arsenopyrite will increase marginally based on the higher feed grade of the sorted material.

For the standard flow-sheet, tailings from both arsenopyrite and pyrite cleaning circuits are combined and fed to the last stage of flotation, sulphide scavenger to recover residue sulphides prior to disposal. Flotation tailings will be separate from gold refining tailings management. Tailings impoundment will be located on site as a managed facility for life of mine operations.

The use of a sequential flotation circuit provides the most flexibility with respect to potential changes in the mineralogy within the Bradshaw Deposit as well future opportunities to treat feed materials at different mills.

17.2.3 Pressure Oxidation (POX)

Flotation concentrates are arsenopyrite and pyrite which are treated using pressure oxidation to oxidize the sulphides to access the gold in the solids.

The concentrate slurry is pumped into a continuously operating autoclave. Once in the autoclave, the slurry is subjected to high temperature (+200°C) and injected with high pressure oxygen from an oxygen plant. After about 60 minutes, approximate 98-99% of arsenopyrite and pyrite components in the concentrate dissociate to free the contained gold. The majority of the iron and arsenic is solubilized first and then precipitates under pressure forming scorodite ($FeAsO_4 \cdot 2H_2O$), a chemically stable form of arsenic.

The oxidized slurry from the autoclave is discharged, cooled and thickened separating solids from the acidic solution. A portion of the acidic solution from the thickener is recycled to the autoclave feed tank to help condition the new concentrate and remove carbonates. The balance of the thickener overflow is neutralized with limestone and lime prior to being pumped to tailings area. The thickened gold bearing POX residue is neutralized to a desired pH level suitable for cyanidation and then pumped to a cyanidation circuit for gold recovery.

Filtration of the thickened POX residue slurry prior cyanidation is being investigated, which will be necessary if the POX product is to be treated in a different location.

17.2.4 Cyanidation and Gold Refining

Neutralized POX residue is pumped a conventional CIL (Carbon-in-Leach) circuit where leaching and adsorption of gold are carried out simultaneously. Cyanide required for leaching gold is added to the circuit, while milk of lime is added to maintain slurry at the desired pH level. Barren activated carbon is added to the last tank and advances via carbon advance pumps counter-currently to the slurry. As the gold is leached, it is adsorbed by the carbon. Air is sparged from the bottom of each tank into slurry to maintain adequate dissolved oxygen levels in the pulp.

Carbon pregnant with gold from the first CIL tank is pumped to a loaded carbon screen where the loaded carbon is separated from the slurry. The slurry falls by gravity back to the 1st CIL tank. The loaded carbon is transferred to a bin from where the loaded carbon is transferred to the stripping stage.

Tailings from the last CIL tank overflows to a carbon safety screen which prevents the loaded carbon from getting lost. The screen undersize flows by gravity to a pump-box, and is then pumped to the tailings dewatering area.

Cyanidation of the POX residue results in high gold extractions of over 98%. Loaded carbon is processed via a pressure stripping and gold refining plant. The expected overall recovery of gold from the ore is approximately 93%.

17.2.5 Cyanide Destruction

Following cyanidation, residual cyanide contained in the leached slurry is destroyed via a SO_2/air cyanide destruction circuit. Cyanide destruction discharge is then thickened and sent to a conventional tailings containment area for impoundment.

17.2.6 Tailings

The autoclave discharge solution from the thickener overflow is neutralized with limestone and lime prior to being pumped to the tailings area. Disposal is in a general cyanidation tailings area or in a separate designated area. Water is recycled from the tailings impoundment area for reuse in the processing plant.

17.3 Processing Plant Options

Gowest has been exploring the possibility of bringing the project into production earlier and at reduced capital outlay through toll milling. In 2012 Gowest and Xstrata (Glencore) entered into a memorandum of understanding (MoU) to complete internally a preliminary engineering study (PES) primarily aimed at evaluating the viability of toll-milling through the currently dormant D-division at the Kidd Operation Concentrator.

In late 2014 Gowest retained AMEC Forster Wheeler (AMEC) to execute a prefeasibility level assessment of Kidd Mill option No.1 (Mill A), and a conceptual study of using alternative milling operation in the Timmins area. The conceptual study included the following options,

- Toll milling through an alternative nearby mill referred to as “Mill B” .
- Construction of a new process plant at a Timmins brownfield site.

The milling options explored in this report consider the production of an arsenopyrite concentrate as final product, which would be shipped to a refinery for gold recovery. To allow for comparison of the options above, similar treatment rates and flow-sheets were considered. AMEC was retained by Gowest to perform an analysis of the different options available. This section includes key findings from the AMEC Pre-Feasibilty Study Report (Bradshaw Ore Toll-Milling Study) and an Internal Concepts Study Report (Bradshaw Ore Treatment Alternatives).

17.3.1 Process Design Basis

Key design parameters for the toll milling are given in Table 17.1 below.

Table 17.1: Summary of Toll Milling Process Design Parameters

Design Parameters	Units	Value
Nominal processing rate	t/a	180,700
Nominal production of gold (avg 7 years LOM)	oz/a	40,500
Average feed grade to processing plant	g/t	7.61
Average plant daily throughput	t/d	500
Crushed ore size (P80)	mm	varied
Primary grinding size (P80)	µm	75
Regrinding size (P80)	µm	25
Crusher work index (CWI)	kWh/t	14.5
Abrasion index (Ai)	gram	0.135
Rod mill work index (RWI)	kWh/t	16.6
Bond ball mill work index (BWi)	kWh/t	15.3
Flotation gold recovery	%	96 *
Flotation mass pull	%	8 to 10 *
Estimated Au concentrate grade (Aspy Conc.)	g/t	90 *

**Data are estimated based on recent test work done for toll milling option, to be confirmed in the next phase.*

The mass balance calculations were performed using the Metsim® simulation software. The model is based on the latest flowsheets as presented in AMEC's Prefeasibility Study Report.

17.3.2 Toll Milling Option 1 (Kidd Operations - Concentrator)

This option is about the use of toll milling for Gowest to process Bradshaw ore using the 'D'-Division (D-circuit) at Glencore's Kidd Operations. Toll milling with this option will produce an arsenopyrite gold concentrate as the final product. The gold concentrate produced will be filtered and dewatered in the concentrator and then trucked offsite for third-party processing. Flotation tailings will be sent to the current tailings management area (TMA) for disposal.

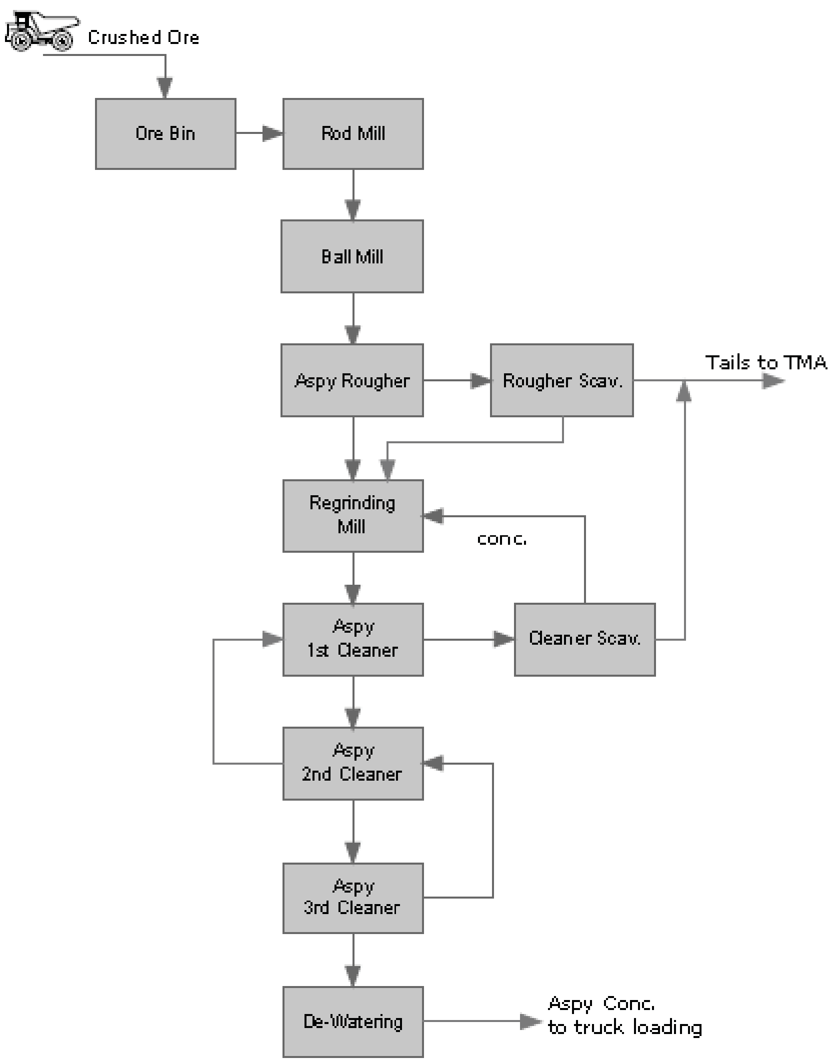
The 'D' circuit has a design capacity of 90 t/h. The operations team at the mill proposed a campaign based operation schedule so the D circuit will maintain a process rate near 90 t/h (nominal). The current 'D'-circuit can be modified to accommodate the Bradshaw ore process flow-sheets as depicted in AMEC's Prefeasibility Study Report

The flow-sheets for this option include the following:

- 100-F-001 Ore Receiving and Storage
- 200-F-001 Primary Grinding
- 300-F-001 Arsenopyrite Rougher and Regrinding
- 300-F-002 Flotation Scavengers
- 300-F-003 Arsenopyrite Cleaners
- 310-F-001 Concentrate De-watering / Truck Loading

A block diagram for toll milling is presented in Figure 17.2.

Figure 17.2: Bradshaw Ore Toll Milling Block Diagram



17.3.3 Process Description (Option 1)

Crushing, Screening and Ore-Sorting

For a description of this area refer to Section 17.2 above.

Ore Receiving and Storage Area (Dwg. 100-F-001)

The crushed ore is delivered in 40t highway trucks to the Kidd Mill site and processed in the existing 'D' circuit. The maximum ore size is expected to be less than 1 inch (<25mm). The ore is dumped directly to a truck dump hopper, 100-HPR-001, from where it discharges to a feeder. This feeds the ore to an ore receiving belt conveyor, 100-CVB-001 (existing No.6 conveyor), which is equipped with a belt scale for flow rate measurement. The ore is transferred from ground level to the top of the coarse ore bins, 100-BIN-011, with an active capacity of 3200 tonnes. An additional 1500 tonne bin capacity could be available if a transfer conveyor belt plow was utilized. Ore discharges from the fine ore bins onto four of seven belt feeders, 100-FEB-011 to -017 (existing 14DA to 14DG), which feed ore to mill feed transfer conveyors 100-CVB-003/004 (existing 15D and 16D), followed by 100-CVB-005 (existing 17D). Conveyor 100-CVB-005 feeds ore to the rod mill and is equipped with a belt scale (100-SLB-002) measuring the feed to the grinding circuit.

Grinding Circuit (Dwgs. 200-F-001, 300-F-001)

The D circuit has two comminution circuits, primary grinding of crushed ore and regrinding of rougher concentrate. Bradshaw ore processing requires both circuits. The primary grinding circuit consists of a rod mill and ball mill for two stage grinding. Ore is ground to give a size distribution of 80% passing 75 microns (P80) prior to being sent to the flotation circuit. The regrind mill circuit reduces the rougher flotation concentrate particle size to approximately 80% passing 25 microns (P80) in order to improve the cleaning efficiency.

Primary Grinding Circuit (Dwg. 200-F-001)

The primary grinding of Bradshaw ore is completed using the rod mill, 200-MLR-010 (existing 420-2101-04) followed by two ball mills, 200-MLB-020/021 (existing 420-2102-04 and 430-2101-04), operating in parallel. The rod mill, with dimensions of 3.5 metre diameter and 5.3 metre length, is driven by an 800 hp motor. The ball mills, each 3.7 metre diameter and 5.5 metre length, are powered by two 1500 hp motors. The rod mill and one ball mill, 200-MLB-020, discharge to a common pump-box, 200-SMP-001 (existing D-24). Ball mill 200-MLB-021 discharges to a pump-box 200-SMP-004 (existing D-025) from where slurry is pumped to pump-box 200-SMP-001. Combined slurry is then pumped to the D primary cyclopac for classification. Cyclone overflow with a particle size 80% passing 75 microns passes to the flotation stage. Cyclone underflow is recycled back to the two ball mills by gravity.

The existing primary grinding cyclones are gMAX10-3139. A cyclone supplier has run a simulation for the cyclopac based on the Bradshaw project ore parameters and process conditions, and confirmed that the existing cyclopac in the circuit could be operated with larger 4" vortex finders, 5" overflow adapters and 5" overflow pipes.

A sampler, 200-SAM-001, is provided to collect samples from the cyclone overflow. Sample will be analyzed for metallurgical and accounting purposes.

Milk of Lime is added to the ball mills to bring the slurry pH level to about 11 that is required in the next rougher flotation stage.

Regrinding Circuit (Dwg. 300-F-001)

Aspy rougher concentrate collected in pump-box 300-SMP-001 (existing D-31) is transferred to the regrinding pump-box, 200-SMP-101 (new). The regrinding stage reduces particle size of rougher concentrate from 75 microns to 25 microns (80% passing). Regrinding mill, 200-MLB-120, discharges to pump-box 200-SMP-101. The combined slurry is pumped by regrinding cyclone feed pump to a cyclopac, 200-CYH-125. Cyclone underflow flows back by gravity to the regrinding mill. Cyclone overflow (-25 µm material) discharges to regrinding cyclone overflow pump-box, 200-SMP-102, from where slurry is pumped to slurry distributor (D-10) feeding to the Aspy 1st Cleaner stage.

Note: The capacity of this mill will also be reviewed for upgrading or replacement with a new regrinding mill.

Flotation

The flotation circuit consists of rougher and cleaner stages for recovery of the arsenopyrite gold concentrates. The rougher flotation has a feed with a P80 particle size of 75 microns. The arsenopyrite rougher concentrate is then reground to a P80 of 25 microns for multiple stage cleaning operations. The rougher tailings are discharged to the tailings management area.

Existing D circuit flotation banks will be used for Bradshaw ore flotation. The existing piping system will require re-routing in order to accommodate Bradshaw ore flotation flow-sheets and flow rates.

Rougher Flotation (Dwg. 300-F-001, 300-F-002)

Ground ore slurry is pumped to a conditioning tank, 300-TAK-001 (existing D-3) where slurry is conditioned with flotation reagents, including talc depressant (CMC), copper ($CuSO_4$), frother (MIBC), and promoter (3894). The required condition retention time is about 3 minutes. The conditioned slurry is fed by gravity to two parallel rougher flotation banks via a distributor 300-SPS-001 (existing D-13).

Recent test results showed that a total of 17 minutes (without a scale up factor) is required for Bradshaw ore rougher flotation. Because of the existing equipment volume limitation, the rougher stage is divided into rougher flotation and rougher scavenge flotation to achieve the total retention time.

The two existing banks, (D Bulk Rougher East and West), have total of 18 bulk rougher cells. They are Wemco 84 models with an original volume of 4.2 m^3 per cell. They were later modified to increase the volume to 5.8 m^3. These cells will remain in place and be used as Aspy Rougher Cells (300-FLO-201 to 209, 300-FLO-210 to 218), which will provide approximately 30 minutes of retention time.

Rougher concentrate from two rougher banks is collected in pump-box 300-SMP-001 (existing D-31). Rougher tails are collected in pump-box 300-SMP-002 (existing D-32) from where slurry is pumped to rougher scavenger, 300-FLO-221 to 225.

Rougher scavenger cells are the existing five D-circuit bulk cleaner No.1 west cells. These are Denver DR-100 cells, 2.8 m^3 per cell. This stage is employed to provide four additional minutes rougher flotation time (or as a rougher flotation scavenger). Scavenger concentrate is collected in the Aspy Rougher Concentrate Pump-box (D-31) 300-F-001, where it combines with concentrate from the rougher flotation. Scavenger tails are collected in pump-box 300-SMP-004 (D-40) and will be pumped to the final tailings transfer pump-box.

Arsenopyrite Cleaners (Dwg. 300-F-003)

Three cleaner stages are adopted for Bradshaw ore flotation based on flotation test work results. The purpose of these cleaners is to improve gold grade and thereby reduce the cost of shipping to third party processing.

Existing copper rougher banks have total of 12 Denver DR 24 cells, six in the north bank and six in the south bank. Volume of each cell is approximately. 1.4 m^3. These cells were used for copper/nickel separation. They can be used as Aspy first cleaner cells, 300-FLO-251 to 256, 300-FLO-301 to 306. The two banks are fed from the existing stream distributor, 300-SPS-003 (D-10).

The D circuit has two bulk cleaner No. 2 banks installed in parallel. The west bank with 5 Denver DR 24 cells can be used as Aspy second cleaners (300-FLO-261 to 265). The east bank with 5 Denver DR 24 cells will be used as the third cleaners (300-FLO-271 to 275).

Aspy First Cleaner Scavenger (Dwg. 300-F-002)

The D circuit bulk cleaner No.1 east (Denver DR-100 x5 cells, 2.8 m^3 per cell) can be used as the first cleaner scavenger. Slurry feed to the scavenger is pumped from the pump-box 300-SMP-003 (D-33).

Concentrate Dewatering and Handling Area (Dwg. 310-F-001)

The existing nickel concentrate thickener, 18.3 metres in diameter, will be used for the dewatering of final arsenopyrite concentrate. The thickener underflow will be pumped to a concentrate holding tank, 310-TAK-001, which is the existing nickel holding tank. The slurry will be pumped from the holding tank to a filter for further dewatering.

The filter press for concentrate dewatering will be sized to operate on an 8-hour per day schedule. The truck loading schedule should average 2 to 3 trucks per day depending on truck load size. The trucks can be filled up daily with the filter press dewatered concentrate slurry in 8 hours.

As the details of the project evolve with engineering plans and further analysis, the concentrate handling routing, methods, and frequency can be better defined by Gowest for Glencore.

Reagent Preparation Area

The reagent preparation and delivery systems required to provide reagents for processing Bradshaw ore are listed below:

- Lime System
- MIBC
- $CuSO_4$
- CMC
- Promote 3894
- Flocculant Preparation System

The Kidd Operations- Concentrator has a well-organized area for reagent storage and mixing. The bulk of the plant reagents are unloaded and stored in a reagent area. After preparation and mixing, they are pumped to the head tank room above the control room with overflows returning to the reagent area. Metering pumps are located beneath the head tanks to distribute reagents to various addition points. The lime system uses a circulating loop through the plant to feed the addition points.

Some modifications and/or additions to the reagent areas may be required for the purpose of processing Bradshaw ore.

17.3.4 Toll Milling Option 2 - Timmins Area Mill

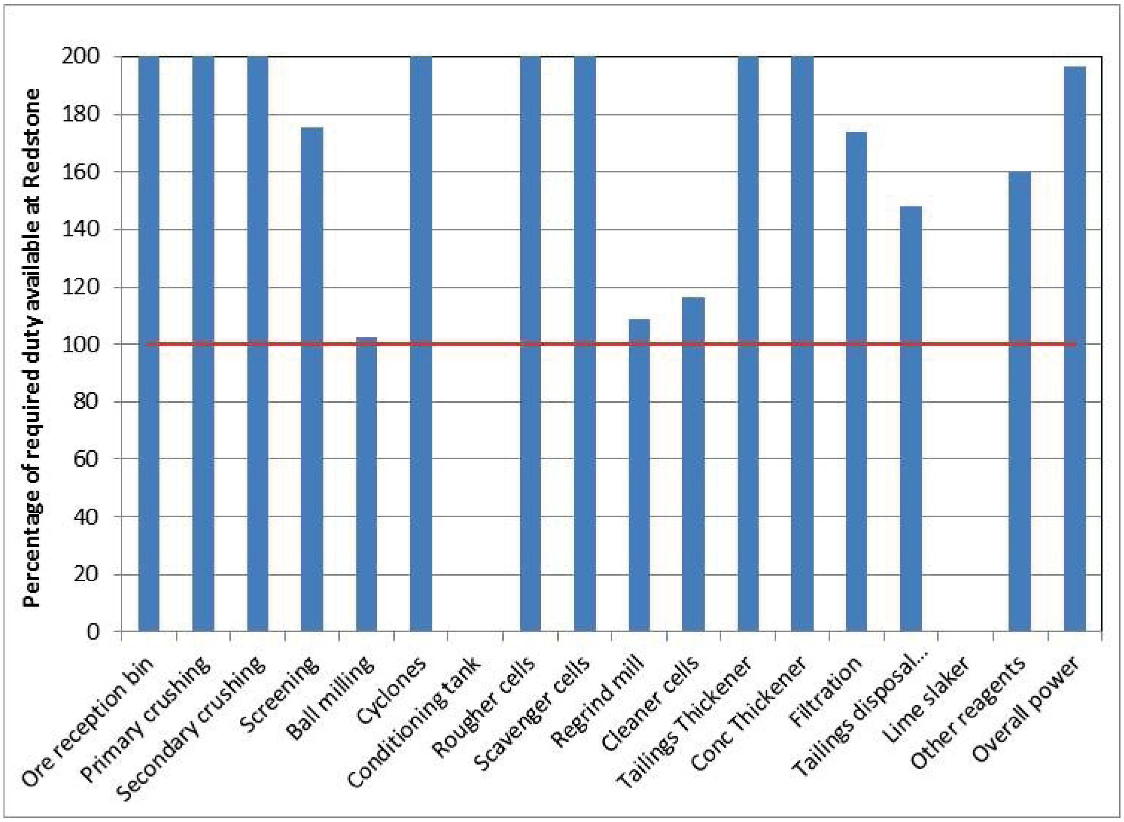
This tolling option is about an alternative operating plant (Mill) in the Timmins area that could process Bradshaw feed material. Gowest personnel and AMEC consulting engineers visited the facility in 2014. The objective of the visit was to assess the general condition of the facility and to obtain capacity information on the Mill to accommodate the Bradshaw flowsheet and throughput.

The Mill is currently toll treating other material and has a flowsheet that closely matches the intended Bradshaw flowsheet (essentially grinding and flotation followed by filtration of the final product). This Mill was in operation during the time of the visit and was well maintained and operated by the owner. It was constructed about a decade ago and has operated more or less continuously ever since and is hence in good working condition. The plant is designed for various milling streams with a potential through-put of up to 1500 tpd.

Design parameters and the flowsheet for this Mill are the same as for the Toll Milling Option 1 provided in Table 17.1: and Figure 17.2.

This Mill includes a crushing plant therefore the Bradshaw mine could deliver coarser material if required. The capacity of each section is assessed and results are presented in Figure 17.3. The maximum value of the Y-axis was set to 200% i.e. wherever the installed capacity of the Mill exceeds the required capacity by more than double it is shown truncated to 200% for visual presentation purposes.

Figure 17.3: Comparison of Existing Mill's Installed versus Required Capacity per Operation



*Note: This chart is based on the existing mill capacity of 1500 tpd which is depicted at the 100% redline.

The mill description henceforth is based on treating 1500 tonnes of material per day, the plant is more than sufficient for handling the tonnage rate of the Bradshaw ore feed material.

Crushing and Screening

The 30"x42" primary jaw crusher can be used mainly as a "chute" or can be used to crush feed, depending on the crushing at the mine site.

The secondary crushing circuit will receive ore at a F80 of 60 mm and will produce a screen undersize at a P80 of approximately 8 mm. A smaller transfer size is targeted for this plant because it became apparent that this Mill has reduced grinding capacity. Using the crushing work index of 14.5 kWh/t and published data by Metso, it is estimated that this HP400 crusher would consume 112 kW from its motor. The HP400 is driven by a 400 hp (315 kW) motor that has more than enough power to perform this duty. The small targeted product will demand a small closed side setting of 13 mm and would result in a circulating load of 58% in the secondary circuit. The conveyor belts have not been assessed whether they can accommodate this circulating load. However, given that the primary crusher can be used to reduce the feed size to the HP400 significantly, it is assumed that an 8 mm product would be achievable. The same argument applies to the capacity of the double deck screen.

Ore Storage

The two ore storage bins ahead of the primary Ball Mills can each accommodate 800 t of coarse ore. At a total surge capacity of 1600 t these bins provide more than sufficient surge capacity between the mine and the plant. This significant overcapacity would practically decouple the mining and treatment operations.

Grinding

This Mill currently has two operating and one standby 10' x 13' ball mills. Two of these would be fed using existing conveyor belts as primary mills while the third would be assigned to perform the secondary grinding or regrinding duty in the Bradshaw flowsheet. With the average Bond Ball Mill Work Index of 15.9 kWh/t and the sizes as stated, each of the two parallel mills will draw 547 kW of pinion power (734 hp).

The capacity of all of the unit operations at the Mill exceeds the required capacity. Only two of the ball mills are fed by a conveyor belt, the third mill will act as a secondary mill receiving cyclone underflow as opposed to crushed fresh feed, a modification that would be relatively simple to implement.

The third 10' x 13' ball mill could be used for regrinding of rougher and scavenger concentrates. However, if a vertical stirred mill (such as Metso's 1250 hp Vertimill) is supplied for regrinding it would require a rectangular footprint of approximately

6 x 10 metres (including maintenance access and pumps) – it is conceivable that such an area would be available indoors at the Mill.

The third mill is driven by a 560 kW motor. This is adequate power to act as a regrind mill in the process flowsheet.

The current operation at this Mill uses a mix of different sized cyclones divided between two easily accessible cyclopacs. The size of these cyclopacs appeared to be more than sufficient to accommodate more cyclones for the grinding circuits.

Flotation

There is currently no dedicated conditioning tank at the Mill. However, many installed rougher tanks would not be required for the Bradshaw ore and can hence be used for conditioning duties. If required there is sufficient space on the platform next to the cyclones to install a new conditioning tank.

The Mill currently has four banks of roughers, with each bank having eight of 14.3 m^3 rougher cells. Based on a preliminary mass balance and the residence time estimated from test results the Bradshaw ore would require only 14.9 cells. Hence, two of the four available banks would suffice.

Given the number of free rougher cells it was assumed that the cleaner scavenger section could consist of three of the rougher cells in one of the two unassigned banks. The residence time required for the scavenging duty calculates to a total live volume of 17 m^3 only. Using three 14.3 m^3 cells would provide a large excess residence time thus compensating for any possible short circuiting due to the small number of cells.

This Mill has just one line of small cells for concentrate cleaning. To circumvent this it is assumed that the remaining five cells in the bank of large cells can be used as the first cleaner bank. This will require some modifications to the fifth tank to seal it off from the bottom three tanks such that this bank can be effectively split into two separate and autonomous banks. Alternatively, the fourth unused rougher bank can be used. The total volume required for all three stages of cleaning is estimated at 80 m^3, as this arrangement will provide the required volume as shown in Figure 17.2 above. This proposed arrangement still leaves eight large 14 m^3 cells unassigned so there is more than sufficient capacity at the Mill to perform the required flotation operation.

The first cleaner bank would thus consist of five 14.3 m^3 cells while the second cleaner bank would consist of six 2.8 metres (100 ft^3) cells and the third cleaner would consist of four 1.4 m^3 (50 ft^3) cells. The 3rd cleaner bank's total available volume falls well short of the volume required but it is assumed that this is not a significant issue given the excess total volume available.

Thickening and Filtration

Two thickeners at the Mill have diameters of 12.2 and 7.9 metres and will provide more than sufficient capacity for settling both the Bradshaw ore flotation tailings and concentrate respectively. By comparison, the theoretically required diameter of the tailings thickener is 7.7 metres, while the concentrate requires a smaller 1.8 metre diameter thickener. The concentrate thickener provides ample surge capacity ahead of the filtration and concentrate load out operations.

It is understood that the Larox 1.6 PF filter installed at this Mill has seven chambers while a four-chamber unit would be sufficient for the Bradshaw application. The model and number of chambers available needs to be confirmed in the future.

Miscellaneous

This Mill does not currently have a lime slaker. A lime slaking and distribution system would need to be obtained. One possibility would be to buy a lime slaking skid and install this in a suitable area at the plant. A second possibility would be to add quicklime pebbles via a feeder onto the ball mill feed conveyor to adjust the operating pH.

It has been assumed that all of the other reagent mixing and dosing systems are available at this Mill. It is also assumed that the process, raw and gland seal water systems are also adequate for the Bradshaw duty as is the air service. These assumptions are justified by the similarity in unit operations, flowsheet and throughputs between the current operation and the proposed treatment of Bradshaw ore.

The total power consumption required to process Bradshaw ore at this rate is estimated at 3.0 MW. This is well below the 6.0 MW capacity of the main power distribution substation at the Mill.

In conclusion, this Mill provides an excellent fit to the required flowsheet and capacities for the Bradshaw ore. The potential limitation is with the existing grinding power, but only when the plant would treat up to 1500 tonnes per day, which is a long-term concern.

New Plant Option- Kidd Brownfield Site

This option considers the construction and operation of a complete gold plant at the Kidd brownfield metallurgical property. It is anticipated that re-use of the site would offer many advantages, from extending existing permits to savings on infrastructure. Advantages specifically include the following which should already be readily available at the site:

- Certificate of approval (air).
- Process heating permit.
- Standby generators permit.
- Area dust control permit.
- Certificate of approval for tailings disposal.
- Permit to draw water.
- Rail access.
- Power – substation.
- Water – pumping equipment.
- Reduced site clearance and preparation.

This option would require more capital costs and longer engineering and construction time compared with the other options. For Gowest, this option would only be considered if toll milling was not available for long term operations in the Timmins area. For purposes of this study, Gowest is pursuing the toll milling option and have obtained costs of this option.

Toll Gold Refining

Producing Dore Gold Bars

Gowest will produce a high grade gold concentrate product that can be shipped in bulk with a low moisture content. The quantity of concentrate is dependent on the quantity of ore processed at the mill and the feed grade to the mill from the mine.

The gold concentrate can be either shipped by truck or rail depending on the location of the processing facility.

Potential facilities currently available to Gowest for processing the concentrate are as follows:

1. Cobalt Refinery, North Cobalt. United Commodity memorandum of understanding to receive Gowest gold concentrate at process in expanded plant. Current plans at the plant include expanding the refinery to process higher rates of feed tonnage to match or exceed the potential production from the Bradshaw Deposit.

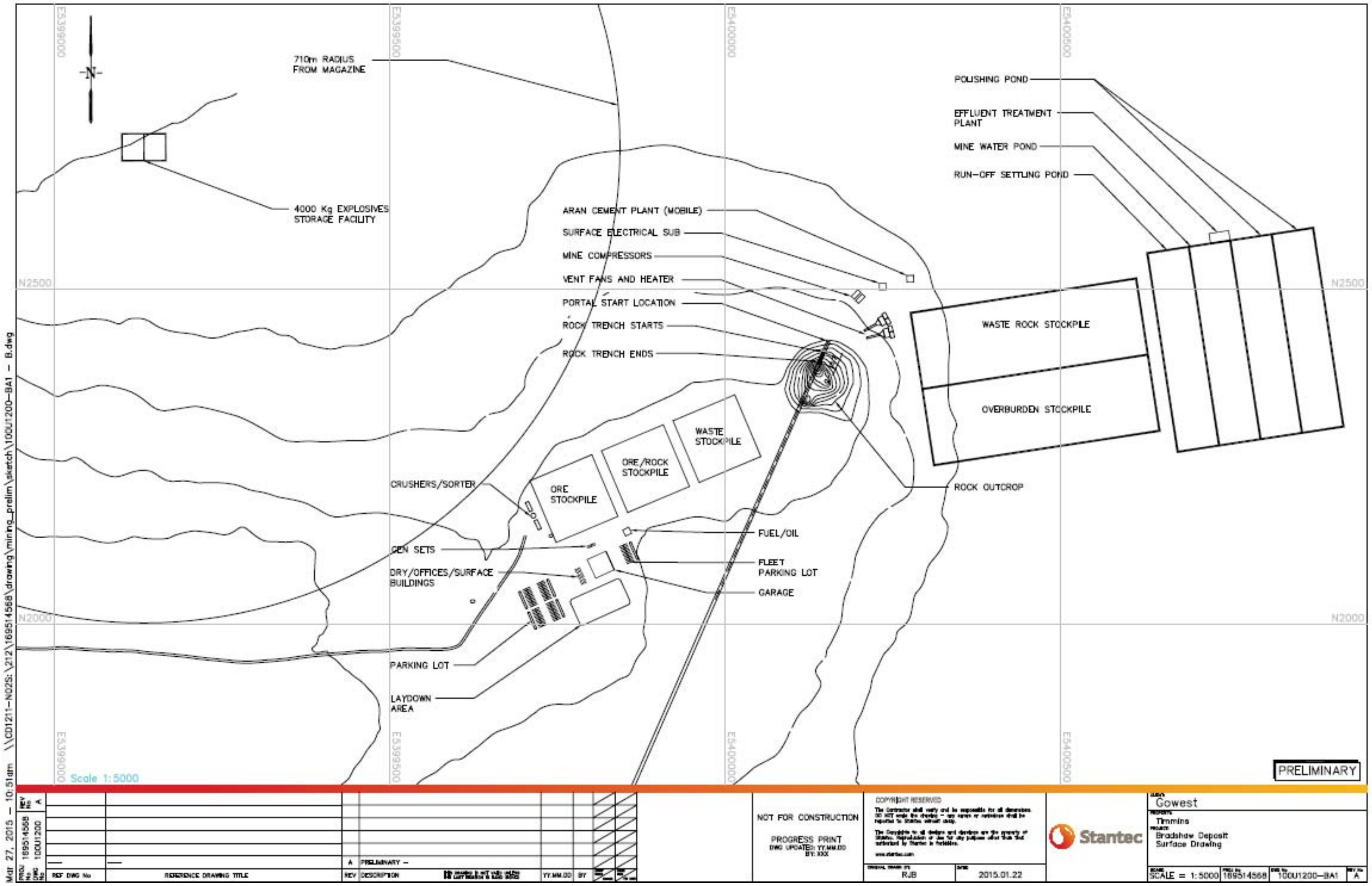
2. Barrick and Newmont Gold Refineries in Nevada, USA
3. Overseas via shipping container, currently being explored with various commodity traders.

Gowest has been in ongoing discussions with the above potential facilities to process the gold concentrate. These discussions require further details on how contracts for gold processing would be constructed. Preliminary costs were provided by the Cobalt Refinery and have been used in the financial analysis in this study.

18.0 PROJECT INFRASTRUCTURE

This section describes the mine site infrastructure that will be constructed. Efforts have been made to minimize the area that will be disturbed by the construction of these facilities and to minimize the capital cost. The general arrangement of mine site surface infrastructure is shown in Figure 18.1. Detailed drawings can be found in Appendix F – Mine Drawings and details on compressed air, de-watering, water discharge design and calculations are found in Appendix H – Air and Water.

Figure 18.1: Surface Site Plan Including Infrastructure



18.1 Mine Site Surface

The mine site surface infrastructure will include the following:

- Roads
- Mine Dry and Office Trailers
- Sprung Building and Laydown Area
- Maintenance Building
- Compressor Plant
- Surface Water Distribution System
- Parking Area
- Waste, Mix Development and Ore Storage Areas
- Surface Settling Sumps
- Ore Sorter and Crushing Plant
- Power Distribution System
- Main Fresh Air Fan and Heater Systems
- Mine Air Heating
- Main Return Air Fans
- Mine Water Treatment Facility
- Portal Access to Underground
- Cement/Backfill ("Aran" or equivalent)

18.1.1 Roads

Existing Road to Site

The existing road to the Bradshaw site will be used by personnel, material delivery units, and haulage trucks transporting material to the mill. The current road will be used during the bulk sample phase. $250,000 of upgrades to the road such as widening, top layer and proper culverts and ditching will be required once the the project becomes an operating mine. The road will be maintained by a contractor year round.

Site Access Road

A 2,000 metre site road allowance or $536,000 has been included to service the various infrastructures required during the bulk sampling and operating mine phases. The roads will be maintained to support service vehicles and underground truck travel to the waste and mill feed storage areas. The area is relatively flat and additional roads will be constructed to the fill batch plant, sorting area, and returnair raise (RAR) locations. The road will be maintained by a contractor throughout the mine life.

18.1.2 Mine Dry and Office Trailers

A mine dry and office facility will be located near the ore stockpile and portal area. A 30 person mine dry will be constructed with portable trailers and will be installed for the bulk sample phase and will be later expanded to accommodate 60 construction and development personnel during the mining phase.

The facility will provide space for a meeting/assembly area for mine operations and maintenance crews, offices/work areas for all mine technical, supervisor, and management personnel, and mine rescue equipment maintenance and storage.

18.1.3 Sprung Building/Laydown Area

An allowance of $69,000 for a "Sprung" building (40 m x 50 m) has been included for this project. Items such as ventilation fans and ducting, ground support, mobile equipment parts and other items not required to be stored in a heated facility (warehouse) will be kept in the "Sprung" building/laydown area.

18.1.4 Maintenance Shop

The shop and warehouse building will be approximately 12.5 x 25 metres in size. The shop will have three bays to service trucks and other surface and underground equipment. It will be equipped with a 10 tonne capacity overhead crane and will provide adequate space for the storage of maintenance tool cabinets and other items required for maintaining the mobile fleet. An allowance of $177,000 of capital has been included for this building.

18.1.5 Compressor Installation

The compressed air system will supply compressed air throughout the mine. The selected portable compressors are air-cooled, two-stage rotary screw type at 1400 cfm (400 hp) each. Only one compressor with a spare is required during the bulk sample phase. There will be a total of three compressors installed, two operating and one spare in the mine operating phase. Each compressor will provide 1,400 cfm (125 psig) to supply the estimated average demand of 2,040 cfm with a peak demand of 2,660 cfm. A 9,690 litre (2,560 USgal) air receiver and an oil and water separator will also be incorporated into the compressed air system during the mining operations. All compressors and accessories will be located near the portal area. A total $92,000 of capital has been allocated for the installation and piping the portable compressors to the underground compressed line distribution system.

18.1.6 Surface Water Distribution System

The surface water distribution system will provide fire water to surface buildings, process water to the batch plant, and process water underground for the mining equipment. The process water will be pumped from existing surface diamond drill holes to a water tank and distributed to the various facilities.

The water tank will contain 500 m^3 (132,086 USgal) of water and this volume is capable of supplying the fire suppression system to the surface buildings at 53.6 L/s (850 USgpm) of flow for 90 minutes, and the underground service water system with a 3.5 hour supply at the estimated peak demand. The bottom portion of the tank will serve as the fire suppression water supply reservoir. Service water will only be drawn out of the top portion of the tank to ensure that the water volume required for fire suppression is available at all times.

There will be two 20 HP pumps (1 Duty, 1 Spare) used to supply service water for the surface buildings, the batch plant and service water underground. Average batch plant water consumption is estimated to be 3.15 L/s (50 USgpm) and underground peak water consumption is estimated to be 16.5 L/s (261 USgpm). Each of the pumps will deliver up to 20.2 L/s (320 USgpm).

A total of $70,500 has been allocated to this project for the pumps and water tank. The water source is expected to be nearby from the West Buskegau River and pump locations have not been determined at the time of writing this report. This cost estimate will need to be updated once additional information is available.

18.1.7 Parking Area

Two parking areas will be provided. A common parking area will be sized to accommodate site personnel and visitors and will be located to the entrance to the mine site.

A parking area for surface and underground equipment will be located adjacent to the maintenance shop building. Surface service vehicles and underground equipment will park in the area during shift change.

18.1.8 Ore, Mix Development and Waste Rock Storage Areas

Three pads, 100 x 100 metres, will be constructed for the ore, mix development and waste rock material. Each pad will be capable of accommodating, 5000 tonnes of material and $360,000 has been allocated for this construction.

A larger area for waste and overburden (240 metres x 340 metres) is sized to accommodate 2 million tonnes of material (816,000 m^3). A portion of the waste will be used as backfill to fill underground stopes. A ditch system surrounding this area will be constructed to collect and sample water to be treated at the waste water treatment plant if required.

18.1.9 Surface Water Settling Sumps

The settling sumps will be performed in two stages: first for the bulk sample period and a larger facility for the operation of the mine. The construction of the smaller settling sumps will occupy an area of 18,750 m^2 and the larger settling sump will occupy an area of 56,250 m^2. A total cost of $1.435 million has been included in the capital estimate and provides a total area of settling sumps of 250 metres x 300 metres or 75,000 m^2.

18.1.10 Ore Sorter and Crushing Plant

A total of 2.45 million tonnes of ore and mix development material will have to be crushed to -2" and processed through the ore sorting equipment before being shipped to the mill.

The crushing plant and ore sorting equipment will be located close to the ore and mix development material pads. $500,000 has been included in the capital estimate for the installation of this equipment.

18.1.11 Power Distribution System

Power for the Bradshaw site will be initially supplied by diesel generator sets for the bulk sample phase and Year 1. Electrical power from the Ontario grid will be available in Year 2 once the new power line and switchgear are installed.

Sufficient grid capacity and a connection point are assumed available at the intersection of mine site road and Hwy 655. Included with the power line are fiber optic services from Hwy 655 for business system/internet service. It is assumed that sufficient bandwidth exists at the Hwy 655 intersection connection point. Hydro One has been contacted, and a New Customer Connection Request has been submitted on behalf of the project such that Hydro One can evaluate internally whether sufficient system infrastructure and capacity is available. A total of $6.5 million (including 30% contingency) of capital has been included to establish a power line to the mine site.

One genset will be required (Cat model C175-20) for the bulk sample and will located close to the fuel farm and stockpile areas. A second gen set (Cat model

C175-16) will be required as the mine is expanding to lower levels and until power grid from Ontario Hydro is available. A total of $430,000 has also been added to the project for the installation of the necessary gensets and required switchgear.

The main site electrical switchyard and substation will be located close to the portal area. The electrical distribution system will initially include a diesel generator for the first two operating years and will connect to the Ontario grid via overhead lines for the remainder of the mine life. General site lighting will be a combination of power line pole mounted fixtures and building mounted fixtures at the offices, shop, and other miscellaneous buildings.

Underground facilities will be code compliant and most electrical equipment will be located within mine power centres (MPC) located adjacent to the point of utilization (e.g. dewatering sumps, level substation).

A dual circuit feeder will be provided for underground power, allowing for leap-frogging of the feeders during development and balancing production loads later on. The total load for surface and underground, approximately 5 MVA will include dewatering and mining of two stopes on three levels. Operating costs have been assumed @ $0.08/kwhr.

The estimate assumes that the electrical feeders will be installed in the ramp, but they can be routed down the RAR if preferred. The cables will be terminated in junction boxes at each level.

Underground communication will be by leaky feeder, and hardwired telephone. There will be no advanced automation or communication systems.

18.1.12 Main Fresh Air Fan and Heater Installations

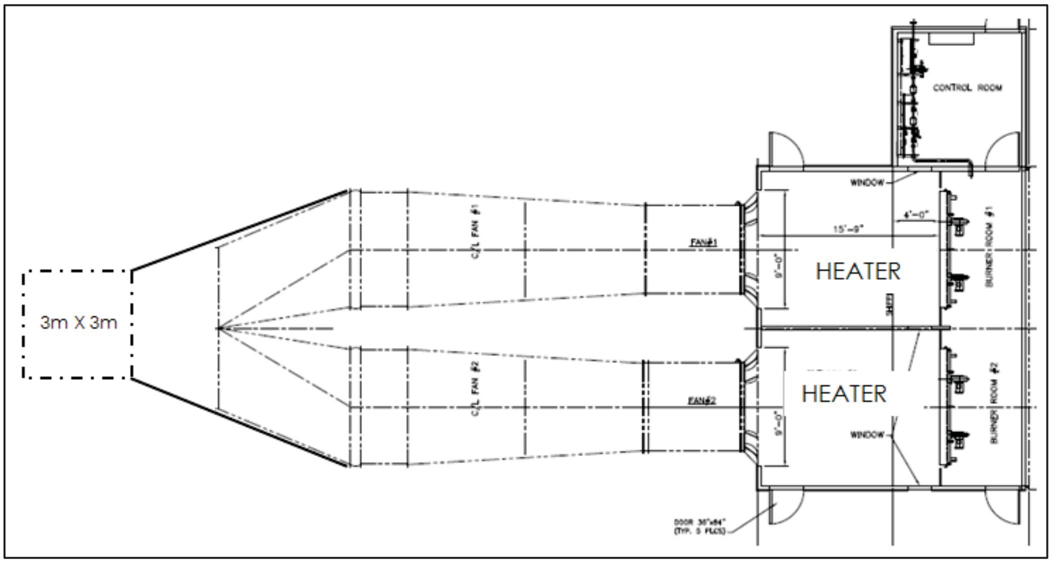
Figure 18.2 illustrates the main fan and heater installation on surface (beside the portal) consisting of:

- Inlet hood with safety screen.
- Rectangular inlet silencer to reduce the noise level to 85 dBA at 3 metres from inlet.
- Two 5.9 MW (18.5 MM BTU/hour) direct fired propane mine air heaters come with control room, and transition section comes with access doors.
- Vane axial mine fans at 37 kW (50 HP) each comes with inlet bell with screen and outlet evase.
- Flex connection.
- Gravity operated back-draft dampers.
- Transition section from fan outlet to 3 metres x 3 metre elbow.

- 90 degree x 3 metres (10 ft.) x 3 metres (10 ft.) elbow to connect to 3 metres (10 ft.) x 3 metres (10 ft.) raise collar.
- Air volume and pressure monitoring devices.

The design will provide an air-lock door system to allow access for egress.

Figure 18.2: Typical Intake Axial Fan/Direct Fired Heating System



The main intake fan maximum operating duty point is listed in Table 18.1: .

Table 18.1: Intake Raise Fan Duty Points

System	Number of Fans	Operating Duty Point	Connected Fan kW	Mine Air Heater
Duty	2	189 m³/s (400 Kcfm) at 124.5 Pa (0.5" WG) SP	2 x 37 kW (50 HP)	2 x 5.9 MW

18.1.13 Main Return Air Fan Installations

The main exhaust raise is sized at a 3.7metre (12 ft.) x 3.7metres (12 ft.) to keep the resulting velocities out of the critical velocity range 7-12 m/s (1400-2400 fpm) for exhaust raises. The proposed exhaust raise system will also serve as the secondary egress route from underground workings as it is proposed to be equipped with a man-way.

The type of man-way installed in the raise will dramatically impact the raise resistance and therefore the surface main fans operating duty point pressure which impacts the electrical energy consumption.

The air velocity within the 3.7 metres (12 ft.) x 3.7 metres (12 ft) raise at the design volume of 170 m^3/s (360,000 cfm) is estimated at 2.8 m/s (2519 ft./min). This velocity will make travel through the system at full air volume difficult; therefore, in the event secondary egress is required, the variable frequency drives on the main RAR fans can be adjusted to lower airflow and velocity to allow safe passage. The operating points for the return air fans are summarized in Table 18.2.

Table 18.2: Main Return Air Fan Duty Points

System	Number of Fans	Operating Duty Point	Connected Fan kW
Duty	2	85 m^3/s (180 Kcfm) at 2.6 kPa (10.4" WG) SP	2 x 300 kW (400 HP)

18.1.14 Mine Air Heating

The underground air heating system will be located in the short raise near the portal and will heat the mine air to a nominal temperature of 3°C during the winter months. The heating system capacity is designed for a 48°C (86°F) temperature range to allow for heating of the mine air at low ambient temperatures (-45°C has been recorded in the Timmins area).

The direct propane fired system will include two 5.9 MW (18.5MM BTU/hour) heaters with a temperature rise of 48°C (86°F), common control room and will be complete with valve trains, electrics, and two x 37 kW (50 HP) variable frequency drives for the heater raise fans.

The propane system will include a three 90,000 litre tank farm, vaporizers, pump, cement slab and piers, piping, and fencing.

A total of $1 million of capital has been allocated for the fan/heater and RAR fan installations.

18.1.15 Mine Water Treatment Facilities

Domestic Water Servicing

This system will provide water for use by personnel working at the mine and visitors, for eyewash, safety, showers, hand-washing, toilets and urinals; but will not be used for drinking or cooking consumption. Water for drinking will be provided by bottled potable water from commercial sources.

Sanitary Wastewater Servicing

Sanitary facilities will be provided for use by the mine employees and visitors, and include handwash sinks, toilets and urinals, and safety showers.

The estimated wastewater flows will be less than 10 m^3/d and will be discharged to a subsurface septic leach field. The final designs for this system were not available and the costs will be added at the next level of study.

Mine Supply Water Treatment

The average water demand for underground process use is estimated at 31 m^3/hr. A pumping capacity of 40 m^3/hr will be provided to allow for peak demand periods.

The process water supply is assumed to be from several surface diamond drill holes located close to the surface facilities.

Mine Process Water Discharge Treatment

A treatment plant will be provided to treat water discharged from the settling ponds. There are several discharge points available for the treated water with the preferred point being at the nearby West Bakegan River.

18.1.16 Portal Access to Underground

A portal will be constructed on the surface outcrop, north of the deposit near the proposed infrastructure and storage pads. The portal will provide primary access to underground and allow for haulage of workers and material to surface. The ramp will be constructed at a maximum grade of -15% to a size of 5.5 metres by 5.0 metres supported by rebar bolts and screen. A total of $307,000 has been included to establish the portal for this project.

18.1.17 Aran Plant Installation

The backfilling of stopes will be required to mine the Bradshaw Deposit. The backfill will consist of cement slurry mixed with development waste rock. The Aran plant is a portable plant comprised of a cement silo and a batching tank to mix the cement slurry. The cement slurry will be pumped into a pipeline to feed underground tanks, stored close to the stope requiring backfill. The cement slurry will be pumped and mixed with the development rock as dumped into the stopes.

$1.6 million of capital has been included to set up the Aran plant, associated pipeline and the procurement of two underground mixing tanks.

18.1.18 Underground Infrastructure

Underground infrastructure has been kept to a minimum in order to reduce capex and opex costs. No major garage, tire bays or wash bays have been included in the mine design. All equipment repairs will be performed in the maintenance garage located on surface.

Mine Dewatering

Water from mining operations, backfill decant, and groundwater will be collected in sumps located throughout the mine. The water collected in the sumps will be pumped to surface using the cascading dewatering system. A dewatering rate of 190 m^3/hr (840 USgpm) has been assumed during full mine production. During development, submersible pumps will remove water from the mine, until such time as the main pump stations are established.

A semi-dirty cascading pump system is proposed for the Bradshaw project. The use of submersible pumps and elimination of dirty water sumps will reduce the capital cost and standardizes the pump sizes for the project. Each level sump will be connected with boreholes and will drain to the larger cascade sump located three levels below. As shown in Table 18.3, the proposed cascade dewatering system is comprised of the following.

Table 18.3: Sump Location and Pumping Rates

Sump Location	Flowrate		Sump Size	Sump Length	TDH	Equiv Pressure	Pump Power	
45 Level	300 m^3/h	1,320 USgpm	432 m^3	36 m	54 m	575 kPa	80 Kw	107 HP
135 Level	270 m^3/h	1,180 USgpm	336 m^3	28 m	116 m	1,235 kPa	154 Kw	207 HP
225 Level	270 m^3/h	1,180 USgpm	240 m^3	20 m	116 m	1,235 kPa	154 Kw	207 HP
315 Level	300 m^3/h	1,320 USgpm	432 m^3	12 m	120 m	1,279 kPa	83 Kw	111 HP
405 Level	80 m^3/h	310 USgpm	64 m^3	8 m	90 m	1,679 kPa	62 Kw	83 HP
495 Level	80 m^3/h	310 USgpm	64 m^3	8 m	90 m	1,679 kPa	62 Kw	83 HP

Each cascading sump will collect water from the level and boreholes sumps above. This semi-dirty type of dewatering system is less expensive to build but requires agitators (usually compressed air) to keep solid material in suspension. This will increase the wear and tear on the pumps but will eliminate the sumps from being "sanded out." The cascading sumps will be driven at a -20% grade to allow for LHD access to clean out the slimes.

An allowance of 210 metres (4 metres x 4 metres) or $450,430 has been included in the development costs for the cutting of the level sumps.

Refuge Stations

The cost of three refuge stations has been included in this project. The specific location for each refuge station has not been determined (will be developed on every fifth level) for this study but an allowance for development of 40 metres has been included in the development costs ($85,600), and $75,000/each or $225,000 for the refuge stations construction.

Explosives and Accessories Magazine

The cost of three powder and fuse magazines has been included in this project. The specific location for the magazines have not been determined (will be developed on every fifth level) and an allowance for development of 48 metres has been included in the development costs ($103,000) and $75,000/each or $225,000 for the magazine construction.

Level Storages

A development allowance has been added to each level development costs as shown in Table 18.4.

Table 18.4: Level Storages

Description	Quantitiy	Size	Total Metres	Total Costs
Ore Storage	16	4 m x 4 m x 10 m	160	$343,184
Waste Storage	16	4 m x 4 m x 10 m	160	$343,184
Electrical Substation	16	4 m x 4 m x 10 m	160	$343,184
Level Sump	16	4 m x 4 m x 8 m	128	$274,547
Supply Storage	16	4 m x 4 m x 6 m	96	$205,910
Total	16		704	$1,510,000

Secondary Egress

The main egress to the mine will be the main ramp accessing each individual levels. A secondary means of egress will be located within the RAR system from the bottom mining level to surface. The escapeway will be comprised of ladders and landings with a screened wall separating the manway and the 3.7 metre by 3.7 metre return air raise (RAR). A total of $4.35 million has been allocated to sustaining capital for the development of the RAR and the construction of the secondary egress.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Neither Gowest nor Stantec has conducted a market study in relation to the gold doré that will be produced by the Bradshaw Project. Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers.

19.2 Commodity Price Projections

Commodity pricing is based on base case metal prices and exchange rates consistent with current consensus estimates.

This economic analysis was based on a gold price of US$1,200 /oz using a US$/C$ exchange rate of $0.80. This is in line with the average gold price over the last five years (May 28, 2010 to May 29, 2015) of US$1,433 /oz and the US$/C$ exchange rate averaging $0.96.

19.3 Contracts

Gowest currently has no binding agreements or contracts in place for the development of the Bradshaw Gold Deposit.

While preparing the economic analysis, non-binding “budget” quotations were received for a number of key cost items that were typical of and consistent with standard industry costs including;

- Mining contractor rates
- Mine site crushing rate
- Transportation rates for ore and concentrate
- Toll milling rates
- Gold concentrate refining cost terms

For purposes of this study, Stantec developed the costs for a mining contractor supported operation as a basis to develop operating costs that can be used in the financial analysis. Gowest Gold expects that terms contained within any sales contract that could be entered into for the sale of gold would be typical of and consistent with standard industry practices, and be similar to contracts for the supply of gold elsewhere in the world. In the opinion of the QPs, Gowest Gold will be able to market gold produced from the Bradshaw Gold Project at the prevailing world price through a variety of buyers available to other gold producers in the Timmins area.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

Unless otherwise noted, the following sections have been summarized from the 2009-2014 Environmental Baseline Report Gowest Bradshaw Project by Golder Associates (2015a).

20.1.1 Overview

Gowest recognized the importance for initiating environmental baseline studies and the permitting process at an early stage, and as such, environmental baseline studies were carried out between 2009 and 2014. The objective of the environmental baseline studies was to document existing conditions at the Project site prior to development. The initial phase (Phase I) of the baseline studies was undertaken in 2009 and 2010, to collect existing background information and carry out limited field baseline data collection programs, including a small-scale geochemical evaluation program, and a limited hydrology/surface water quality program. Phase II of the baseline studies involved the completion of programs initiated in Phase I as well as remaining baseline field programs (e.g. hydrogeology, groundwater quality, aquatic biology, vegetation, and wildlife, carried out between 2010 and 2014).

20.1.2 Geochemistry

The primary objective of the geochemical characterization program was to provide sufficient data for the evaluation of the environmental behaviour of the various waste materials expected to be produced during mining and mineral processing at the Bradshaw Deposit. Components of this objective included:

- Identification of mine materials (i.e. waste rock and ore) that may generate acid rock drainage (ARD) and/or metal leaching (ML).
- Quantification of mineral reactions to develop metal leaching rates.
- Identification of key factors that could influence site water quality; and
- Provide input to engineering design and other environmental evaluations for the Project.

The geochemical characterization included the following tasks: a review of the site geology and general properties of mine materials and data evaluation and characterization of ARD/ML potentials of all collected materials. Geochemical characterization was completed using samples selected from the diamond drill core to represent specific rock types, and to evaluate the characteristics of geological materials on a scale that is relevant to proposed mining operations. The scope of work was consistent with the guidance documents that have gained regulatory acceptance in jurisdictions around the world.

- The geochemical work on the waste rock indicates that the argillaceous sediment samples have potential for acid generation based on static test data. The argillaceous sediments are a very minor lithology (<5% of the total rock) at the Bradshaw Deposit. The remaining rock types are expected to be non-acid generating. In general the neutralization potential of waste rock samples was high, consisting primarily of reactive carbonate minerals. Acid generation of the ore is not an issue based on the results of Neutralization Potential Ratio (NPR), Carbonate Neutralization Potential Ratio (CaNPR) and Non Acid Generating (NAG) pH values. Furthermore, the neutralization potential of the ore was high, and consists primarily of reactive carbonate minerals. Metal leaching, most notably arsenic, does appear to be an issue with the ore and waste rock, as observed in both short-term leach and humidity cell tests. The waste rock lithologies reporting the highest arsenic concentrations in the leach tests were the argillaceous sediment and the unaltered mafic volcanics.

20.1.3 Hydrogeology

The purpose of the hydrogeological program was to document existing hydrogeological conditions at the site prior to development, establish baseline groundwater quality, determine groundwater flow direction and identify potential groundwater users in the area. The baseline hydrogeological study included the following tasks:

- Background review and data gathering.
- Determination of groundwater monitoring well requirements (overburden and shallow bedrock).
- Drilling, installation and development of monitoring wells.
- Sampling of monitoring wells.
- Assessment of groundwater quality and flow direction and potential groundwater users in the area.

Five monitoring wells were installed in June 2010 on the Project site. The stratigraphy encountered in the wells was peat/organics, clay/silt, and till overlying bedrock. Water quality samples and water level measurements were taken from October 2010 to August 2012. Groundwater elevation is near surface, with some artesian conditions encountered at the site. Hydraulic conductivity estimates for the screened intervals of the wells ranged from 3.3 x 10-7 to 2.1 x 10-5 m/s, with shallow bedrock hydraulic conductivities being slightly higher than that of overburden.

Groundwater users in the area are very limited with no known active wells being located within 10 km from the center of the site. Shallow groundwater flow in the eastern portion of the site is likely flowing east towards the West Buskegau River and west in the western portion of the site towards a wetland area which flows northwards and eventually reports to the West Buskegau River.

20.1.4 Meteorology And Hydrology

The objective of the baseline hydrology program was to document existing climate conditions, streamflow and water elevation data to provide and understanding of site hydrology over a range of conditions and drainage basin sizes. The baseline hydrology program included the following tasks:

- Desktop review and basic analysis of available regional information;
- Selection of station locations and installation of equipment;
- Station monitoring; and
- Comparison of regional and local hydrologic information.

Stream discharge was collected at four stream crossings around the Project site from September 2010 to November 2011 and in the winter of 2014. Water elevation (stage) and discharge data were used to develop stage-discharge rating curves at each of monitoring locations. The rating curves at the stations were used in conjunction with the continuous stage (water level) recorders that were installed to estimate a continuous discharge record over the program period.

Despite the fact that station locations were not optimal due to site accessibility limitations around the Project area, it was found that for the recorded period, the site hydrology is comparable to the regional flow regime.

20.1.5 Water Quality

The baseline ground water, surface water and sediment quality program included the following tasks:

- Collection of water samples and limnological data at six surface water locations;
- Collection of sediment samples at surface water locations; and
- Collection of ground water samples and limnological data, and measure water levels at eight monitoring wells.

Due to seasonal variability, four time periods were sampled throughout each year to determine seasonal trends and changes. The sampling occurred from September 2010 to August 2012.

Fifty-four surface water samples were collected from six locations at the Project site. The surface water is generally acidic to near-neutral and organic, which is characteristic of the low-flow meandering watercourses of the wetland and coniferous forest environment. Several metal levels were elevated with respect to the receiving water quality criteria of the Canadian Council of Ministers of the Environment (CCME) and Provincial Water Quality Objectives (PWQO) in all stations throughout all sampling seasons including iron, aluminum, cadmium and copper. Phosphorus was also elevated with respect to the PWQO criteria with concentrations classified as meso-eutrophic to oligotrophic according to the CCME guideline. Sporadic concentrations (7%) of free cyanide were elevated with respect to the receiving water quality guidelines (CCME and PWQO) at some stations. Some phenol, cobalt and lead concentrations (below 17%) were elevated with respect to receiving water quality criteria (CCME and PWQO). Sporadic selenium, zinc and Escherichia coli concentrations (below 6%) were elevated with respect to CCME and PWQO. There was no apparent overall geographical trend observed in water quality. A seasonal trend was observed with the highest numbers of exceedances with respect to CCME and PWQO criteria occurring in the winter sampling period (e.g. February).

Twenty-eight sediment samples were collected from six locations at the Project site. Several metals were elevated with respect to the Provincial Sediment Quality Guidelines and CCME at all stations including cadmium, chromium, copper, iron, manganese, nickel. All concentrations were below the CCME Probable Effects Level. A correlation between water quality and grain size and/or sediment quality is not apparent.

Sixty groundwater quality samples were collected at the Project site. The groundwater was generally near-neutral and within the receiving water quality criteria ranges (CCME and PWQO). The groundwater samples had consistently elevated concentrations of phosphorus (92% exceeded the PWQO criteria value of 0.02 mg/L) and were classified according to the CCME guidelines to be hyper-eutrophic, indicating that the groundwater is nutrient-rich. Dissolved iron concentrations were elevated with respect to the CCME and PWQO criteria in greater than 90% of samples. Several metal concentrations were elevated with respect to the CCME and PWQO criteria including dissolved aluminum, dissolved arsenic, dissolved cobalt, and dissolved boron (i.e., greater than 25% of samples). Sporadic concentrations of copper, molybdenum, phenols, tungsten and vanadium exceeded the CCME and/or PWQO criteria at one or more stations.

20.1.6 Terrestrial Ecology

The terrestrial ecology study involved a phase one records review and a phase two field mapping (plant community mapping and breeding bird point count surveys). The phases were completed to provide a baseline characterization of the terrestrial environment within the Project site. They provide preliminary baseline data as support for permit applications and the basis for comparison with future studies.

Based on the results of the records review and field surveys, the following points relative to the Project site were highlighted:

- A total of 84 plant species were inventoried during the detailed plant community surveys.
- Broad beach fern has a provincial rank of vulnerable (S3) and was observed in two Eastern White Cedar – Black Spruce- Species Rich: organic soil (ES13r) ecosites. These small ecosites are located in the northwest part of the Bradshaw Property away from any current proposed mining development.
- There is potential for 13 provincially listed wildlife species, one federally listed wildlife species, and one species listed as provincially vulnerable known to occur in the study area.
- Based on the information available, olive-sided flycatcher and rusty blackbird are assessed as having a moderate-high to high potential to occur within the site study area. Of these species, only the olive-sided flycatcher was documented in the study area.
- A total of 55 species of birds were recorded during the breeding bird surveys.
- The study has provided Gowest with an understanding of the terrestrial flora and fauna within the site and can be used as a basis for more effective scoping of future baseline studies and ongoing monitoring studies.

20.1.7 Aquatic Ecology

The aquatic ecology work consisted of collecting representative information for portions of the West Buskegau River system located upstream, within and downstream of the Project site. The following tasks were completed:

- Fish Habitat Mapping and Supporting Data Collection at five locations within an unnamed tributary of the West Buskegau River that flows along the western boundary of the property (2011) and at sampling locations SW2 (potential discharge location) and SW4 (downstream of the potential discharge location) on the West Buskegau River (2014).
- Fish Community Assessment at five locations within the western tributary of the West Buskegau River (2011) and at SW2 and SW4 (2014).

- Benthic invertebrate community and algal community sampling at SW2 and SW4 (2014).

The results of the aquatic ecology field program indicate that the unnamed tributary to the West Buskegau River supports small fish species while the main channel of the West Buskegau River also supports sport fish (e.g., walleye and northern pike). Neither of the SW-2 or SW 4 locations assessed contained potential spawning habitats for these sport fish species, although only two discrete sections of the West Buskegau River were assessed during baseline studies. Catch data indicates that the West Buskegau River Tributary and the main River are cool water streams.

Benthic invertebrate community and algal community sampling on the West Buskegau River provide background information required for permitting and for comparative purposes in future monitoring studies following approval of a final discharge location. Results indicate location SW2 supports a benthic invertebrate community of moderate density (>5000/m²) and moderate to low richness, with 10 benthic invertebrate families and one phylum represented, total mean richness was 18. Location SW4 supports a low density (<5000/m²) benthic invertebrate community and moderate richness represented by 18 families, one phylum and one subclass of invertebrates. Mean total richness was 23. Based on the 2014 assessment at SW2 and SW4, both benthic invertebrate communities appear to be characteristic of warm water habitats of low to moderate productivity. The benthic invertebrate communities at SW2 and SW4 are similar and are likely representative of much of the West Buskegau River and other local and regional watercourses of similar geomorphology. Highly sensitive indicator species typically associated with sensitive (e.g., cold water habitats) are not abundant at either location (Seyler, 2014).

20.1.8 Archaeology

Golder Associates Ltd. conducted a Stage 1 archaeological background study for the Bradshaw Project area in late 2014. This assessment was conducted as part of the environmental baseline studies for Advanced Exploration. The objective of the Stage 1 assessment was to compile all available information about the known and potential cultural heritage resources within the study area and to provide specific direction for the protection, management and/or recovery of these resources, consistent with Ministry of Tourism, Culture and Sport guidelines. The Stage 1 archaeological assessment resulted in the determination that archaeological potential exists within the Project area 150 m either side of the West Buskegau River. It is recommended that this area of archaeological potential be subject to an archaeological survey prior to any ground disturbance activities in that area. No further assessment is recommended for areas not found to exhibit archaeological potential, away from the West Buskegau River (Davidson, 2014).

Golder Associates Ltd. Conducted a Stage 2 archaeological assessment for the Bradshaw Project area (Davidson, 2015). The objectives of the Stage 2 archaeological assessment were to provide an overview of archaeological resources on the property, and to determine whether any of the resources might be artifacts and archaeological sites with cultural heritage value or interest and to provide specific direction for the protection, management and/or recovery of these resources. Areas recommended for Stage 2 assessment were surveyed by shovel test pitting. The Stage 2 test pit survey of the areas of archaeological potential within the Bradshaw Project area did not result in the identification of any archaeological remains. No further archaeological assessment is recommended for the Bradshaw Project area.

20.2 Project Permitting

Several different types of permits, authorizations and/or licenses are required to develop a typical mining project in Ontario. To obtain the necessary permits, a proponent is required to submit applications, with the relevant technical supporting documentation, to a number of different regulatory agencies. On June 25, the Minstry of Northern Development and Mines (MNDM) accepted the Gowest Closure Plan for Advanced Exploration at the Bradshaw. Subsequently Gowest have filed all the necessary permits for an Advanced Exploration underground bulk sample program with the various government agencies. :

Ontario Ministry of the Environment and Climate Change

- Environmental Compliance Approval (ECA) - Industrial Sewage Works, and Air and Noise;
- Permits to Take Water (PTTW) – groundwater and surface water takings;
- Hazardous Waste Information Network Registration;

Ontario Ministry of Northern Development and Mines

- Closure Plan;

Ontario Ministry of Natural Resources and Forestry

- Application for a Work Permit that requires Lakes and Rivers Improvement Act Approval;
- Application for Easement;
- Forest Resource License;

Fisheries and Oceans Canada determined that a Fisheries Act authorization is not required given that serious harm to fish can be avoided by following standard measures. In addition, Gowest or their contractor, will obtained a permit from Natural Resources Canada for explosive use and storage.

Gowest met with Provincial Ministry officials on June 14, 2010 to share the draft Project Definition and provide regulators with an opportunity to comment. On February 27, 2014 Gowest met with MNDM, Ministry of the Environment and Climate Change (MOECC) and Ministry of Natural Resources and Forestry (MNRF) to review their revised Project Definition. On April 25, 2014, Gowest met with MOECC to discuss the path moving forward for the ECA application for Industrial Sewage Works. Gowest also conducted a field visit to the Project site with representatives from MOECC and MNDM on October 29, 2014. On November 26, 2014 Gowest met with MOECC to solicit feedback on the proposed content of the Preliminary Water Management Plan, submitted as a step of the application for the ECA for Industrial Sewage Works. A meeting was held on December 19, 2014 with MNRF and MNDM. Discussion at the meeting included land use, the Forest Resource License, the Lakes and Rivers Improvement Act application, and other permitting considerations.

20.3 Community Consultation

20.3.1 General

Gowest continues to engage and consult with the local communities, including First Nations and the Métis community. Through meetings, site tours and regular communications, Gowest strives to ensure engagement with all members of the local communities. The following sections have been summarized from the Closure Plan for Advanced Exploration – Bradshaw Project by Golder Associates (2015).

20.3.2 First Nations Consultation

The MNDM identified the following Aboriginal groups to be consulted regarding the mine development of the project:

- Matachewan First Nation
- Mattagami First Nation
- Taykwa Tagamou Nation
- Metis Nation of Ontario - Timmins Metis Council
- The Metis Nation of Ontario - Northern Lights Metis Council

Gowest has been engaging and consulting with the communities identified above and Tribal Councils that expressed an interest since 2010.

Gowest has a strong relationship with the Wabun Tribal Council that coordinates mineral development communication for both the Matachewan and Mattagami First Nations.   As such, Gowest has engaged Wabun Tribal Council to assist with coordinating the consultation activities with Matachewan and Mattagami. These consultation activities have included:

- Memorandum of understanding between Matachewan and Mattagami First Nations and Gowest for mineral exploration signed in October 2011.
- Meetings, letters, and e-mails directly with Matachewan and Mattagami First Nation leadership as well as community members since 2010.
- Field visit to the Project site and an exploration overview of the Project in November 2012 and August 2014.
- Distributing the Draft Advanced Exploration Closure Plan to Matachewan and Mattagami First Nations in September 2014.
- Open houses on October 6, 2014 at the Matachewan First Nation and on October 7, 2014 at the Mattagami First Nation to provide their communities with information about the Project and Closure Plan.   Invitation from Gowest in November, 2014 to attend Open House held on December 1, 2014 in Timmins.
- Copy of the archaeology report Stage 1 was provided in December, 2014.
- Invitation from Gowest on June 16, 2015 to participate in the archaeology study Stage 2 to take place on June 23, 2015.

- Response from Matachewan and Mattagami First Nations to Gowest's Closure Plan in December 2014 indicating no issues with the Closure Plan but that the Project would affect their traditional territory.
- Discussions with Matachewan and Mattagami First Nations related to development of an Impact Benefit Agreement (IBA).

Key issues identified by Matachewan First Nation include project footprint, schedule and reporting compliance. Key issues identified by Mattagami First Nation include the mine life, total tonnage, definition of bulk sample, and treatment for acid generating material from the waste rock. Also, there were questions about the environmental and archaeological studies done to date. Gowest will continue to work with Matachewan First Nation and Mattagami First Nation to address the potential effects on their traditional territory and work towards an IBA

Advanced Exploration consultation activities with Taykwa Tagamou Nation started in 2014 when MNDM identified that their Aboriginal and Treaty rights could be impacted by the Project; however Gowest had made initial contact with Taykwa Tagamou Nation in 2010. Significant efforts have been made to consult with Taykwa Tagamou Nation and key consultation activities have included:

- Initial early Gowest letter in 2010 about exploration activities on the project.

- Letter from Gowest in July 2014 advising the Project is currently moving into the advanced exploration stage followed by a meeting in August, 2014 at Gowest's Toronto office.
- Distribution of the Draft Closure Plan to Taykwa Tagamou Nation in September 2014.
- Field visit to the project site and a meeting at the Gowest Timmins office in October, 2014.
- Meeting in October, 2014 reiterating the desire to enter into a memorandum of understanding.
- Meeting held in November, 2014 to discuss the exploration agreement and review of the closure plan.
- Invitation from Gowest in November, 2014 to attend Open House held on December 1, 2014 in Timmins. Copy of the archaeology report Stage 1 was provided in December, 2014.
- A Taykwa Tagamou Nation report with comments to the draft Closure Plan was received March 25, 2015.
- Gowest responded to Taykwa Tagamou Nation on their draft Closure Plan comments on April 17, 2015.
- Invitation from Gowest on June 16, 2015 to participate in the archaeology study Stage 2 to take place on June 23, 2015.

Discussions with Taykwa Tagamou Nation related to the development of a memorandum of understanding, and potential effect of the Project on Aboriginal rights and interests are ongoing.

Gowest first initiated contact with the Metis Nation of Ontario (MNO) in 2010. Consultation activities increased in 2014 when MNDM identified the MNO Timmins Métis Council and Northern Lights Métis Council Aboriginal and Treaty rights could be impacted by Gowest's proposed the advanced exploration plan. Key consultation activities have included:

- Meetings, letters, and e-mails that have dealt with introducing and updating the Project, MNO's preferred consultation approach and MNO's rights within the Project area;
- Introductory letter from Gowest in July 2014 advising the Project is currently moving into advanced exploration stage followed by a meeting in September 2014 at Cedar Meadows in Timmins.
- Field visit to the project site as part of the Archaeology Study (Stage 1) in October 2014. Copy of the archaeology report Stage 1 was provided in December, 2014.
- Distribution of the Draft Closure Plan to MNO in September 2014.
- Invitation from Gowest in November, 2014 to attend Open House being held on December 1, 2014 in Timmins.

- Teleconference in January, 2015 where MNO indicated support of the Closure Plan.
- Inviation from Gowest on June 16, 2015 to participate in the Archaeology Study Stage 2 to take place on June 23, 2015.

Since distribution of the Draft Closure Plan in September 2014, Gowest has continued to follow up with MNO to solicit feedback and comments on the Draft Closure Plan. MNO has indicated support to the Closure Plan but a letter indicating this has not been received to date. To date, MNO leadership has identified Aboriginal rights within the Project area, however, they have not specified impacts on their Aboriginal or Treaty rights.

20.3.3 Public Consultation

Gowest is an active member of the local community with an exploration office in Timmins, Ontario that offers local residents an easily accessible location to learn about Gowest and the Project. In addition to the Aboriginal consultation activities described above, Gowest has carried out stakeholder consultation activities including communications and meetings with the City of Timmins, ongoing consultation with relevant Ontario government regulatory agencies.

Gowest held an Open House in Timmins on December 1, 2014 at the McIntyre Community Arena Auditorium to provide information about the Bradshaw Project and an Advanced Exploration Closure Plan. The open house provided the public with geological and engineering details about the Project, the environmental permitting and approvals process, details on the environmental baseline studies being completed in support of the Bradshaw Gold Project, and information about the Closure Plan. Over 100 people attended the Open house with 42 people signing into the Open House and five people completing comment forms. Gowest received positive feedback on the environmental studies and Closure Plan of the Project. Some issues identified either verbally or through written comments included: potential effect of mining and transportation on Ice Chest Lake cottagers (Gowest stated road access to Project would not be on the Ice Chest Lake road), the use of overburden during closure activities (Gowest stated overburden would be utilized to aid vegetation of any waste rock piles during closure activities) and storage of diesel fuel for power generators if used (Gowest stated government approved storage containers would be utilized during operations and an emergency spill action plan would be in place).

21.0 CAPITAL AND OPERATING COSTS

All costs have been estimated in Q1 2015 Canadian dollars and are to a prefeasibility economic assessment level of accuracy of -15% to +25% percent. Costs have been calculated from first principles, or based on recent experience for similar installations at other project sites. Equipment and material purchase prices are based on recent and escalated vendor quotes. All underground mine development and operations work will be completed by a contractor labour force. All surface capital construction will be completed by contractors. The facilities will be managed by the owner over the life of the mine.

21.1 Capital Costs

Capital costs are defined as all project and sustaining costs incurred during the pre-production period and up to the end of Year 8. Initial project capital will occur during the pre-production period and Year 1, and will comprise the cost of surface installations. With the addition of closure costs in year 8, the total capital costs 1ill $16.52 million. Sustaining capital pertains to all underground ramp, level and infrastructure costs up to Year 4.

The capital costs include closure costs, surface road upgrades, surface infrastructure construction, surface power line, underground infrastructure development and underground infrastructure construction. A contingency of 18% has been applied to the project capital during the pre-production period and Year 1. No contingency has been added to sustaining capital. The capital costs have been summarized in Table 21.1. Additional capital cost details are included in the Appendices.

Table 21.1: Bradshaw Budgeted Capital Costs

Description	Est. Cost (millions)
Bradshaw Surface Road Upgrades	$0.250
Bradshaw Surface Infrastructure Construction	$6.314
Bradshaw Surface Ventilation Infrastructure Construction	$1.027
Underground Infrastructure Development (sustaining capital)	$37.225
Underground Infrastructure Construction	$0.600
Surface Power Line Grid Costs	$6.500
Closure Costs	$0.350
Contingency (On project capital only – First two years) - 18%	$1.474
Gowest – Bradshaw Capital incl. Contingency – Prefeasibility Study	$53.740

21.1.1 Ramp Development

Ramp development quantities have been based on Stantec's mine design, including a main ramp accessing 16 individual levels at 30 m intervals. The ramp will begin at surface and end at 495 level, 495 metres below surface. Each active production level in the mine will be accessed by the ramp and the ventilation system.

The estimated unit cost for ramp development has been developed from first principles using labour rates (including wages, overtime, bonus, and allowances), mobile equipment rental and operating costs (fuel and lubricants, spare parts, tires, buckets), consumable materials, services materials (piping, ventilation ducting, electrical cables), and calculated productivities. The ramp development unit costs do not include haulage of the waste rock (identified separately). The ramp development unit costs are based on an average 5 metre per day advance rate. This advance rate is consistent with Stantec experience with many projects. The 5.0 metre x 5.5 metre ramp development unit cost is summarized in Table 21.2.

Table 21.2: Ramp Development Unit Cost

Description	Metres	$/M	Total $ (millions)
Main Ramp (5 m x 5.5 m) to 495 Level	3,924	$5,601	$21.978

21.1.2 Waste Infrastructure and Ore Development

Waste infrastructure development quantities have been based on Stantec's mine design. Waste development will include lateral waste access to the mineralized zones, FAR access, and underground facility development. The underground facility development will include:

- Ore, rock and supplies storages, as well as dewatering sumps and electrical substation for every level
- Lunchroom/refuge station, as well as a explosives and accessories magazine on every fifth level
- An intermediate and main dewatering sump for the entire mine envisaged in this study

The lateral waste development unit costs do not include. The waste development unit costs which have been based on an average 7 metre per day advance rate (multiple heading). The 4 metre x 4 metre waste infrastructure development unit rate is summarized in Table 21.3.

Table 21.3: Waste Infrastructure Development Unit Cost

Description	Metres	$/M	Total $ (millions)
Level Access Drift and Services (5 x 5.5)	1,322	3,513	$4.644
Level Waste X-Cut (4 x 4)	1,018	3,016	$3.070
Level FAR Access (4 x 4)	1,301	2,447	$3.182

21.1.3 Raise Development

Raise development quantities have been based on the underground longitudinal section. All Return Air Raise (RAR) development will include an escapeway with platforms and ladders (included in raise unit rate). RAR excavation and escapeway equipping will be completed by contractor alimak crews.

The estimated unit cost for raise development has been developed from first principles using labour rates (including wages, overtime, bonus, allowances and overhead), mobile equipment operating costs (fuel and lubricants, spare parts, tires, buckets), alimak machine rental, consumable materials, services materials (piping, ventilation ducting, electrical cables), and calculated productivities. The unit costs do not include haulage of the waste rock (identified separately) or primary mucking of waste rock (this will be completed by the owner). The raise development unit costs are based on an average 1.0 metre per day advance rate (includes setup and teardown) and accounts for pilot raising and second pass raise slashing. This advance rate is consistent with Stantec experience with other studies. The 3 metre x 3 metre RAR alimak development unit rate is summarized in Table 21.4.

Table 21.4: Alimak Raises Development Unit Cost

Description	Metres	$/M	Total $ (millions)
Surface to 495 Level (3 m x 3 m)	525	8,288	$4.351

21.1.4 Equipment Purchases

No capital equipment was purchased for this project. All surface buildings, shops, fans, compressors and pumps are rental units and are included in the project contractor indirect operating costs. All underground mining equipment, fans and pumps are rental/leased units and are included in the mining operating costs and the contractor indirect operating costs.

21.1.5 Surface Infrastructure

Surface infrastructure construction will be completed by contractor personnel. Surface infrastructure estimates are developed from first principles and quotes from suppliers. The surface infrastructure costs are included in the Table 21.5.

Table 21.5: Surface Infrastructure Capital Costs

Description	Est. Cost (millions)
Pre-Production Period Site Road Access (2000 m)	$0.536
Ore Pad Construction	$0.120
Surface water settling sump (250 m x 300 m)	$0.359
Generator Installation	$0.215
Office/washroom trailers	$0.149
Mtce Building (12.5 m x 25 m)	$0.177
Air Compressor Installation	$0.046
Pump House	$0.070
Total Pre-Production Period	**$1.672**
Production Period	
Aran Plant Installation	$0.925
Backfill Pipeline	$0.206
Concrete mixing tanks	$0.500
Safety/Mine Rescue Equipment	$0.650
Material Handling System	$0.500
Waste Pad	$0.120
Mix Development Pad	$0.120
2nd Surface water settling Sump (250 m x 300 m)	$1.077
2nd Generator installation	$0.215
Fuel Farm	$0.065
2nd Office/washroom Trailers	$0.149
Storage Sprung Building (40 m x 50 m)	$0.069
2nd Air Compressor Installation	$0.046
Total Production Period	**$4.642**
Main Surface Road Upgrade	$0.250
Power Line Grid Installation	$6.500
Gowest – Bradshaw Surface Capital	**$13.064**

21.1.6 Underground Infrastructure and Construction

Infrastructure construction will include underground facilities required to support production. Infrastructure cost estimates have been developed from first principles and include direct labour, materials, and equipment operating. All construction will be completed by contractor crews. The underground infrastructure costs are included in the Table 21.6.

Table 21.6: Underground Infrastructure Capital Costs

Description	Est. Cost (millions)
Ventilation Infrastructure	
Pre-Production Period	
Heating Plant	$0.060
1st set of Ancillary fans	$0.095
2nd set of Ancillary fans	$0.020
Production Period	
Ancillary Fans	$0.050
Main Surface fans	$0.071
Elect Controls for Main Surface fans	$0.030
2nd Heating Plant	$0.060
Vent Bulkheads (16 levels @ $40,000/ea)	$0.640
Total Ventilation Infrastructure	$1.026
Underground Infrastructure Lunchroom/Refuge Station (Total of 4)	$0.300
Powder/Fuse Magazine (Total of 4)	$0.300
Total Underground Infrastructure	$0.600
Gowest – Bradshaw Underground Capital	$1.626

21.1.7 Milling Capital

Milling capital cost has not been included for this project since the ore will be milled through a custom milling arrangement budgeted at 35.00/tonne. The sorted ore tonnes from the stope and mixed development will be trucked to the Kidd mill, located 20 km from the mine site. A trucking cost of $5.00/tonne has been applied to move the sorted material to the Kidd mill site.

21.2 Operating Costs

Operating costs include all mine and mill costs, other than sustaining capital costs. The life of mine operating costs will include both direct and indirect costs. The direct operating costs are summarized in Table 21.7. Additional operating cost details are included in the Appendices.

Table 21.7: Budgeted Operating Cost Summary

Description	Estimated Costs
Operating	
Operating Development Cost	$55,304,454
Owners Cost	$31,278,979
Indirect Cost	$51,304,841
Mining Operating Cost	$59,632,904
Milling Operating Cost (Gowest)	$43,647,805
Definition Diamond Drilling	$1,500,000
Ore Transportation to the Mill (Gowest)	$6,235,401
Smelting/Refining Costs (10% of Recovered Gold Production)	$42,619,296
Total Bradshaw Operating Costs (no capital included)	$291,523,680

The "all-in" estimated average life of mine direct operating cost (not including capital costs) per tonne is $163.10/tonne for the underground operation.

21.2.1 Direct Operating Costs

The underground direct operating costs include waste development to access stopes, sill development and stope production activities. All costs not directly related to mine construction, development, and production activities, have been included in the owner and contractor indirect operating costs.

21.2.2 Operating Development

Underground operating development quantities have been based on Stantec's mine design and include sill development along the mineralized zones as well as operating waste development to access the mineralization.

The operating development unit costs have been based on a 6.0 metre per day advance rate. This advance rate is consistent with Stantec experience. The 4 metre x 4 metre sill development unit rate is summarized in Table 21.8.

Table 21.8: Sill Development Unit Cost Summary

Description	Metres	$/M	Total $ (millions)
Level Waste Development	8,732	2982.1	$26.039
Level Ore Development	10,799	2710.0	$29.265
Total	19,531	2831.6	$55.304

21.2.3 Mining Operating Cost

The direct costs related to longhole stoping include the labour, consumable material/supplies, and equipment operating and maintenance associated with:

- Drilling, loading, and blasting longholes (including drop raises).
- Mucking from the stope with a 6yd LHD and tramming to a remuck.
- Truck Haulage ore to surface and backfill haulage to the level remucks.
- Backfilling with a mixture of consolidated and unconsolidated waste rock.
- Operation of a backfill plant.
- Material handling on surface, crushing and ore sorter.

The direct costs for longhole stoping are summarized in Table 21.9.

Table 21.9: Mine Operating Costs and Unit Costs

Description	Total $ (millions)	Tonnes	$/tonne
Production Drilling and Blasting	$18.870	1,787,295	$10.56
LHD Mucking	$8.441	1,787,295	$4.72
Underground Truck Haulage	$13.031	1,787,295	$7.29
Backfilling Concrete Material	$1.823	1,787,295	$1.02
Backfill-Back Haulage	$3.258	1,787,295	$1.82
Material Handling, Crusher and Sorter	$12.365	1,787,295	$6.92
Aran Batch Plant Operation	$1.844	1,787,295	$1.03
Total	$59.632	1,787,295	$33.36

21.2.4 Contractor Indirect Operating Costs

The Bradshaw Deposit will be mined with contractor labour. All costs not directly related to mine development, construction, or production activities have been included in the indirect operating costs. Indirect operating costs have been developed from first principles and recent experience. The indirect operating costs consist of the following:

- Support labour (built up labour rates including wages, overtime, bonus, and allowances)
- Indirect hourly labour

- Contractor Staff operating
- Contractor Engineering
- Administration
- Mobile equipment operating costs (fuel and lubricants, spare parts, tires, buckets)
- Supplies and services (i.e. dry operating, road maintenance, offices supplies, etc.)
- Maintenance and supplies for equipment (i.e. fans, heaters, pumps, compressor, etc.)

Indirect operating costs and labour are summarized in Table 21.10 and Table 21.11.

Table 21.10: Indirect Operating Costs

Description	Total $ (millions)	Tonnes	$/tonne
Indirect Staff	$32.705	1,787,295	$18.30
Indirect Building/equipment Operating and Rental Costs	$18.600	1,787,295	$10.41
Total	$51.305	1,787,295	$28.71

Table 21.11: Indirect Contractor Labour Costs

Position	Quantity	Total Annual Cost ($)	LOM Cost ($)
Contractor Indirect Labour			
Mine Superintendent	1	$261,798	$2,027,757
Underground Supervisor	2	$740,074	$5,732,237
Safety Supervisor	1	$201,445	$1,560,288
Clerk	1	$127,193	$985,171
Engineering Surveyor	2	$402,799	$3,119,875
Mobile Mechanic Lead	2	$612,470	$4,743,881
Mobile Mechanic	2	$442,453	$3,427,016
Electrical Lead	2	$612,470	$4,743,881
Electrician	2	$442,453	$3,427,016
Dryman	2	$356,313	$2,937,888
Total	17	$4,199,468	$32,705,012

21.2.5 Owner's Operating Costs

All costs not directly related to mine development, construction, or production activities have been included in the owner's operating costs. Owner's operating costs have been developed from first principles and experience. The owner's operating costs consist of the following:

- Owner's staff
- Engineering and geology
- Mine air heating
- Generator and electrical power (5MVA capacity)

Electrical costs were calculated at $0.08 kwhr. Owner's operating costs and staff numbers are summarized in Table 21.12 and Table 21.13.

Table 21.12: Owner's Operating Costs

Description	Total $ (millions)	Tonnes	$/tonne
Owner's Staff	$10.678	1,787,295	$5.97
Surface Building Heating	$1.811	1,787,295	$1.01
Ventilation Underground Heating	$3.574	1,787,295	$2.00
Generator and Electrical Power	$15.216	1,787,295	$8.51
Total	$31.279	1,787,295	$17.49

Table 21.13: Owner's Staff

Position	Quantity	Total Annual Cost ($)	LOM Cost ($)
Bradshaw			
Mine Manager	1	$288,350	$1,802,174
Mine Engineer Planner	1	$237,250	$1,482,802
Mine Geologist and Sampler	2	$474,500	$2,965,603
Security/First Aid	2	$328,500	$2,053,110
Total	6	$1,328,600	$10,678,139

21.2.6 Milling Operating

Milling costs have been provided by Gowest. The process plant operating costs were calculated to be $35.00/tonne milled. The material will be processed at the Kidd milling facility, located approximately 20 kilometres away.

21.2.7 Refining and Metallurgical Costs

The refining and metallurgical costs were provided by Gowest. 10% of the total gold ounces have been removed from the revenue stream to pay for this cost. The cost per tonne equivalent is $42.619 million or $23.83/tonne.

21.2.8 Ore Transportation Costs to the Mill

The ore transportation cost to the mill was provided by Gowest. A $5.00/tonne has been applied to all sorted tonnes being transported from the mine site to the mill located at the Kidd Operations. Trucking sorted material will range from a low of 110 to a maximum of 620 tpd.

Total ore transportation cost of $6,235,401 has been applied to this project.

21.2.9 Diamond Drilling

There are 30,000 metres or $1,500,000 of definition drilling that have been included for the Bradshaw project. About 2,500 metres of definition drilling will begin once the ramp and the first two levels are established for the bulk sample. The definition drilling will be increased to 5,000 metres per year as the mine ramp and additional levels are developed. Drilling will occur throughout the mine life with the exception of the final year.

22.0 ECONOMIC ANALYSIS

22.1 Basis of Evaluation

Stantec has assessed the project based on a discounted cash flow model. The cash flow from the production plan (base case) has been forecast and the resulting Net Present Value (NPV) calculated. The sensitivity of the NPV to key changes in the base case assumptions was assessed.

22.2 External Factors

22.2.1 Exchange Rate

All costs, revenues, etc., in this study are expressed in Canadian dollars. Costs are based on constant, 2015 Canadian dollars with no provision for escalation or inflation.

An exchange rate of C$1.00 = US$0.80 was used.

22.2.2 Weighted Average Cost of Capital (Discount Rate)

A discount rate of 5% has been used in the NPV calculations based on current interest rates and the rates used in other 43-101 compliant reports.

22.2.3 Metal Prices

A base case gold price of US$1,200/oz. (C$1,500/oz.) was used throughout the life of the project.

22.2.4 Taxes

The Bradshaw Project is subject to mining rights payments, taxes, and municipal, provincial and federal income taxes. The project is eligible for a tax credit for exploration activities. Stantec calculated the post NPV prior to taxation. Mining rights payments, taxes, and tax credits were evaluated in this study by Gowest.

22.2.5 Royalties

The following royalty is associated with the Bradshaw Deposit:

- Royalty of 1.5% to New Texmont or payout of $3.5 million at full production. Gowest has elected to payout the $3.5 million at Year 3 of the life of mine plan and this payout is reflected in the financial analysis.
- Royalty of 1% to Gold Royalties Corp starting at Year 3 of the life of mine plan.

22.2.6 Selling Expense

The following allowances have been included in the financial analysis and were provided by Gowest:

- $35/tonne for milling charges
- $5/tonne for ore transportion to the mill and refining facilities
- 10% of gold production or $24/tonne of ore for the for gold refining charges

22.3 Internal Factors

22.3.1 Production Schedule

The life of mine production profile is listed in Table 22.1.

Table 22.1: Mine Production Profile (LOM)

	Production Profile LOM									Total
	0	1	2	3	4	5	6	7	8	
Dev Tonnes (t)	11,334	31,630	37,798	53,264	50,332	51,497				235,855
Dev Grade (g/t Au)	5.52	3.63	4.30	4.36	3.82	4.35				4.19
Stope Tonnes (t)	39,562	49,695	144,019	188,695	25,0448	197,468	302,046	302,550	76,957	1,551,440
Stope Grade (g/t Au)	5.78	5.49	4.56	5.31	4.64	4.48	4.95	4.88	5.89	4.92
Stope and Dev Prod (t)	50,896	81,325	181,817	241,959	300,780	248,965	302,046	302,550	76,957	1,787,295
Stope and Dev Grade (g/t Au)	5.72	4.77	4.50	5.10	4.50	4.45	4.95	4.88	5.89	4.82
Incremental Dev (t)	27,444	11,2871	123,841	133,920	148,877	119,700	0	0	0	666,253
Incremental Dev Grade (g/t Au)	1.31	1.31	1.31	1.31	1.31	1.31	1.31	0	0	1.31
Gold from Dev and Stope Production (Au Oz)	9,366	12,467	26,327	39,666	43,520	35,648	48,036	47,492	14,579	277,101
Gold from Incremental Dev (Au Oz)	1,152	4,736	5,197	5,619	6,230	5,023	0	0	0	27,957
Total Au Oz	10,518	17,203	31,523	45,286	49,751	40,671	48,036	47,492	14,579	305,058
Avrg Stope and Dev Production (tpd))	139	223	498	663	824	682	828	829	855	675

22.3.2 Operating Costs

The average life of mine operating costs are listed in Table 22.2.

Table 22.2: Average Life of Mine Operating Costs

Description	Quantity	UoM	Unit Cost	Estimated Cost (millions)
Level Development Costs	19,531	Meter	$2,832/m	$55.304
Owners Costs	3,006	Day	$10,404/day	$31.279
Indirect Costs	3,006	Day	$17,065/day	$51.305
Mining Operating Costs	1,787,295	Tonne	$29.90/tonne	$59.633
Mill Operating Costs	1,247,080	Tonne	$35/tonne	$43.648
Refining Operating Costs	1,787,295	Tonne	10% of Au production ($23.75/tonne equivalent)	$42.619
Definition Diamond Drilling Costs	1,787,295	Tonne	$0.84/tonne	$1.500
Ore Transportation Costs	1,247,080	Tonne	$5.00/tonne	$6.235
			Total Operating Costs	$291.523
			Underground Tonnes (mill feed)	1,787,295
			Total Operating Cost per Tonne	$163.10

22.3.3 Capital Costs

The LOM capital costs are summarized in Table 22.3.

Table 22.3: Life of Mine Capital Costs

Description	Estimated Cost (millions)
Bradshaw Surface Road Upgrades	$0.250
Bradshaw Surface Infrastructure Construction	$6.314
Bradshaw Surface Ventilation Infrastructure Construction	$1.027
Underground Infrastructure Development (sustaining capital)	$37.225
Underground Infrastructure Construction	$0.600
Surface Power Line Grid Costs	$6.500
Closure Costs	$0.350
Contingency (On project capital only – First two years) - 18%	$1.474
Gowest – Bradshaw Capital incl. Contingency – Prefeasibility Study	$53.740

22.3.4 Financial Evaluation

The Bradshaw evaluation indicates an IRR of 32%, and using a discount rate of 5.0%, an NPV of $49.75 million before taxes, and returning a NPV of $36,495,879.00 and an IRR of 27.3 % after tax.

Table 22.4 provides a summary of the project economics and associated parameters.

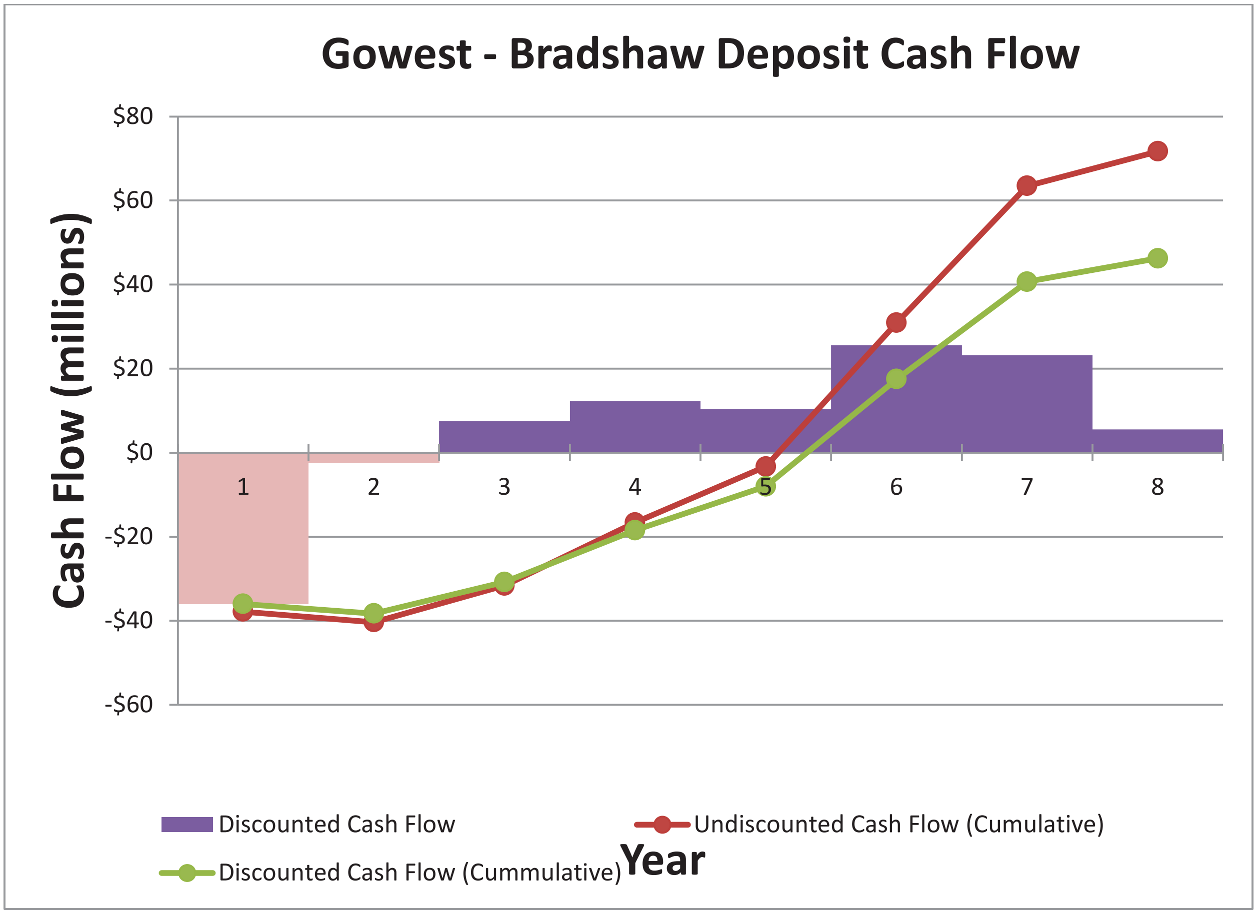
Table 22.4: Bradshaw Project Economics (before and after taxes)

Item	Value
Forecast Gold Price (C$)	$1,500
Mine Tonnes	1,787,295
Exchange Rate	C$1.00 = US$0.80
Mined Grade	4.82 g/t
Mill, refining and Ore Sorting Recoveries	93%
Mine Recovered Ounces (including incremental development mineralized material)	305,058
Produced Ounces from Mill	284,129
Gross Revenue to Operations (C$)	$419,205,783
Operating Costs (C$)	$291,523,680
Capital Costs (C$)	$53,740,688
Net Cash Flow (C$)	$73,941,414
Net NPV (5%) – Before Taxes (C$)	$49,750,509
IRR (%) – Before Taxes	32.0%
Net NPV (5%) – After Tax (C$)	$36,495,879
IRR(%) – After Tax	27.3%

22.3.5 Life of Mine Cash Flows

The LOM cash flow summary is illustrated in Figure 22.1.

Figure 22.1: Bradshaw Project – Cumulative Cash Flow Graph (Pre-tax)



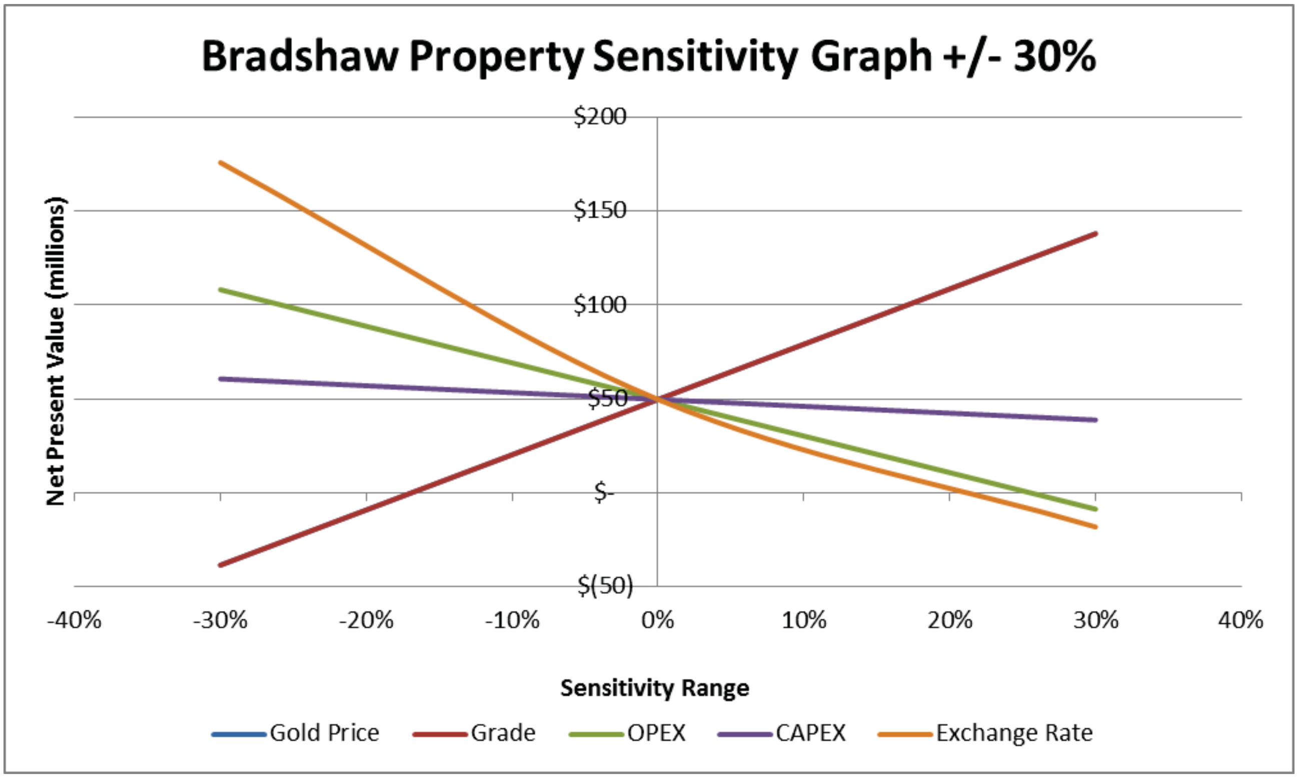
22.4 Sensitivities

The capital and operating costs, gold price, average mine grade and exchange rates were varied to observe the influence of these parameters on project profitability. Each parameter was changed over a range of ±10-30% from the base case assumptions.

Figure 22.2, illustrate the influence of changes in these parameters on the project NPV. Increases and decreases in the price of gold, gold grade and exchange rate have the greatest effect on the NPV and IRR. Altering the opex and capex figures have a lesser impact on the project economics.

Note: The gold price and the gold grade variables are the same figures and plot one on top of the other. Thus only the gold grade is plotted but the gold price line is underneath it.

Figure 22.2: Change in Project NPV (5%) versus ± 10% Incremental Change in Variables (Pre-tax)



**Note that the curve for gold price and grade overlap each other and appear as one line.*

The sensitivity of the NPV and IRR to the gold price is listed in Table 22.5.

Table 22.5: Gold Price Sensitivity

Gold Price $/oz (C$)	NPV	IRR
$1,050	($38,614,471)	-15.0%
$1,200	($9,159,478)	0.0%
$1,350	$20,295,516	16.0%
$1,500	$49,750,509	32.0%
$1,650	$79,205,503	50.0%
$1,800	$108,660,496	71.0%
$1,950	$138,115,490	96.0%

23.0 ADJACENT PROPERTIES

The reader is cautioned that the information in this section is not necessarily indicative of the mineralization on the property that is the subject of this report.

23.1 SGX Resources Inc. and San Gold Corporation's North Timmins Property

SGX Resources Inc. and San Gold Corporation's each own 50% of the Timmins North Property. It lies immediately south of and is contiguous to Gowest's Frankfield Property.

The Tully gold (formerly known as the Nickel Offsets Deposit) deposit lies within the North Timmins Property and was discovered in 1968 by McIntyre Mines Limited while testing conductive horizons for base metal mineralization potential. The deposit is located approximately 2 km to the south of the Bradshaw Deposit and is associated with conductive mineralization, mainly graphite and disseminated sulphides within a shear zone adjacent to ultramafic volcanic rocks. The shear zone trends at N080°E, dips steeply north, and is interpreted to be have a strike length of 1.6 km on the property.

This mineralized structure appears to be localized in tuffaceous mafic volcanic rocks (shear zone) adjacent to the contact between Porcupine Assemblage sedimentary rocks to the north and Kidd-Munro Assemblage mafic / ultramafic volcanic rocks to the south. Gold occurs in the native form along with subordinate amounts of disseminated pyrite and arsenopyrite within, and marginal to, the quartz carbonate veins. Historic diamond drilling indicated that three vein systems occur over a 25-50 m true width for a strike length of more than 450 m, and to a depth of more than 335 m. Both hanging wall and footwall vein systems are parallel to shearing foliation and a middle vein system is approximately perpendicular to these bounding vein systems.

A new geologic interpretation of the Tully deposit was developed within the last two years by SGX Resources geologists using drill sections obtained by previous operators to show that gold mineralization at Tully occurs in a repeatable pattern of quartz stockwork veins hosted by a mafic tuff unit. The drill-indicated strike length at Tully is now more than 1,000 metres and has been drilled to depths of more than 600 metres, remaining open along strike and to depth. SGX Resources website is (http://www.sgxresources.com).

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information to add.

25.0 INTERPRETATION AND CONCLUSIONS

The prefeasibility study evaluated the indicated mineral resources to determine the tonnage delivered to the mill. The study results indicate that ore reserves of 1,787,295 tonnes grading 4.82 g/t (277,101 ounces Au) will be mined over a nominal eight year mine life. In addition, there are 666,253 tonnes @1.31 g/t of mineralized material where 27,957 ounces of Au can be economically extracted by running the incremental material through the sorter. This amounts to 305,508 ounces of gold available to mine with a life of mine production summary shown in Table 25.1.

Table 25.1: Bradshaw Estimated Life of Mine Production Profile

	Production Profile LOM									Total
	0	1	2	3	4	5	6	7	8	
Dev Tonnes (t)	11334	31630	37798	53264	50332	51497				235855
Dev Grade (g/t)	5.52	3.63	4.30	4.36	3.82	4.35				4.19
Stope Tonnes (t)	39562	49695	144019	188695	250448	197468	302046	302550	76957	1551440
Stope Grade (g/t)	5.78	5.49	4.56	5.31	4.64	4.48	4.95	4.88	5.89	4.92
Stope and Dev Prod (t)	50896	81325	181817	241959	300780	248965	302046	302550	76957	1787295
Stope and Dev Grade (g/t)	5.72	4.77	4.50	5.10	4.50	4.45	4.95	4.88	5.89	4.82
Incremental Dev (t)	27444	112871	123841	133920	148877	119700	0	0	0	666253
Incremental Dev Grade (g/t)	1.31	1.31	1.31	1.31	1.31	1.31	1.31	0	0	1.31
Gold from Dev and Stope Production (Au Oz)	9366	12467	26327	39666	43520	35648	48036	47492	14579	277101
Gold from Incremental Dev (Au Oz)	1152	4736	5197	5619	6230	5023	0	0	0	27957
Total Au Oz	10518	17203	31523	45286	49751	40671	48036	47492	14579	305058
Avrg Stope and Dev Production (tpd))	139	223	498	663	824	682	828	829	855	675

The results of the financial analysis indicate that the resources can be extracted at an estimated average underground operating cost of $163.10 per tonne with a total estimated (initial and sustaining) capital cost of $53.741 million or $30.07 per tonne. Using the consistent gold price of $1,200 US per ounce and a currency exchange rate of C$1.00 = US$0.80, the project generates a positive cash flow with an NPV of $49.75 million (discounted at 5%) and an IRR of 32.0 percent before taxes, and an NPV of $36.50 million (discounted at 5%) and an IRR of 27% after taxes.

25.1 Risks

Gold prices are subject to significant fluctuation and are affected by factors beyond the control of Gowest and might affect the forecast cash flow.

The mineral resources reported in this study have been estimated based on the information provided from the sampling of core from diamond drilling. There is some risk related to the grade continuity of the mineralization within the accuracy of the current interpretation. This risk will be reduced once an underground bulk mining program has been completed. During this program, up to 50,000 tonnes of development and stope production will be mined and processed. The results of the bulk sample will determine:

- If the mined grade is comparable to the estimates made to date.
- If the dilution and mining recovery assumptions are correct.
- If the ore and mineralization along the various zones are continuous and within the stope shapes chosen.
- If the ore can be mined within the 30 metre stope height chosen.
- If a 2 metre stope width can be maintained with no significant overbreak or dilution.
- The effectiveness of the ore sorting equipment is comparable to previous testing results and recoveries.
- Predicted process recovery rates are comparable to previous test results.

Currency fluctuations are also affected by factors beyond the control of Gowest and from our sensitivity calculations, can greatly affect the forecast cash flow.

The metallurgical and process recoveries are based on limited testing. Further variability testing of the deposit, and custom processing of a bulk sample from an underground program, will confirm metallurgical conditions and efficiencies following the next stage of study.

The ore sorting recoveries are based on limited testing. Further testing of the ore sorting equipment during the bulk sample will confirm the recoveries of the production ores as well as the mixed development material.

Operating and capital costs have been estimated based on industry benchmarks and best practice and have been estimated to at least an AACE class 3 estimate of -15%/+25% accuracy. Further study and design development will improve the accuracy of these cost estimates.

Social, political, and environmental factors are all considered to be low risk factors.

There are some significant electrical power savings to the Bradshaw Project when connected to the existing electrical grid. Gowest is currently consulting with Hydro One and $6.5 million (including a 30% contingency) has been allocated to connect to the local hydro grid. Any delays or additional costs will impact the financials of this project. Consultations with Hydro One will determine the potential risk to the project and ensure the proposed power requirements are included in Ontario Hydro's planning.

Surface and/or underground geotechnical evaluations were not available for this study. Local geotechnical assumptions were made for this study. Geotechnical conditions during the bulk sample will confirm the quality of rock underground and on surface to validate the assumptions made for this study.

26.0 RECOMMENDATIONS

Stantec recommends proceeding to the next phase of work. This will involve a bulk sample (advanced exploration) followed by a feasibility study.

Prior to proceeding with the feasibility study, conversion of additional resources through additional diamond drilling from the inferred to indicated category will improve the viability of the project. In accordance with NI 43-101 standards, inferred resources are not eligible for consideration at the prefeasibility stage because they are not considered economically viable.

The mineral resources reported in this study have been estimated based on the information provided from diamond drilling and surface sampling. There is some risk related to the grade continuity of the mineralization within the accuracy of the current interpretation. This risk would be reduced by an underground bulk sampling program. During this program, a portion of the resources would be mined and processed. The mined grade may then be compared to the resource estimates. A bulk sampling program should be planned following the next phase of study and assuming the project financial analysis is still positive at that stage. A study with recommendations is depicted in Appendix I – Bulk Sample. The gross capital cost for the underground bulk sampling program is estimated to be $27.2 million, but after gold credits are applied , the net cost of the bulk sample is $12.5 million. (Table 26.1:).

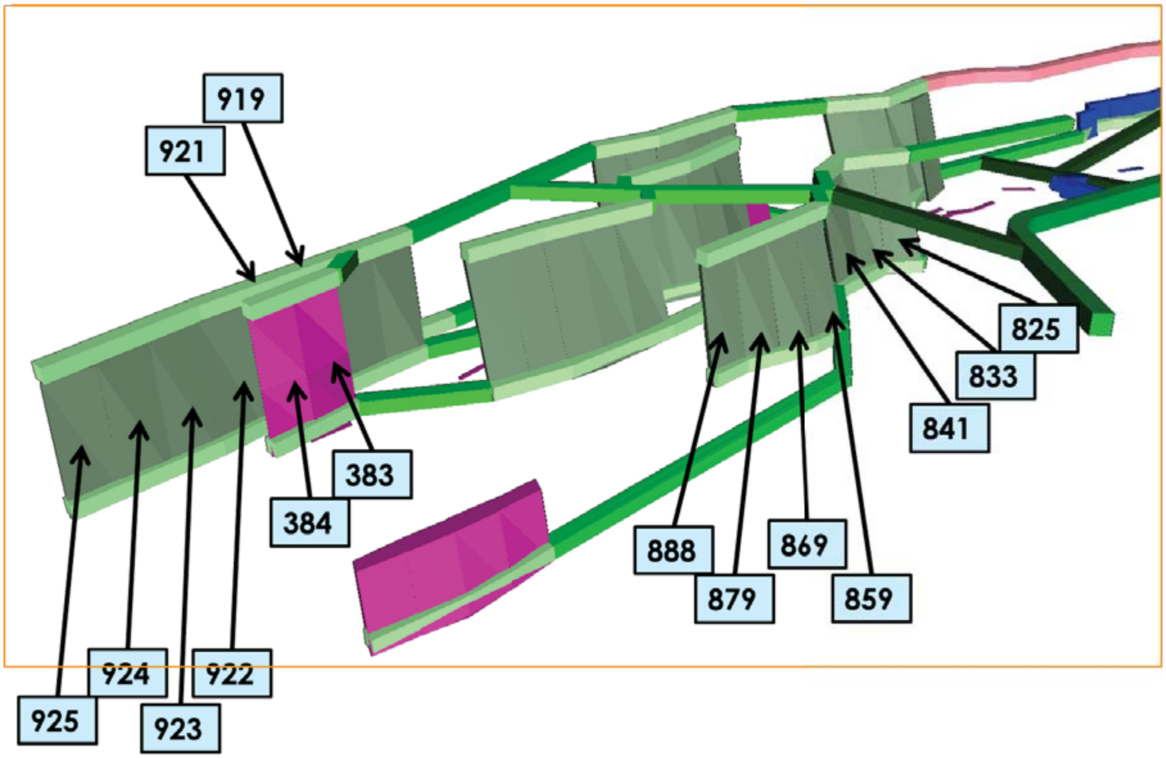
Table 26.1: Summary of Bulk Sample Capital and Operating Costs

Item	Total ($million)
Surface Infrastructure and Facilities (incl 18% cont)	$9,152,194
Capital Lateral Development	$4,236,332
Level Dev Cost	$2,358,419
Mining Operating Cost	$1,503,497
Owner's Costs	$3,339,277
Contractor Indirects	$3,524,714
Milling Costs	$1,572,988
Refining Cost	$1,468,500
Gross Total Cost before deducting Gold Revenue	$27,155,922
Revenue derived from Gold Recovered	(14,693,667)
Total Net Cost after Gold Revenue	$12,462,255

The bulk sampling program will include surface and underground infrastructure including development to access and mine 50,900 tonnes of material (represents several stopes including sill development). The bulk sample is envisioned to be extracted from the upper part of the mine between 45 and 75 Levels, on the east side of the ore body. The

30 metre high stopes will occur within the different ore zones (MZ2, HWZ1, HWZ2, HWZ3 and HWZ4) and will provide valuable geotechnical information (Figure 26.1).

Figure 26.1: Bulk Sample Program Long Section Sketch



The bulk sample program will require a total of 365 days to complete. The start date of the bulk sample program follows the completion of this prefeasibility study and financing from partners but may be earlier pending board approvals.

The proposed feasibility (to be completed concurrently with the bulk sample) should include:

- Collection of geotechnical, dilution and mining recovery data.
- Confirmation of ore distribution and grades.
- Confirmation of metallurgical and ore sorting assumptions.
- Confirmation of ground water assumptions.
- Continue with environmental monitoring and the gathering of base data. There is sufficient data to apply for the environmental permits required to carry on more in depth mining. The permits should be prepared and submitted at this phase.
- Updated block model from new drilling information.

Additional recommendations which may improve the economics of the project include:

- Connection of the Bradshaw Deposit to the local electrical grid. Gowest is currently in talks with Hydro One to determine the feasibility and final costs. Revisions to the economics will be required once the final capital costs are determined.
- The parameters used to determine the block model cut-off grade should be updated for the next level of study. Optimizing the cut-off grade used to define the resource may improve the rate of return for the project.
- Milling facilities exist within trucking distance of the Bradshaw project. Trucking and milling costs will require revisions once a milling facility has been determined.
- The Bradshaw Project does not include the Sheridan Deposit. Future studies related to the Bradshaw Project should assess the economic benefits of including the two deposits.
- Capex has been minimized through the use of contractor equipment and rentals. Trade off studies should be undertaken to review capital versus rental/leased equipment costs.
- The bulk sample will allow a larger sample for the ore sorter and metallurgical testing. The bulk sample will also confirm the ore sorter and metallurgical recoveries and costs. Mine contractor performance, toll milling services and smelting/refining services can be evaluated to select/confirm best performance and optimize costs.
- Good sampling techniques and tight geological control can identify additional economic areas for gold mineralization. The sorter provides an avenue to salvage and recover additional ounces of gold from low grade areas within the development drifts accessing the blast-hole stopes after they are backfilled.
- Good geological sampling of the mineralized low grade areas will require good QA/QC protocals and should be developed during the course of the Bulk Sample Program.
- Since the deposit is open at depth and along certain areas of the periphyry of the deposit, diamond drilling should be concentrated along the margins of the deposit, and face sampling of the development drifts along the low grade mineralized areas can point to secondary targets within the deposit.
- Further Rock Mechanics testing should be conducted within the ore zone and for the HW zones to characterize the Rock Strengths in each of the ore zones. The design parameters used the worst case scenario for the Main Zone due to the proximity to the FW contact of the weaker ultra-mafic rocks. Core examination indicates that more competent rock strengths may be present in the HW zones, and may allow longer stope spans in the design before backfilling. This would introduce cost savings and mining efficiencies.

27.0 REFERENCES

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No references for these sections.

27.2 Sections 4,5,6,7,8,9,10,11,12,14 and 23

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27.3 Section 13 and 17

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27.4 Section 20

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28.0 CERTIFICATE OF QUALIFICATIONS

Stantec

Certificate of Qualified Person
Noris Del Bel Belluz P.Geo.

To accompany the report titled "Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study original date July 15, 2015, amended date September 15, 2015.

- I am a graduate of the University of Toronto, located in Toronto, Ontario with a Bachelor of Science degree in Geology, 1978.
- I am a member in good standing of the Association of Professional Geologists in the province of Ontario (#2377).
- I have been continuously employed in the mining industry since 1980. I am an employee of Stantec Consulting Ltd. since Oct. 2013.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to "Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study" includes 29 years of experience in the mining industry, having worked in copper-gold skarn and porphyry deposits, orogenic, lode and Carlin-type gold deposits, nickel deposits and MVT, SEDEX and VMS base metal deposits in North America, SE Asia and Asia. I have also been in the consulting industry for 6 years working as a consultant and Study Manager. My expertise was acquired working at mining operations from a Project geologist to the Manager of Mine Geology, Hydrology and Geotechnical, delineating and evaluating mineral deposits and providing Technical services to mine operations at large open pit and underground mines. As a Study Manager, I have been responsible for executing NI 43-101 reports, from Resource Evaluations to Feasibility Studies
- I am responsible for the preparation and coordination of this report and contributing in the preparation of Chapters 1, 2, 15, 22, 25 and 26 of this technical report titled "Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study" original date July 15, 2015, amended date September 15, 2015. In addition, I conducted a site inspection of the Bradshaw Deposit on 13 November 2014.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and form 43-101F1, as well as the Repeal and Replacement of National Instrument 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 Technical Reports, and Companion Policy 43-101CP (April 08, 2011) and this Technical Report has been prepared in compliance with these instruments and forms.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 24 day of September, at Mississauga, Ontario.

N. Del Bel Belluz

Signature of Qualified Person

PROFESSIONAL GEOSCIENTIST
NORIS DEL BEL BELLUZ
PRACTISING MEMBER
2377
ONTARIO

Stantec

Certificate of Qualified Person
Michel St-Laurent, P. Eng.

To accompany the report titled "*Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study*" original date July 15, 2015, amended date September 15, 2015.

I, Michel St. Laurent P.Eng., a Consulting Engineer with Stantec Consulting Ltd. with a business address at 1760 Regent Street, Sudbury, Ontario, Canada, P3E 3Z8, do hereby certify that:

- I am a graduate of Queen's University, located in Kingston, Ontario with a Bachelor of Science degree in Mining Engineering, 1977.
- I am a member in good standing of the Association of Professional Engineers of the province of Ontario (#43995018).
- I have been continuously employed in mining industry since my graduation. I have been a consulting engineer with Stantec Consulting Ltd. since May 2009.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to "Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study" includes 38 years of experience in the mining industry, having worked in the copper nickel mines of the Sudbury area. My expertise was acquired in mining operations as a planner, senior engineer and chief mine engineer where I was also involved with underground design, operations and maintenance at several different mines. I also contributed to the deepening of existing mines and project evaluations including capex/opex estimations and financial evaluations through my role within the Technical Services department of a mining company.
- I am responsible for the preparation of 1, 2, 3, 15, 16, 18, 21, 22, 25, 26 and 27 of this technical report titled "*Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study*" original date July 15, 2015, amended date September 15, 2015. In addition, I conducted a site inspection of the Bradshaw property on November 13, 2014.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and form 43-101F1, as well as the Repeal and Replacement of National Instrument 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 Technical Reports, and Companion Policy 43-101CP (April 08, 2011) and this Technical Report has been prepared in compliance with these instruments and forms.
- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 24 day of September 2015 at Sudbury, Ontario.

Signature of Qualified Person

REGISTERED PROFESSIONAL ENGINEER
M. A. ST. LAURENT
PROVINCE OF ONTARIO

Certificate of Qualified Person

Neil N. Gow B.Sc. (Hons.), P.Geo.

As an author of this report entitled "Bradshaw Gold Deposit – NI 43-101 Technical Report and Prefeasibility Study" prepared for Gowest Gold Ltd., original date July 15, 2015, amended date September 15, 2015, I hereby make the following statements:

My name is Neil Neville Gow and I am a Consulting Geologist. My office address is 678 Powell Court, Burlington, ON L7R 3E8.

• I have received the following degrees in Geological Sciences:

• BSc.(Hons) 1966 University of New England, Armidale, NSW Australia

• I am a registered Professional Geologist (#433) in the Province of Ontario. I am also a member of:

a. The Prospectors and Developers Association of Canada (PDAC)

b. Society of Economic Geologists (SEG).

c. Australasian Institute of Mining and Metallurgy (AusIMM)

• I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI-43-101") and certify by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. My relevant experience for the purpose of this report is:

• Mineral Resource and Mineral Reserve audit Homestake Mine, SD

• Mineral Resource Estimate, Dome Mine, Timmins, ON

• This report is based on my personal review of technical reports and other data supplied by the Issuer, on discussions with the Issuer and its representatives, discussions with the geologists working for Geological Team on the property. I have visited the property previously. A visit was made March 11, 2011, and more recently August 28, 2015

• I have been practicing as a professional geologist for more than thirty years.

• I have previously co-authored a report dated November 15, 2012 and available on the Sedar website under Gowest Gold Ltd.

• I am responsible for Sections 4 to 12, 14, 20, 23 and 24 of this technical report titled "Bradshaw Gold Deposit – NI 43-101 Technical Report and Prefeasibility Study" original date July 15, 2015, amended date September 15, 2015.

• I am not aware of any material fact with respect to the subject matter of this report which is not reflected in "the Report" the omission to disclose which makes this report misleading.

• I am independent of the issuer applying the tests set out in section 1.5 of National Instrument 43-101. I have red National Instrument 43-101 and Form 43-101F1 and this report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

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

Signed and dated this 24th day of September 2015 at Toronto, Ontario.

Signature of Qualified Person

PROFESSIONAL GEOSCIENTIST
NEIL GOW
PRACTISING MEMBER
433
ONTARIO

Certificate of Qualified Person
Peimeng Ling M.SC., P.Eng

I, Peimeng Ling, M.Sc., P.Eng. (Ontario), do hereby certify that:

1. I am President of Peimeng Ling & Associates Limited (CofA #100183418) with an office located at 39 Clovercrest Road, Toronto, Ontario, Canada, M2J 1Z5
2. I hold the following academic qualifications:

B.Eng. (Chemical Engineering)	Zhejiang University, PRC	1982
M.Sc. (Chemical Engineering)	University of Toronto, Canada	1994

3. I am a registered Professional Engineer in good standing of Professional Engineers of Ontario (Registration Number 90444985). I am a member of The Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
4. I have over 20 years of direct experience with precious and base metals mineral and hydrometallurgical processing in Canada, USA, Brazil, and Russia including test work, project feasibility study, process design, plant design, environmental compliance, and financial evaluation with a variety of deposit types including gold, silver, copper, zinc, nickel, cobalt, vanadium, platinum-group metals and industrial minerals. Additional experience includes the completion of various NI 43-101 technical reports for vanadium, talc-magnesite deposit projects.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am an author of the technical report titled: "Bradshaw Gold Deposit - NI 43-101 Technical Report and Prefeasibility Study" prepared for Gowest Gold Ltd., original date July 15, 2015, amended date September 15, 2015, (the "Technical Report"). I am responsible for the preparation of Sections 13, 17 and 19. I have visited the Bradshaw Gold Project between December 7th and 9th, 2011.
7. I have previous involvement with the issuer or involvement with the property that is the subject of the Technical Report. I was the author of the technical report titled: "Preliminary Economic Assessment on the Frankfield Gold Project, Tully Township, North-Eastern Ontario" for Gowest Gold Ltd. dated Feb 15, 2012.
8. As of the effective date of the technical report, to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15 day of September, 2015.

Signature of Qualified Person

LICENSED PROFESSIONAL ENGINEER
P. LING
PROVINCE OF ONTARIO

Certificate of Qualified Person

David G. Brown, P. Geo

I, David G. Brown, P.Geo., do hereby certify that:

1. I am a Principal employed by:

Golder Associates Ltd.
141 Adelaide Street W., Suite 910
Toronto, ON, Canada M5H 3L5
Telephone: 416-366-6999
Email: david_brown@golder.com

2. I contributed to a report titled "Bradshaw Gold Deposit NI 43-101 Technical Report and Prefeasibility Study" original date July 15, 2015, amended date September 15, 2015. (Technical Report).
3. I graduated with a Bachelor of Science degree in Chemistry and Environment and Resource Studies from University of Waterloo in 1990. I graduated with a Master of Science degree in Earth Science from University of Waterloo in 1996.
4. I am a registered professional geoscientist with the Association of Professional Geoscientists of Ontario.
5. I have worked as an environmental geoscientist for a total of 25 years, 18 of which have been in the mining industry. I have experience in baseline investigations, characterizing mine wastes (tailings, waste rock and process residues), environmental monitoring programs, site rehabilitation and closure, and environmental assessment and permitting related to proposed, existing and closed mining facilities.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am responsible for summarizing the environmental baseline studies, project permitting and community consultation for the Bradshaw Project as described in Section 20 of the Technical Report, original date July 15, 2015, amended date September 15, 2015. I visited the property on November 11, 2009.
8. I have been involved in environmental baseline data collection, preparation of environmental permits and government and community consultation associated the Bradshaw Project since 2009.
9. As of the date hereof, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for as listed in Item 7 above contains all technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Signed and dated this 24th day of September, 2015.

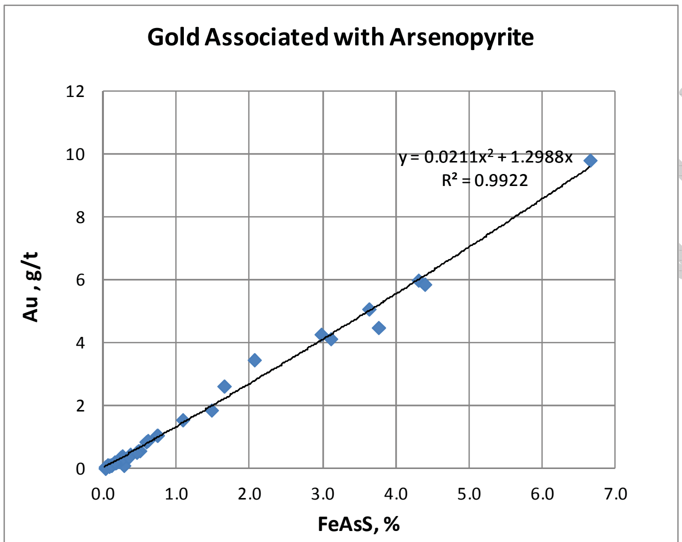
David Brown

Signature of Qualified Person

APPENDIX A
SORTER

Investigation of Ore Sorting Techniques for Gowest Gold's Bradshaw Project

Early metallurgical testing has shown that the gold in Gowest Bradshaw deposit is closely associated with arsenopyrite (FeAsS), such that the higher the concentration of FeAsS, the higher the gold grade (see Figure below).



As a possible route to reduce underground mining dilution, transportation and milling costs, Gowest initiated a comprehensive program to evaluate whether the rocks from the Bradshaw gold deposit are amenable to an array of potentially suitable automated ore sorting techniques which include visible spectrum optical sorting (**Optical**), Dual Energy X-Ray Transmission sorting (**DEXRT**), conductivity/magnetic susceptibility sorting (**EM**), and X-Ray Fluorescence Spectroscopy sorting (**XRF-S**).

1. Preliminary Ore Sorting Investigation - Benchtop Test

In June 2010, Gowest Gold asked SGS, Lakefiled to engage Terra Vision (later acquired by CommodasUltrasort of Germany in Aug. 2010) in Quebec City to perform preliminary ore sorting investigation.

The objective of this test was to take a sample from Gowest's Bradshaw property, characterise the rocks in the sample using several sorting sensors and then determine whether these characteristics can be used to sort the rocks by grade or another metallurgical property of interest. These results can then used to determine if there is a sorting characteristic that warrants further investigation for full scale sorting.

The sensors used in this first preliminary test were:

1) DEXRT - Images acquired with a dual energy X-ray transmission (DEXRT) system
2) EM – Conductivity and magnetic susceptibility acquired with a multi-frequency sensor.

The results of DEXRT and EM tests are presented in following two tables. Each "Class" represents a group of rocks that have similar DEXRT or EM characteristics and can be separated from the others.

DEXRT Sorting – Masses, Grades and Recoveries for Each Class.

Class	Mass (g)	Au (g/t)	Mass in Class (%)	Cumulative Mass Sorted to Conc. (%)	Au Conc. (g/t)	Au Tails (g/t)	Au Distribution (g)	Au Distribution (%)	Au Cumulative Recovery (%)
1	1133.10	10.90	10.59	10.59	10.90	0.99	0.0124	56.53	56.53
2	1122.30	4.91	10.49	21.08	7.92	0.47	0.0055	25.22	81.75
3	2221.00	0.89	20.76	41.84	4.43	0.32	0.0020	9.05	90.80
4	2945.90	0.36	27.54	69.38	2.82	0.29	0.0011	4.85	95.65
5	2048.90	0.26	19.15	88.53	2.26	0.34	0.0005	2.44	98.09
6	1227.10	0.34	11.47	100.00	2.04	0.00	0.0004	1.91	100.00
Subtotal	10698.30	**2.04**	100.00				0.0218	100.00	

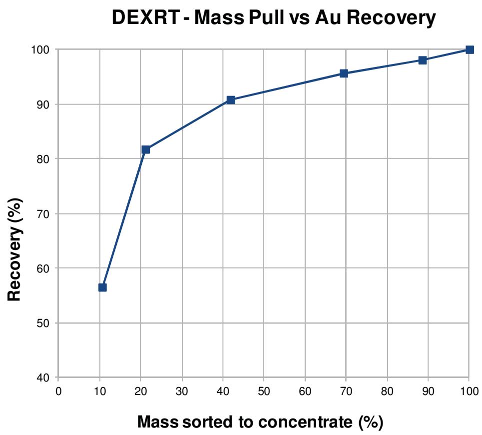
EM Sorting – Masses, Grades and Recoveries for Each Class

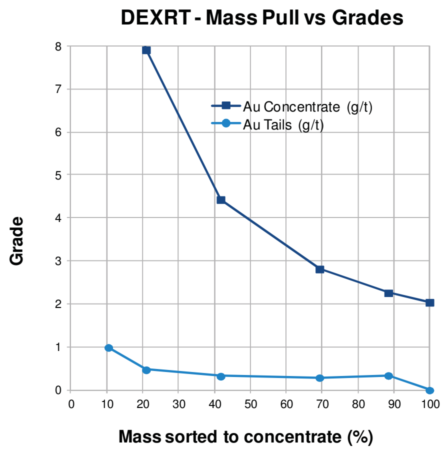
Class	Mass (g)	Au (g/t)	Mass in Class (%)	Cumulative Mass Sorted to Conc. (%)	Au Conc. (g/t)	Au Tails (g/t)	Au Distribution (g)	Au Distribution (%)	Au Cumulative Recovery (%)
3	4710.50	6.13	18.01	18.01	6.13	0.44	0.0125	81.27	81.27
2	4561.00	0.47	40.33	58.34	1.58	0.36	0.0021	13.95	95.23
1	2036.70	0.36	18.01	76.36	1.36	0.00	0.0007	4.77	100.00
	11308.20	**1.36**	76.36				0.0154	100.00	

The DEXRT test recovery curves shows that almost all of the Au (96%) is contained in the first four of six classes, which accounts for only 69% of the mass. The DEXRT grade curve shows that if the first four classes were sorted to the sorter concentrate it would result in a grade of 2.82 g/t in 69% of the mass while the tails of such a sort would be 0.29 g/t.

There are only three classes in EM test as there was no measurable conductivity response and a magnetic susceptibility response at the lower limit of sensitivity of the sensor. It is unlikely that the three classes could be separated with a sensor based sorter.

Mass pull vs recovery and mass pull vs gold grade using DEXRT sorting are shown in figures below. The recovery curves shows that almost all of the Au (96%) is contained in the first four of six classes, which accounts for only 69% of the mass. The grade curve shows that if the first four classes were sorted to the sorter concentrate it would result in a grade of 2.82 g/t in 69% of the mass while the tails of such a sort would be 0.29 g/t.





Details of this preliminary test can be found in a report by Terra Vision, "2010.06.15_SGS-GoWestGold-Frankfield_Property-rev0".

2. Second Benchtop Amenability Test

A second bench-top screening tests was completed in September 2011 using rock samples from the Bradshaw deposit.

The objective of this test was to take a set of specimens from the Gowest's Frankfield project, characterize the rocks in the set using conductivity and magnetic susceptibility (EM), color (Optical), X-Ray Fluorescence Spectroscopy (XRF-S) and Dual Energy X-ray Transmission (DEXRT) features and determine whether these features can be used to sort the rocks to upgrade the gold values. These results can then be used to determine if there is a sorting characteristic that warrants further investigation for full scale sorting tests.

Gowest shipped split core rock specimens to CommodasUltrasort in Quebec City. 100 rocks were chosen at random from the specimens.

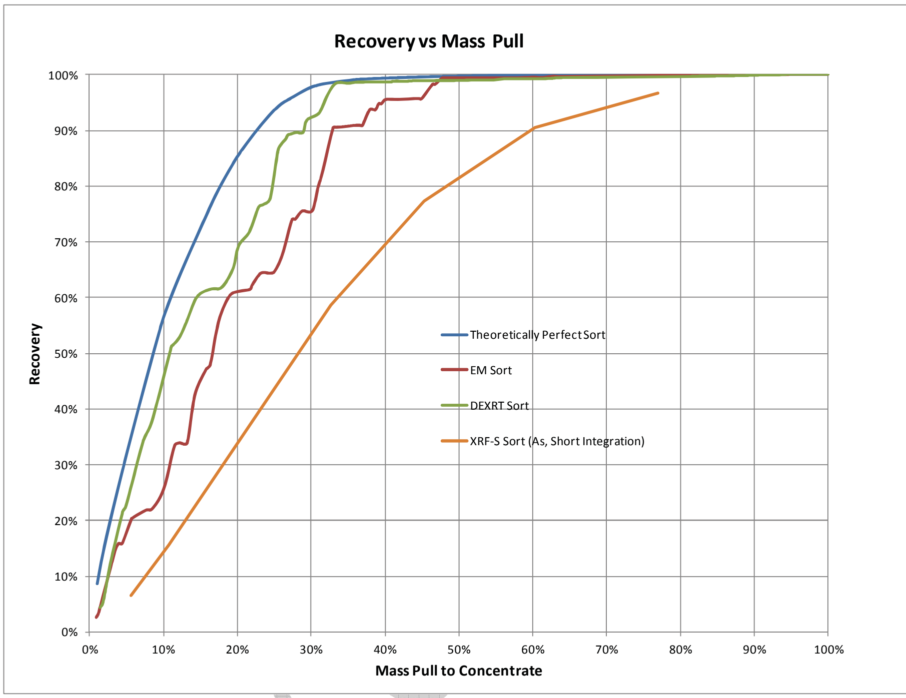
Features of the 100 rocks were then acquired with the following sensors:

1) DEXRT - A dual energy Heimann 6040i x-ray scanner was used to acquire the x-ray transmission (DEXRT) characteristics for each rock.
2) OPTICAL - A benchtop optical sorter was used to acquire an image of each rock.
3) EM – The conductivity and magnetic susceptibility response was acquired with the GDD MPP EM2S+ probe.
4) XRF-S – The results were acquired with our benchtop test rig system configured to simulate a full scale XRF-S sorter.

The 100 rocks were assayed at ALS Global in North Vancouver, BC.

The figure below shows the Theoretically Perfect Recovery to be 99% of the gold at a 33% mass pull to concentrate. The Theoretically Perfect Recovery curve for this sample was obtained by ordering the rocks in descending gold grade. The tests showed that DEXRT sorting appeared to have the best potential for sorting by grade as the DEXRT recovery curve approaches the theoretically perfect curve at approximately 35% mass pull to concentrate. The DEXRT sort showed a 98% gold recovery at 33% mass pull to concentrate.

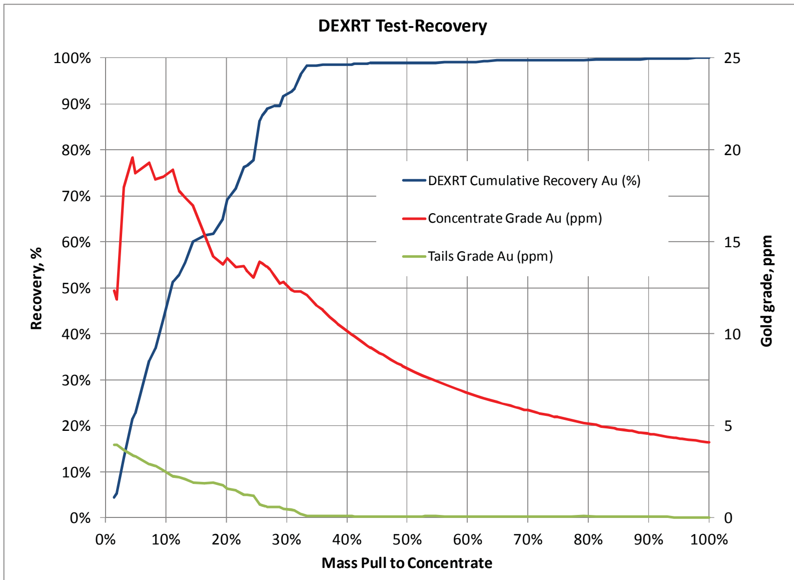
The specimens are also amenable to sorting with the XRF-S sensor used in this study, although the upgrading is not as significant as with the DEXRT sensor. A mass pull of 60% to concentrate resulted in 90% recovery of the gold.



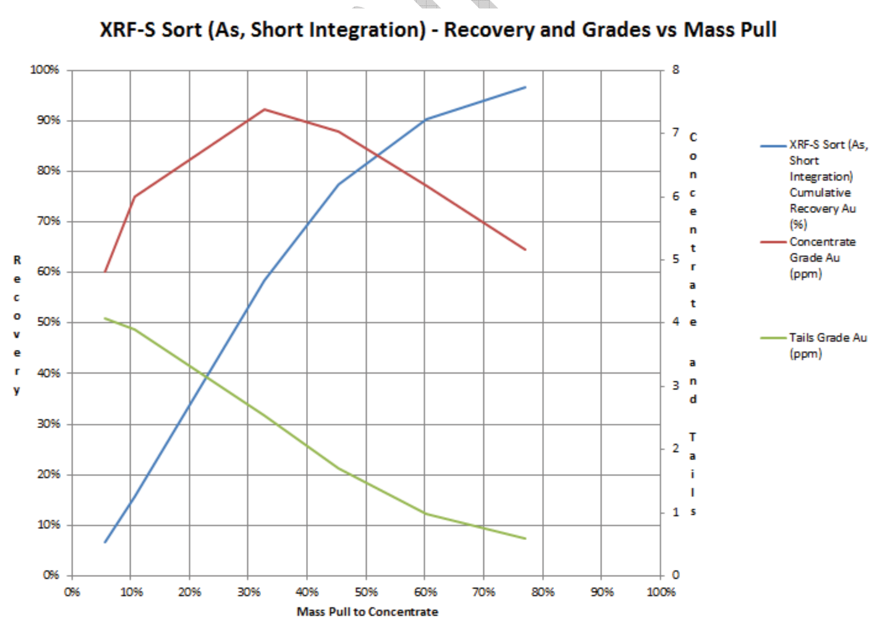
Although the figure above appears to show EM sorting with similar performance to the DEXRT sensor the sort results from this test for this EM sensor are only applicable at the laboratory scale and not useful for a full scale sorter.

Optical sorting results have not been included in the above figure as no pattern or visual characteristic could be defined that was usable by an optical sorter. **The specimens were not amenable to Optical sorting**.

The figure below is the result of the composite (crushed to a diameter of less than 0.75 inch) consisting of a wide range of Bradshaw drill core intersections ranging in gold content from 0 g/t (waste rock) to over 10 g/t (high grade main zone) and averaging 4 g/t. Despite the relative low gold content of the composite, results from the testwork confirmed an extremely efficient separation by DEXRT test. Greater than 50% of the rock mass was rejected resulting in a final crushed rock product containing 12-15 g/t gold with only 2-3% gold losses.



For the XRF test, as shown in figure below a mass pull of 60% to concentrate would result in a recovery of 90% of the gold. For this same mass pull the concentrate would grade 6.8ppm gold with tails of 0.99 ppm gold. The graph shows the Au recoveries and grades, however the sensor was not able to sort directly for gold grade.



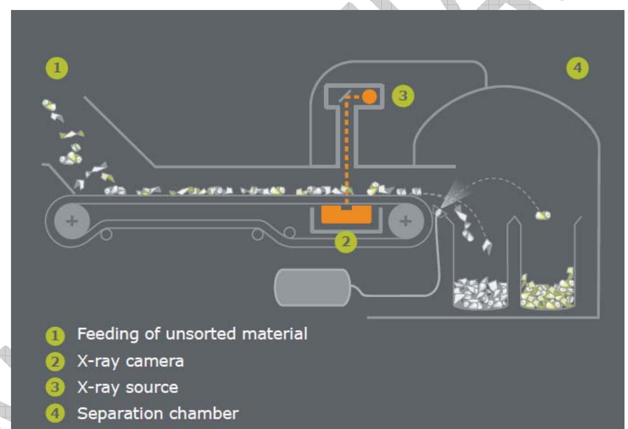
A detailed discussion of this benchtop sorting program and results for each sensor can be found in the test report, "**Preliminary Ore Sorting Investigation Benchtop Amenability Test: X-Ray Transmission, Optical, Conductivity and X-Ray Fluorescence for for the Frankfield Project**, 2011.09.12 (file name: 2011.09.12_GoWest_Frankfield)", by CommodasUltrasort, as well as the appendices to the report.

3. Pilot Test - 2012

A pilot test was conducted in April, 2012. One-ton bulk sample taken from the Frankfield Bradshaw (previuous Frankfiled East) deposit was provided by GoWest and sorted on automated sorters at the CommodasUltrasort lab in Wedel, Germany.

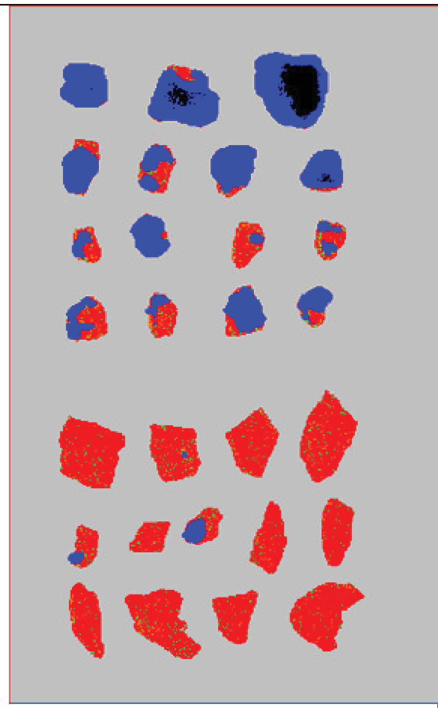
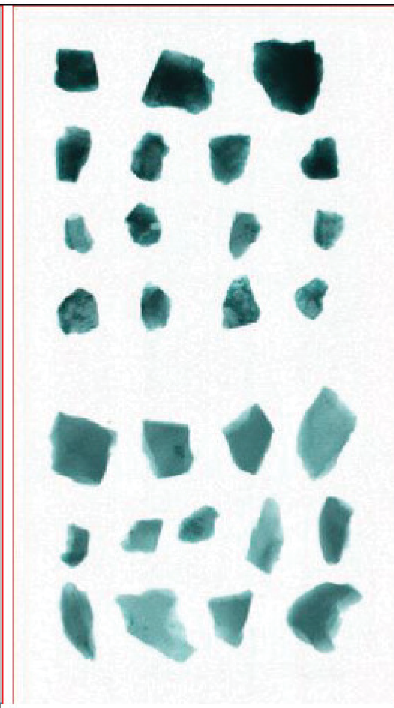
1) Test System - XRT Sorter

The two pictures below show a Slag Secondary XRT belt machine that was used in pilot test to scan and sort samples and a schematic of the functional principals of the XRT sorter.



The sorter functions by using a broad-band electrical x-ray source that is applied to the material to be sorted while it is moving along the belt. The X-ray sensor system below the material produces a digital image of the material being sorted, using two different energy bands. The X-ray attenuation through the material is different within the two bands and depends on both the thickness and the density of the material. An image transformation of the density images of the two bands then makes it possible to classify each pixel according to atomic density. Classification proceeds relative to a reference density, to which the system has been calibrated. Depending on the classification the selected particles are either "ejects", diverted upwards by air jets (Material Stream A) or "accepts" in the other stream (Material Stream B). It is important to note that "eject" refers to the material that the system has been configured to blow out of the material stream; this can be either the waste or the product. Figure below shows an example of a XRT image and the transformed image used to determine whether a particle is an "eject" or "accept".

Classified XRT image (left) and original raw XRT image (right).

Example of high grade particles. Particles with visible Pyrite inclusions. Examples of low grade particles. Particles with no visible inclusions.		

2) Test Samples and Procedure

Gowest shipped four barrels of rock specimens to the CommodasUltrasort lab in Wedel, Germany. These barrels contained specimens of four grades ranging from low to high grade (soapstone, waste, low, and high) as well a range of sizes (1/4”-5/8”, 5/8”-1”, 1"-1½”). Subsamples were used to create a training set. The training set was created by passing rocks in the subsamples through a benchtop XRT sorter and then classifying the rocks into four predetermined categories based in their grade.

Each grade was assigned a label (see Table below) in which it was tested for each of the three size fractions.

Sample	***Grade***
Material A	*Soapstone*
Material B	*Waste*
Material C	*High Grade*
Material D	*Low Grade*

Individual test images of the training sets were taken to enable the sorting unit to separate the material into:

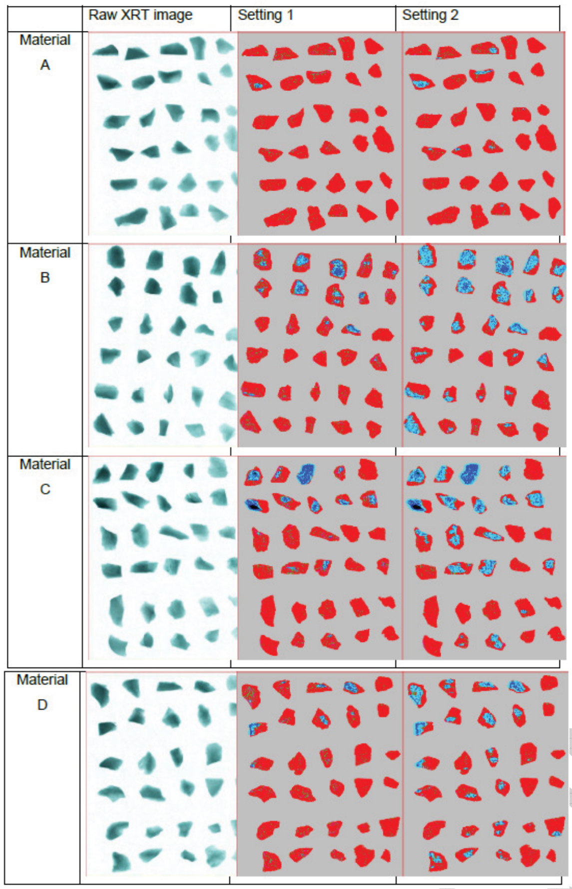
Higher grade ore as “Ejects*”
Lower grade ore as “Accepts*”

Classified XRT 5/8”-1” “ejects’ and ‘accepts’: Tomra Testing Laboratory, Germany.

“Ejects” and “Accepts” refer to the physical ejection of the material in the sorter not to the value of the material. For example a sorter can be configured to eject either ore or to eject waste.

Each material was tested twice using a different setting each time. The first setting was more selective of the high grade material that it “ejected”, while the second setting was less selective and “ejected” more material. After these tests were completed, a mixed input fraction from each grain size was generated and sorted. Figure below displays images of the classes for the size range of 1”-5/8”. Blue inclusions are indicative of high atomic density, “ejected” specimens.

XRT images and settings for all grades, size fraction 1”-5/8”.

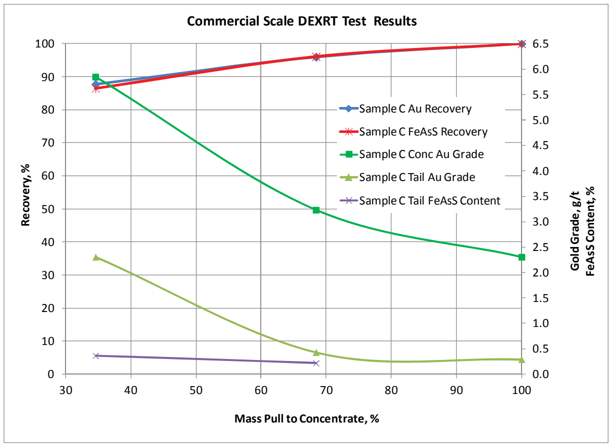


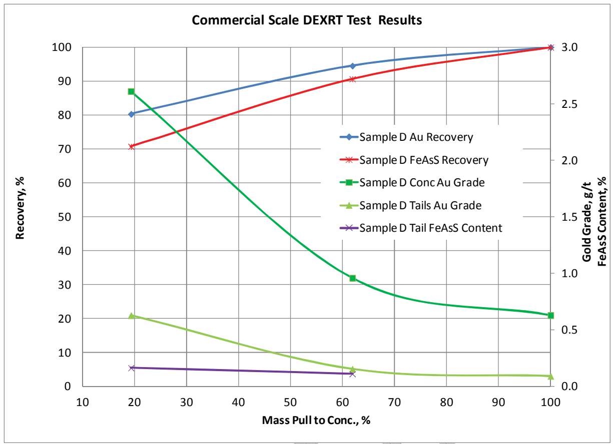
3) Test Results

This test used widely available production scale DEXRT ore sorting equipment and was performed under commercial operating conditions. The crushed material is transferred at high speeds along a conveyor belt in front of an x-ray sensor that analyzes the signatures of individual rocks to detect the FeAsS in the crushed ore. The sensors then trigger a series of individually controlled air jets to separate the uneconomic ore with less than approximately 0.3 g/t gold.

All of the sorted material processed in this test was sent to SGS Lakefield for assay.

The figures below presented results for high grade (C) and low grade (D) samples, showed that the equipment used was able to detect FeAsS down to 0.1-0.2%,





4. Conclusion and Future Work

The tests showed that DEXRT sorting appeared to have the best potential for sorting by grade as the DEXRT recovery curve approaches the theoretically perfect curve at approximately 35% mass pull to concentrate..

The specimens are also amenable to sorting with the XRF-S sensor, although the upgrading is not as significant as with the DEXRT sensor.

Based on TOMRA's expert, it is better to focus on XRT sorting. The bulk tests showed that this is a very effective technology for sorting Gowest's material and it is a mature technology as compared to XRF-S sorting. With regards to throughputs, TOMRA is putting a lot of effort into building high capacity XRT sorters which will effectively remove any advantage that XRF-S has for the Bradshaw deposit.

Test Report

DEXRT Sorting of Gold Ore

Client: GoWest Gold Ltd.

by

Markus Dehler, Wedel, June 12th 2012

Revision 1 by Sarah Rathay

Customer:	GoWest Gold Ltd. 80 Richmond Street West, Suite 1400 Toronto, Ontario M5H 2A4 Represented by Darren Koningen and Garth Wilcox
Test operator:	Mr. Markus Dehler
Sales contact:	Mr. Matthew Kowalczyk
Site:	Test facility of CommodasUltrasort GmbH Feldstraße 128 D-22880 Wedel Germany
Test date:	April 23-27, 2012

Content

1. Introduction

Automated optical ore sorting equipment is used by major mining companies for aggregates, industrial mineral, base metal and precious metal projects. Ore sorting systems allow mining operations to reduce dilution, lower strip ratios and transport costs, remove contaminants and optimize process flows by separating rocks on a particle by particle basis. CommodasUltrasort, a division of TOMRA Sorting, offers ore sorting solutions from initial amenability testing and characterization to equipment sales and complete sorting lines. CommodasUltrasort, the world leader in sorting systems for mining is allied with SGS Lakefield and Hazen and Inspectorate Metallurgical Services to offer ore sorting amenability studies as part of their standard scoping studies, as well as having worked with numerous other reputable labs such XPS Services, Mintek, Met-Solve Laboratories, and G&T Metallurgical Services . Our global presence with offices in Canada, Russia, Germany, South Africa, and Australia guarantees mining operations local ore sorting expertise and service.

This report describes the results of the automated sorting tests performed on samples from the GoWest Frankfield Project. The samples were provided by GoWest and sorted on automated sorters at the CommodasUltrasort lab in Wedel, Germany.

2. Sorting Task

The work described in this report describes X-Ray Transmission (XRT) sorting tests conducted at the CommodasUltrasort test facility in Wedel, Germany on gold ore samples from the GoWest Frankfield Project. The report follows an initial benchtop amenability test from the Frankfield Project (see CommodasUltrasort report *2011.09.12_GoWest_Frankfield.pdf*). The sorter used is a Slag Secondary XRT belt production scale sorting system.

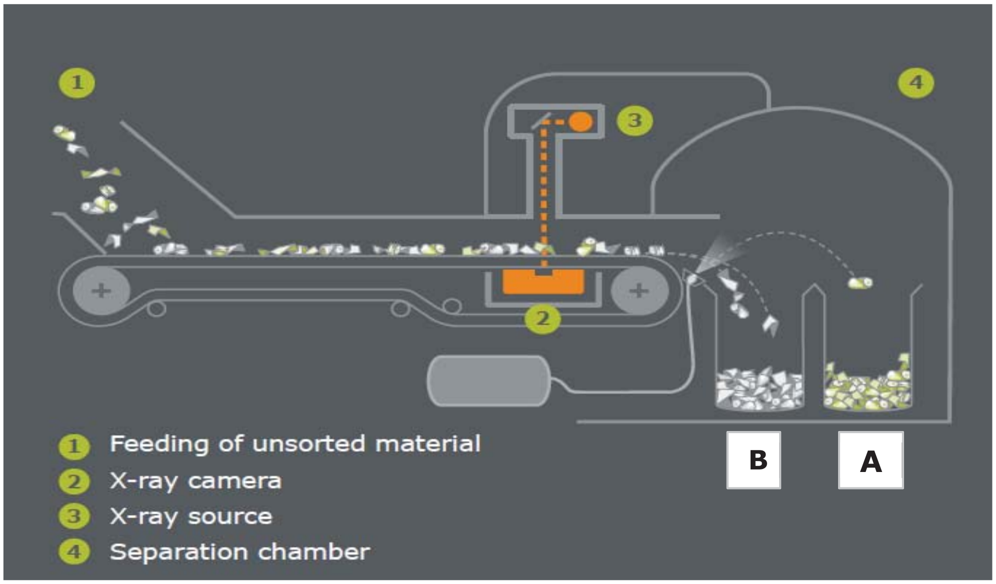
3. Test systems

XRT Sorter

The samples were scanned and sorted using the Slag Secondary XRT belt machine. The following images give an overview of the equipment used for the tests. Figure 1 shows a picture of the actual XRT sorter that was used for the tests. Using a high energy X-ray source and twin sensors as detectors, the equipment is able to measure the atomic density of the material passing through it. Figure 2 shows a schematic of the functional principals of the XRT sorter.

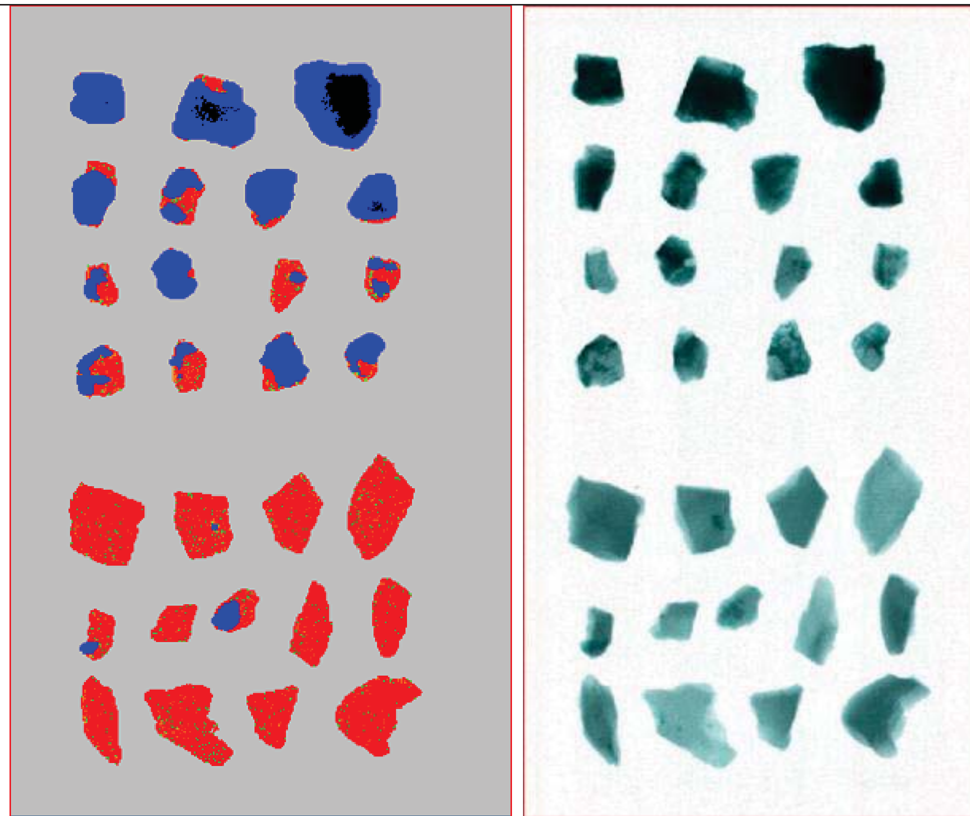
Figure 1: XRT Sorter used in the CommodasUltrasort test facility in Wedel, Germany.

Figure 2: A *schematic of* the belt-style *XRT sorter*.



The sorter functions by using a broad-band electrical x-ray source that is applied to the material to be sorted while it is moving along the belt. The X-ray sensor system below the material produces a digital image of the material being sorted, using two different energy bands. The X-ray attenuation through the material is different within the two bands and depends on both the material's thickness and density. An image transformation of the density images of the two bands then makes it possible to classify each pixel according to atomic density. Classification proceeds relative to a reference density, to which the system has been calibrated. Depending on the classification the selected particles are either “ejects”, diverted upwards by air jets (Material Stream A) or “accepts” in the other stream (Material Stream B). It is important to note that “eject” refers to the material that the system has been configured to blow out of the material stream; this can be either the waste or the product. Figure 3 shows an example of a XRT image and the transformed image used to determine whether a particle is an “eject” or “accept”.

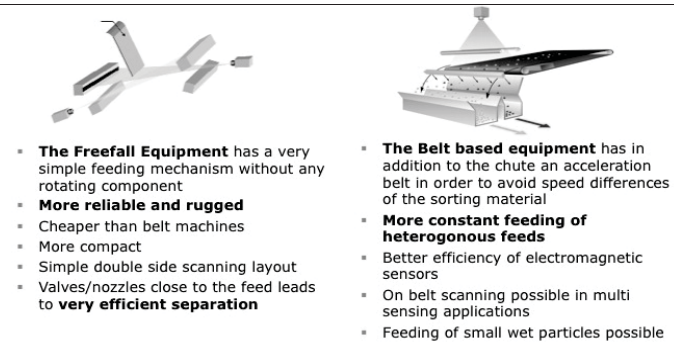
Figure 3: Classified XRT image (left) and original raw XRT image (right).

Example of high grade particles. Particles with visible Pyrite inclusions. Examples of low grade particles. Particles with no visible inclusions.	

Because the XRT sorter uses X-rays that pass through the particles and are a measure of the attenuation through the entire rock, XRT separation is independent of surface quality of the material to be sorted or its moisture. Surface properties such as color and texture and/or contaminations such as dirt, dust, paint, etc. are irrelevant to the detection.

There are two types of sorters available for mining applications – belt and chute. Figure 4 illustrates the difference in these two styles of machines. More details on the principles of operation of XRT sorters can be found in Appendix A.

Figure 4: Machine types "Chute" and "Belt"



4. Material

GoWest shipped four barrels of rock specimens to the CommodasUltrasort lab in Wedel, Germany. These barrels contained specimens of four grades ranging from low to high grade (soapstone, waste, low, and high) as well a range of sizes (-1½"+1", -1"+5/8", -5/8"+1/4"). Each grade was assigned a label (see Table 1) in which it was tested for each of the three size fractions.

Table 1: Labels of specimens.

Material A	*Soapstone*
Material B	*Waste*
Material C	*Low Grade*
Material D	*High Grade*

5. Test Procedure

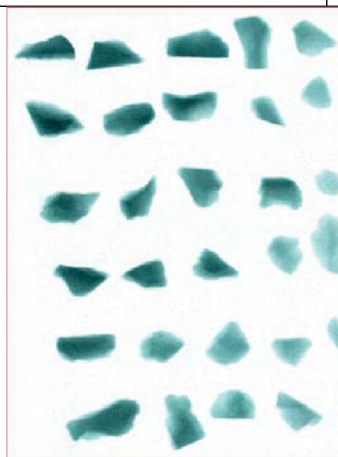
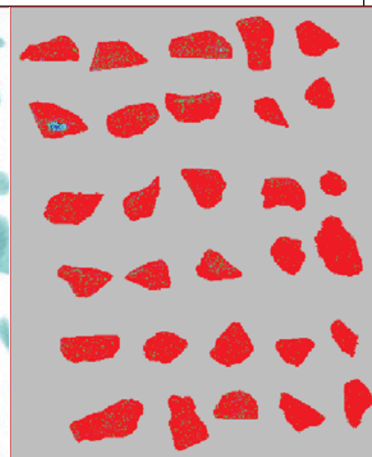
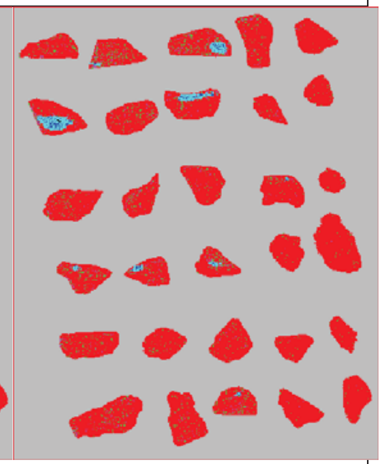
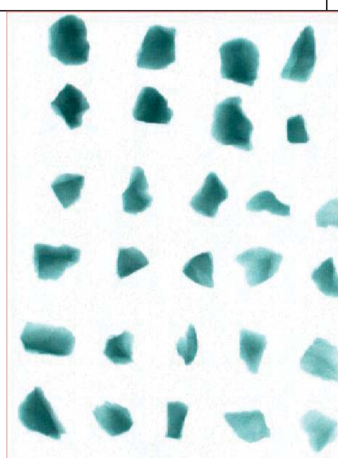
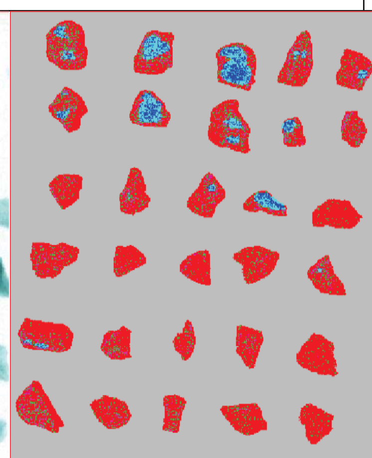
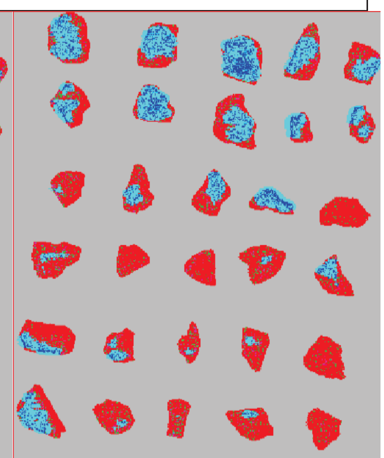
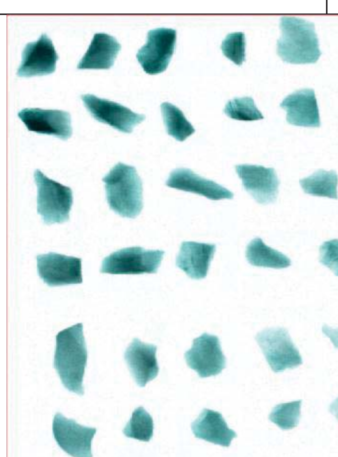
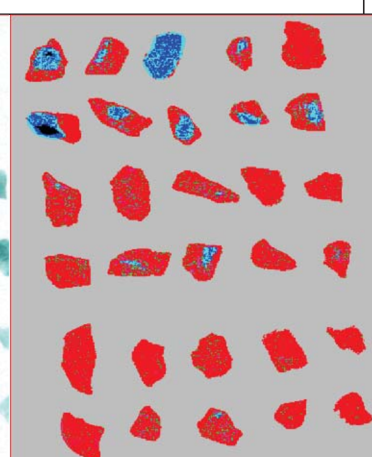
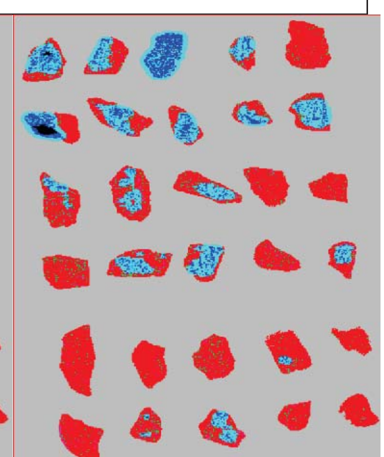
Subsamples received at the CommodasUltrasort lab in Wedel, Germany were used to create a training set. The training set was created by passing rocks in the subsamples through a benchtop XRT sorter and then classifying the rocks into four predetermined categories based in their grade. The four categories were Material A (soapstone), Material B (waste), Material C (high grade), and Material D (low grade). Individual test images of the training sets were taken to enable the sorting unit to separate the material into:

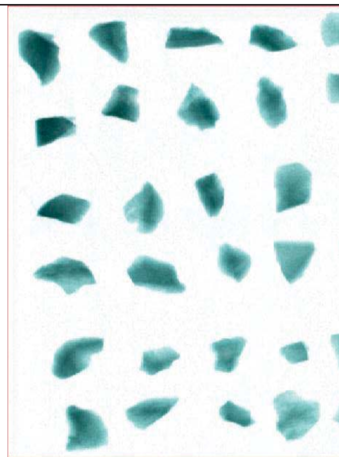
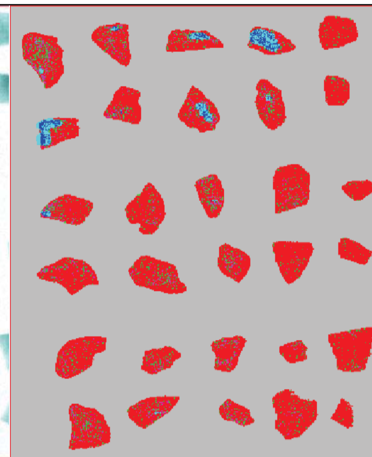
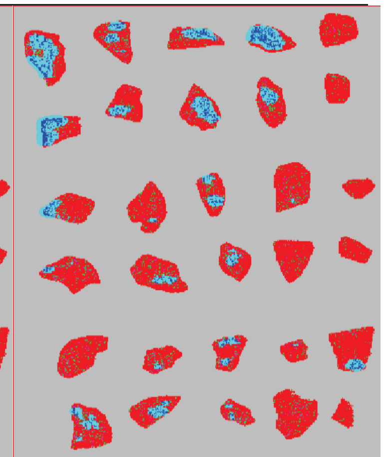
1. Higher grade ore as “Ejects*”
2. Lower grade ore as “Accepts*”

*Please note that “Ejects” and “Accepts” refer to the physical ejection of the material in the sorter not to the value of the material. For example a sorter can be configured to eject either ore or to eject waste.

Each material was tested twice using a different setting each time. The first setting was more selective of the high grade material that it “ejected”, while the second setting was less selective and “ejected” more material. After these tests were completed, a mixed input fraction from each grain size was generated and sorted. Figure 5 displays images of the classes for the size range of 1”-5/8”. Blue inclusions are indicative of high atomic density, “ejected” specimens.

Figure 5: XRT images and settings for all grades, size fraction 1”-5/8”.

	Raw XRT image	Setting 1	Setting 2
Material A			
Material B			
Material C			

Material D			

6. Test Results

6.1 Grain size -5/8"+1/4"

Material A (soapstone)

Input [kg]	29.75

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	29.6	

Reject		
	Mass	Percentage [%]
Concentrate	0.15	0.5

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	28.75	96.6

Reject		
	Mass	Percentage [%]
Concentrate	0.85	2.9

Material B (waste)

Input [kg]	49.95

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	37.45	

Reject		
	Mass	Percentage [%]
Concentrate	12.5	25.0

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	20.95	41.9

Reject		
	Mass	Percentage [%]
Concentrate	16.5	33.0

Material C (high)

Input [kg]	40.3

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	27.5	

Reject		
	Mass	Percentage [%]
Concentrate	12.8	31.8

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	10.75	26.7

Reject		
	Mass	Percentage [%]
Concentrate	16.75	41.6

Material D (low)

Input [kg]	30.4

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	22.7	

Reject		
	Mass	Percentage [%]
Concentrate	7.7	25.3

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	13.15	43.3

Reject		
	Mass	Percentage [%]
Concentrate	9.55	31.4

Test with mixed material

Input:

Material	A	B	C	D
Mass [kg]	20	0	20	20

Mix

Input [kg]	59.7

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	48.6	

Reject		
	Mass	Percentage [%]
Concentrate	11.1	18.6

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	36.2	60.6

Reject		
	Mass	Percentage [%]
Concentrate	12.4	20.8

6.2 Grain size -1”+5/8”

Material A (soapstone)

Input [kg]	24.55

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	23.35	

Reject		
	Mass	Percentage [%]
Concentrate	1.2	4.9

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	21.8	88.8

Reject		
	Mass	Percentage [%]
Concentrate	1.55	6.3

Material B (waste)

Input [kg]	34,3

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	26.5	

Reject		
	Mass	Percentage [%]
Concentrate	7.8	22.7

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	10.8	31.5

Reject		
	Mass	Percentage [%]
Concentrate	15.7	45.8

Material C (high)

Input [kg]	58.4

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	38.65	

Reject		
	Mass	Percentage [%]
Concentrate	19.75	33.8

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	18.5	31.7

Reject		
	Mass	Percentage [%]
Concentrate	20.15	34.5

Material D (low)

Input [kg]	39.75

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	31.6	

Reject		
	Mass	Percentage [%]
Concentrate	8.15	2.5

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	14.9	37.5

Reject		
	Mass	Percentage [%]
Concentrate	16.7	42.0

Test with mixed material

Input:

Material	A	B	C	D
Mass [kg]	20	0	20	12

Mix

Input [kg]	51.95

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	41.4	

Reject		
	Mass	Percentage [%]
Concentrate	10.55	20,3

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	28.0	53,9

Reject		
	Mass	Percentage [%]
Concentrate	13.4	25,8

6.3 Grain size -1½”+1”

Material A (soapstone)

Input [kg]	34.9

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	34.6	

Reject		
	Mass	Percentage [%]
Concentrate	0.3	0.9

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	33.4	95.7

Reject		
	Mass	Percentage [%]
Concentrate	1.2	3.4

Material B (waste)

Input [kg]	60.0

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	39.7	

Reject		
	Mass	Percentage [%]
Concentrate	20.3	33.8

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	22.2	37.0

Reject		
	Mass	Percentage [%]
Concentrate	17.5	29.2

Material C (high)

Input [kg]	79.25

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	54.1	

Reject		
	Mass	Percentage [%]
Concentrate	25.15	31.7

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	33.1	41.8

Reject		
	Mass	Percentage [%]
Concentrate	21.0	26.5

Material D (low)

Input [kg]	51.85

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	38.6	

Reject		
	Mass	Percentage [%]
Concentrate	13.25	25.6

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	22.3	43.0

Reject		
	Mass	Percentage [%]
Concentrate	16.3	31.4

Test with mixed material

Input:

Material	A	B	C	D
Mass [kg]	20	0	20	20

Mix

Input [kg]	59.9

Step 1

Accept		
	Mass [kg]	Percentage [%]
Tailings	48.9	

Reject		
	Mass	Percentage [%]
Concentrate	11.0	18.4

Step 2

Accept		
	Mass [kg]	Percentage [%]
Tailings	39.2	65.4

Reject		
	Mass	Percentage [%]
Concentrate	9.7	16.2

CommodasUltrasort GmbH,

Wedel, 12.06.2012

Markus Dehler
Test facility Manager

Appendix A – DEXRT Ore Sorter Principles of Operation

DEXRT Sorting - Hardware

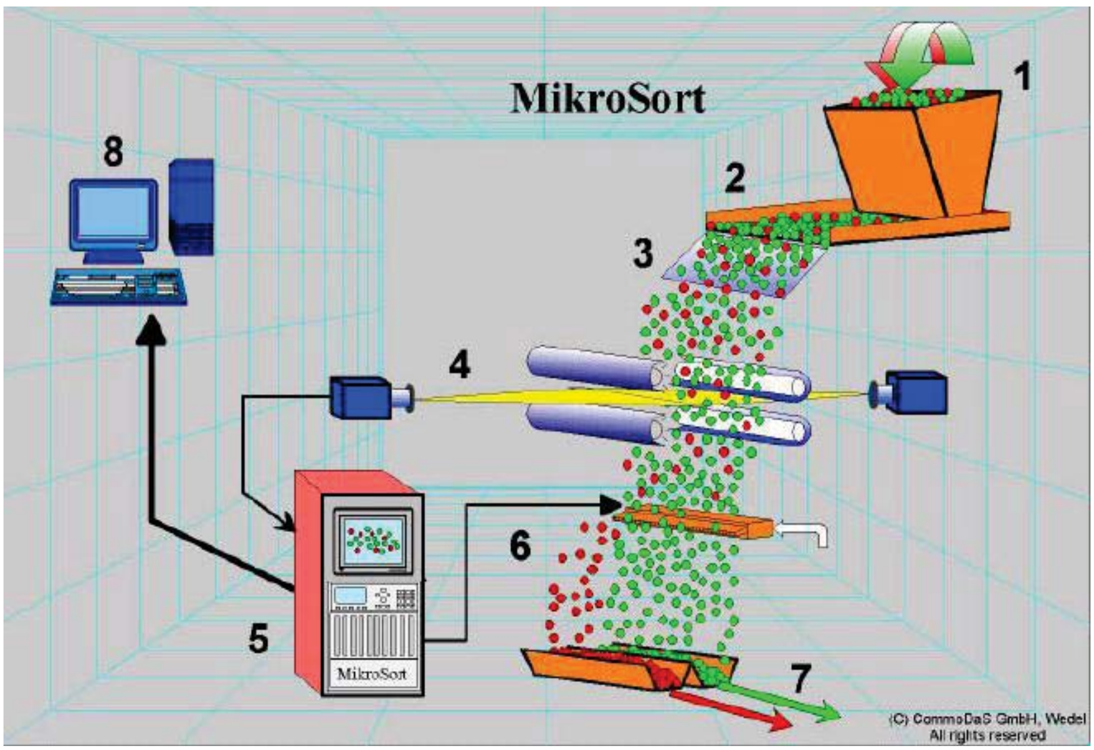
The sensor based scanning of a product stream for the sorting of free-flowing materials is an established technology, which has become widely used in plastic, steel, glass recycling and ore preparation.

Figure B1: A Pro model XRT chute machine

Figures B2 and B3 show the principle of operation of a DEXRT sorting system. The dual energy X-ray transmission method (DEXRT) principle is known to most people from airport baggage inspections. CommodasUltrasort has drawn upon this basic principle to design a sensor system particularly adapted to sorting. A broad-band electrical x-ray source is applied to the material to be sorted while it is moving along the measuring track or falling from the chute. The X-ray sensor system below the material produces a digital image of the material being sorted, using two different energy bands. The X-ray attenuation through the material is different in the two bands and depends on both the material's thickness and density.

An image transformation of the density images of the two bands then makes it possible to classify each pixel according to atomic density. A classification of the ore pieces on the basis of their content is feasible. Classification proceeds relative to a reference density, to which the system has been calibrated.

Figure B2: CommodasUltrasort flow schematic

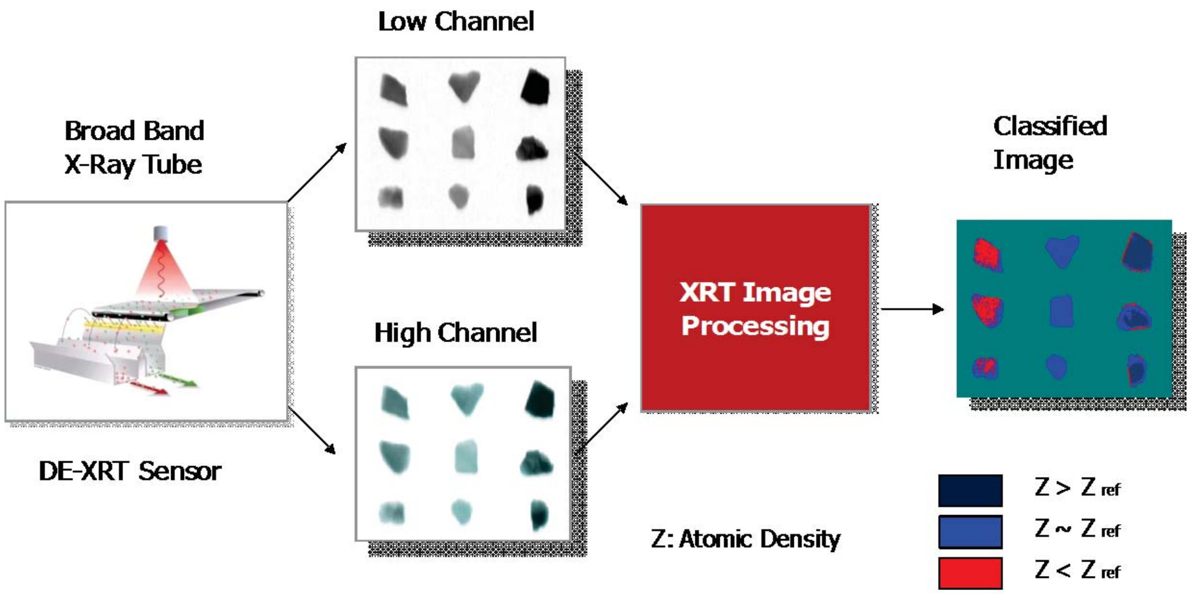


1. Material feed from hopper
2. Vibratory pan feed to spread particles into a monolayer
3. Chute for presentation of particles in front of sensors
4. Scanning of particles. This image shows cameras, however other sensors are available including but not limited to x-ray transmission, x-ray fluorescence, x-ray fluorescence spectroscopy, conductivity and near infra-red spectroscopy sensors
5. High speed image/signal processor
6. Array of high speed air jets for separation of individual particles
7. Separated process streams

8. Remote user interface.

The sensor's high spatial resolution of 0.8mm or 1.6mm (depending on the model) also permits the evaluation of particle shape, particle size, material thickness and texture of the gray-scale image (inclusions of various densities). X-ray transmission image processing provides a highly efficient sensor system for classifying materials. The Pro XRT and ROM XRT machines work with an electric x-ray source featuring a maximum acceleration voltage of 160 KV. Depending on the given densities, this equipment permits the sorting of materials whose can be up to 35mm (iron ore) or 80mm (coal) thick. Depending on the type of application, the achievable throughput rate will lie between 5 and 50 tons per hour. Since market introduction in mid-2005, more than 60 XRT sorting machines are in operation worldwide.

Figure B3: Dual energy X-ray Transmission image processing.



Preliminary Ore Sorting Investigation
Benchtop Amenability Test:
X-Ray Transmission, Optical, Conductivity and X-Ray Fluorescence for for the Frankfield Project

Client:

Gowest Gold Ltd.
80 Richmond Street West, Suite 1400
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Report:

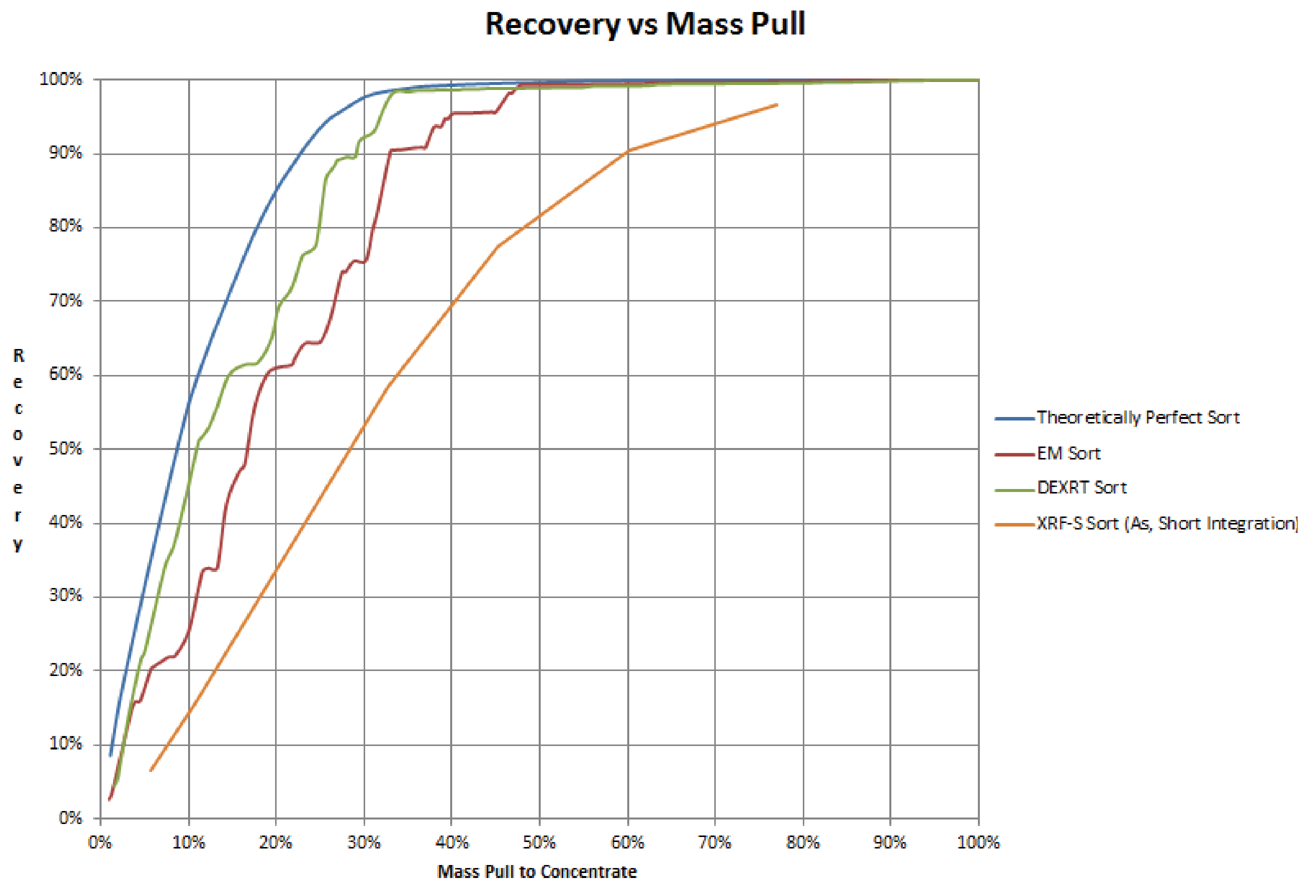
2011.09.12_GoWest_Frankfield

Table of Contents

1 Executive Summary

Gowest Gold Ltd. (Gowest) engaged CommodasUltrasort to investigate a set of rocks from the Frankfield gold project for their amenability to visible spectrum optical sorting (**Optical**), Dual Energy X-Ray Transmission sorting (**DEXRT**), conductivity/magnetic susceptibility sorting (**EM**), and X-Ray Fluorescence Spectroscopy sorting (**XRF-S**). Figure 1 shows the results of the tests. This graph shows the relationship of the recovery of the gold versus mass for each of the sensors in comparison to the theoretically perfect recovery curve.

Figure 1: Recovery Curves for Gold for a "Perfect Sort", EM, DEXRT, Optical and XRF-S Sort



The **Theoretically Perfect Recovery** curve for this sample was obtained by ordering the rocks in descending gold grade.

The tests showed that **DEXRT** sorting appeared to have the best potential for sorting by grade as the DEXRT recovery curve approaches the theoretically perfect curve at approximately 35% mass pull to concentrate. For example figure 1 shows the theoretically perfect recovery to be 99% of the gold at a 33% mass pull to concentrate. The DEXRT sort showed a 98% gold recovery at 33% mass pull to concentrate.

The specimens are also **amenable to sorting with the XRF-S** sensor used in this study, although the upgrading is not as significant as with the DEXRT sensor. As can be seen in figure 1 a mass pull of 60% to concentrate resulted in 90% recovery of the gold.

Although figure 1 appears to s**how EM sorting with similar performance to the Optical sensor the sort results from this study for this EM sensor are only applicable at the laboratory scale and not useful for a full scale sorter**. This is further detailed in section 5.2 of this report.

Optical sorting results have not been included in figure 1 as no pattern or visual characteristic could be defined that was usable by an optical sorter. **The specimens were not amenable to Optical sorting**.

A more detailed discussion of the benchtop sorting results for each sensor is found in the following sections of this report. All of the figures and detailed calculations shown in this report and can be found in Appendix A.

Note that the masses used to generate the curves are based on the 'as received' set of rock specimens. Therefore if the distribution of the grades within the set is not representative of the distribution of grades in the proposed mine plan and resource model, then the data within the graphs will not be representative of the average grades and recoveries likely to be encountered. The results presented in this report should only be used to determine whether the specimens are amenable to sorting. Now that it has been shown that specimens were amenable to **DEXRT** sorting future work should include preparation of a sample that is representative of the feed that an ore sorting installation would treat and a test of this material on a full scale sorter.

2 Introduction

CommodasUltrasort supplies automated ore sorting systems and expertise world-wide through our offices in North America, Australia, Germany, South Africa and Russia. CommodasUltrasort provides services ranging from benchtop sorting amenability studies to the design and installation of production scale automated sorters for the base and precious metals industries, industrial minerals, energy and gemstone miners and processors. The benchtop sorting studies can determine the characteristics that separates various minerals on the basis of their radioactivity, conductivity, magnetic susceptibility, color, texture, X-ray fluorescence spectroscopy features, near infra-red characteristics, dual energy X-ray transmission features and UV fluorescence. CommodasUltrasort uses a combination of data acquisition systems with proprietary classification algorithms that simulate full scale sorter performances, thus revealing the potential impact of sorting on mineral upgrade. Assaying the rocks and correlating the grades to measured sorting characteristics determine if any of these above-mentioned sensors can reliably segregate minerals of interest. Standard production scale ore sorting equipment can then be specified or programmed to use this information, to accept or reject individual rocks based on these differentiating characteristics.

3 Objective

The objective of this test was to take a set of specimens from the Gowest's Frankfield project, characterize the rocks in the set using **conductivity and magnetic susceptibility (EM)**, **color (Optical), X-Ray Fluorescence Spectroscopy (XRF-S)** and **Dual Energy X-ray Transmission (DEXRT)** features and determine whether these features can be used to sort the rocks to upgrade the gold values. These results can then be used to determine if there is a sorting characteristic that warrants further investigation for full scale sorting tests.

4 Description of Work

The sorting simulation was performed using the following methodology.

4.1 Sample Preparation and Measurement

1. Gowest shipped split core rock specimens to CommodasUltrasort in Quebec City. The memorandum provided by Gowest describing the specimens sent to CommodasUltrasort is included in Appendix E of this report.
    a. 100 rocks were chosen at random from the specimens for the study.

2. Features of the 100 rocks were then acquired with the following sensors:
    a. DEXRT - A dual energy Heimann 6040i x-ray scanner was used to acquire the x-ray transmission (DEXRT) characteristics for each rock. The system has been modified to provide the raw x-ray image data for analysis.
    b. OPTICAL - A benchtop optical sorter was used to acquire an image of each rock. This benchtop image acquisition system has been designed to reproduce the physical resolution and lighting environment typical of a full scale optical sorter.
    c. EM – The conductivity and magnetic susceptibility response was acquired with the GDD MPP EM2S+ probe.
    d. XRF-S – The results were acquired with our benchtop test rig system configured to simulate a full scale XRF-S sorter.

3. The 100 rocks were assayed at ALS Global in North Vancouver, BC.

4. The recovery curves for the theoretically perfect sort for gold was plotted and the data from each sensor was analyzed with the goal of matching the theoretically perfect recovery curve. The theoretically perfect recovery curve was calculated by ordering the rocks from highest grade to lowest. Grade in this study consistently refers to gold content unless otherwise specified.

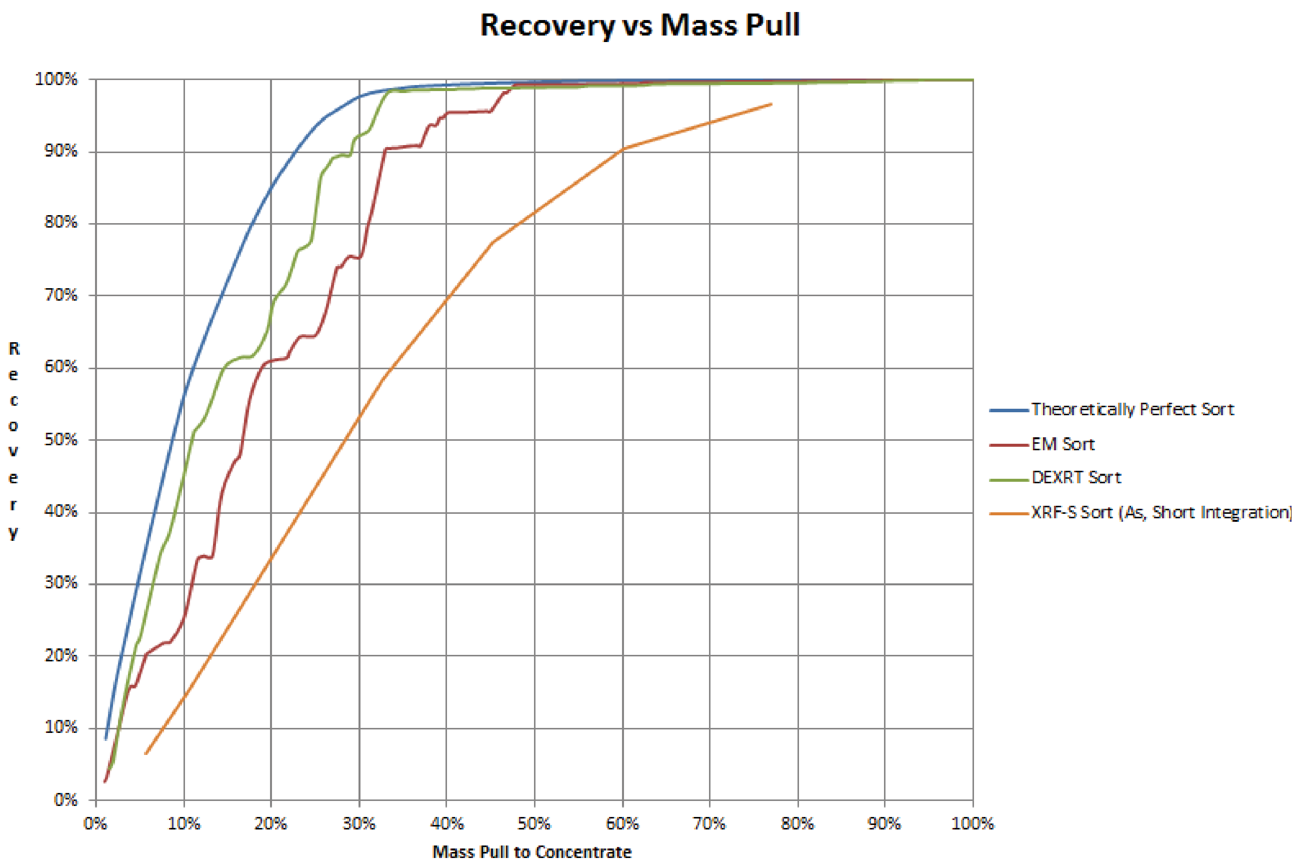
5 Discussion and Results

All of the graphs and calculations presented in this section can be found in Appendix A as both graphs and tables. The complete assay results are also included in Appendix A and Appendix D. Figure 2 shows the recovery curves for each of the sensors as well as the perfect recovery curve.

It is important to note that the masses used to generate the curves are based on the 'as received' sample of rocks. Therefore if the distribution of the grades within the sample is not representative of the distribution of grades in the proposed mine plan and resource model, then the data within the graphs will not be representative of the grades and recoveries likely to be encountered.

As noted in the Executive Summary the **DEXRT sorting results are close to the theoretically perfect sort. The specimens were also amenable to sorting by XRF-S, although not to the degree exhibited in the DEXRT sorting.** There are no Optical sorting results to report as no relation between visual features and gold grade were found. Although the EM recovery curve in figure 2 appears to show good sorting results, **the specimens were amenable to sorting by the EM sensor used in this study at laboratory bench scale. However the EM sort results achieved with this sensor will not translate to a full scale sorter**. The sorting results for each sensor are detailed below.

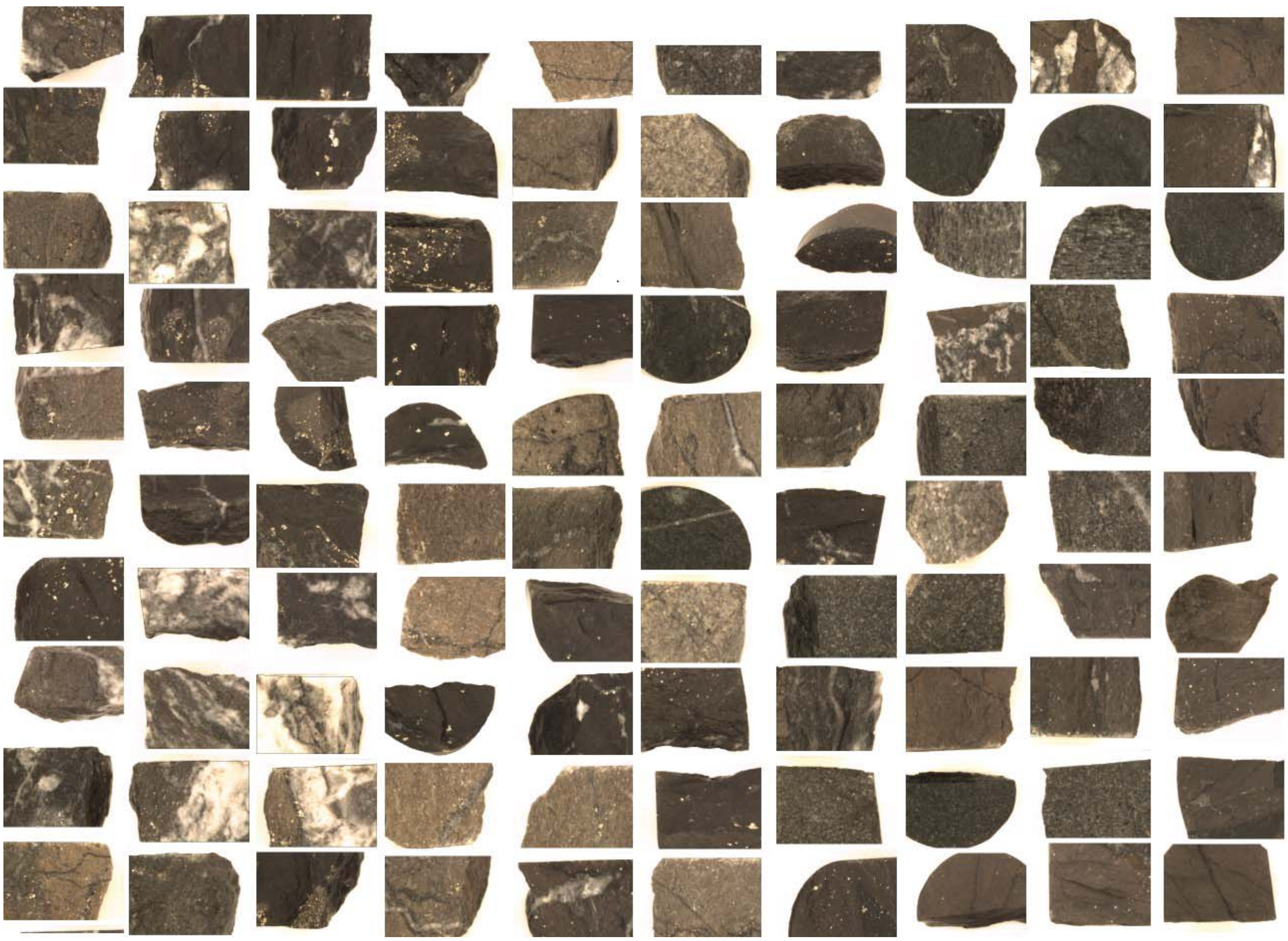
Figure 2: Recovery Curves for Gold for a "Perfect Sort", EM, DEXRT, Optical and XRF-S Sort



5.1 Visible Spectrum Optical Sorting (Optical)

High resolution photographs were taken of all samples. The photos were analysed using proprietary CommodasUltrasort image processing software (PACT). The images of all of the rocks were arranged in descending gold grade and analysed for a colour feature with a relation to the grade. No correlation between gold grade and features was found. Therefore the specimens are **not amenable to Optical sorting**.

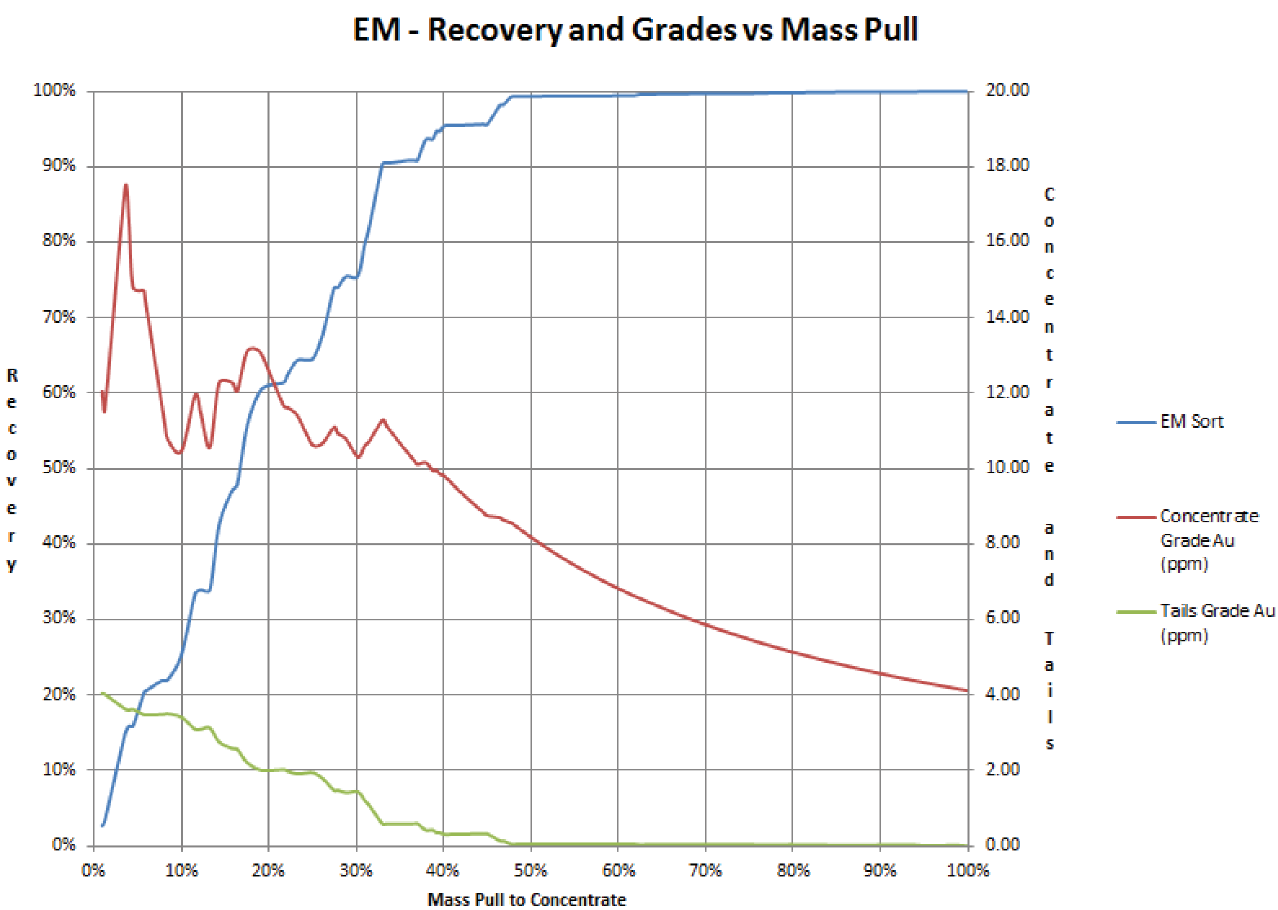
Figure 3: Rocks ordered from highest to lowest grade with highest grade rock being in the top left of the first column and the lowest being the bottom rock in the tenth column.



5.2 Conductivity and Magnetic Susceptibility Sorting (EM)

Conductivity and magnetic susceptibility measurements were taken for each rock using the GDD MPP-EM2S+ multi-frequency sensor. There was no conductivity response measurable with the instrument used for this test, therefore the rocks were ordered only by their increasing magnetic susceptibility. The sensor used in this study was able to differentiate the specimens by their magnetic susceptibility, however the magnetic susceptibility values measured ranged from 0.0002 SI units up to 0.0105 SI units. These values would be difficult to measure on a full scale sorter therefore it does not appear that the specimens are amenable to EM sorting except at the laboratory benchtop scale. For the sake of completeness the results for the EM sort when the specimens are ordered by increasing magnetic susceptibility is shown in figure 4.

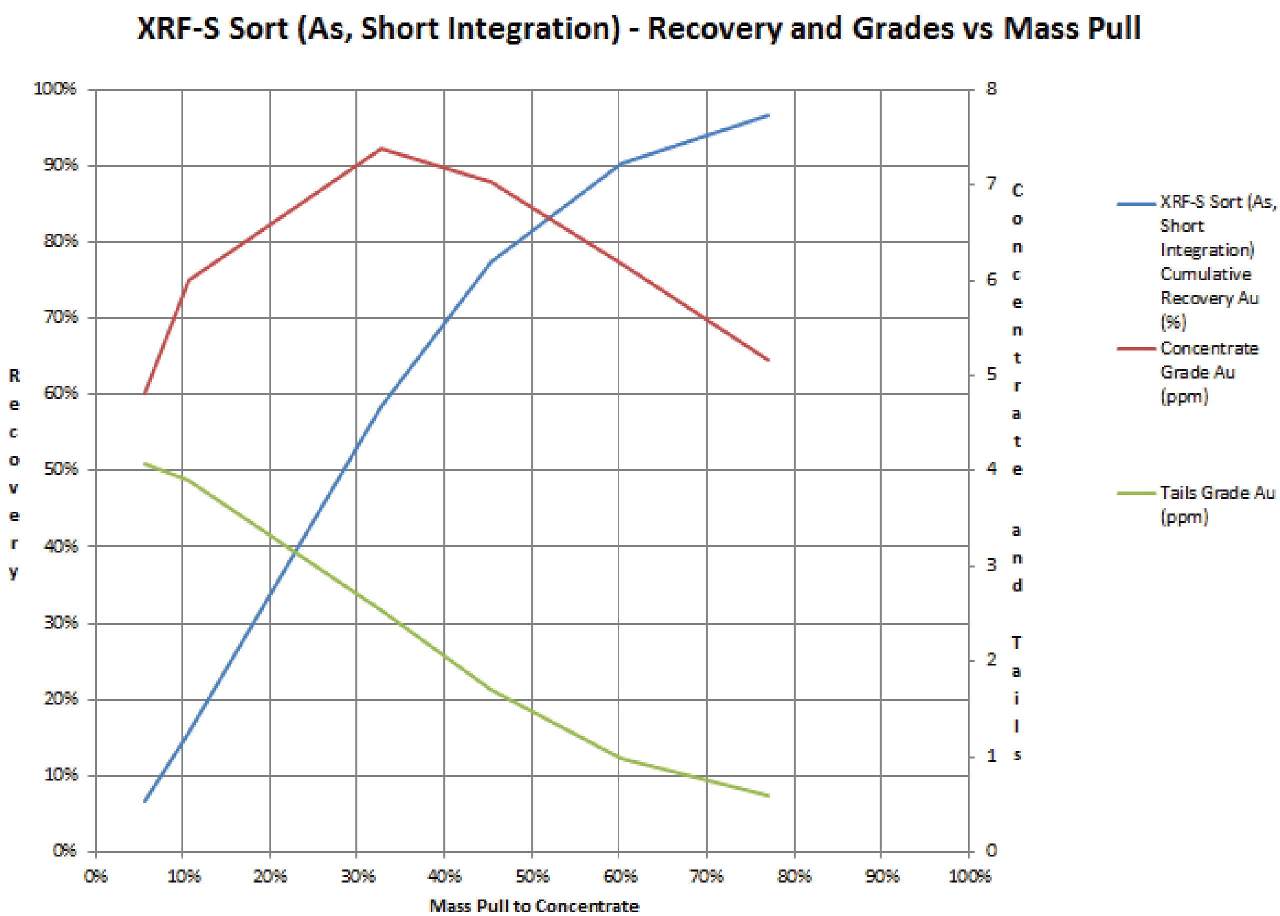
Figure 4: Recovery, concentrate and tails grades for EM sorting



5.3 X-Ray Fluorescence Spectroscopy Sorting (XRF-S)

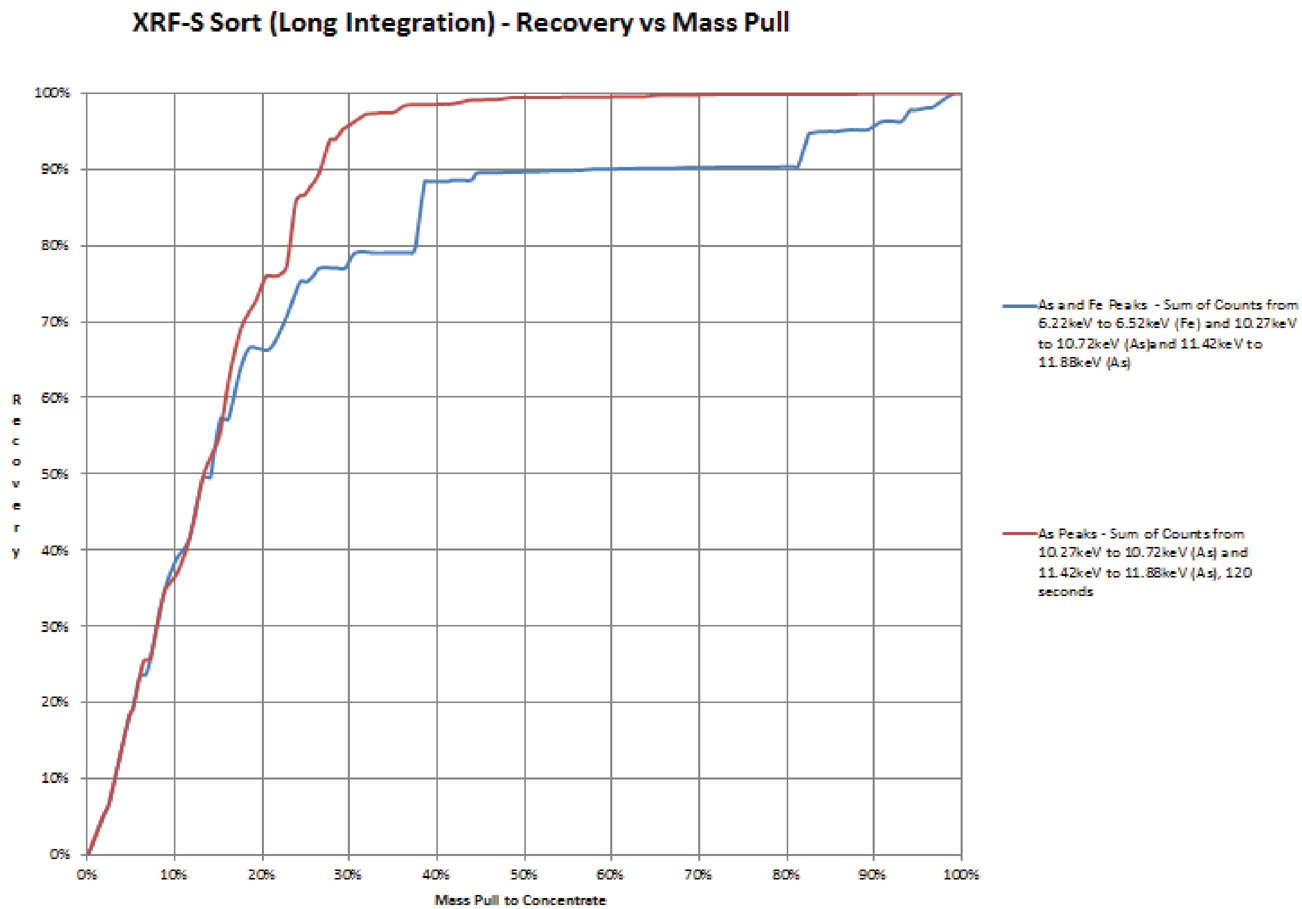
X-Ray fluorescence spectroscopy measurements were taken with a benchtop unit that imitates the sensor and source setup of a full scale XRF-S sorter. The parameters of the X-ray source and sensors used for this experiment are detailed on the tab "XRF-S Spectra - Raw Data" in the spreadsheet in Appendix A. Figure 5 shows the Au grade and recovery curves for the best XRF-S sort. As shown in figure 5 a mass pull of 60% to concentrate would result in a recovery of 90% of the gold. For this same mass pull the concentrate would grade 6.8ppm gold with tails of 0.99 ppm gold. The graph shows the Au recoveries and grades, however the sensor was not able to sort directly for gold grade. Rather because the gold is associated with arsenopyrite the sensor was configured to sort for arsenic which is a proxy for gold. Gowest suggested using this arsenopyrite to gold relationship for the XRF-S sorting test.

Figure 5: Recovery, concentrate and tails grades for XRF-S sorting using arsenic emission peaks (from 10.27keV to 10.72keV and 11.42keV to 11.88keV)



Several different analyses were done to determine the best XRF-S spectra to use for the sort. Figure 6 shows the comparative results for two of these analyses: (1) Arsenic peaks only and (2) Arsenic and iron peaks. Figure 6 illustrates that the sorting results are improved when only the arsenic peaks are used to sort for gold content.

Figure 6: Comparison of recovery vs mass pull for XRF-S sorting at different energy levels



Because X-ray fluorescence spectroscopy only covers a limited surface in each sample, the assumption is that that there is a linear relationship between the amount of arsenic measured on the surface of the specimen and the amount of arsenic inside the specimen. Figure 7 shows the relationship between arsenic measured on the surface of the sample and the total amount of arsenic in the rock.

Figure 7: XRF-S measurements for counts/s at the arsenic fluorescence peaks vs arsenic content of each specimen

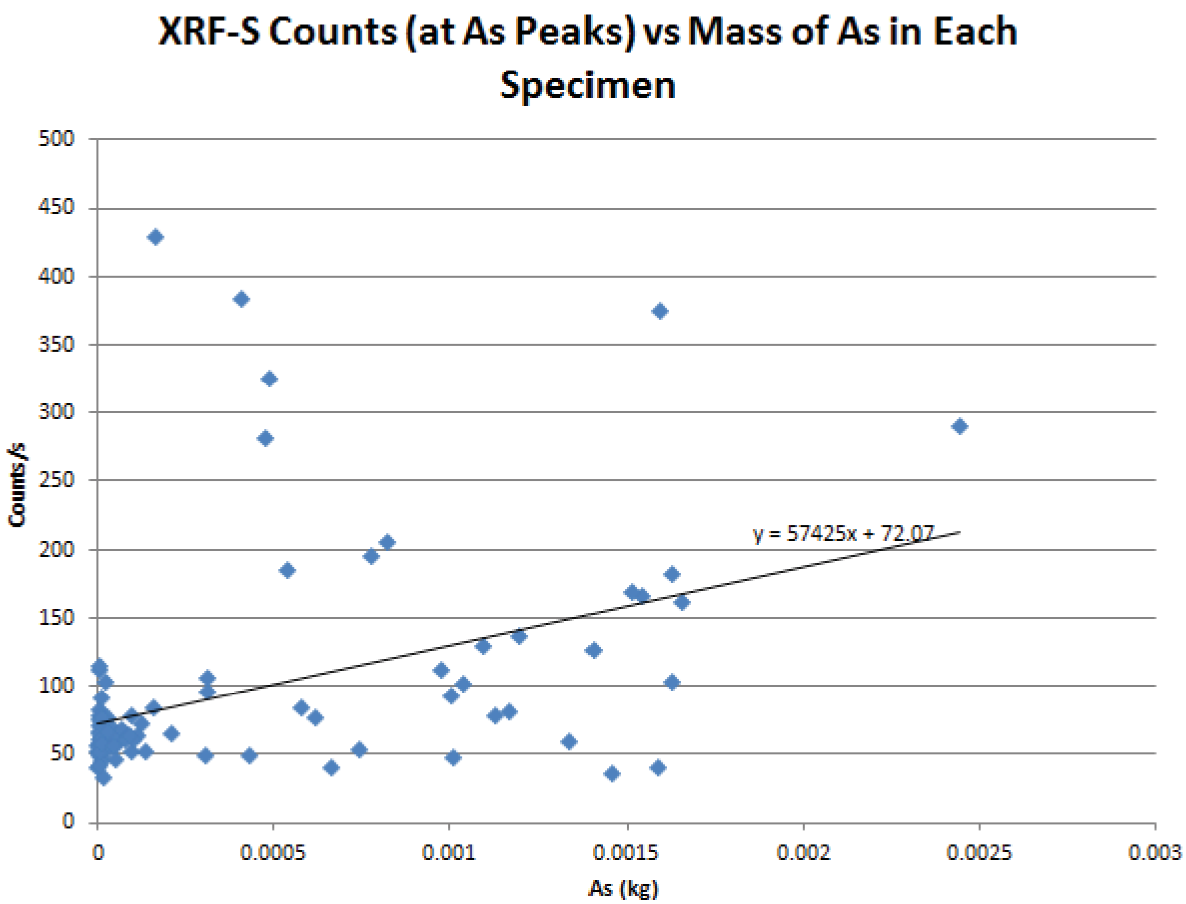
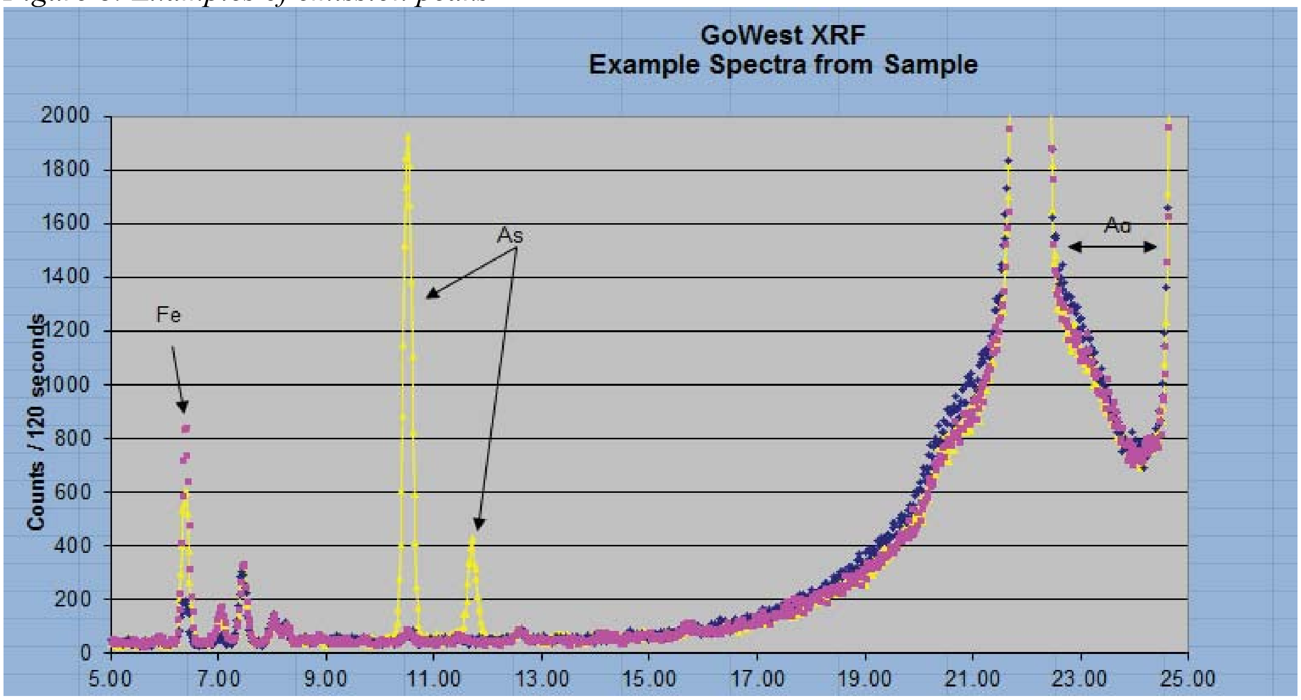


Figure 8: Examples of emission peaks



The relationship shown in figure 7 provides a means to model and predict the estimated number of counts per second that a full scale sorter is expected to measure on passing rocks for the whole range of arsenic grades. This number is adjusted to take into account the standard 0.2 second of total rock exposure to sensor array found on most sorters (based on typical sorter belt speed and sensor size).

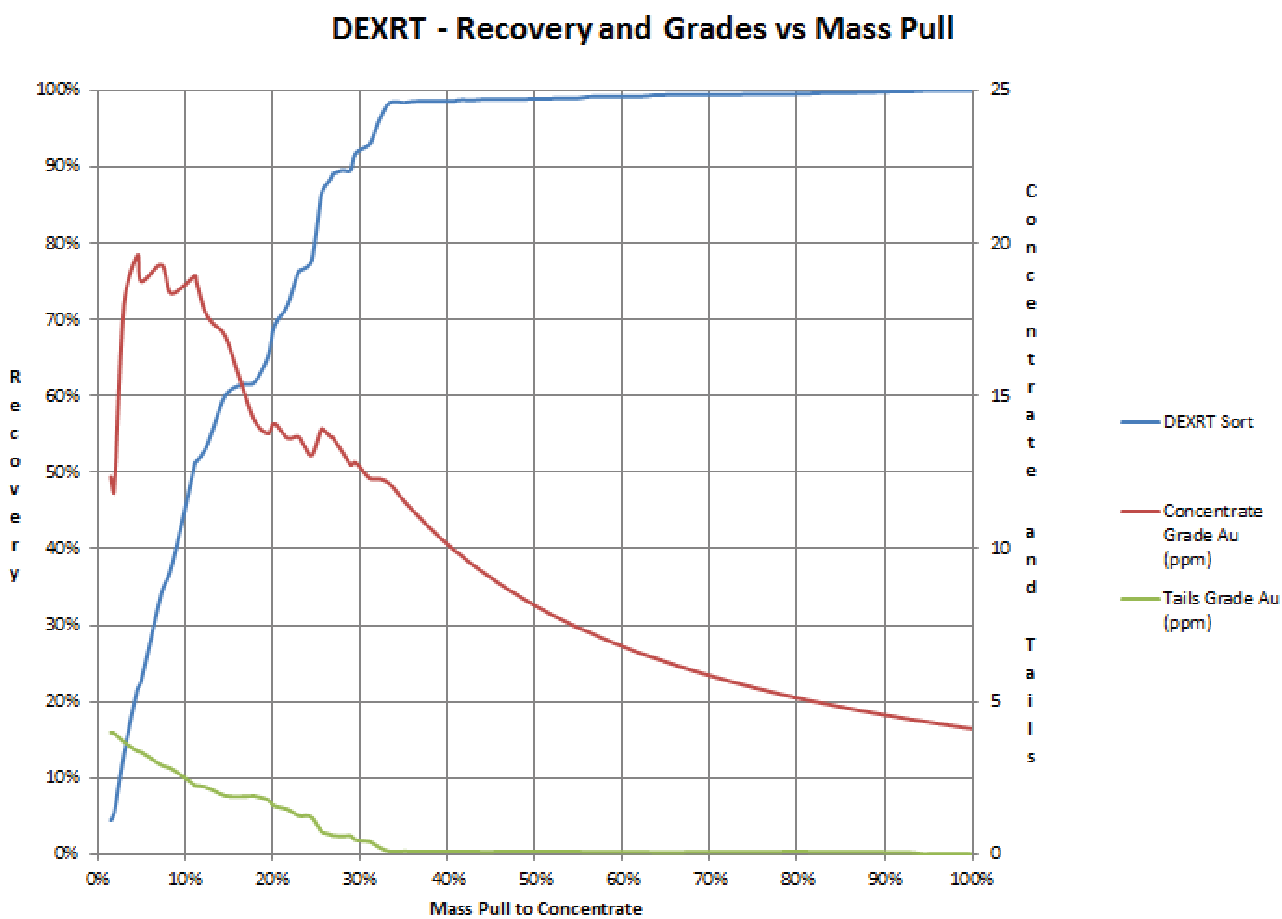
A series of sorter set points, which dictate the sorter to send all rocks with counts above these cut-off points to the concentrate, were arbitrarily selected to simulate the resulting recoveries and concentrate grades from the sample rocks. The detailed table is available in Appendix A. In this table, the predicted counts for each rock (detected by a hypothetical sorter) were sorted from highest to lowest. Also, a statistical error associated with each predicted count was determined, and a corresponding Gaussian probability function was calculated to determine the average probability of each rock passing the different selected cut-off points of the sorter. To better simulate the operation of a sorter, the cut-off points themselves were corrected to compensate for rock surface area (presumed to be proportional to mass in this study). For a study where physically larger rocks are available a function for surface area vs mass can be determined to account for oddly shaped fragmentation.

It is worth noting that a benchtop XRF-S test rig is an idealized environment which will normally give better sorting results than a full scale sorter, but there are several optimizations that could be made on a full scale sorter to optimize for the arsenic readings. Now that it is known that the 10.27keV to 10.72keV and 11.42keV to 11.88keV energy ranges, which are the emission peaks for arsenic, are representative of the gold content certain parameters could be changed to reduce the measurement error. For example, reducing the spectral resolution which would in turn increase the number of counts detected may increase sensitivity to arsenic and, by extension, to gold.

5.4 Dual Energy X-Ray Transmission Sorting (DEXRT)

This set of specimens was amenable to DEXRT sorting. Upgrading was possible with DEXRT sorting and in fact the **DEXRT sorting results were very close to the theoretically perfect sort**. Figure 9 shows the recovery and grade curves for the specimens if sorted by DEXRT. For example, the theoretically perfect sort could pull 33% of the mass to concentrate and recover 99% of the gold. A DEXRT sort of these specimens with a similar mass pull of 33% to the sorter concentrate would result in a recovery of 98%. At this 33% mass pull to the DEXRT sorter concentrate the concentrate grade would be 12.1ppm of gold with tails of 0.1 ppm of gold.

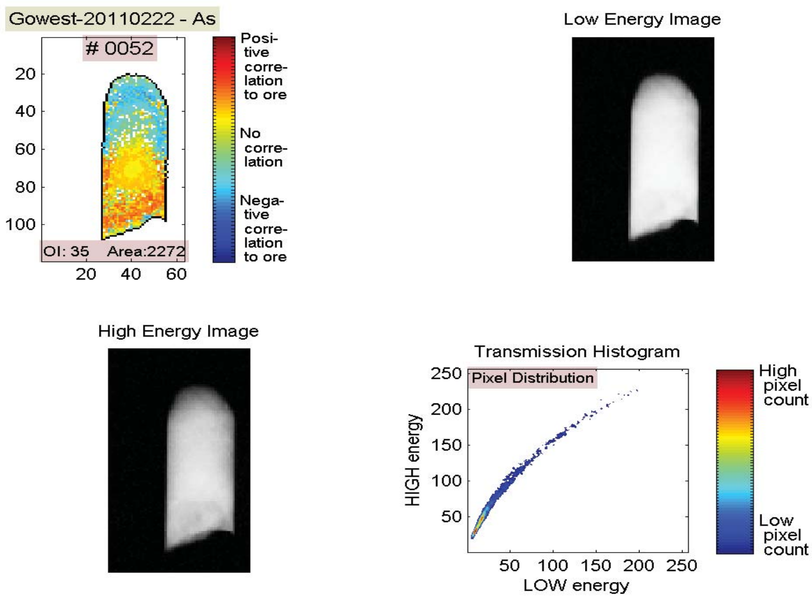
Figure 9: Recovery, concentrate and tails grades for DEXRT sorting



To create the DEXRT recovery curve shown in figure 9 the rocks were analyzed and assigned a DEXRT "Ore Index". An explanation of the DEXRT ore index and some examples of DEXRT images (raw and processed) are shown on the following pages.

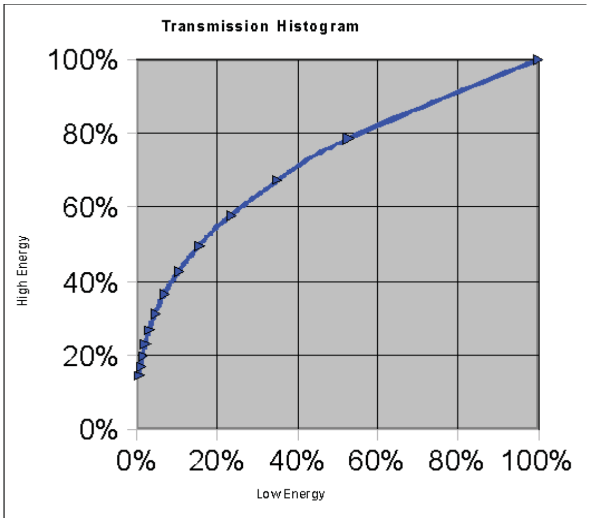
Figure 10 shows one example of the high and low energy images that were captured for each of the 100 rocks with a dual energy x-ray transmission sensor. Fort each rock a 2D Transmission Histogram was created (lower right) from the low and high energy images (upper right and lower left respectively). It shows the distribution of all the High/Low Energy combinations that are found inside the rock image. The interpretation of the 2D Transmission Histogram is explained below.

Figure 10: Example of a Low/High Energy distribution (2D Transmission Histogram) for a given rock

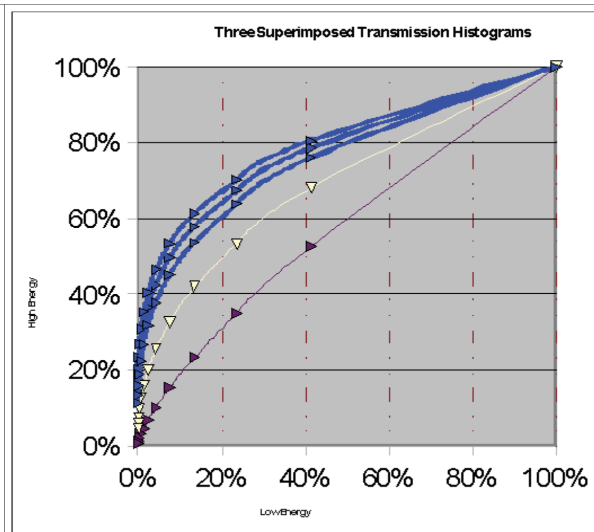


To help illustrate the use of the 2D Transmission Histogram in Figure 10, idealized curves of homogeneous materials are shown below in Figure 11. Figure 11 shows (a) the idealized histogram for a wedge shaped block of homogeneous material and (b) the idealized curves of different perfectly homogeneous wedge shaped materials. The lower left corner of the graph indicates low transmission (thick material) whereas the upper-right corner signals high transmission (thin material). The shape of the curve is what reveals the atomic density of the material. Figure 11 (b) shows several superimposed histograms for homogeneous materials with different average atomic numbers. As the average atomic number of the material increases the curve will shift up and left. The inverse is true for decreasing atomic number. In other words, for a given x-ray signal wavelength (or energy), the transmission level through an object will depend on its composition and thickness. However, when using dual-energy x-ray transmission, the relationship between the high-energy and low-energy x-ray signals is independent of thickness over a specific range, and reflects the average atomic density only.

Figure 11: Idealized Transmission Histograms



(a)

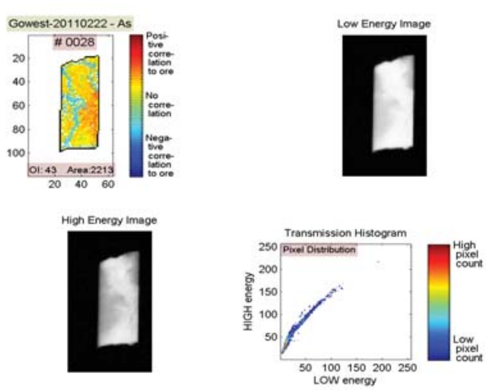


(b)

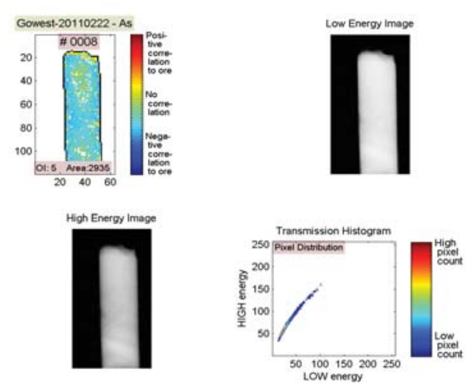
The 2D transmission histograms for each rock can be mapped against a weighted reference histogram for the sensor characteristics. The weighted reference histogram is also referred to as the Ore Index Reference Map, which is created from the set of histograms and assay results for the sample. When the 2D histogram for an individual rock is mapped against the reference histogram an “Ore Index” for each rock can be calculated.

Figure 12 shows Ore Indices mapped to images of rocks. The upper left quadrant of figure 12 (a) shows a rock with a high Ore Index, while the upper left quadrant of figure 12 (b) shows a rock with a low Ore Index. In the Ore Index images red indicates a higher likelihood of rocks containing elements with higher atomic numbers (for example Co, Zn, Ni, Fe, Cu, As,,...).

Figure 12: Example of Ore Index for a given rock



(a) Rock with high Ore Index (assumed to be high in mineralization)



(b) Rock with low Ore Index (assumed to be low in mineralization)

A benchtop DEXRT sorter is an idealized environment which will usually give better sorting results than a full scale sorter. Moreover, it should be noted that the calculation of the total Ore Index is subject to a few sources of error:

1. The Ore Index Reference Map used in this study to calculate the Ore Index for each rock was built using data from a small 100 rock set.
2. The edges of the rocks were ignored due to irregular edge effects produced by the x-ray system, making it difficult to reliably interpret density values in these specific areas of the rocks. Therefore the edges of the rocks are eroded in the image and ignored.
3. The sensitivity of the x-ray sensors may fluctuate over time, yielding different intensities for the same rock. These fluctuations were however minimal in this data since all the images within a set were acquired over a short time. Additionally a calibration was applied to every image to compensate for the changes in sensor characteristics.

Appendix A – Sort Results

The contents of this appendix can be found in the file:

2011.09.12_GoWest_Frankfield_AppendixA.xls

Appendix B – Complete set of Optical and DEXRT Images

The contents of this appendix can be found in the file:

2011.09.12_GoWest_Frankfield_AppendixB.zip

Appendix C - Glossary

Colour Histogram

Two-dimensional graph showing the distribution of the pixel colour values in a digital image. Each vertical-axis value or bin in a histogram represents the percentage of image pixels corresponding to the horizontal-axis intensity level [1 (lowest / darkest) – 256 (highest / brightest)] for that particular color component.

The color histogram of a rock image illustrates the percentage of pixels in the rock image at each different intensity value found in that image. For an 8-bit gray-scale image, there are 256 different possible intensities, and the histogram graphic has 256 bins showing the number of pixels with each gray-scale value. Color images are decomposed into three gray-scale images, each comprising 256 intensity values, and each representing one of three color components. For color images, the histograms for the three color components are often combined into one graphic with three superimposed curves.

Histograms are useful color descriptors. They suffer the drawback of containing a great deal of information, some useful and some redundant. Once classes of minerals have been generated in the training process, it is important to look back and determine which portions of the histogram contribute the most to differentiating the rocks.

Color Models

A color image can be viewed as a ‘stack’ of three component images. That is, each color image pixel is formed of three values, referring to a specific color attribute, depending on the color model.

- RGB color space : Each color pixel is a triplet corresponding to the red, green and blue components.
- HSI : In the HSI color model, H (hue) is the attribute describing pure color (tint), whereas S (saturation) gives a measure of how much a color is diluted by white light. I refers to the intensity component.
- YCbCr : Luminance (intensity) information is represented by Y, while colour (chromatic) information is stored as blue chroma Cb and red chroma Cr.

In the HIS and YCbCr color models, intensity is described separately and is statistically uncorrelated to the color (chromatic) information.

Conductivity (electrical)

Is a measure of a material's ability to conduct an electric current. Measured in mhos/m.

Counts (gamma)

Is a measure of the individual gamma radiation emissions measured with a scintillation counter (also called a scintillator.

DEXRT – Dual Energy X-Ray Transmission

Dual-energy X-ray transmission is an imaging technique, which consists of combining two images acquired at two distinct X-ray energies. The images can be analyzed to obtain both density and atomic number, thus to provide information about material composition, or at least to improve image contrast.

Lookup Table (Class Lookup Table)

Table showing, for a given sort, the class assignment of each rock.

Magnetic Susceptibility

A number that characterizes the intensity of the magnetization of a substance in the presence of an ambient magnetic field. Measured in SI or CGS dimensionless units.

MPP HF Response

A value measured by the MPP-EM2S+ probe. Indicates the amount of conductive material in the sample being investigated.

Near Infrared Spectroscopy (NIR)

A spectroscopy method for analysis of the characteristic spectra in the near infrared region of light that can be used to identify specific molecular structures.

Scintillator (or Scintillation Counter)

The sensor, called a scintillator, consists of a transparent crystal that fluoresces when struck by ionizing radiation. A photomultiplier tube measures the light from the crystal. The photomultiplier tube is attached to an amplifier and other electronic equipment to count the signals produced by the photomultiplier.

Sorter

Mechanized system that continuously separates ore into two or more streams of differing characteristics by applying sensors to analyze the visual characteristics of individual rocks. Rocks are initially fed into a surge hopper, from which they travel down a conveyor belt to a vibratory feeder that spreads the rocks into a single layer of randomly spaced particles. In the case of the an optical sorter, rocks reach the sorter and free fall off of a chute, passing through a sensor viewing zone. Each individual falling particle is scanned by a line-scan camera at least two thousands times per second, and a complete picture of it is built up and stored as it moves through the viewing area. This method of acquiring images is similar to scanners and photocopy machines except that, in the case of sorters, the objects (rock particles) are moving in front of a fixed camera system (instead of the camera and lighting moving across the object). A microprocessor then analyzes the exposed surface of each rock, and makes a decision whether or not to reject the particle from the stream. An array of high-speed compressed air jets are used to reject the individual rocks. **There are other types of sorters, but with sensors that measure physical characteristics other than colour, such as radiometric, conductivity, x-ray fluorescence spectroscopy, near infra-red spectroscopy, magnetic susceptibility or atomic density (dual energy x-ray transmission sorting or DEXRT)**

Supervised Classification

Most commonly, supervised learning generates a model that maps input objects (rocks) to desired outputs (classes or composites). Supervised learning requires *a priori* knowledge of the classes and the classification of the input objects. First the classes of the input set are defined, then the supervised classifier is taught to recognize these classes. The accuracy and efficiency of the supervised classifier can then be tested with another set of known inputs (rocks)

For example, in a supervised classification ore sorting test, every rock would be photographed and individually assayed. Next the images would be analyzed to determine which visual characteristics could be used to determine the grade of the rock. The disadvantage of using a supervised classification in a scoping study is, of course, that it is much more onerous because each individual rock needs to be assigned a grade or a class prior to analyzing the images of the rocks.

Texture Information - Histogram Statistics :

One simple approach to describing rock texture is to recover and analyze histograms (pixel value distributions) of the rock images in all of the color space components. Histogram statistics, as the name suggest, rely exclusively on the statistical properties of the overall distribution of pixel intensities and colors, and gives us an idea of the rock overall coarseness, granulation and regularity. Each color component histogram contains 256 gray-scale values, but retrieving its main features reduces the amount of information from 256 to 6 texture descriptors which we refer to as statistical texture descriptors :

- Mean (*m*) : A measure of average color or intensity
- Standard deviation (*CNT*) : A measure of average contrast
- Roughness (*R*) : Relative roughness / smoothness of a rock surface
- Third moment (*u-3*) : Measures the skewness of a histogram
- Uniformity (*U*) : Measures uniformity
- Entropy (*e*) : A measure of randomness

Texture Information - Wavelet-Based :

The weakness of histogram statistics lies in the fact that they do not take into account the spatial relationship between pixels. A grainy salt-and-pepper rock surface may yield the same statistical measures as another rock with large inclusions, provided they have the same range of colours. Wavelet decompositions allow the retrieval of the spatial frequency information from a rock image, discriminating between periodic and non-periodic texture patterns. In multi-resolution analysis (through the use of wavelets), one is able to retrieve each spatial frequency contribution to the image texture at each image point. This information is used to generate the wavelet-based texture descriptors.

Unsupervised Classification

Method of machine learning that creates a model to fit observations. The machine is given a set of inputs (rock images), but no set of outputs (assignments to different classes) from which it could learn to make associations. Instead, the system creates a model and subsequently a set of classes out of the inputs, based on their inherent common statistical characteristics. It can be said to partition, or cluster, a set of inputs into 'natural' categories of shared identifying features.

Throughout this study, unsupervised learning refers to the partitioning (clustering) of a sample of rocks, into classes of similar visual traits, without prior specification on what particular color, texture, brightness, size and shape settings or thresholds to base the decision on. The machine automatically separates the sample into classes based on the most striking visual distinctions, or more technically speaking, on those visual descriptors with the highest statistical variance. Different sorts (class assignments) are generated by limiting the range of visual descriptors that the machine can derive its results from. For example, one sort may be based on texture only, while another may rely on colour and shape attributes, etc.

Unsupervised learning is the method used in this study. The advantages of using such an approach are the following :

> Flexibility : Material can be classified with no prior knowledge of their mineral grade or of the mineral processing to be applied to the ore. Thus different processing scenarios can be examined, using the proper sort (out of the many class assignments generated) that best applies to each scenario explored. For example, a sort based on color characteristics may offer the best overall affiliation with mineral grade, but texture-based sorting would best apply to a specific mineral processing method.
>
> Simplicity : There is no need to parametrize and fine-tune a setup, as is the case with commercial ore sorters, which require prior knowledge of how rocks must be classified. Because rocks are classified into composites without *a priori* knowledge of the metallurgical or mineralogical properties, it provides a rapid and inexpensive method for exploring ore sorting at the scoping stage of a project.

UVC

Shortwave ultraviolet light (approximately 254nm)

X-Ray Fluorescence Spectroscopy

The analysis of the spectra of the characteristic secondary X-rays that are emitted when a material is bombarded to with X-ray or gamma rays to determine the elemental composition of the bombarded material.

Appendix D – Assay Certificate

This appendix contains the certificate of analysis from the assays of the specimens used in this study.

The contents of this appendix can be found in the file:

2011.09.12_GoWest_Frankfield_AppendixD.zip

Appendix E – Sample Selection Memorandum

This appendix contains the memorandum provided by Gowest describing the specimens sent to CommodasUltrasort for this study..

The contents of this appendix can be found in the file:

2011.09.12_GoWest_Frankfield_AppendixE.zip

APPENDIX B
ROCK MECHANICS

Project No. 169514568 **01 April 2015**

PREPARED FOR:

Gowest

CONCERNING:

Bradshaw Gold Deposit Prefeasibility Study

Bulk Sample Stopes Modelling Review

PREPARED BY:
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1760 Regent Street
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Canada

REVISION NOTES

Revision	Date	Description	Name
A	24-Mar-2015	Document Initiated	D. Thibodeau
A1	25-Mar-2015	Internal Review	D. Thibodeau
A2	01-Apr-2015	Internal Comments Incorporated	D. Thibodeau
B	01-Apr-2015	Issued for Client Review	D. Thibodeau
B1		Client Comments Incorporated	
C		Issue for Client Approval	
0		Issued as Final	

REVISION APPROVAL

Revision	Designation	Name	Date	Signature
A2	Originator	Denis Thibodeau		
A2	Project Manager	N. Del Bel Belluz	01-Apr-2015	Approval on File

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List of Tables and Figures

1.0 INTRODUCTION

Numerical modelling of the proposed bulk sample extraction sequence was completed for the Bradshaw Deposit. The purpose of the sequence review was to understand the impact on the deposit stability by leaving open stopes until full production is started.

2.0 NUMERICAL MODELLING

Modelling parameters for the Meta Volcanic rock were obtained from the core sample testing by Stantec's laboratory in Halifax. The midpoint between minimum and maximum (Rock Mass Rating (RMR)values established in the Stantec report *Bradshaw Gold Deposit Geomechanical Stope and Pillar Review, January 2015 was used* as the Geological Strength Index, 63 for the Meta Volcanic. To obtain Hoek and Brown number m and s values, the meta volcanic was associated to Andesite described at Kidd Creek to obtain a Geological Strength Index to be used in the Rocscience software rock data database. Table 2.1provides the input parameters for the numerical modelling. Figure 2.1 gives the results obtained for the meta volcanic using RocData software.

Table 2.1 Rock Mass Properties Used for Numerical Modelling

Property	Hanging Wall Meta Volcanic	Ore (same as Meta Volcanic)
Unconfined Compressive Strength (MPa)	122	122
Hoek and Brown m value	2.67	2.67
Hoek and Brown s value	0.0164	0.0164
Young's Modulus (Mpa)	72,300	72,300
Poisson Ratio	0.34	0.34

In the absence of measurement, the in situ stress equation was obtained from Maloney et al. (2006). Table 2.2 lists the stress regime used for the numerical modelling. In Table 2.2, Z is the depth below surface in metres.

Figure 2.1 Modelling Parameters for the Meta Volcanic

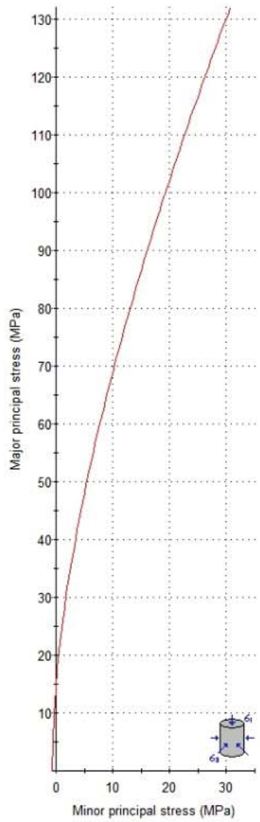
Analysis of Rock/Soil Strength using RocData

Hoek-Brown Classification
intact uniaxial comp. strength (sigci) = 122.7 MPa
GSI = 63 mi = 10 Disturbance factor = 0
intact modulus (Ei) = 72300 MPa

Hoek-Brown Criterion
mb = 2.668 s = 0.0164 a = 0.502

Mohr-Coulomb Fit
cohesion = 7.601 MPa friction angle = 34.32 deg

Rock Mass Parameters
tensile strength = -0.754 MPa
uniaxial compressive strength = 15.561 MPa
global strength = 28.783 MPa
modulus of deformation = 42495.22 MPa



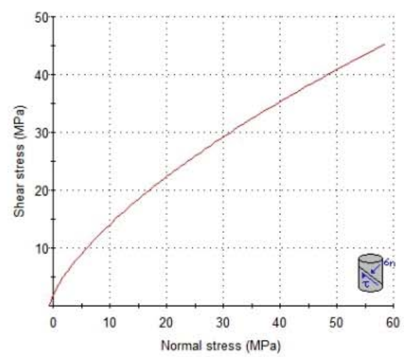


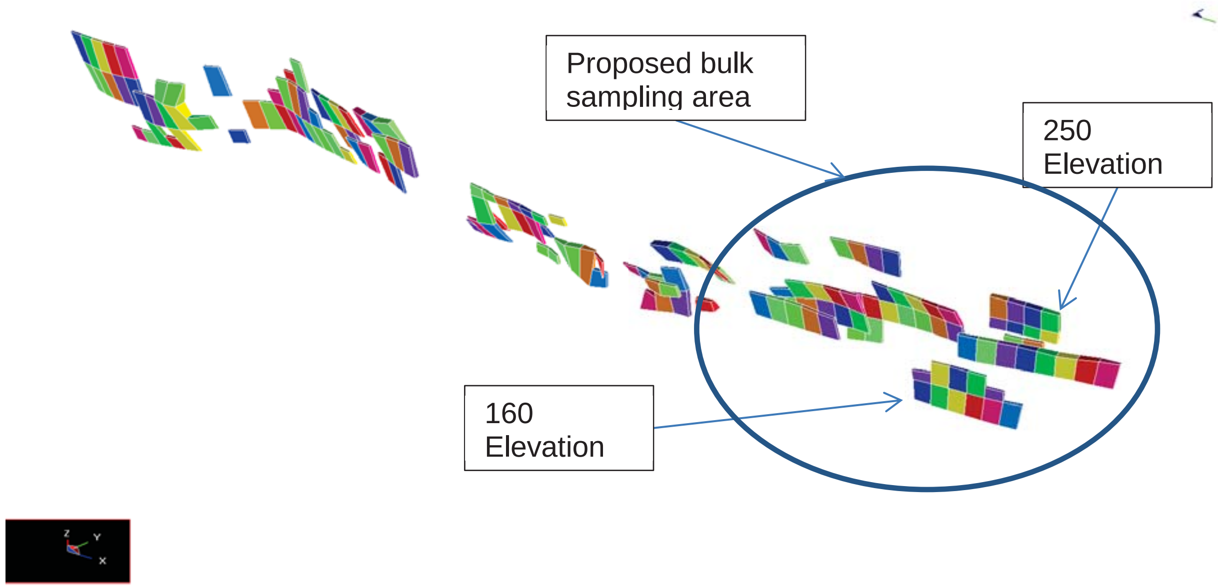
Table 2.2 In Situ Stress Regime

Principal Stress Component	Value (MPa)	Azimuth	Plunge
Major	5.768 + 0.071*Z	344	02°
Intermediate	3.2787 + 0.043*Z	74°	02°
Minor	0.034*Z	292°	85°

Modelled Geometry and Parameters

Numerical modelling was completed to assess the proposed impact of mining stopes for bulk sampling. Figures 2.2 illustrates an isometric view of the model.

Figure 2.2 Isometric View of the Bradshaw Deposit Top Levels with the Proposed Bulk Sampling Area



2.1 Results

Numerical model indicates that mining of the bulk sample will generates tension zone (low confinement) both in the hanging wall and the footwall as shown on Figure 2.3. This indicates potential footwall and or hanging wall unraveling in the stope with time. Given the small dimension of the stope unravelling (if it occurs) will be limited to a small volume around the openings. The swell factor of the rock will eventually provide confinement and stop the caving process due to lack of void. Based on modelling results, unravelling if any is not expected to affect the rock mass above the top-sill elevation as shown on Figures 2.4 to 2.6.

Figure 2.3 Minor Principal Stress at 235 Level all Bulk Sample Stope Mined

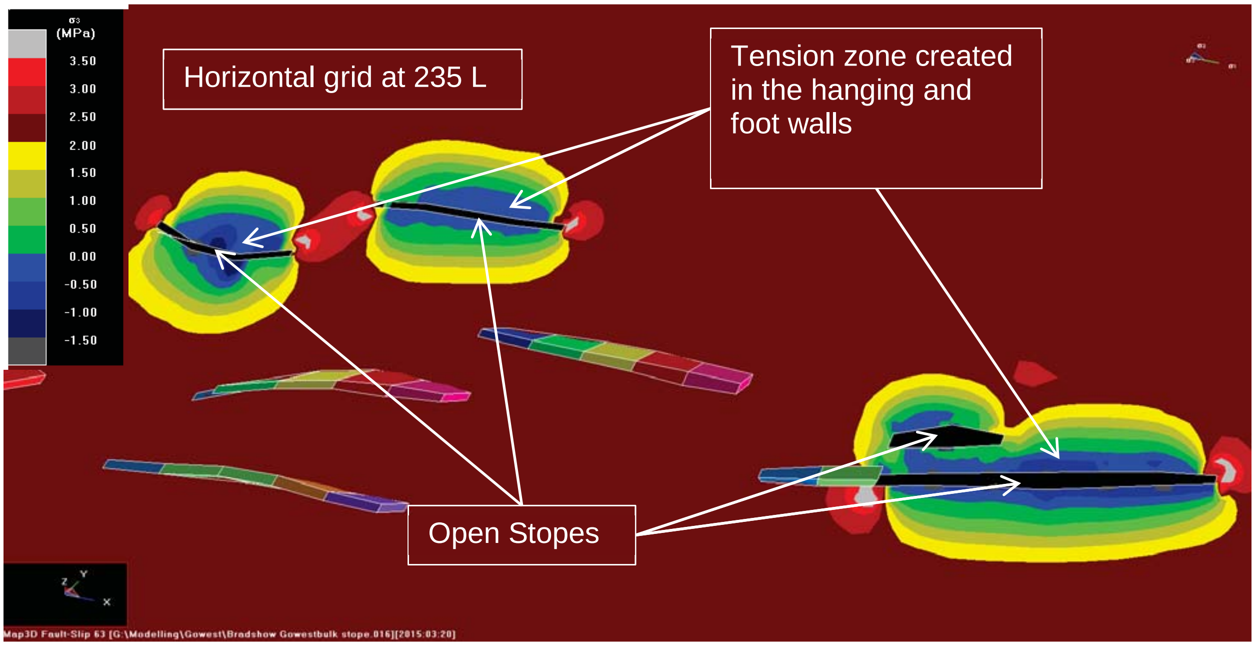


Figure 2.4 Minor Principal Stress on Vertical Section at East End of Bulk Sample Zone

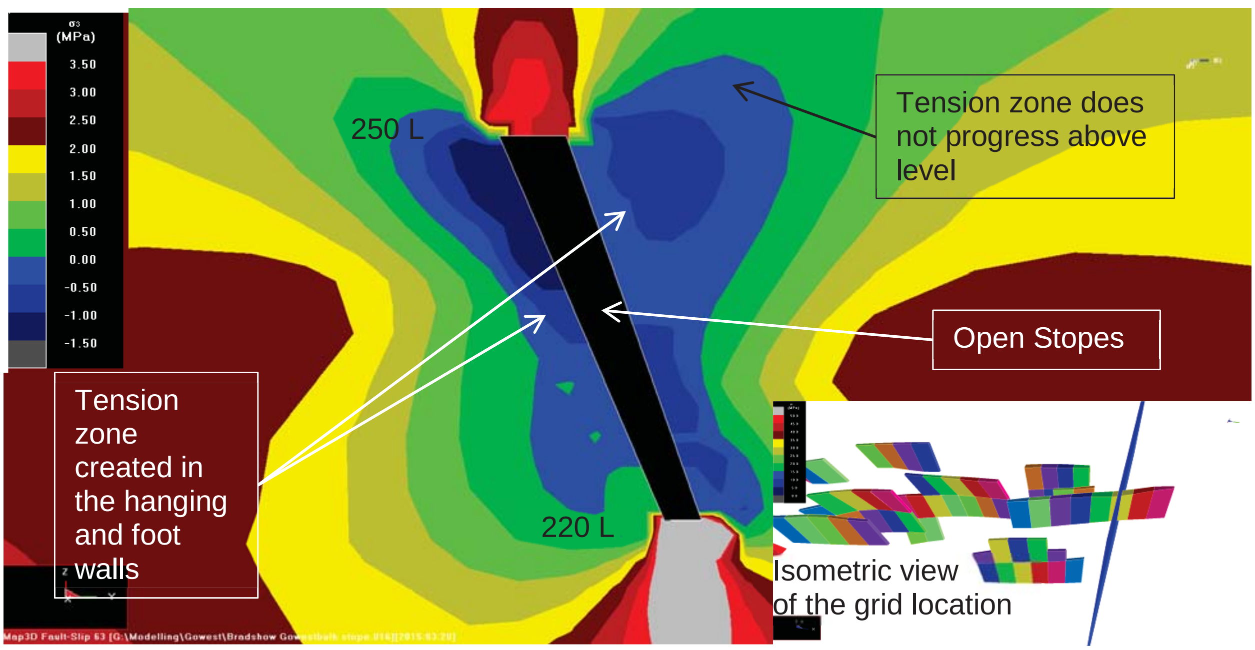


Figure 2.5 Minor Principal Stress on Vertical Section at Center of Bulk Sample Zone

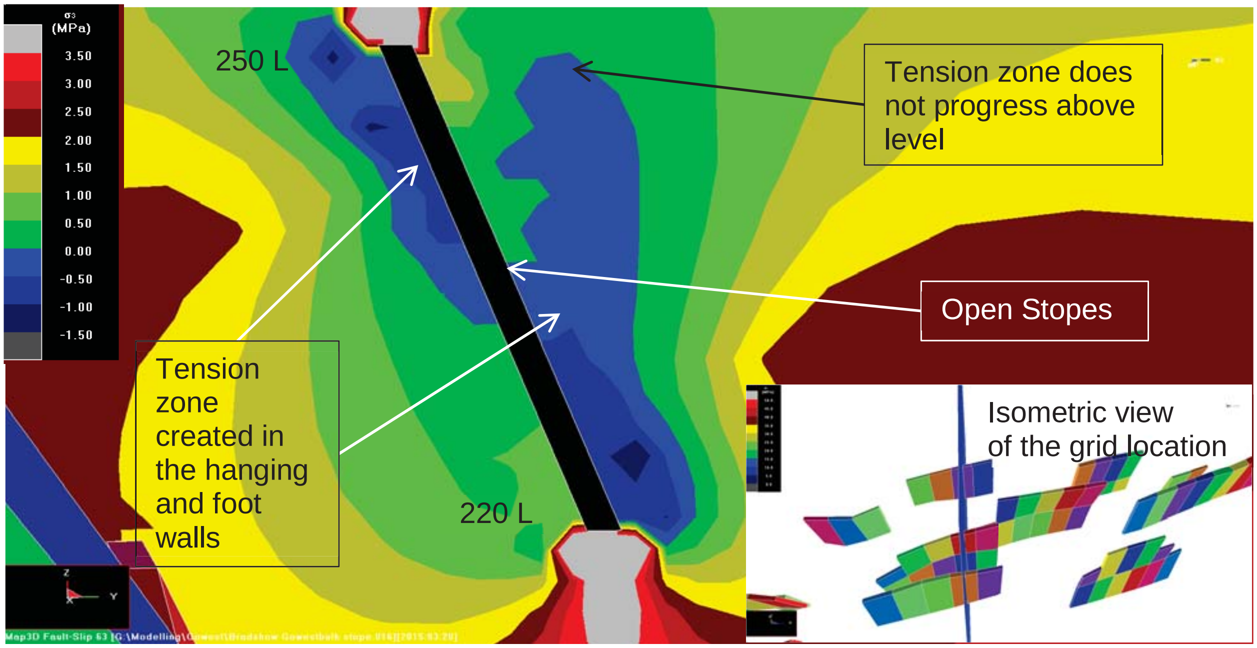
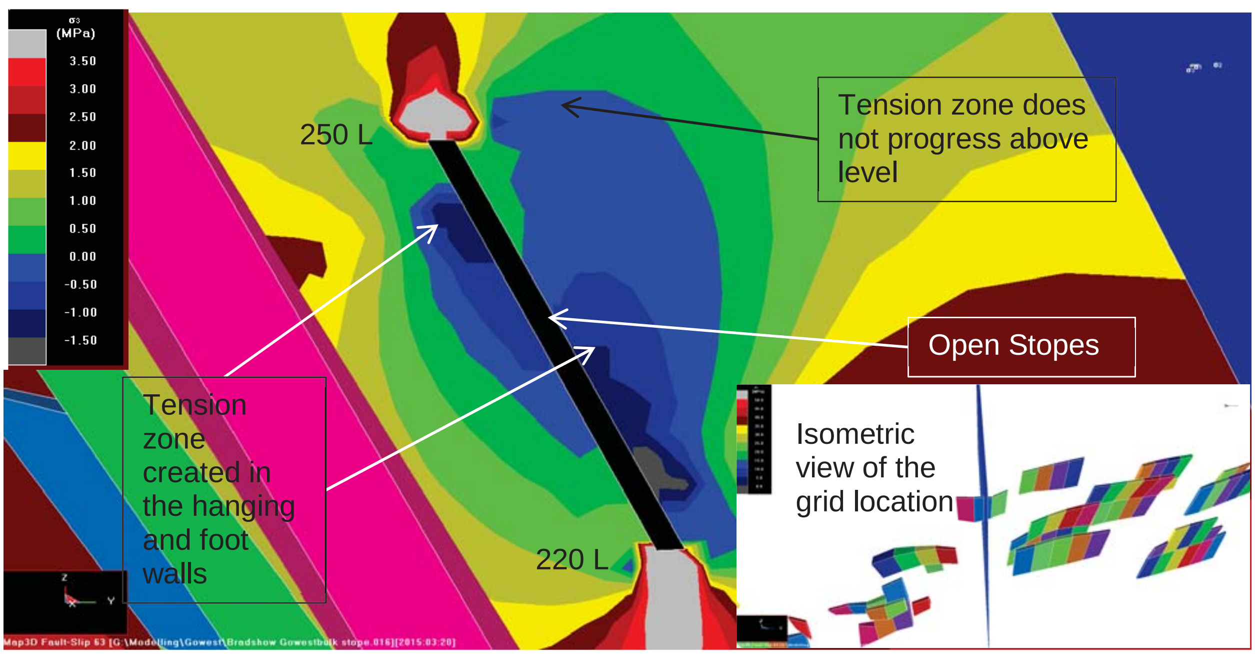


Figure 2.6 Minor Principal Stress on Vertical Section at West End of Bulk Sample Zone



Figures 7 to 12 illustrates that a tension zone is created in an interstitial pillar between bulk sampling stopes. It may be expected that with time the pillar will fail and unravel in the stope causing possible local instability. Given the small dimension of the stopes unravelling (if it occurs) will be limited to a small volume around the openings. The swell factor of the rock will eventually provide confinement and stop the caving process due to lack of void.

Figure 2.7 Minor Principal Stress at 235 Level Creating Interstitial Pillar

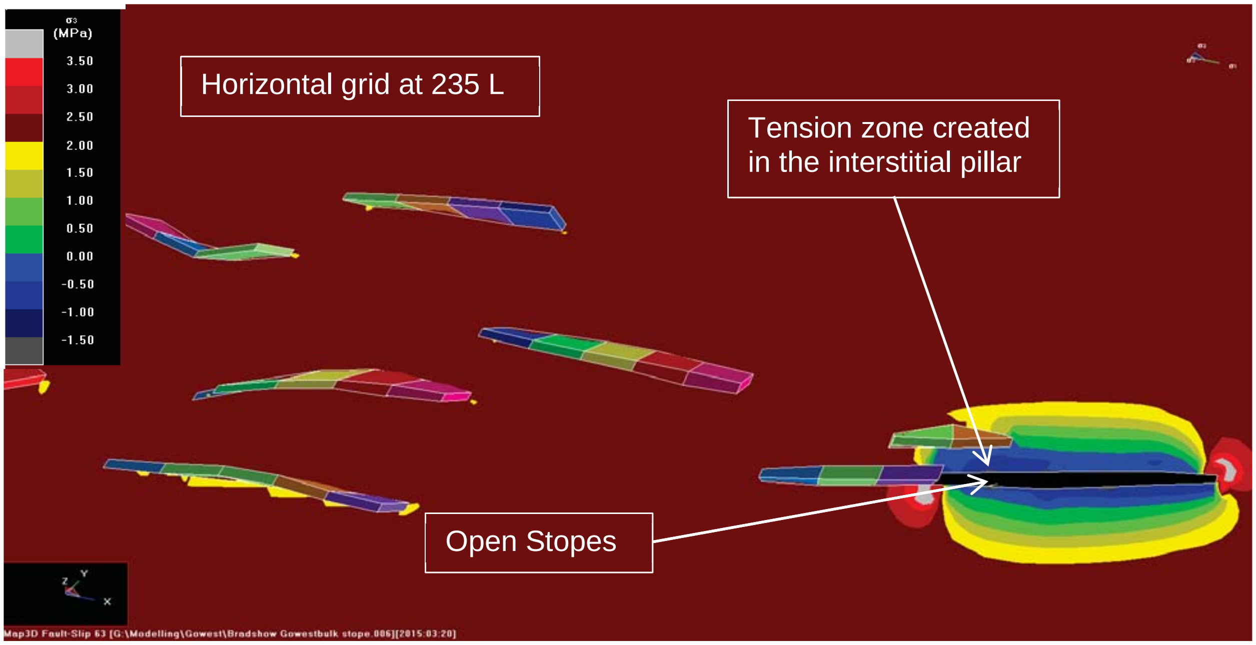


Figure 2.8 Minor Principal Stress on Vertical Section at East End of Interstitial Pillar

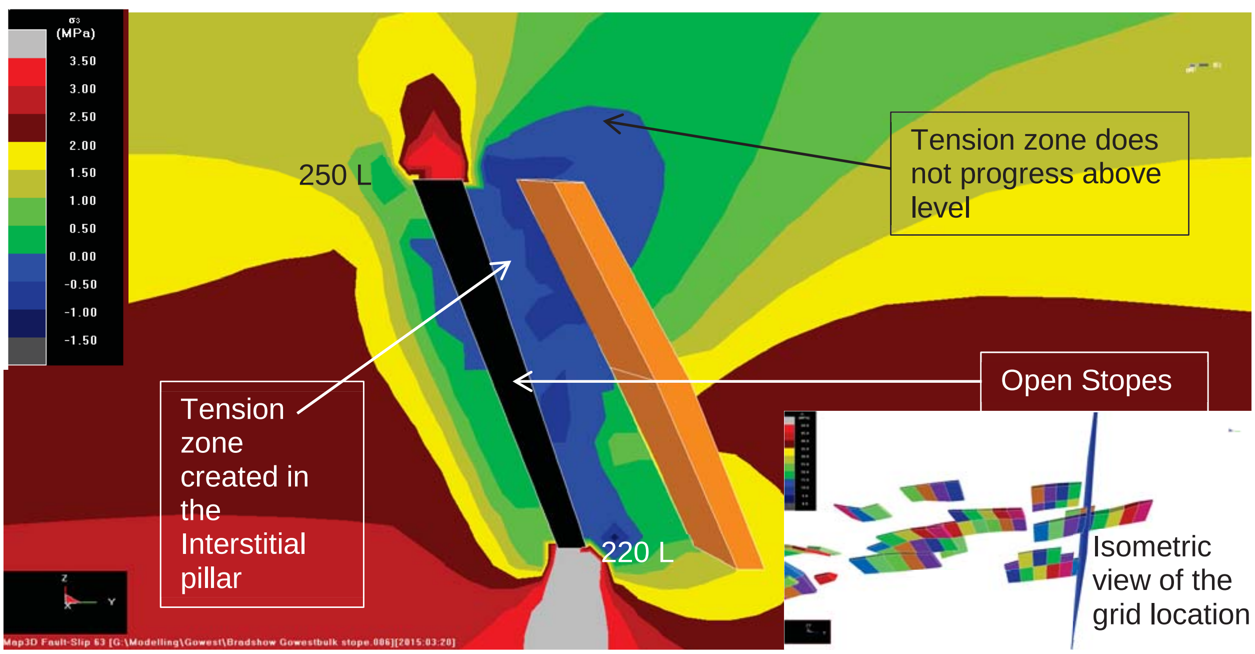


Figure 2.9 Minor Principal Stress at 235 Level Completing Interstitial Pillar

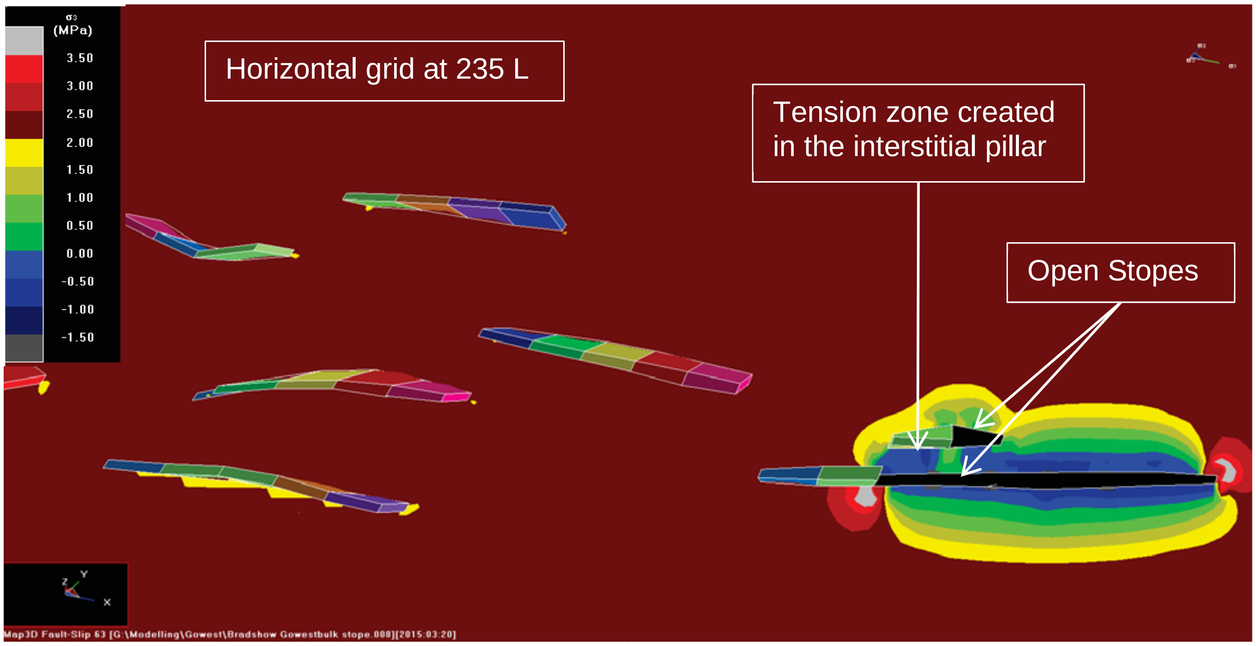


Figure 2.10 Minor Principal Stress on Vertical Section at West End of Interstitial Pillar Completed

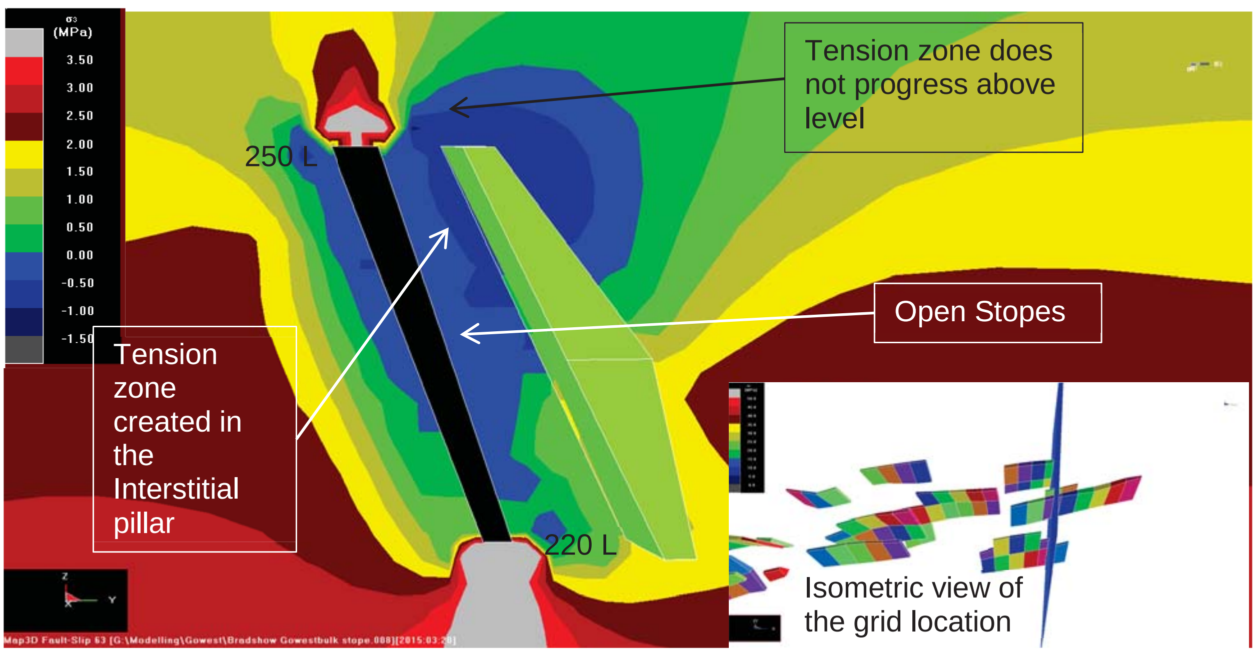


Figure 2.11 Minor Principal Stress at 235 Level with Interstitial Pillar Established

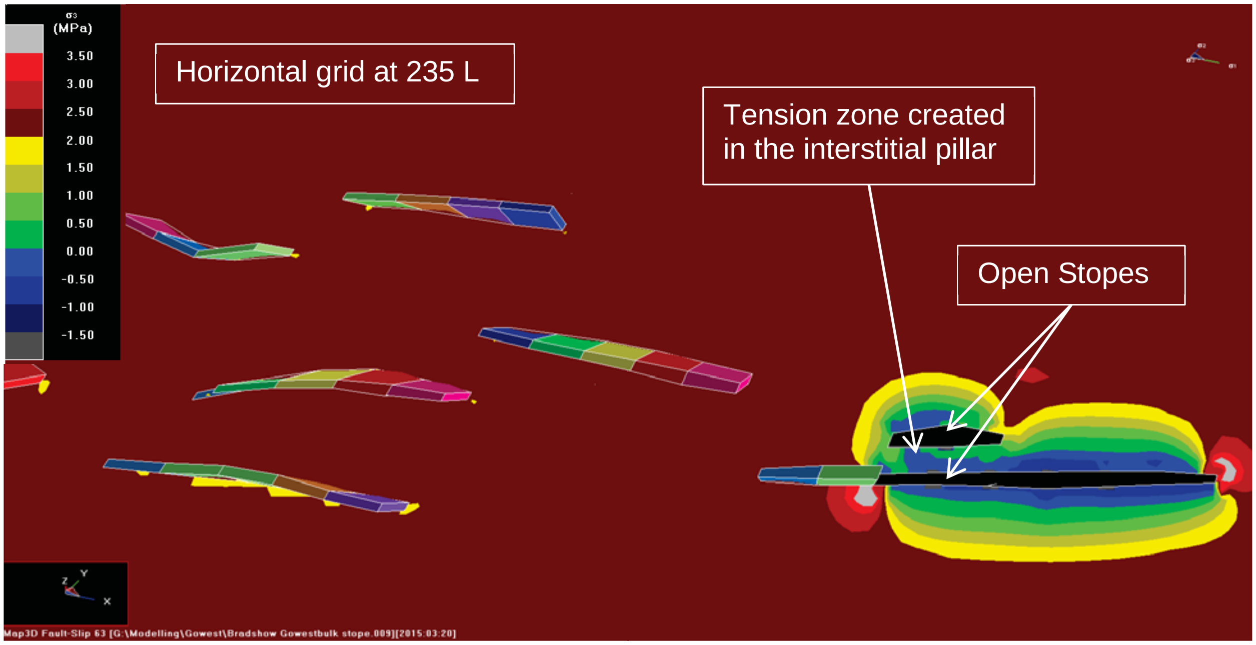
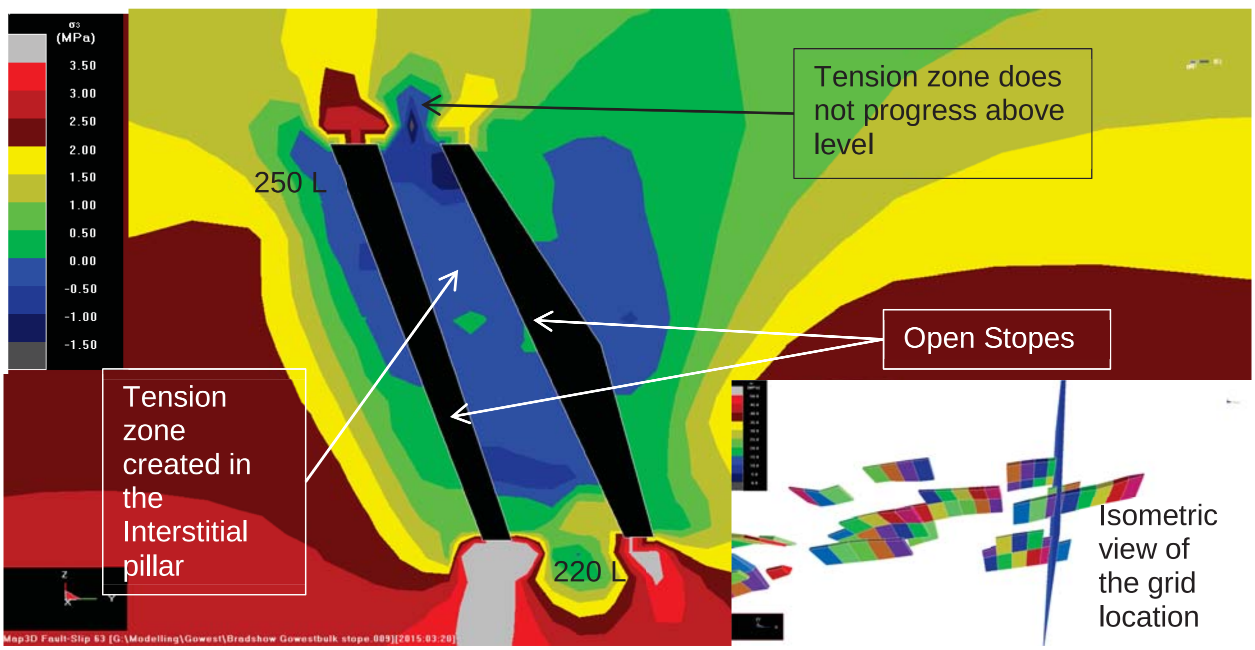


Figure 2.12: Minor Principal Stress on Vertical Section at Interstitial Pillar Established



Due to the time dependency failure around the bulk sample stopes, when the deposit will be brought to full production those areas must be restricted until a drilling program is completed to assess the extent of unraveling and caving around the open stopes.

3.0 CONCLUSION

A numerical modelling review was completed to understand the impact of leaving open bulk sample stopes on the stability of the Bradshaw Deposit. The numerical model indicates that the rock mass above the top-sill elevation will not be affected by the open stopes. Time dependent failure of the rock mass in the Hanging wall, footwall and interstitial pillar may occur. The time dependency failure will probably cause unraveling of the hanging wall, footwall and interstitial pillar in the open stope. Given the small dimension of the stope unravelling (if it occurs) will be limited to a small volume around the openings.

4.0 RECOMMENDATIONS

1. Bulk sampling leaving open stope may proceed.
2. Restrict access to bulk sample open stope area when the mine is brought to full production.
3. If access is required to the bulk sample open stope area, a drilling program is required to establish the extent of unravelling and caving around the open stopes.

5.0 REFERENCES

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Project No. 169514558 **January 23, 2015**

PREPARED FOR:

Gowest

Attention: Mr. Garth Wilcox

CONCERNING:

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1.0 INTROUDUCTION

As part of stage 1 and to the support the mining trade off study, geomechanical stable stope and pillar dimension are required. Stope and pillar dimensions will be based on available geomnechanical data provided by Gowest and on data obtained from adjacent mines (Kidd Creek). For stage 1, empirical techniques will be used to establish ranges of acceptable stope and pillar dimension.

2.0 GEOMECHANICAL DATA REVIEW

Geomechanical data obtained from the deposit site consisted of Rock Quality Designation (RQD) data from diamond drill holes, and laboratory testing perform by Stantec.

For this study, the rock mass was divided in three domains, as per available information:

1. An hanging wall (HW)domain consisting of mafic volcanic
2. A mineralized domain consisting of mineralized mafic volcanic (MinZ)
3. A foot wall (FW) domain consisting of ultramafic (UM).

   Table 1 provide the RQD obtained for the three domain

Table 1: RQD Values

Domain	Average RQD	Standard Deviation	Number of sample
HW MV	96.8	7.52	13,044
MinZ	97.0	6.54	531
FW UM	87.7	18.97	4050

2.1 BARTON (1974) ROCK MASS CLASSIFICATION Q' VALUE

Barton Q' is calculated as follows:

Q' = RQD/Jn * Jr/Ja

Where:

- RQD is the Rock Quality Designation (Deere, 1964),

- Jn is the joint set number,
- Jr is the joint roughness number, and
- Ja is the joint alteration number

Values for Jn, Jr and Ja were obtained from Kidd Creek Mines Yu and Quesnel (1984). Table 2 lists the parameters used and Q' values obtained for the three domains. RQD was obtained from the core logs.

Table 2: Q' Value for Selected Domains

Domain	Equivalent Kidd Creek Rock Type	RQD	Jn	Jr	Ja	Q' Minimum	Q' Maximum
HW MV	Andesite	96.8	9 to 12	1 to 1.5	1 to 2	4.0	16.1
MinZ	Andesite	97.0	9 to 12	1 to 1.5	1 to 2	4.0	16.2
FW UM	UM	87.7	9 to 12	0.5 to 1	2 to 3	1.2	4.8

2.2 LABORATORY RESULTS

Stantec Laboratory in Halifax perform test on core samples. Results are summarized in Table 3

Table 3: Intact Rock Properties

Domain	Average Tensile Strength (MPa)	UCS (MPa)	Young's Modulus (GPa)	Poisson Ratio
HW MV*	13.4	122.7	72.3	0.34
FW UM	3.4	37.3	30.62	0.30

- No mineralized zone samples were available. Values for the HW MV domain will be used for the MinZ domain.

2.3 BIENIAWSKI (1989) ROCK MASS RATING

Bieniawski (1989) RMR was obtained by the following parameters.

- Strength of intact rock parameters,
- RQD,
- Spacing of discontinuities,

- Condition of discontinuities,
- Groundwater, and
- Adjustment for joints orientation with respect to drift azimuth.

Table 4 lists the parameters used and RMR values obtained for the three domains. Intact rock strengths were obtained from laboratory tests. RQD was obtained from the core logs. An average RQD value was obtained by processing the intervals from the diamond drilled holes available for Bradshaw property. Spacing were assumed based on experience in the Canadian Shield. Condition of joints were obtained from Kidd Creek Mine rock description (Yu and Quesnel, 1984)

Table 4: Rock Mass Rating Values

Rock Type	Intact Rock Strength (MPa)	Strength Rating Value	RQD	RQD Rating Value	Joint Spacing (m)*	Joint Spacing Rating Value	Joint Condition Rating Value	Ground Water Rating Value	Drift Azimuth Rating	Maximum RMR	Minimum RMR
HW MV	122.7	12	96.8	20	0.05-1.0	5-15	12-20	10	-5	72	54
MinZ	122.7	12	97.0	20	0.05-1.0	5-15	12-20	10	-5	72	54
FW UM	37.3	4	87.7	17	0.05-1.0	5-15	6-12	10	-5	53	37

*Assumed

3.0 STOPE DIMENSION

3.1 METHODOLOGY

The Stability Graph Method (Potvin, 1988) was used to obtain stable stope dimension. The method consists of comparing the hydraulic radius (Hr) of a stope surface (back, endwall, footwall or hangingwall) to a stability number (N).

Hr = A/P

Where:

- A is the area of the surface and
- P is the perimeter of the surface.

N' = Q' x A x B X C

Where:

- Q' is Barton Rock Mass Classification Index (Barton, 1974) with stress reduction factor and water value set to 1.
- Factor A is the ratio of Unconfined Compressive Strength (UCS) over the induced stress of the surface of interest. Induced stresses were calculated based on a depth of 300 m using unit mass of 2800 kg/ m^3.
- Factor B considers the angle that joints make with respect to the surface of interest. Since joint orientations were not available, the minimum B value was used to consider the worst case scenario.
- Factor C considers slabbing and sliding failure due to joint sets parallel to or intersecting the stope surface. Since joint orientations were not available, the minimum B value was used to consider the worst case scenario.

Table 5 lists the values used to determine N and Hr. The minimum Q' was used to have a conservative approach given the uncertainties for the geomechanical parameters.

Table 5: Estimated Parameters Used to Establish HR and N

Surface	Length or height m	Width m	Q'	UCS MPa	Induced Stress MPa	A	B	C	N'
Stope Back	12	5	4	122.7	42	0.159	0.2	2	0.254
Stope End Wall A	30, 20, 10	5	4	122.7	32	0.268	0.2	2	0.429
Stope End Wall B	30, 20, 10	5	4	122.7	32	0.268	0.2	2	0.429
Stope HW	30, 20, 10	10,11,12,13, 15, 20	4	122.7	3.9	1.000	0.2	2	1.6
Stope FW	30, 20, 10	10,11,12,13, 15, 20	1.2	37.3	10.2	0.252	0.2	2	0.12

3.2 RESULTS

Figure 1 illustrates a typical stope with the terminology used for dimension. Figures 2 to 7 provides the Stability Graph results for the stope dimensions selected. Stope heights were set to 15, 20 and 30 metres. Strike length (width) along the hanging wall and the footwall ranges from 10 m to 20 m. Width from hanging wall to foot wall was set at 5 m. For all cases the end walls (width of 5 m and height of 15, 20 or 30 m) and the back (width of 5 m and length of 10 to 20 m) were stable. Unstable conditions were identified only for the hanging wall and the foot wall (the footwall is considered to be ultramafic rock). Table 6 summarizes the findings of Figures 2 to 7.

For ore entirely within the meta-volcanic rock with 30 m high stope, HW and FW strike length of 13 m is possible. If the FW is in ultramafic the strike length is limited to 10 m for 30 m high stope. For ore entirely within the meta-volcanic rock with 20 m high stope, HW and FW strike length of 15 m is possible. If the FW is in ultramafic the strike length is limited to 11 m for 20 m high stope. For ore entirely within the meta-volcanic rock with 15 m high stope, HW and FW strike length of 20 m is possible. If the FW is in ultramafic the strike length is limited to 12 m for 15 m high stope.

Figure 1: Typical Stope

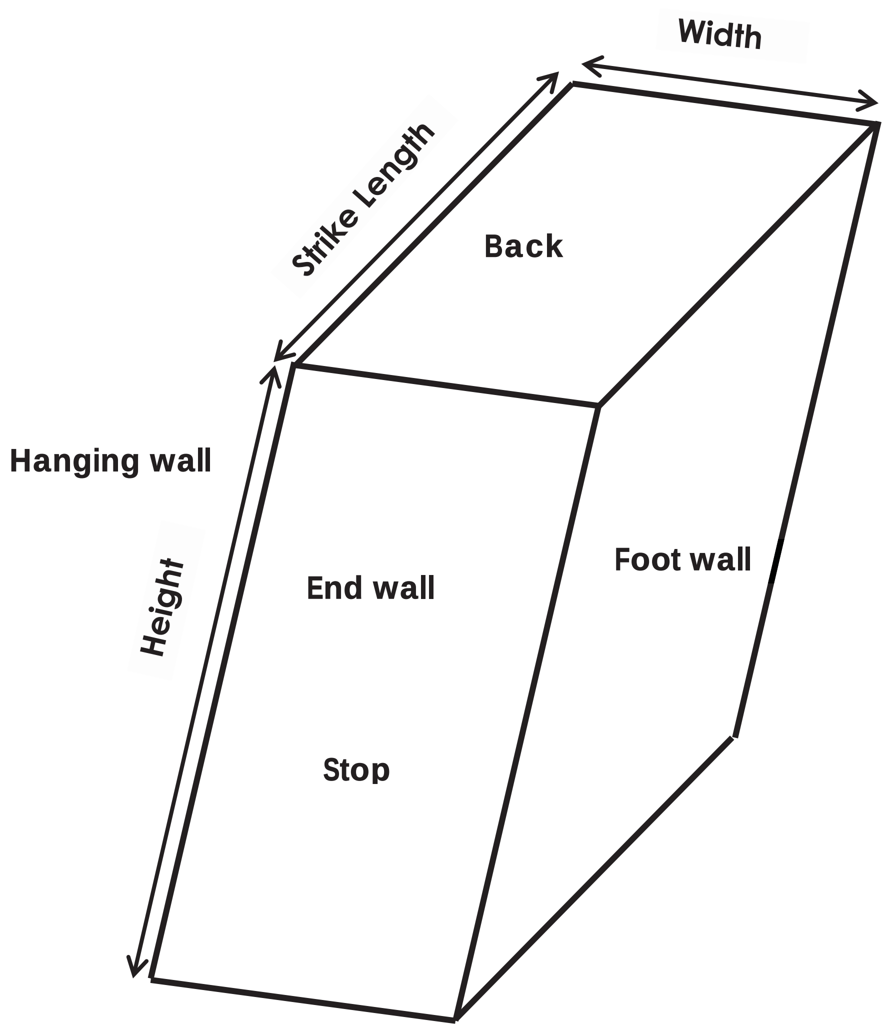


Figure 2: Stable Stope Dimension for 30 m High Level

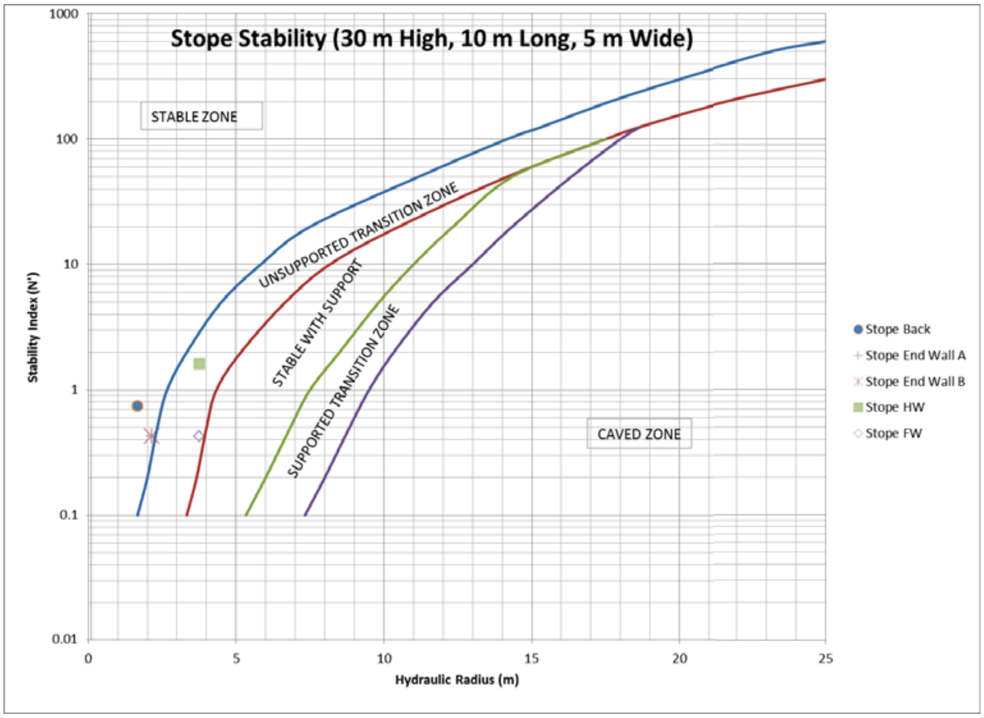


Figure 3: Stable HW for 30 M High Level

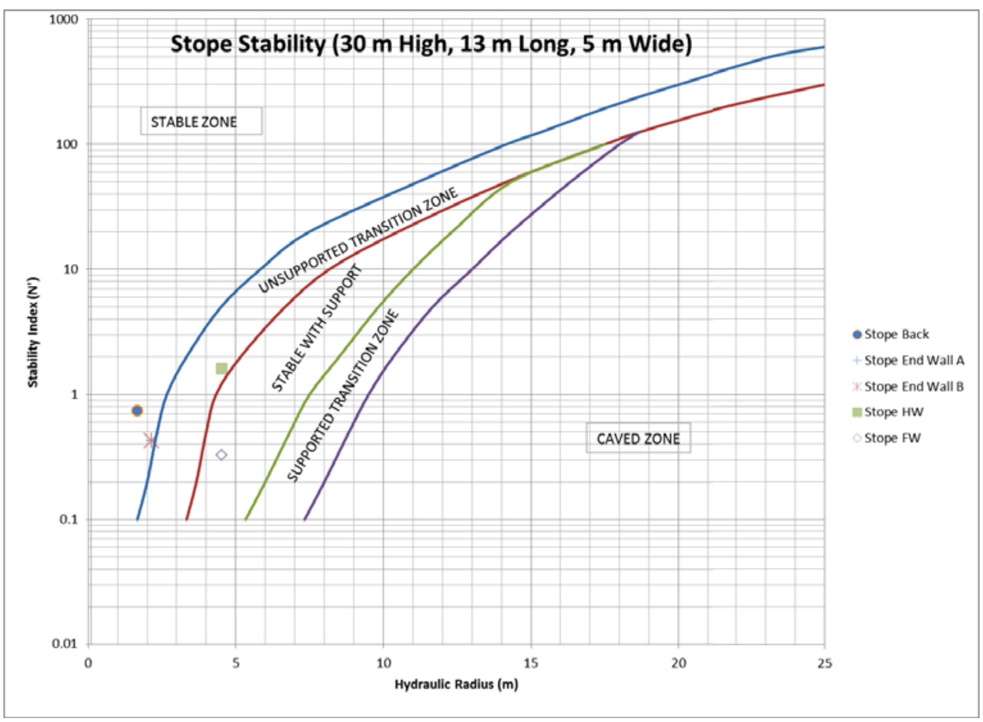


Figure 4: Stable HW and FW for 20 m High Level

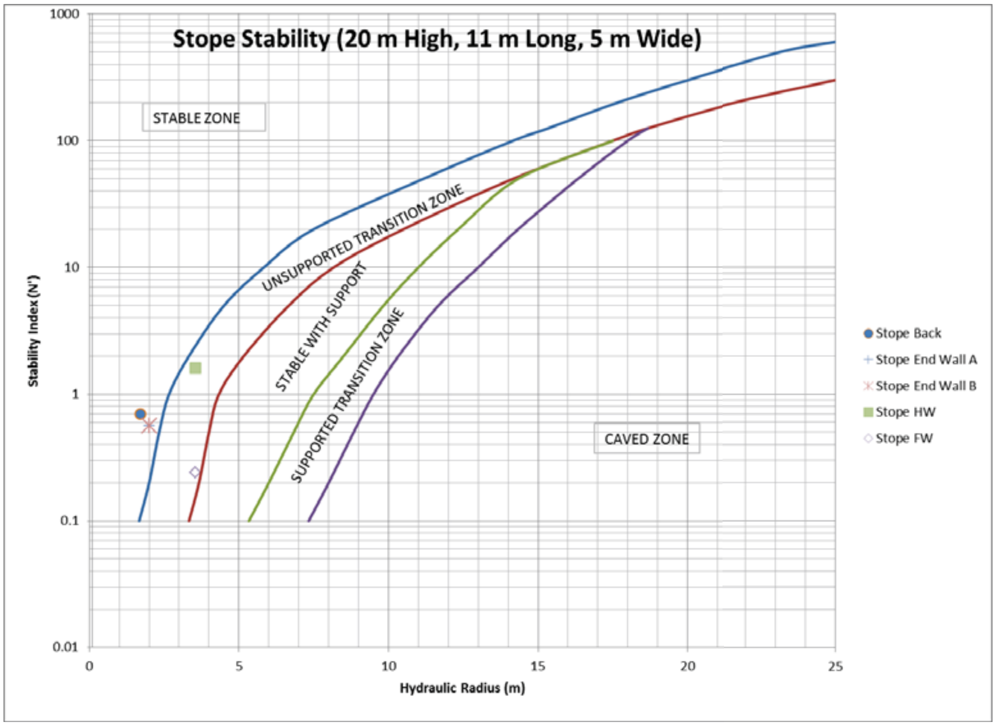


Figure 5: Stable HW for 20 m High Level

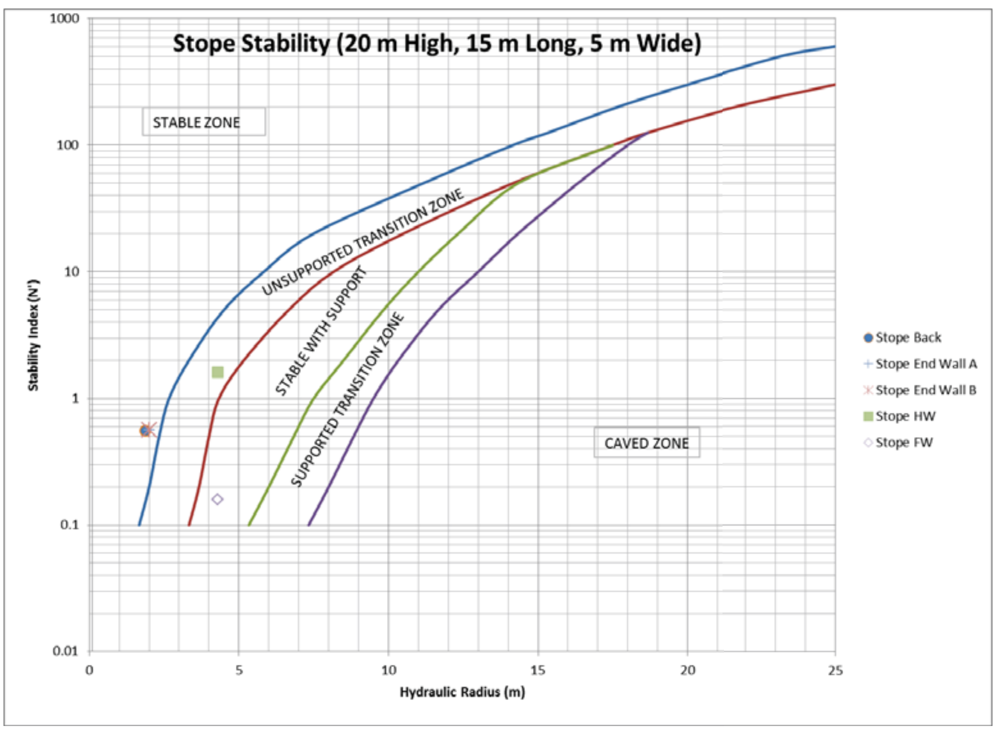


Figure 6: Stable HW and FW for 15 m High Level

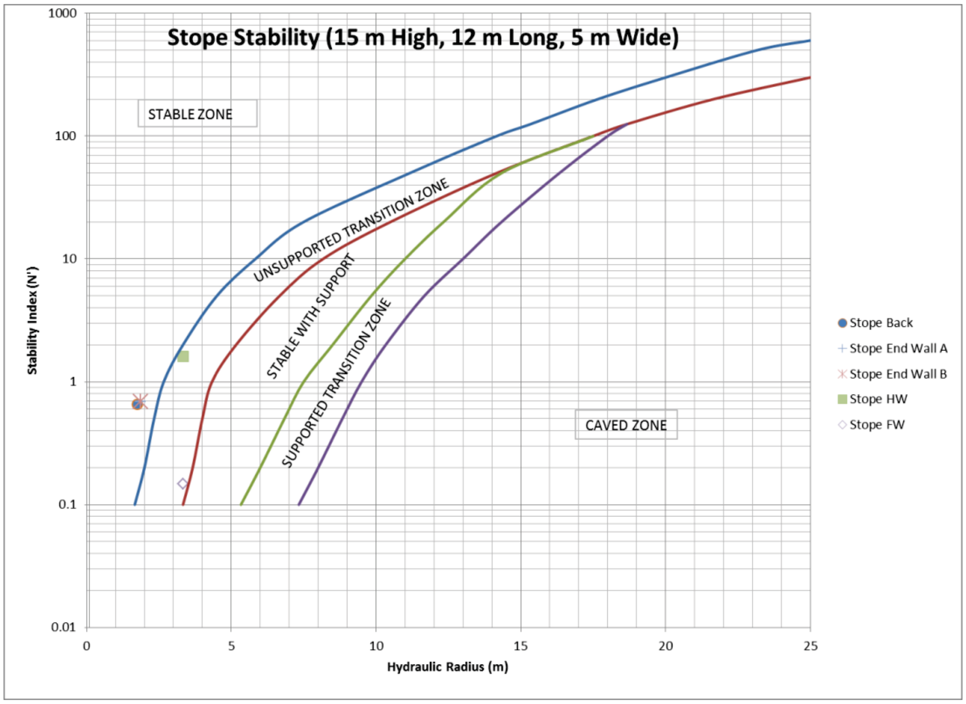


Figure 7: Stable HW for 15 m High Level

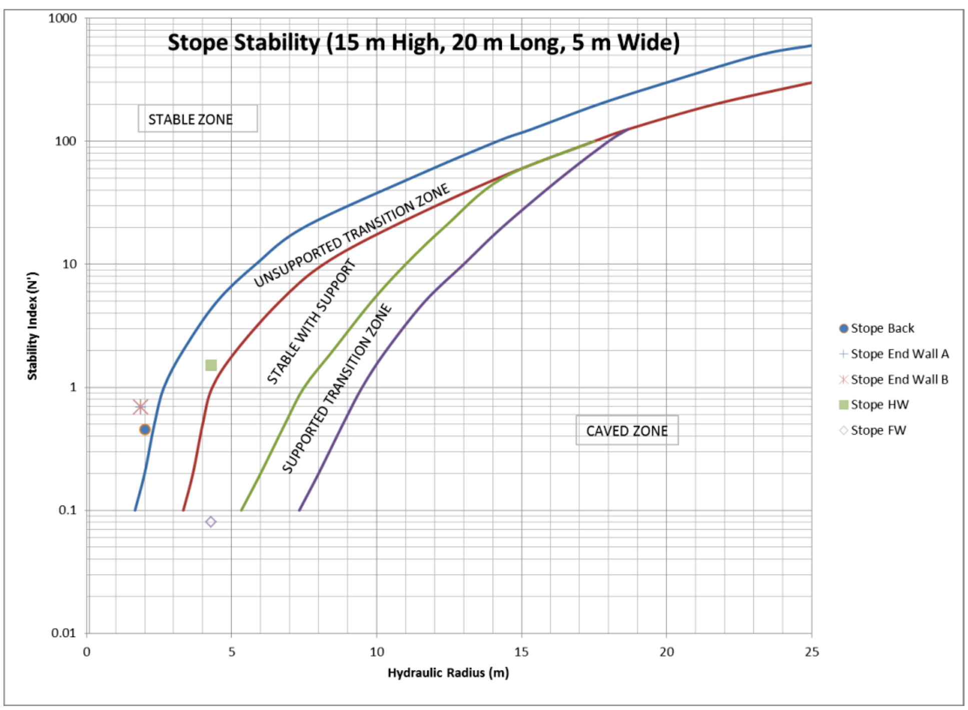


Table 6: Dimension for Stable HW and FW

Stope Height (m)	Length (m)	Stable HW	Stable FW
30	10	Yes	Yes
30	13	Yes	No
20	11	Yes	Yes
20	15	Yes	No
15	12	Yes	Yes
15	20	Yes	No

4.0 PILLAR STABILITY

4.1 INTERSTITIAL AND RIB PILLAR STABILITY

To assess the interstitial and rib pillars stability, empirical approach established by Lunder at al. (1997) was used to determine stable pillar dimension. Stress in the pillar was evaluated as a function of the weight of material at a depth of 300 m using tributary area deterministic approach. Table 7 provides pillar geometry evaluated for stability.

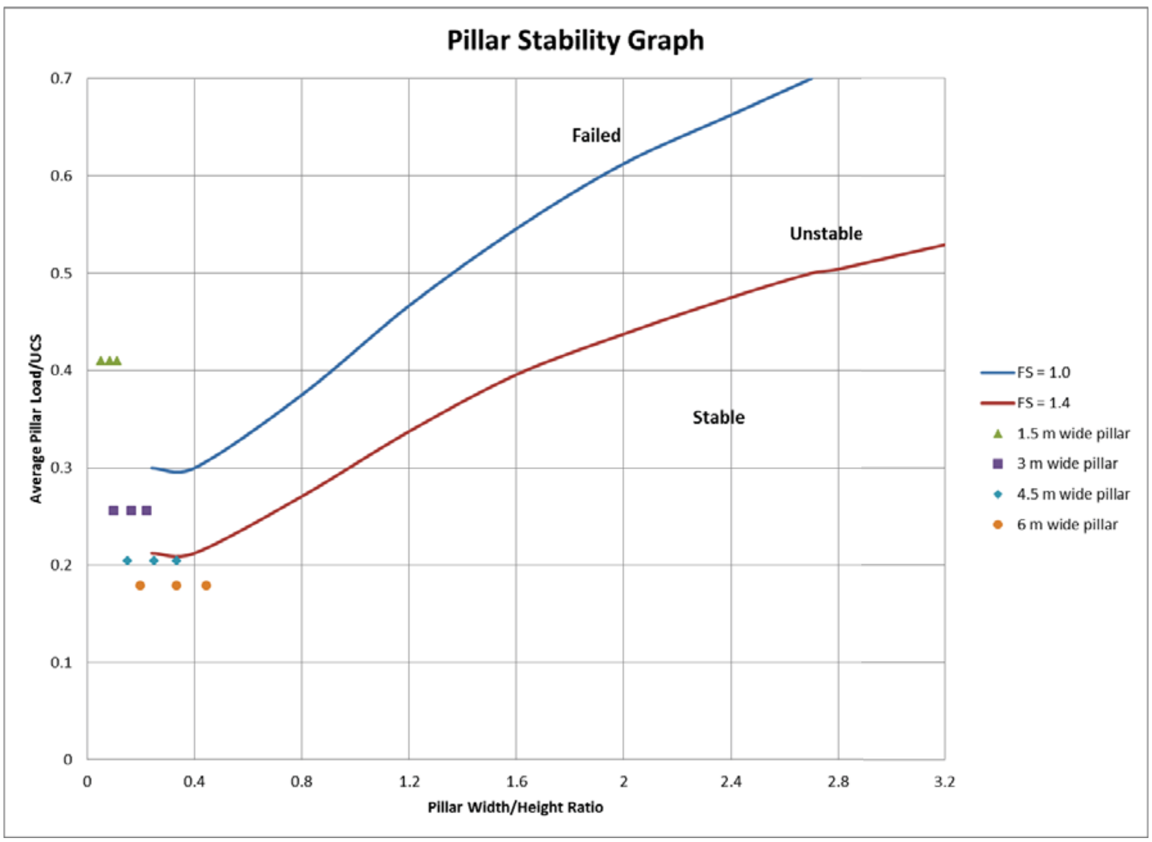
Table 7: Interstitial Pillar Dimensions Evaluated

Height (m)	Length (m)	Width (m)	Width/height Ratio
30	10	1.5	0.05
20	15	1.5	0.08
15	20	1.5	0.11
30	10	3	0.10
20	15	3	0.17
15	20	3	0.22

30	10	4.5	0.15
20	15	4.5	0.25
15	20	4.5	0.33
30	10	6	0..20
20	15	6	0..33
15	20	6	0.44

Figure 8 illustrates stability of pillar when all the ore is extracted and the stopes are left empty. As shown, a minimum pillar width of 4.5 m is required to obtain stable pillars.

Figure 8: Interstitial and Rib Pillar Stability



4.2 SILL PILLAR STABILITY

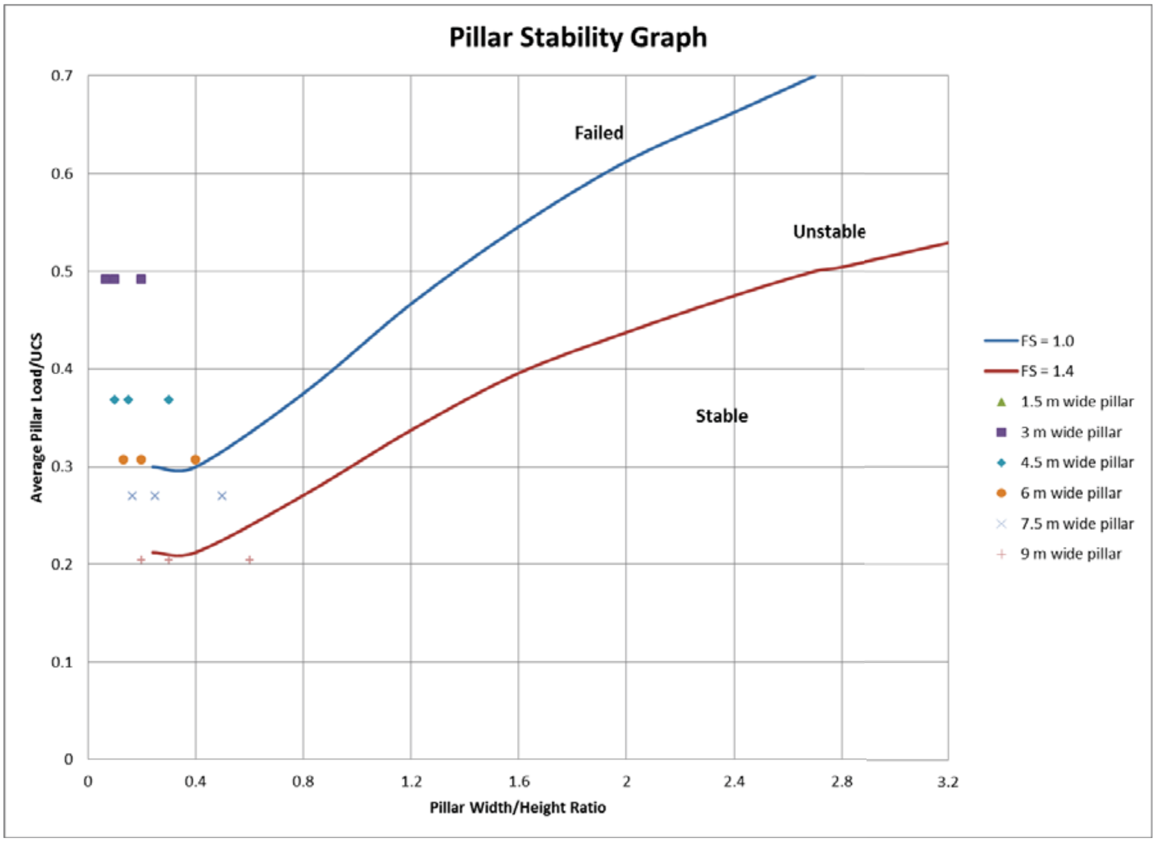
To assess the sill pillar stability, empirical approach established by Lunder at al. (1997) was used to determine stable pillar dimension. Stress in the pillar was evaluated as function of the weight of material at a depth of 300 m using tributary area deterministic approach. Table 7 provides pillar geometry evaluated for stability.

Table 8: Sill Pillar Dimensions Evaluated

Height (m)	Length (m)	Width (m)	Width/height Ratio
15	5	1.5	0.10
30	5	1.5	0.05
45	5	1.5	0.03
15	5	3	0.20
30	5	3	0.10
45	5	3	0.07
15	5	4.5	0.30
30	5	4.5	0.15
45	5	4.5	0.10
15	5	6	0.40
30	5	6	0.20
45	5	6	0.13
15	5	7.5	0.50
30	5	7.5	0.25
45	5	7.5	0.17
15	5	10	0.60
30	5	10	0.30
45	5	10	0.20

Figure 9 illustrates stability of pillar when all the ore is extracted and the stopes are left empty. As shown, a minimum pillar width (thickness) of 9 m is required to obtain stable sill pillars.

Figure 9: Sill Pillar Stability



4.3 CROWN PILLAR DIMENSION

To establish stable crown pillar the empirical scale span method establish by Carter et Al. 2008 was used. Part of this method is to select an acceptable class (Table 9) as function of the mine closure strategy. For this study Class E was selected corresponding to a probability of crown failure of 1 to 5.5 % with minimum monitoring and surveillance of the area. Two Q values were used for the evaluation: a Q value of 4 considering dry conditions with moderate stress and a Q value of 0.8 considering large water inflow and with low stress near surface.

Figure 9 provides the scale span values for Class E of 3.5 and 1.8 for Q vales of 4 and 0.8 respectively. The scale span values allow calculating corresponding crown thickness requirement for given stope span and strike length. Figure 10 illustrates crown thickness requirement for the selected opening width with strike length of 50 m to 140 m. Span in excess of 10 m will require crown thickness greater than 80 m for Q of 0.8 considering wet conditions. For span of 5 m the Crown thickness is 20 m for Q of 0.8. For Q of 4 (dry condition), the crown thickness is 5 m, 20 m and 45 m for span of 5 m. 10 m and 15 m respectively, for strike length greater than 50 m.

Table 9: Acceptable Risk Exposure Guidelines- Comparative Significance of Crown Pillar Failure (Carter et Al. 2008)

Class	Probability of failure (%)	Minimum Factor of Safety	Maximum Scaled Span	Excavation Support Ratio (ESR)	Design Guidelines for Pillar Acceptability/ Service Life of Crown Pillar				
					Serviceable Life	**Years**	**Public Access**	**Regulatory Position on Closure**	**Operating Surveillance Required**
A	50-100	<1	$11.31Q^{0.44}$	>5.0	Effectively zero	<0.5	Forbidden	Totally unacceptable	Ineffective
B	20-50	1.0	$3.58Q^{0.44}$	3.0	Very, very short term (temporary mining purposes only; unacceptable risk of failure for temporary civil tunnel portals)	1	Forcibly prevented	Not Acceptable	Continuous sophisticated monitoring
C	10-20	1.2	$2.74Q^{0.44}$	1.6	Very short term (quasi temporary stope crowns; undesirable risk of failure for temporary civil works)	2-5	Actively prevented	High level of concern	Continuous monitoring with instruments
D	5-10	1.5	$2.33Q^{0.44}$	1.4	Short term (semi-temporary crown, e.g. under non-sensitive mine infrastructure	5-10	Prevented	Moderate level of concern	Continuous simple monitoring
E	1.5-5	1.8	$1.84Q^{0.44}$	1.3	Medium term (semi-permanent crowns, civil	15-20	Discouraged	Low level of concern	Continuous superficial monitoring

					portals, possibly under structures)				
F	0.5-1.5	2.0	$1.12Q^{0.44}$	1.0	Long term (quasi-permanent crowns, civil portals, near-surface sewer tunnels)	50-100	Allowed	Of limited concern	Incidental superficial monitoring
G	<0.5	>>2.0	$0.69Q^{0.44}$	0.8	Very long term (permanent crowns over civil tunnels)	>100	Free	Of no concern	No monitoring required

Figure 10: Scale span values obtained for Class E

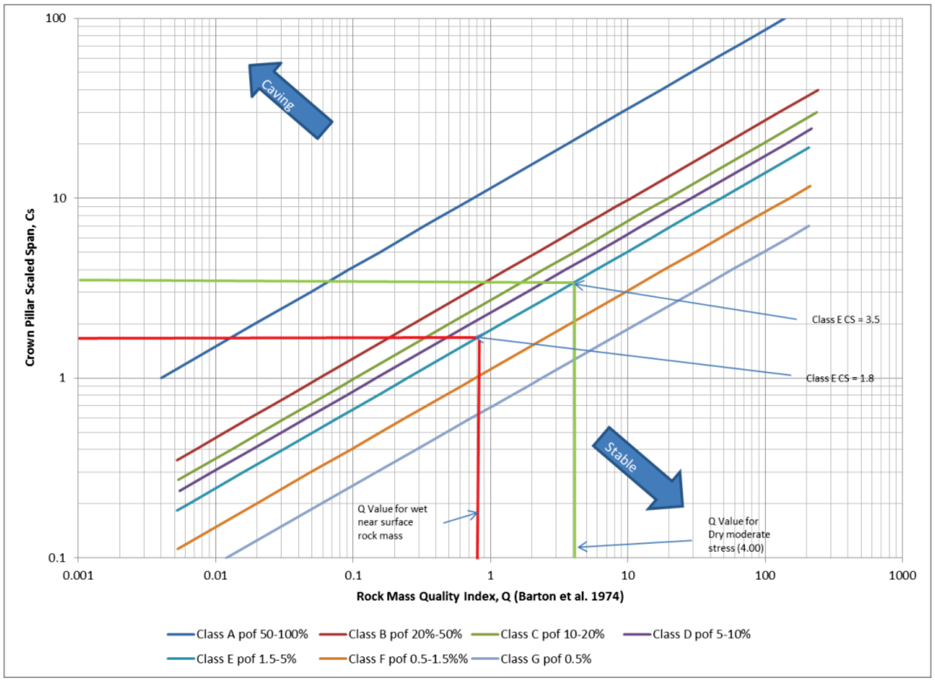
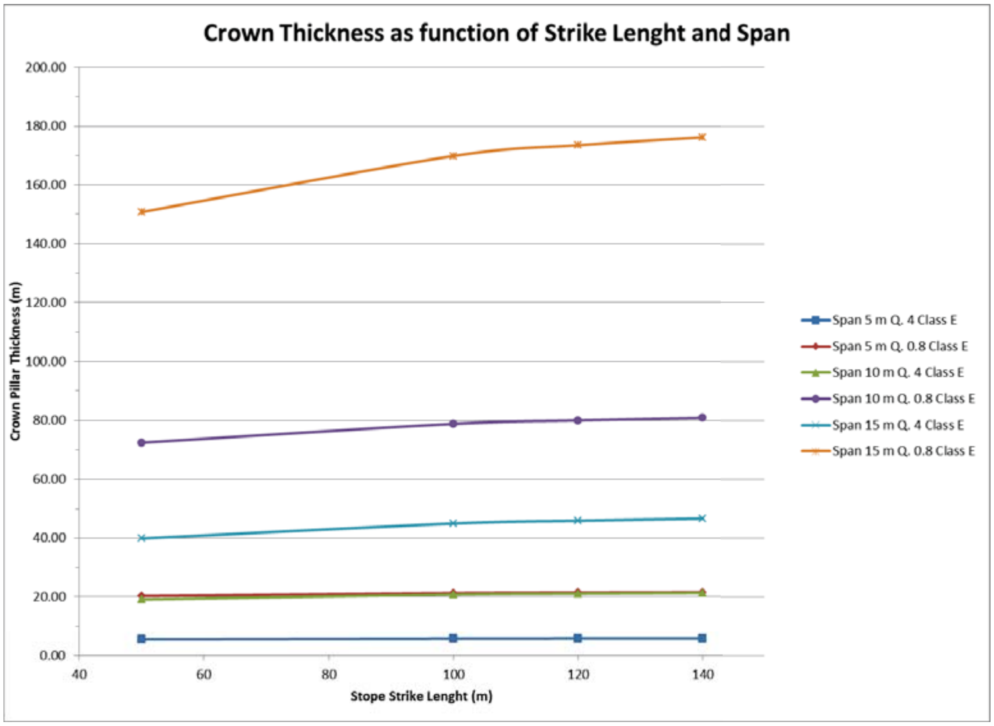


Figure 11: Crown Thickness as a Function of Opening Span and Strike Length



5.0 CONCLUSION

Review of available geomechanical data to estimate stope dimension and the stability of the pillars (interstitial, sill and crown) was completed with data obtained from Gowest. It was found that very little data was available for the Bradshaw Deposit, therefore data from adjacent Kidd Creek Mine were used to perform the analysis.

The deposit was divided in three domains: 1- Hanging wall in meta-volcanic (Q' = 4), 2- mineralization (Q' = 4) and 3- foot wall in ultramafic (Q' = 1.2). Nevertheless, more geomechanical data for intact rock parameters will be useful to increase confidence level in the design, therefore it is recommended to establish a specific geomechanical data acquisition program for Bradshaw deposit to include geomechanical core logging (oriented core are required or acoustic televiewing to obtain information about joints orientation), more laboratory test (to obtain unconfined compressive strength, Young's Modulus, Poisson ratio and tensile strength for the ore, footwall rock, and hangingwall rock). Stress measurement to confirm the stress regime is also required.

Review of the suggested stope dimension through the stability graph method indicates that stope entirely in the meta-volcanic are stable with strike length of 13, 15 and 20 m for level height of 30, 20 and 15 m respectively. Footwall of stope in direct contact with the ultramafic will have a reduced strike length of 10, 11 and 12 m for level height of 30, 20 and 15 m respectively.

Interstitial and rib pillars, independent of level height, must have a minimum width of 4.5 m. Sill pillar must have a minimum thickness of 9 m. The pillar dimensions are based on the available data and are assumed to always be in the meta-volcanic or the mineralized zone.

The scale span empirical method was used to obtain stable crown pillar thickness with low regulatory concern at mine closure. Thickness of the crown pillar is dependent on the presence or absence of water within the bedrock near surface. Strike length does not seem to impact thickness requirements. If water is present, excavation span near surface must be limited to a 5 m span from hanging wall to footwall to have a reasonable crown thickness of 20 m. In the absence of water, excavation span could reach 10 m and crown thickness may range between 5 m to 20 m thick depending on the actual excavation span (e.g. smaller span implies thinner crown pillar).

Nevertheless, all findings mentioned above must be confirm when additional geomechanical data for the Bradshaw deposits become available using more refined geomechanical tools such as numerical modelling.

6.0 RECOMMENDATIONS

1. Stope size in meta-volcanic
   a. 30 m high x 13 m long by 5 m wide
   b. 20 m high x 15 m long by 5 m wide
   c. 15 m high x 20 m long by 5 m wide
2. Stope size in ultramafic
   a. 30 m high x 10 m long by 5 m wide
   b. 20 m high x 11 m long by 5 m wide
   c. 15 m high x 12 m long by 5 m wide
3. Minimum width of 4.5 m for interstitial and rib pillars
4. Minimum thickness of 9 m for sill pillar
5. Crown pillar size
   a. In presence of water: 20 m thick for excavation span of 5 m or less
   b. In absence of water: 5 m to 20 m for excavation span of 10 m or less
6. Establish a specific geomechanical data acquisition program for Bradshaw Deposit that will include:
   a. Geomechanical core logging with oriented core and/ or acoustic televiewing.
   b. Laboratory test of intact rock to obtain unconfined compressive strength, Young's Modulus, Poisson ratio and tensile strength for the ore, footwall rock, hangingwall rock. Perform at least 5 tests per rock type.
   c. Perform stress measurement to confirm the stress regime
7. Perform additional geomechanical study using numerical modelling tools once additional geomechanical data are available.

7.0 REFERENCES

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Barton, N., R. Lien, and J. Lunde. Engineering Classifications of Rock Masses for the Design of Tunnel Support. *Rock Mechanics*, vol. 6, no. 6, 1974, pp. 189 - 236.

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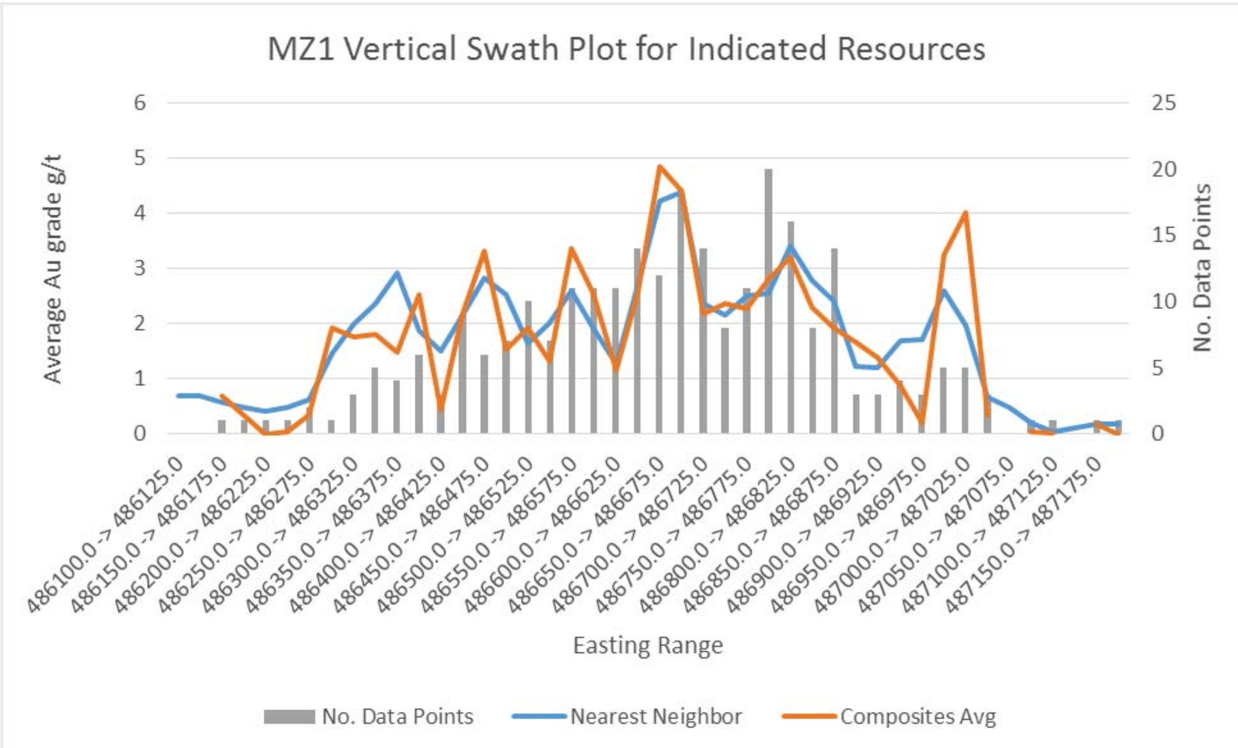
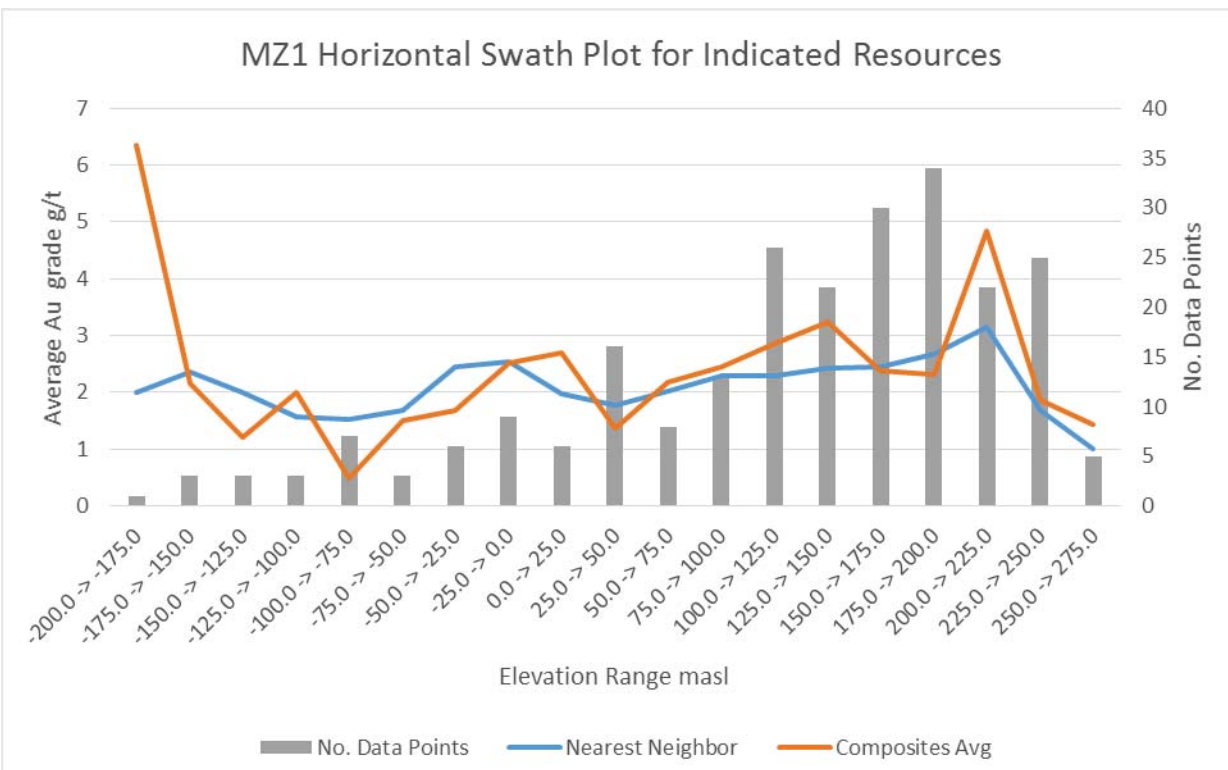
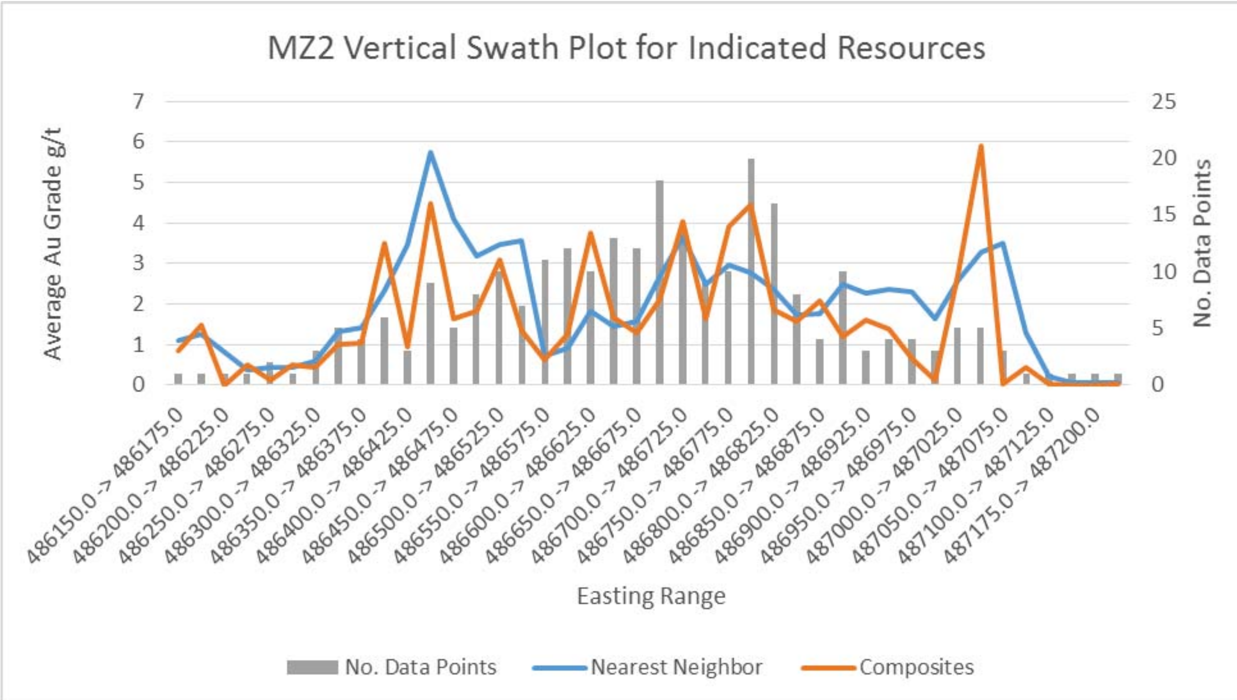
Carter, T.G., Cottrell, B.E., Carvalho, J.L, and Steed, C.M., 2008, Logistic Regression improvements to the Scaled Span Method for dimensioning Surface Crown Pillars over civil or mining openings, Proceedings 42nd US Rock Mechnanics Symposium and 2ndUS-Canada Rock Mechanics Symposium San Francisco.

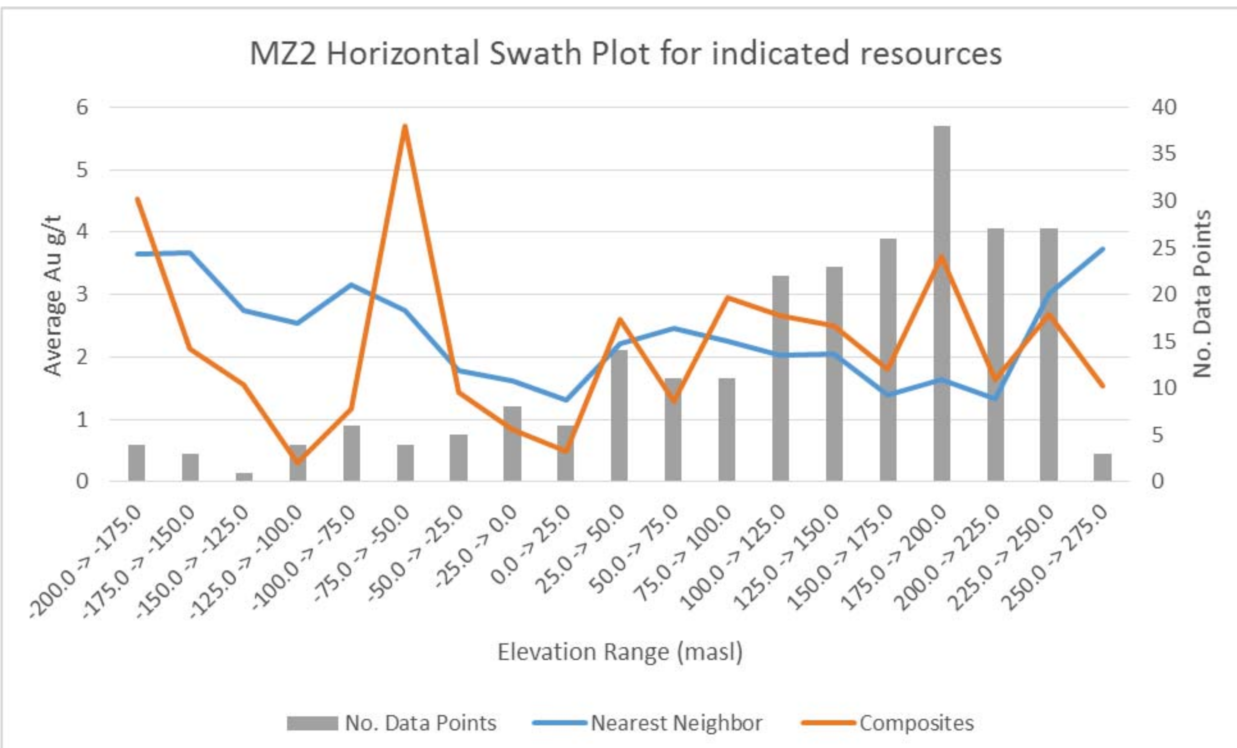
APPENDIX C
GEOLOGY

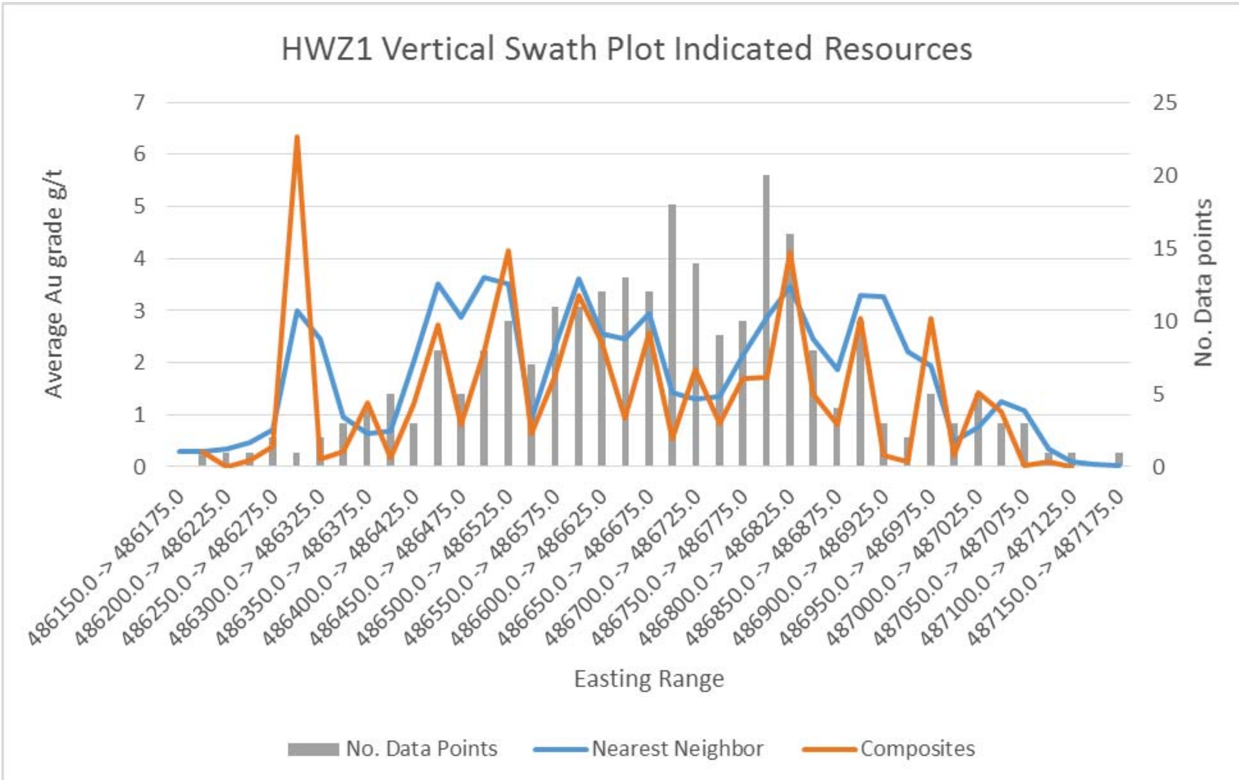
APPENDIX 14-1 SWATH PLOTS






MZ2 Vertical Swath Plot for Indicated Resources
Average Au Grade g/t
No. Data Points
Easting Range
No. Data Points
Nearest Neighbor
Composites


MZ2 Horizontal Swath Plot for indicated resources
Average Au g/t
No. Data Points
Elevation Range (masl)
No. Data Points
Nearest Neighbor
Composites



HWZ1 Vertical Swath Plot Indicated Resources
Average Au grade g/t
No. Data points
Easting Range
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486200.0 -> 486225.0
486250.0 -> 486275.0
486300.0 -> 486325.0
486350.0 -> 486375.0
486400.0 -> 486425.0
486450.0 -> 486475.0
486500.0 -> 486525.0
486550.0 -> 486575.0
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486650.0 -> 486675.0
486700.0 -> 486725.0
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486800.0 -> 486825.0
486850.0 -> 486875.0
486900.0 -> 486925.0
486950.0 -> 486975.0
487000.0 -> 487025.0
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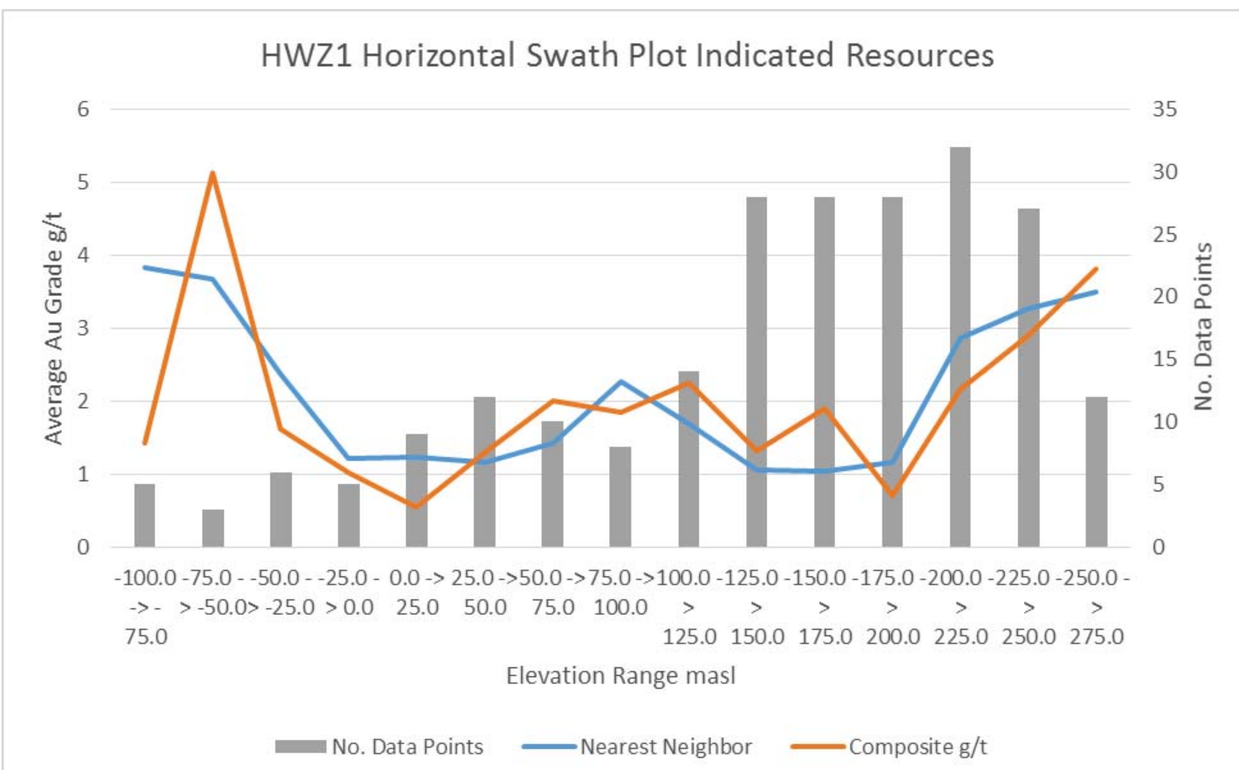
No. Data Points
Nearest Neighbor
Composites



HWZ1 Horizontal Swath Plot Indicated Resources
Average Au Grade g/t
No. Data Points
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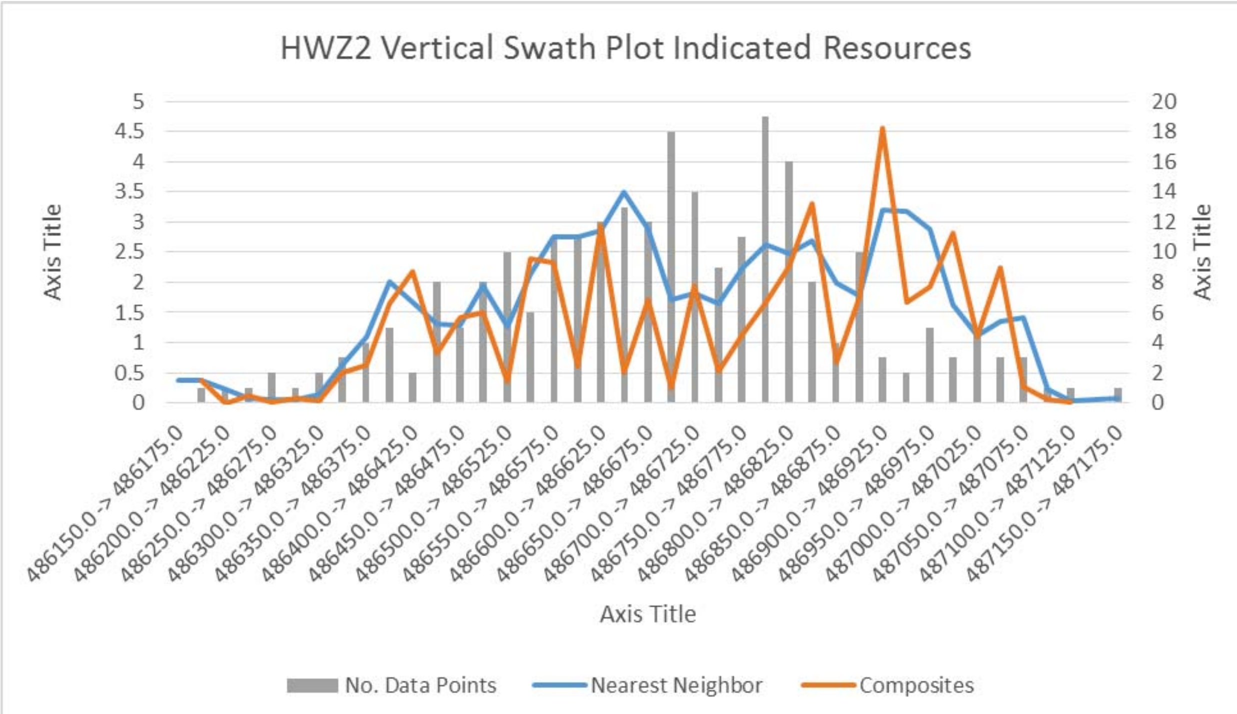
-75.0 -> -50.0
-50.0 -> -25.0
-25.0 -> 0.0
0.0 -> 25.0
25.0 -> 50.0
50.0 -> 75.0
75.0 -> 100.0
100.0 -> 125.0
125.0 -> 150.0
150.0 -> 175.0
175.0 -> 200.0
200.0 -> 225.0
225.0 -> 250.0
250.0 -> 275.0
Elevation Range masl
No. Data Points
Nearest Neighbor
Composite g/t


HWZ2 Vertical Swath Plot Indicated Resources
Axis Title
Axis Title
Axis Title
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486200.0 -> 486225.0
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486400.0 -> 486425.0
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486500.0 -> 486525.0
486550.0 -> 486575.0
486600.0 -> 486625.0
486650.0 -> 486675.0
486700.0 -> 486725.0
486750.0 -> 486775.0
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486850.0 -> 486875.0
486900.0 -> 486925.0
486950.0 -> 486975.0
487000.0 -> 487025.0
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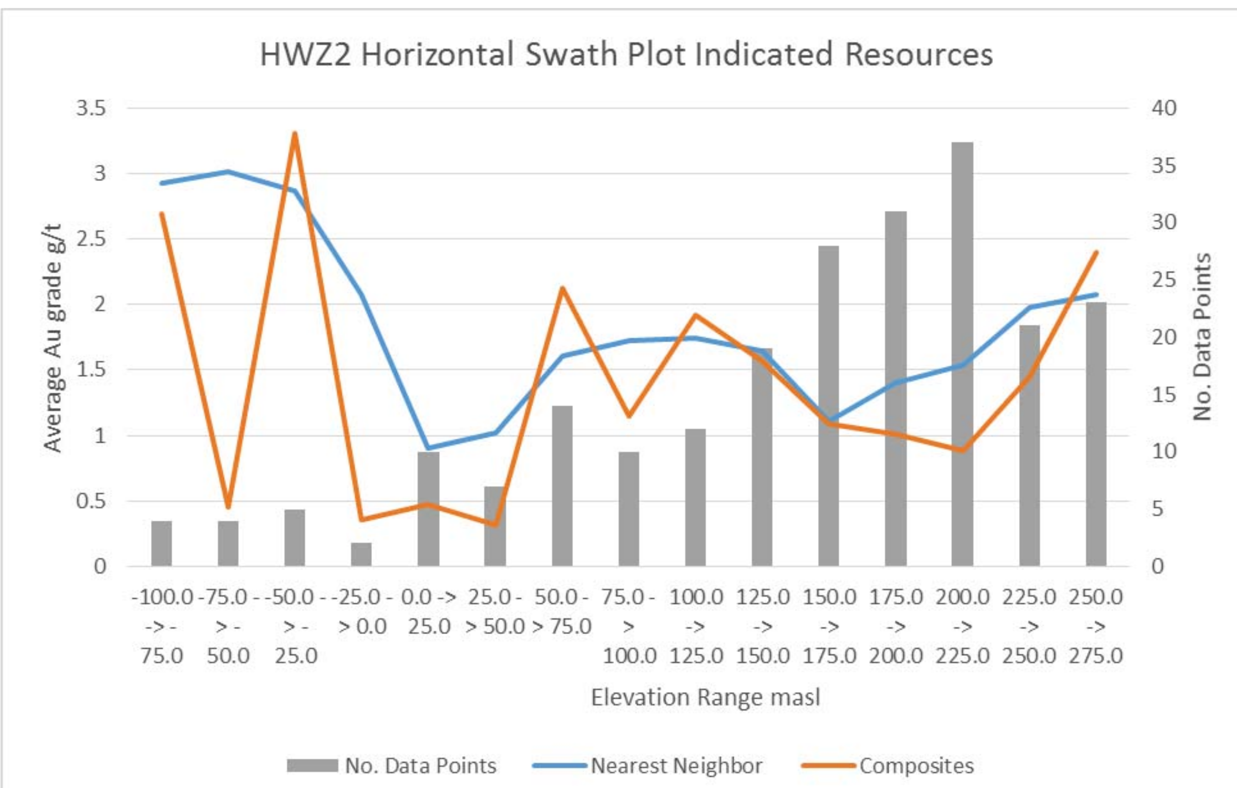
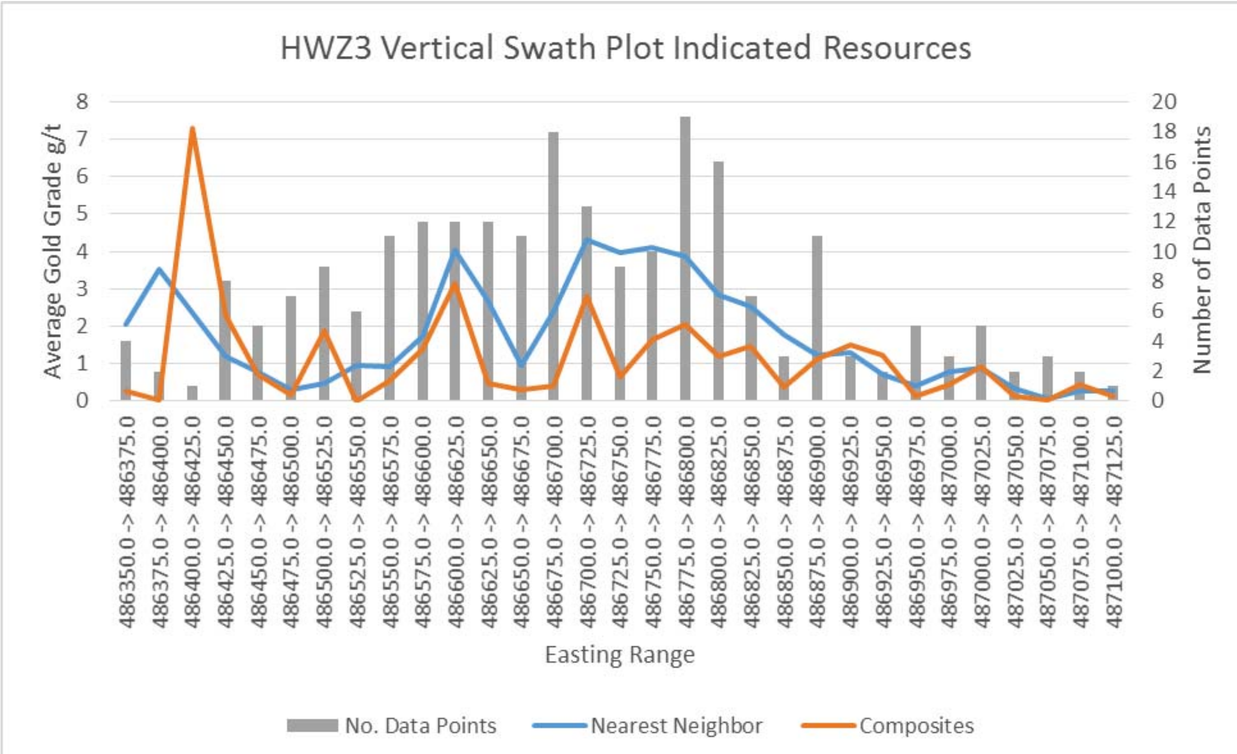
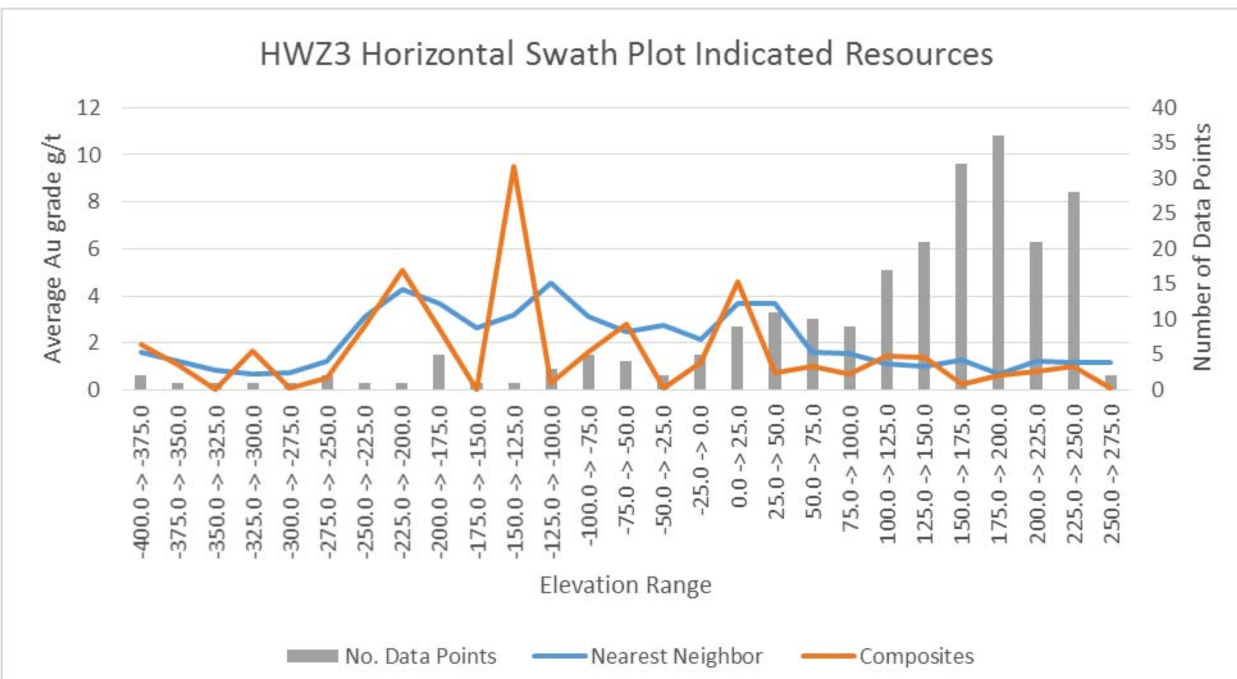
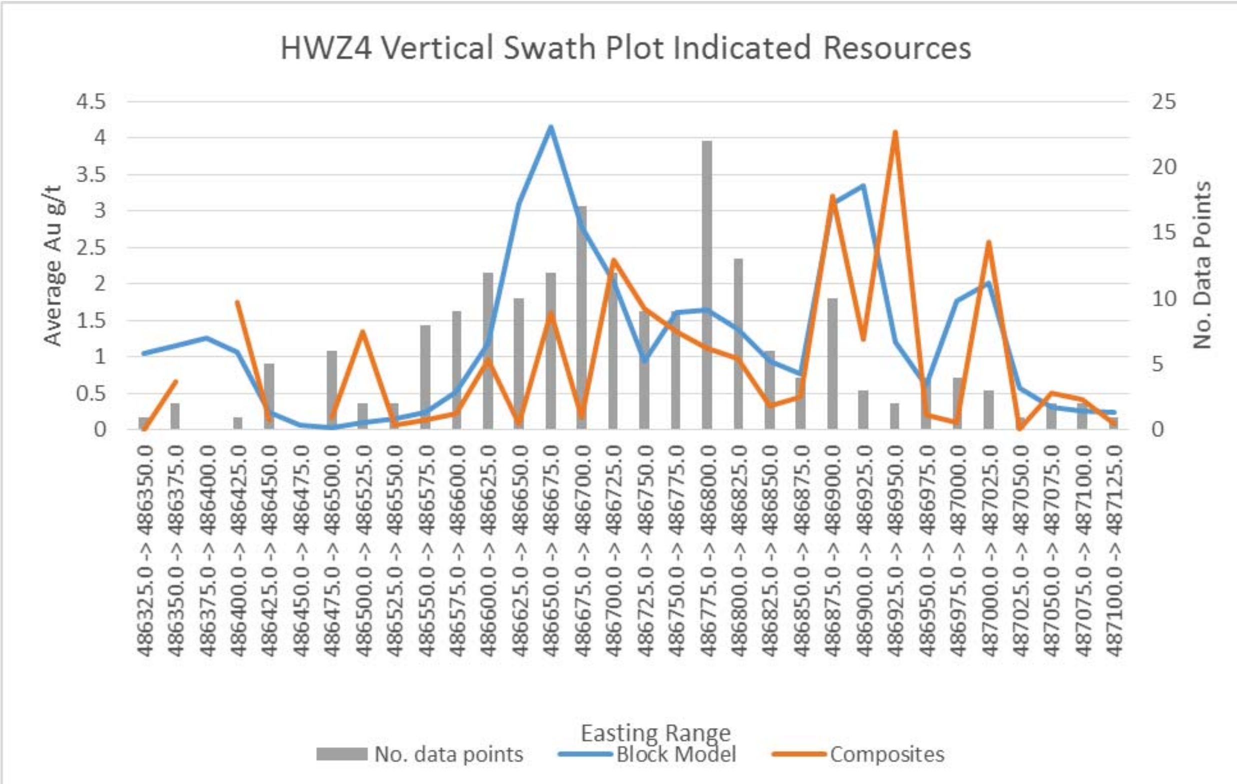
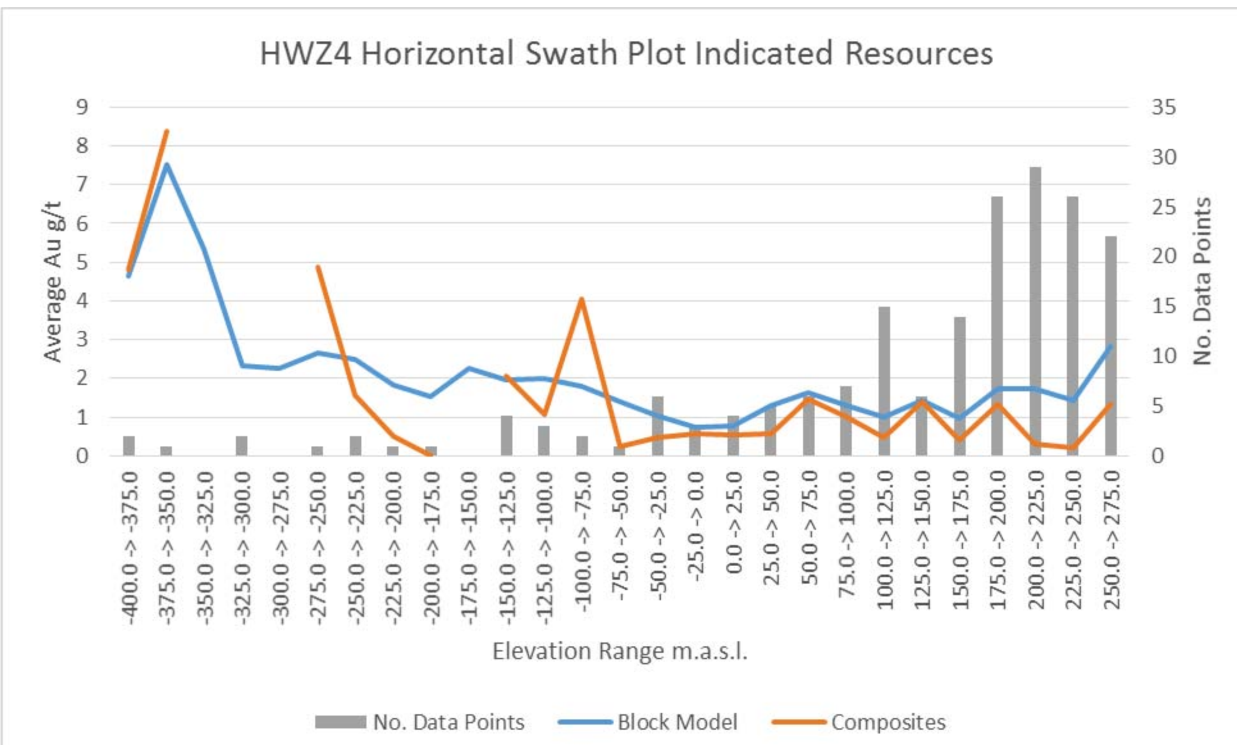
487100.0 -> 487125.0
487150.0 -> 487175.0
No. Data Points
Nearest Neighbor
Composites


HWZ2 Horizontal Swath Plot Indicated Resources
Average Au grade g/t
No. Data Points
-100.0 -> -75.0
-75.0 -> -50.0
-50.0 -> -25.0
-25.0 -> 0.0
0.0 -> 25.0
25.0 -> 50.0
50.0 -> 75.0
75.0 -> 100.0
100.0 -> 125.0
125.0 -> 150.0
150.0 -> 175.0
175.0 -> 200.0
200.0 -> 225.0
225.0 -> 250.0
250.0 -> 275.0
Elevation Range masl
No. Data Points
Nearest Neighbor
Composites

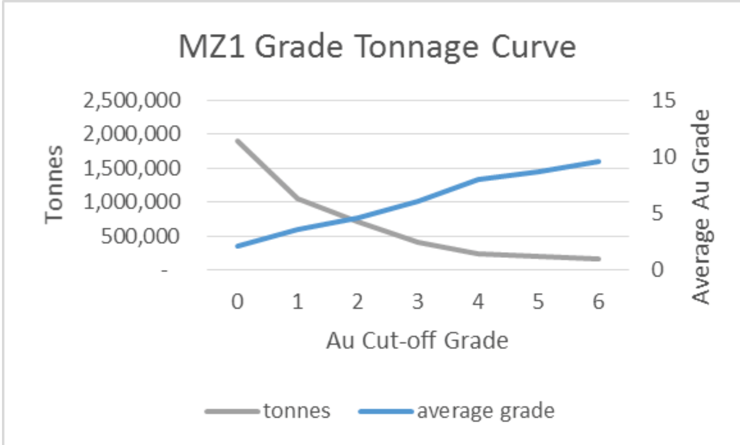

HWZ3 Vertical Swath Plot Indicated Resources
Average Gold Grade g/t
Number of Data Points
Easting Range
No. Data Points
Nearest Neighbor
Composites

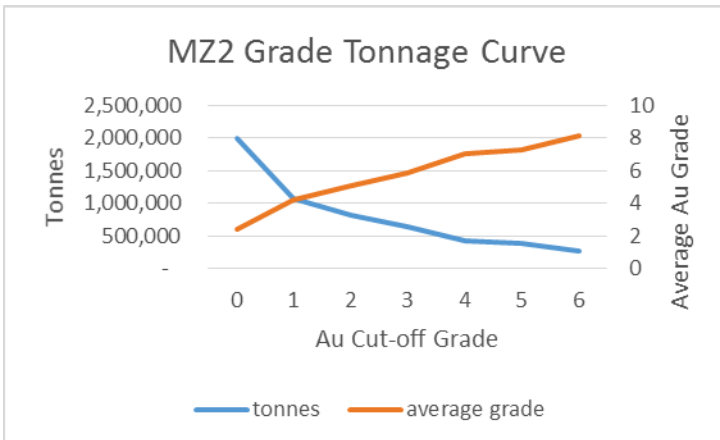

HWZ3 Horizontal Swath Plot Indicated Resources
Average Au grade g/t
Number of Data Points
Elevation Range
No. Data Points
Nearest Neighbor
Composites

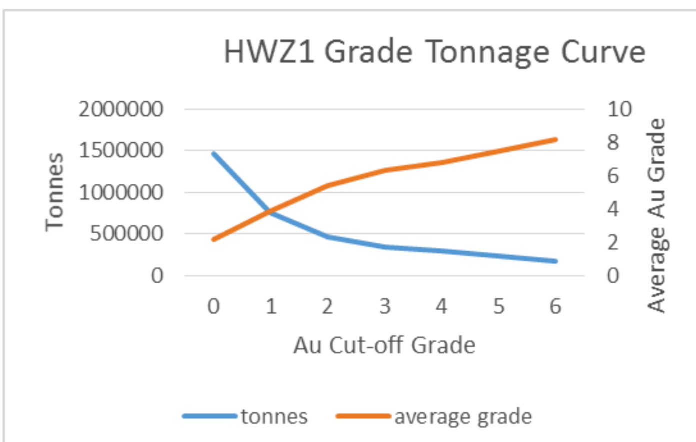

HWZ4 Vertical Swath Plot Indicated Resources
Average Au g/t
No. Data Points
Easting Range
No. data points
Block Model
Composites

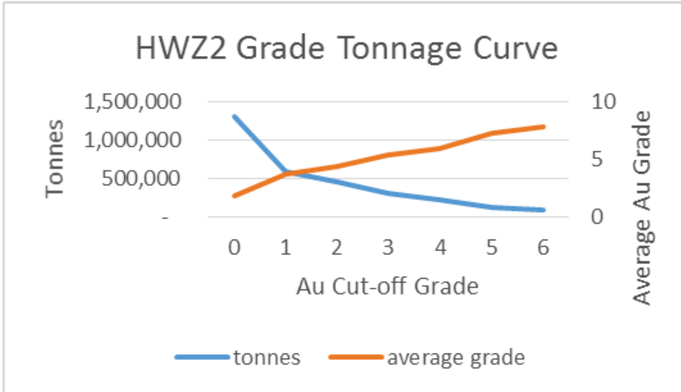

HWZ4 Horizontal Swath Plot Indicated Resources
Average Au g/t
No. Data Points
Elevation Range m.a.s.l.
No. Data Points
Block Model
Composites

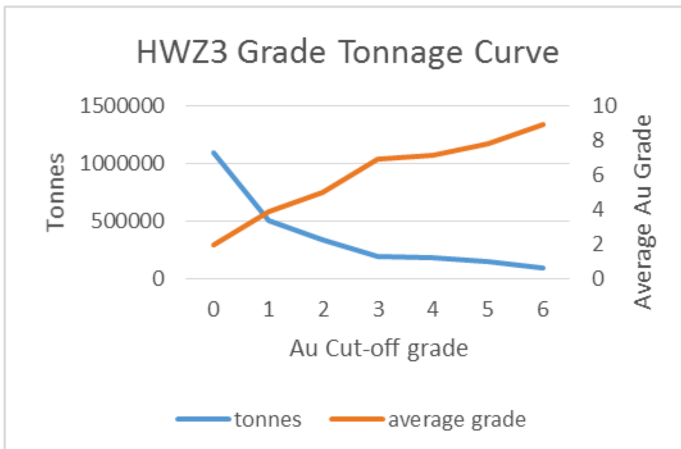
APPENDIX 14-2 GRADE TONNAGE CURVES








HWZ2 Grade Tonnage Curve
Tonnes
1,500,000
1,000,000
500,000
-
0
1
2
3
4
5
6
Au Cut-off Grade
10
5
0
Average Au Grade
tonnes
average grade


HWZ3 Grade Tonnage Curve
Tonnes
1500000
1000000
500000
0
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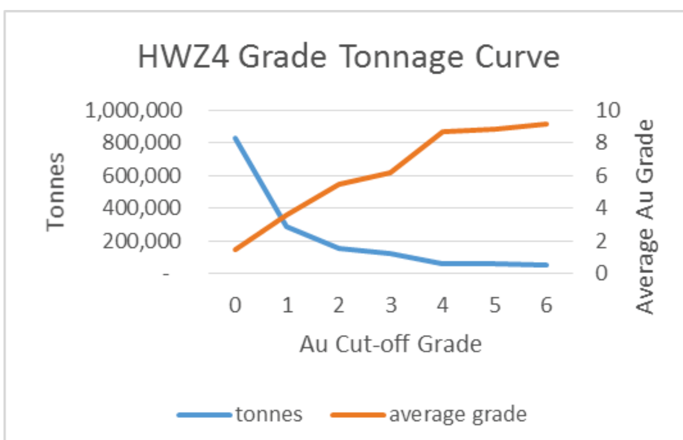
1
2
3
4
5
6
Au Cut-off grade
10
8
6
4
2
0
Average Au Grade
tonnes
average grade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Tonnes
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800,000
600,000
400,000
200,000
-
0
1
2
3
4
5
6
Au Cut-off Grade
10
8
6
4
2
0
Average Au Grade
tonnes
average grade

APPENDIX 14-3 BRADSHAW RESOURCE ESTIMATIONS CUT OFF GRADE SENSITIVITIES

		2 g cut off			2.5 g cut off		
Category	Zone	Tonnes	Au grade	OZ	Tonnes	Au grade	OZ
Indicated	MZ1	712,528	4.61	105,605	564,771	5.23	94,963
	MZ2	828,553	5.06	134,788	687,080	5.65	124,806
	HWZ1	464,062	5.37	80,118	404,807	5.82	75,745
	HWZ2	453,389	4.35	63,408	340,063	5.03	54,993
	HWZ3	335,674	4.95	53,420	207,579	6.65	44,380
	HWZ4	156,677	5.5	27,704	150,111	5.63	27,171
	HWZ5	85,655	5.91	16,275	53,094	8.01	13,673
	HWZ6	62,196	8.61	17,217	55,666	9.36	16,751
	TOTAL	**3,098,734**	**5.00**	**498,535**	**2,463,171**	**5.71**	**452,482**
Inferred	MZ1	893,873	4.94	141,966	753,683	5.42	131,332
	MZ2	1,616,491	3.83	199,046	1,557,872	3.89	194,833
	HWZ1	870,816	4.6	128,785	809,645	4.79	124,684
	HWZ2	650,858	5.89	123,249	566,913	6.45	117,559
	HWZ3	615,024	9.2	181,912	496,642	10.88	173,721
	HWZ4	597,665	5.98	114,905	514,614	6.62	109,527
	TOTAL	**5,244,727**	**5.28**	**889,862**	**4,699,369**	**5.64**	**851,656**

		3 g cut off			3.5 g cut off		
Category	Zone	Tonnes	Au grade	OZ	Tonnes	Au grade	OZ
Indicated	MZ1	412,503	6.14	81,429	287,263	7.40	68,343
	MZ2	634,583	5.88	119,963	485,139	6.71	104,657
	HWZ1	345,637	6.35	70,563	328,791	6.51	68,815
	HWZ2	299,258	5.33	51,281	258,599	5.64	46,891
	HWZ3	194,029	6.93	43,230	188,872	7.03	42,688
	HWZ4	127,096	6.16	25,171	114,593	6.47	23,837
	HWZ5	53,094	8.01	13,673	53,094	8.01	13,673
	HWZ6	55,666	9.36	16,751	31,114	14.07	14,074
	TOTAL	**2,121,866**	**6.19**	**422,059**	**1,747,465**	**6.82**	**382,977**
Inferred	MZ1	331,752	8.64	92,153	331,752	8.64	92,153
	MZ2	1,078,096	4.36	151,121	792,467	4.77	121,529
	HWZ1	693,934	5.16	115,119	579,795	5.53	103,081
	HWZ2	566,913	6.45	117,559	496,901	6.87	109,751
	HWZ3	443,788	11.85	169,073	443,788	11.85	169,073
	HWZ4	514,614	6.62	109,527	514,614	6.62	109,527
	TOTAL	**3,629,097**	**6.47**	**754,553**	**3,159,317**	**6.94**	**705,114**

		4 g cut off		
Category	Zone	Tonnes	Au grade	OZ
Indicated	MZ1	246,777	8.00	63,471
	MZ2	432,095	7.08	98,354
	HWZ1	297,429	6.80	65,024
	HWZ2	220,826	5.96	42,313
	HWZ3	183,975	7.11	42,054
	HWZ4	64,613	8.68	18,031
	HWZ5	53,094	8.01	13,673
	HWZ6	31,114	14.07	14,074
	TOTAL	**1,529,923**	**7.26**	**356,995**
Inferred	MZ1	331,752	8.64	92,153
	MZ2	792,467	4.77	121,529
	HWZ1	489,654	5.88	92,565
	HWZ2	347,417	8.13	90,808
	HWZ3	443,788	11.85	169,073
	HWZ4	514,614	6.62	109,527
	TOTAL	**2,919,692**	**7.20**	**675,655**

APPENDIX D
VENTILATION

Location	Month	Propane Litres			
		Actual 1998	Actual 1999	Budget 1999	Actual 2000
Trout Lake Main Downcast	1	881,839	805,381	981,420	866,550
Note: Budget litres are	2	370,309	388,522	802,245	
based on a set point of 34^0F	3	400,385	233,095	382,000	154,444
and a ten year average temp.	4	73,696	0	0	17,544
	5	0	-32,496	0	
	9	35,505	0	0	28,377
	10	48,292	56,466	0	30,737
	11	183,745	50,024	523,090	127,999
	12	675,207	72,700	912,065	945,300
Total Litres		**2,668,978**	**1,573,692**	**3,600,820**	**2,170,951**
Total $ @ $0.20/litre		**$533,796**	**$314,738**	**$720,164**	
Trout Lake Shaft	1	259,685	221,865	208,525	277,443
Note: Budget litres are	2	159,414	199,159	178,080	631,019
based on a set point of 50^0F	3	178,504	130,832	120,610	166,437
and a ten year average temp.	4	34,112	60,819	56,955	69,417
	5	8,141	13,934	0	44,896
	9	53,098	16,361	0	53,385
	10	48,156	109,476	58,850	64,331
	11	132,638	146,682	139,220	153,159
	12	225,192	527,774	198,355	281,647
Total Litres		**1,098,940**	**1,426,902**	**960,595**	**1,741,734**
Total $ @ $0.20/litre		**$219,788**	**$285,380**	**$192,119**	
Callinan Main Downcast	1	344,612	451,045	580,920	383,000
Note: Budget litres are	2	101,706	308,030	484,740	192,000
based on a set point of 40^0F	3	199,979	147,389	277,210	187,996
and a ten year average temp.	4	0	25,267	61,805	67,701
	10	0	18,957	63,865	
	11	168,351	53,551	346,000	233,001
	12	383,523	339,017	545,780	463,820
Total Litres		**1,198,171**	**1,343,256**	**2,360,320**	**1,527,518**
Total $ @ $0.20/litre		**$239,634**	**$268,651**	**$472,064**	
Konuto Mine Air Heating	1	163,849	233,462	320,425	
Note: Budget litres are	2	155,566	159,772	268,855	
based on a set point of 42^0F	3	149,649	126,794	160,580	
and a ten year average temp.	4	60,821	40,881	46,735	
	5	-17,144	16,315	0	
	6	0	-14,734	0	
	9	0	14,340	0	
	10	96,414	78,178	48,290	
	11	105,427	152,112	196,310	
	12	210,789	210,730	301,930	
Total Litres		**925,371**	**1,017,850**	**1,343,125**	
Total $ @ $0.20/litre		**$185,074**	**$203,570**	**$268,625**	

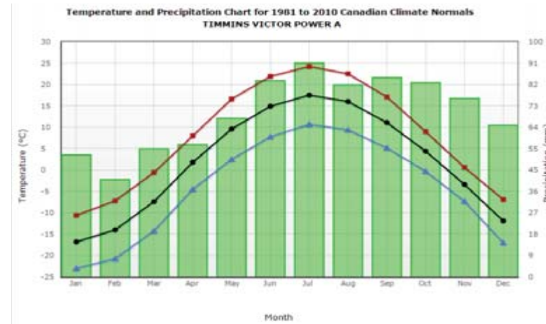
Baseline data - Timmins On **Gowest Project**

input green highlighted boxes

Month	Days in month	Mean average temperature outside (°F)	(°C)
October	31	39.9	4.4
November	30	25.9	-3.4
December	31	10.6	-11.9
January	31	1.8	-16.8
February	29	6.8	-14
March	31	18.7	-7.4
April	30	35.2	1.8
	Max Low	-50.1	-45.6

Area	Temperature setpoint (F)	Temperature setpoint (C)	Airflow (Acfm)	Airflow (M^3/sec)
Gowest Project	35.6	2	400,000	189

$ 0.60 Price of propane



1981 to 2010 Canadian Climate Normals station data Timmins

Temperature

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Daily Average (°C)	-16.8	-14	-7.4	1.8	9.6	14.9	17.5	16	11.1	4.4	-3.4	-11.9	1.8
Standard Deviation	3.6	3.1	2.6	2.3	2.1	1.6	1.2	1.4	1.6	1.5	2.6	3.7	1
Daily Maximum (°C)	-10.6	-7.2	-0.6	8	16.6	21.9	24.2	22.5	17.1	9	0.6	-6.9	7.9
Daily Minimum (°C)	-23	-20.7	-14.2	-4.5	2.5	7.8	10.7	9.4	5.2	-0.3	-7.4	-17	-4.3
Extreme Maximum (°C)	7.6	11.7	19.9	29.9	34.2	38.8	**38.9**	36.7	33.4	28.3	19.8	14.2	
Date (yyyy/dd)	2006/ 27	1994/ 19	1990/ 15	1986/ 28	2010/ 24	1995/ 18	**1975/ 31**	1976/ 20	2002/ 08	1968/ 16	2008/ 05	1982/ 03	
Extreme Minimum (°C)	-44.2	**-45.6**	-37.8	-29.4	-11.1	-3.2	-0.5	-1.7	-6.4	-13	-33.9	-43.9	
Date (yyyy/dd)	1982/ 18	**1962/ 01**	1989/ 03	1964/ 01	1958/ 02	1980/ 19	1992/ 01	1965/ 30	2000/ 28	1981/ 24	1975/ 26	1975/ 19	

Propane Cost Estimate [+ 3 degrees C (37.4 F)]

Basic Calculation is as Follows

Cost of Propane = [(Airflow in Acfm) * (Temp setpoint-Temp outside) * (1.08 Btu/(Acfm*F*hr)) * 24 hr/day * (Days in Month) * ($price/ l of Propane)] / [(21897 Btu / l of Propane)]

Area	Temperature setpoint (F)	Temperature setpoint (C)	Airflow (Acfm)	Propane Cost November	Propane Cost December	Propane Cost January	Propane Cost February	Propane Cost March	Propane Cost April	**Total Winter cost**	**Cost per 1,000 Acfm**	**Cost per degree setpoint**
Gowest Project	35.6	2.0	400,000	$ 82,842	$ 220,349	$298,026	$237,275	$149,013	$ 3,068	**$ 990,572**	**$ 2,476**	**$ 60,512**

Propane Consumption Sheet [+ 3 degrees C (37.4 F)]

Basic Calculation is as Follows

Volume of Propane = [(Airflow in Acfm) * (Temp setpoint-Temp outside) * (1.08 Btu/(Acfm*F*hr)) * 24 hr/day * (Days in Month)] / [(21897 Btu / l of Propane)]

Area	Temperature setpoint (C)	Airflow (Acfm)	Propane Consumption November (Litre)	Propane Consumption December (Litre)	Propane Consumption January (Litre)	Propane Consumption February (Litre)	Propane Consumption March (Litre)	Propane Consumption April (Litre)	**Total Winter Consumption (Litre)**
Gowest Project	2	400,000	138,070	367,248	496,709	395,458	248,355	5,114	**1,650,954**

	1.085 x Flow (CFM) x Temperature Rise (°F) = BTU/hr

Area	Flow (CFM)	Max. Temperature Rise (°F)	BTU/hr
Main Intakes	400,000	85.7	37,185,120

Ramp Development

	1.085 x Flow (CFM) x Temperature Rise (°F) = BTU/hr

Area	Flow (CFM)	Max. Temperature Rise (°F)	BTU/hr
Ramp	78,000	85.7	7,252,791

1981 to 2010 Canadian Climate Normals station data Timmins

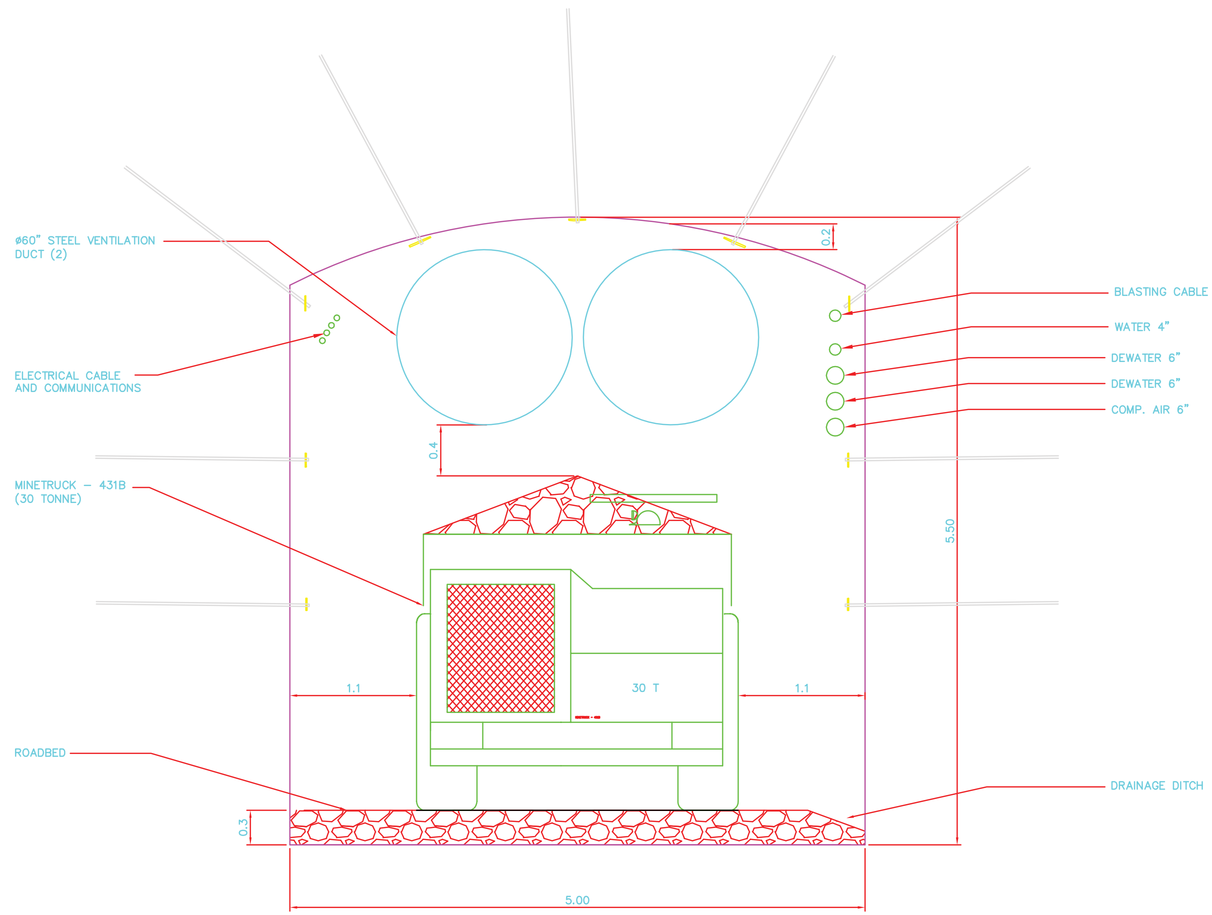
Temperature

	Feb	Mar	Apr	Aug	Sep	Oct	Nov	Dec	Year
Daily Average (°C)	-14	-7.4	1.8	16	11.1	4.4	-3.4	-11.9	1.8
Standard Deviation	3.1	2.6	2.3	1.4	1.6	1.5	2.6	3.7	1
Daily Maximum (°C)	-7.2	-0.6	8	22.5	17.1	9	0.6	-6.9	7.9
Daily Minimum (°C)	-20.7	-14.2	-4.5	9.4	5.2	-0.3	-7.4	-17	-4.3
Extreme Maximum (°C)	11.7	19.9	29.9	36.7	33.4	28.3	19.8	14.2	
Date (yyyy/dd)	1994/ 19	1990/ 15	1986/ 28	1976/ 20	2002/ 08	1968/ 16	2008/ 05	1982/ 03	
Extreme Minimum (°C)	**-45.6**	-37.8	-29.4	-1.7	-6.4	-13	-33.9	-43.9	
Date (yyyy/dd)	**1962/ 01**	1989/ 03	1964/ 01	1965/ 30	2000/ 28	1981/ 24	1975/ 26	1975/ 19	

max low (-f)

-50.1		**85.7**	**temp rise (f)**
-45.6		**47.6**	**temp rise (C)**

APPENDIX E
LEVEL PLANS



DRIFT PROFILE WITH VENTILATION DUCTING & 30 TONNE TRUCK

4.75m WIDE x 5.25m HIGH ARCH BACK DRIFT

PRELIMINARY

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			A	PRELIMINARY –	THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED		

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DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: DATE:

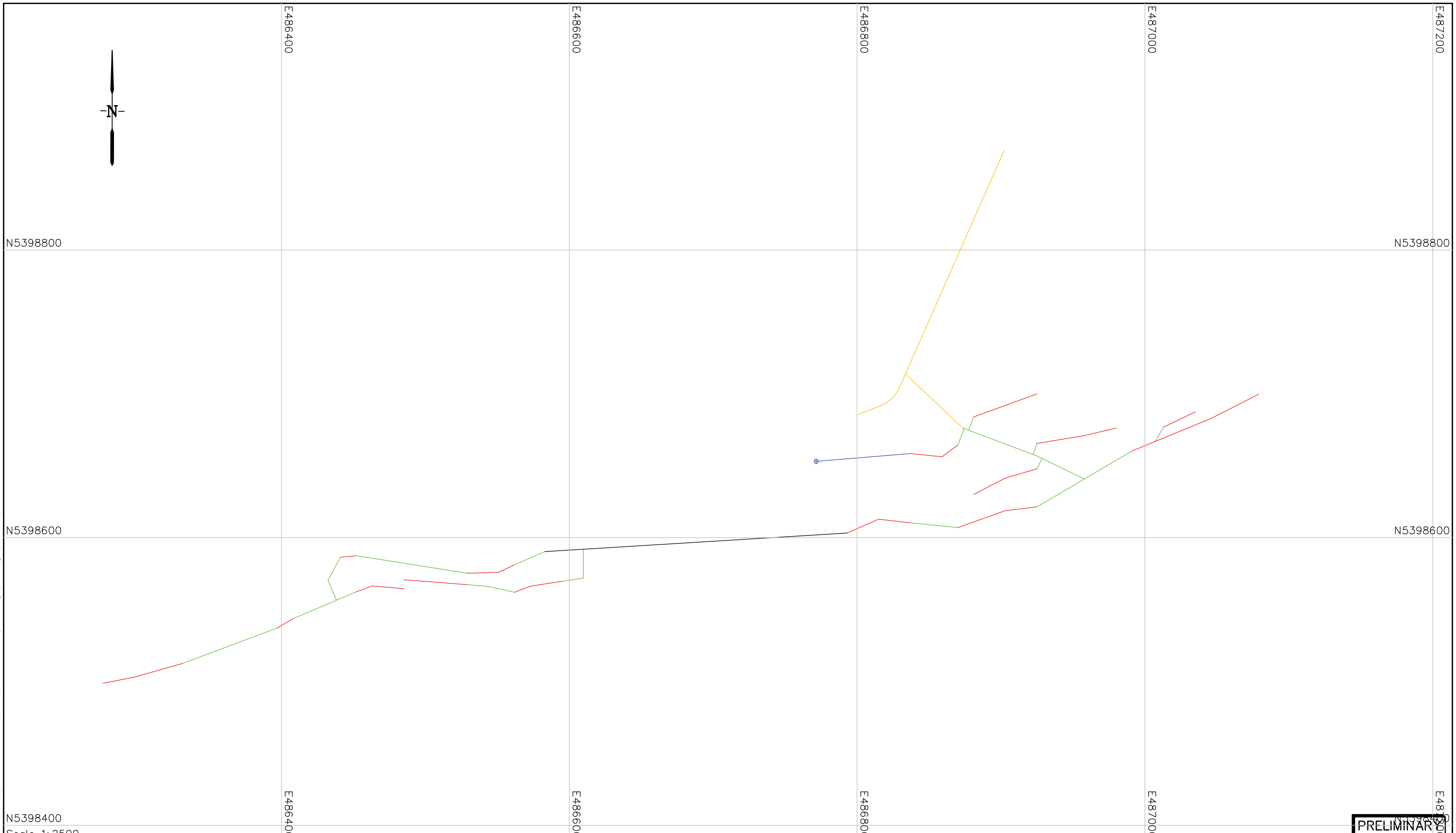
Stantec

CLIENT: Gowest

PROPERTY:

PROJECT:

SCALE: PROJ No DWG No REV No



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PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
45 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
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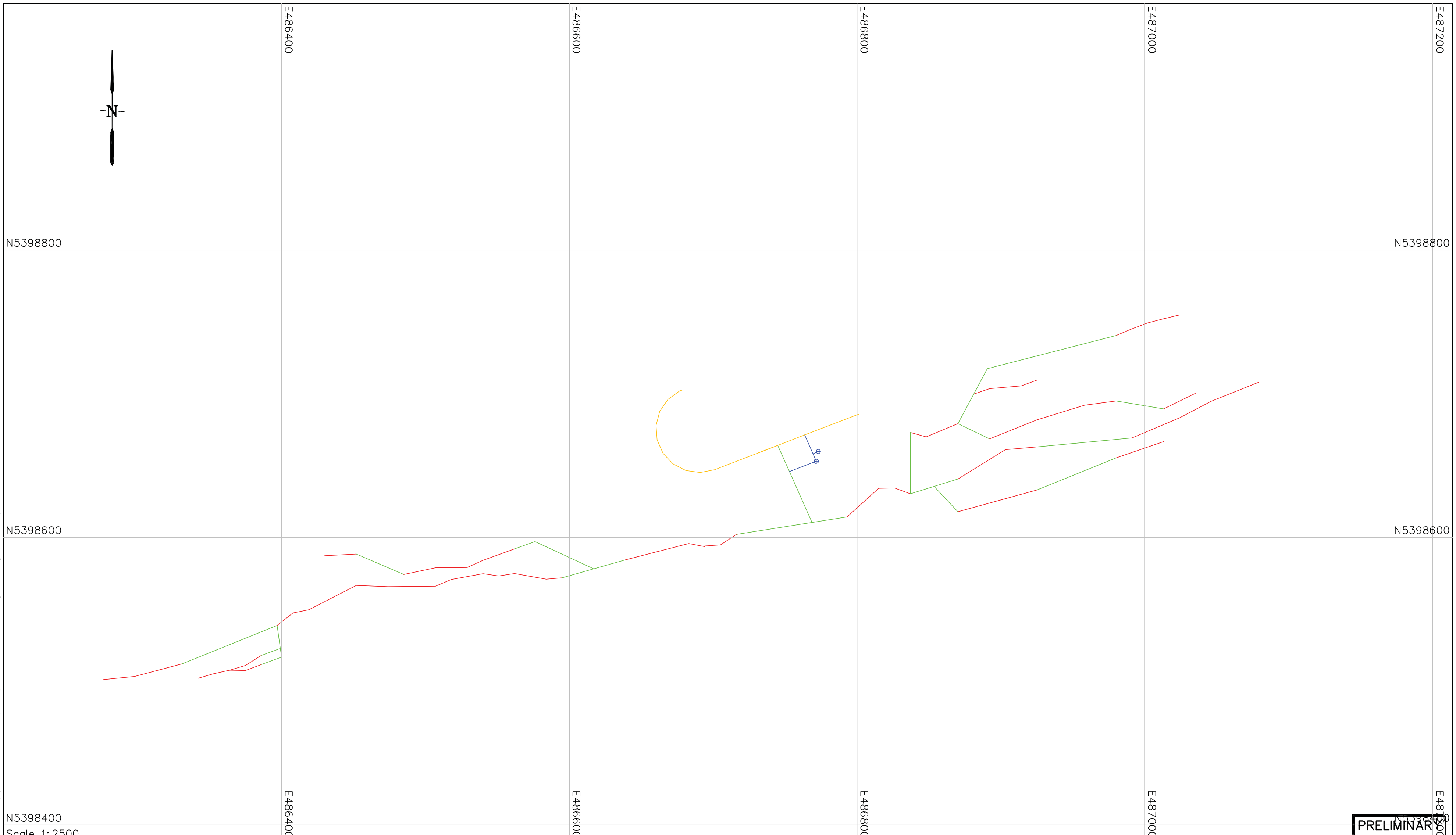

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PROGRESS PRINT
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BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
75 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE: SCALE | PROJ No 169514568 | DWG No 0000000 | REV No A

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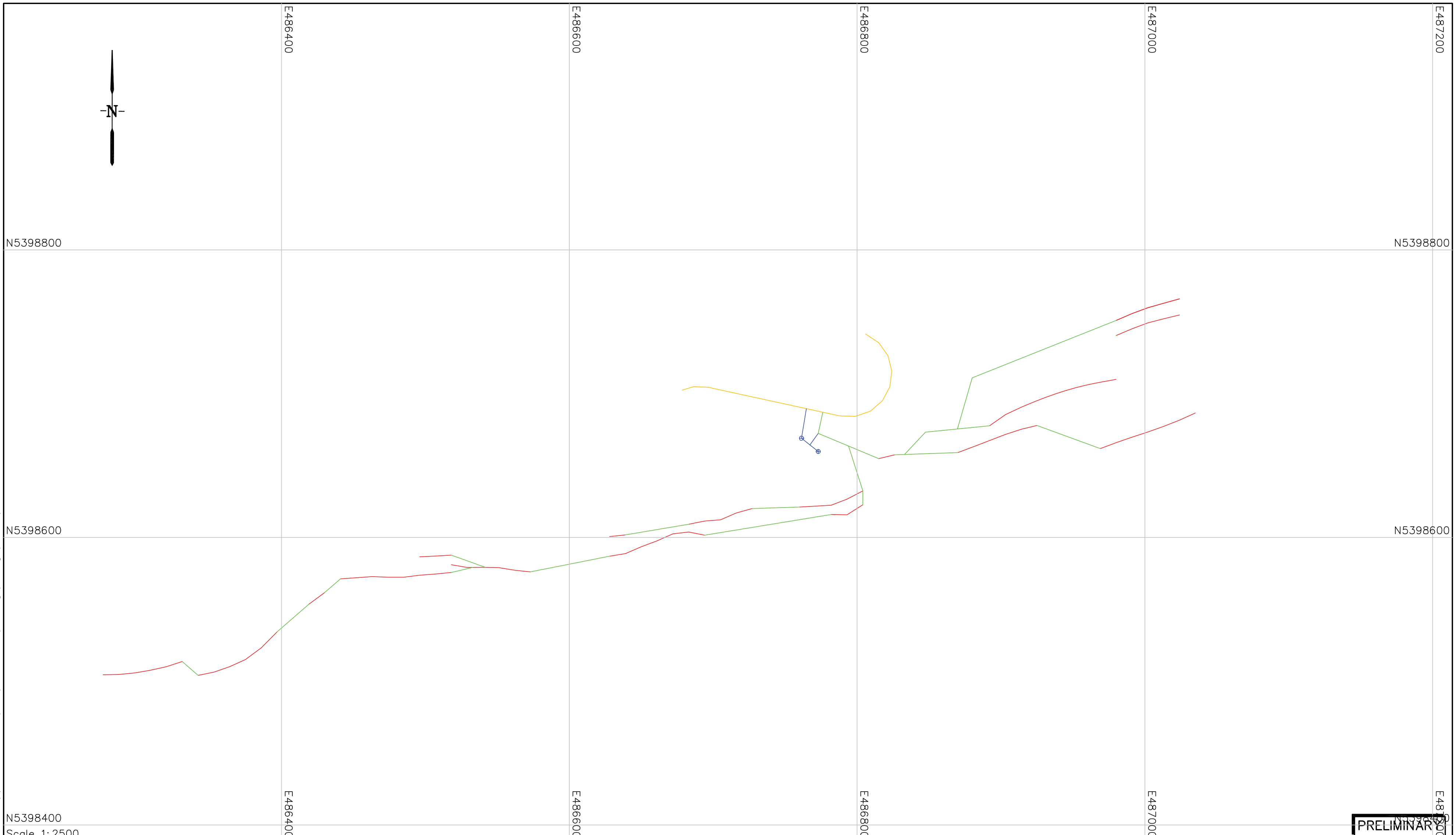

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PROJ No 169514568
DWG No 0000000

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PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
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TITLE LINE 2
TITLE LINE 3

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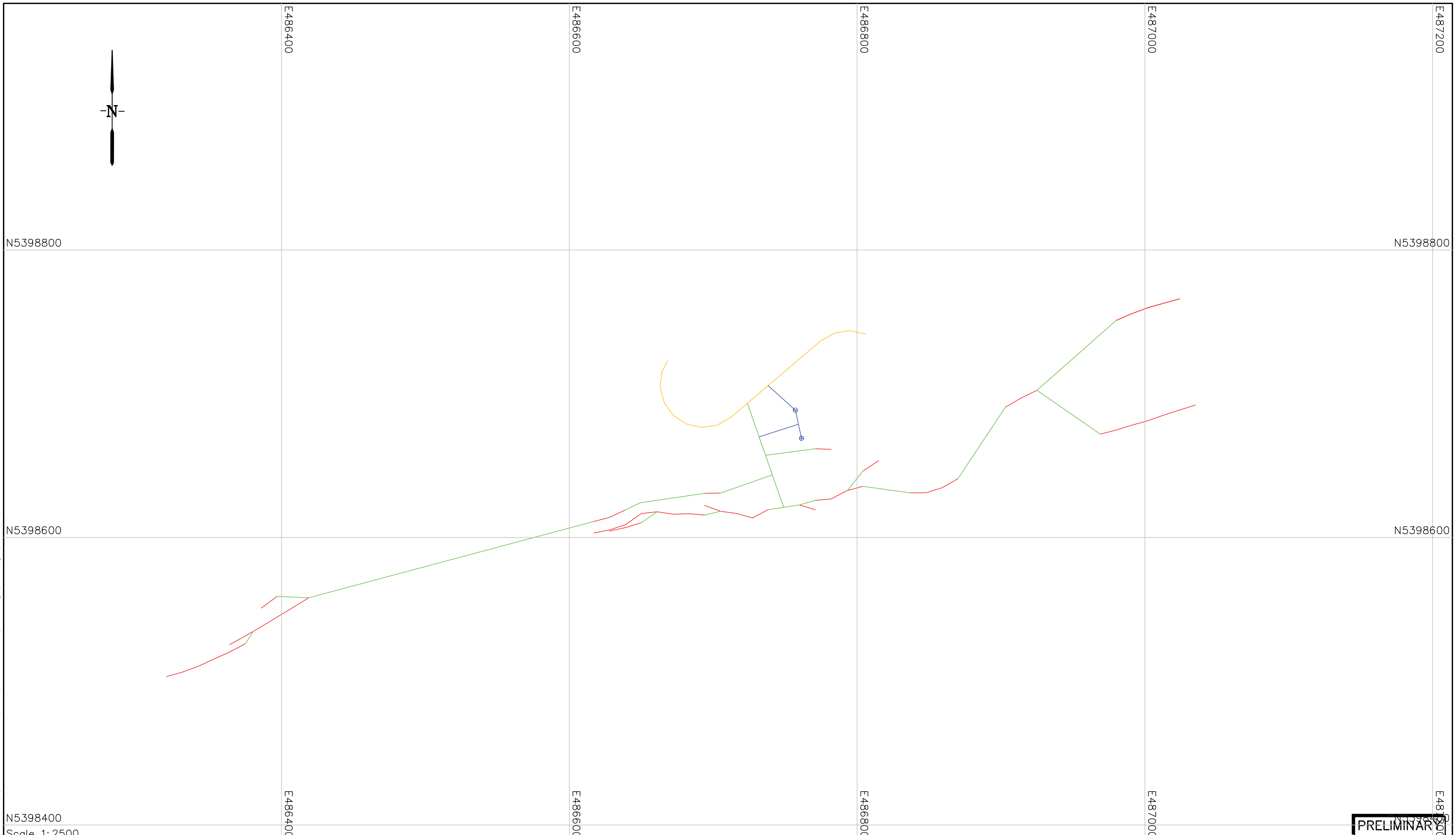

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A	169514568	0000000	---	---	A	PRELIMINARY –		

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DRWN CHKD DSGN ENG CHECK PM MGR

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE-FEASIBILITY
135 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
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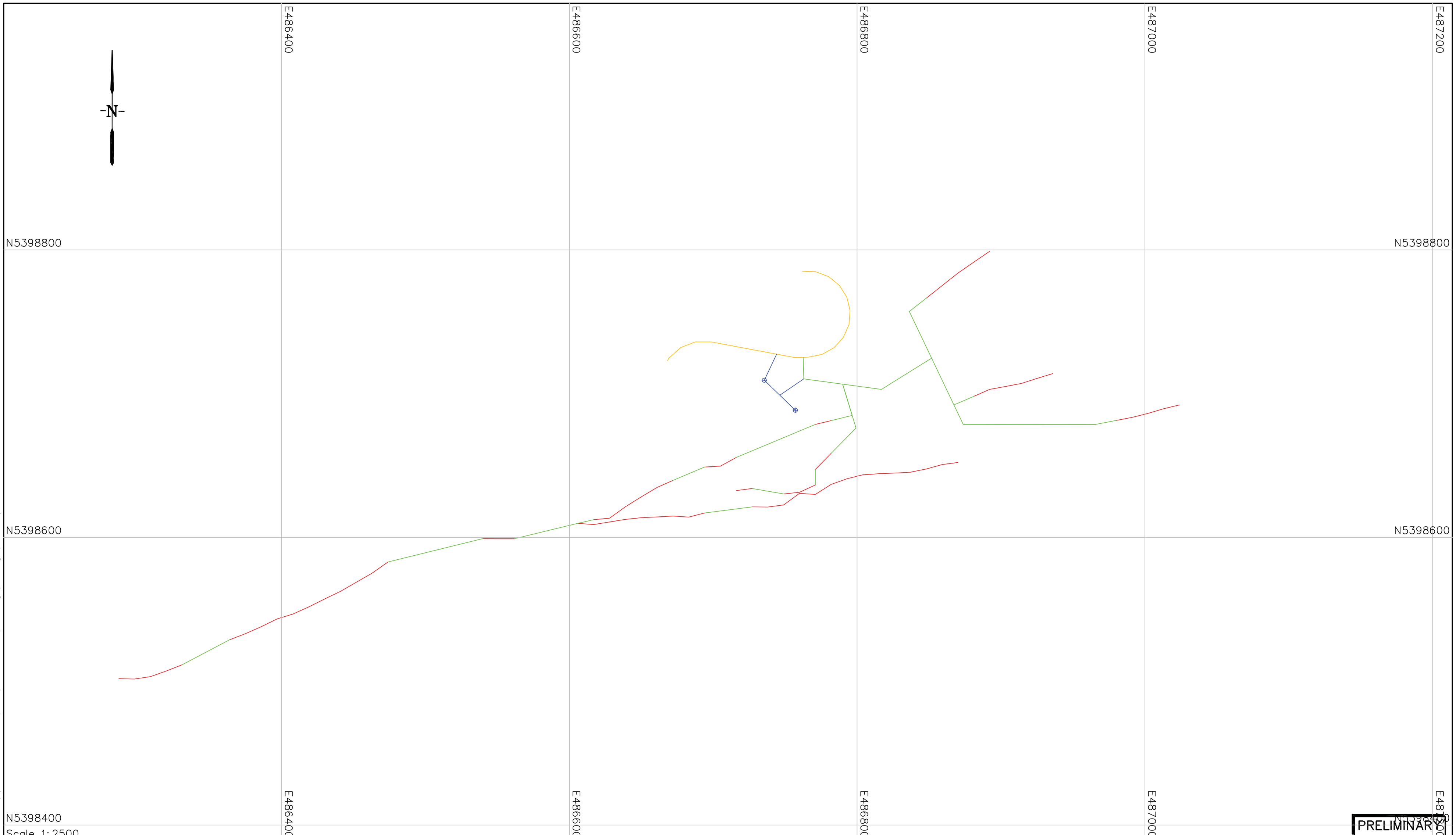

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BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE-FEASIBILITY
165 LEVEL – PLAN
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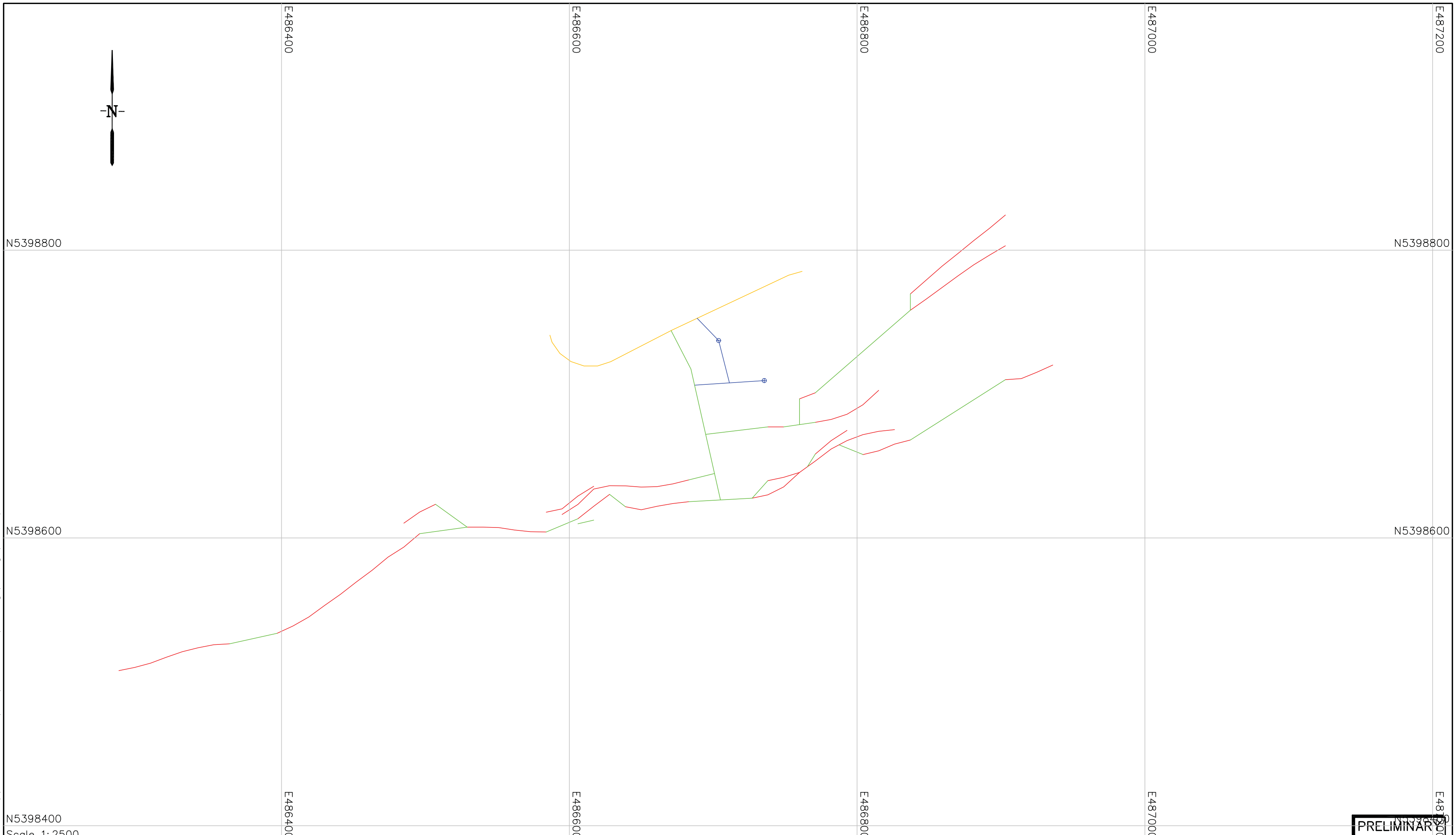

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NOT FOR CONSTRUCTION

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BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
195 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

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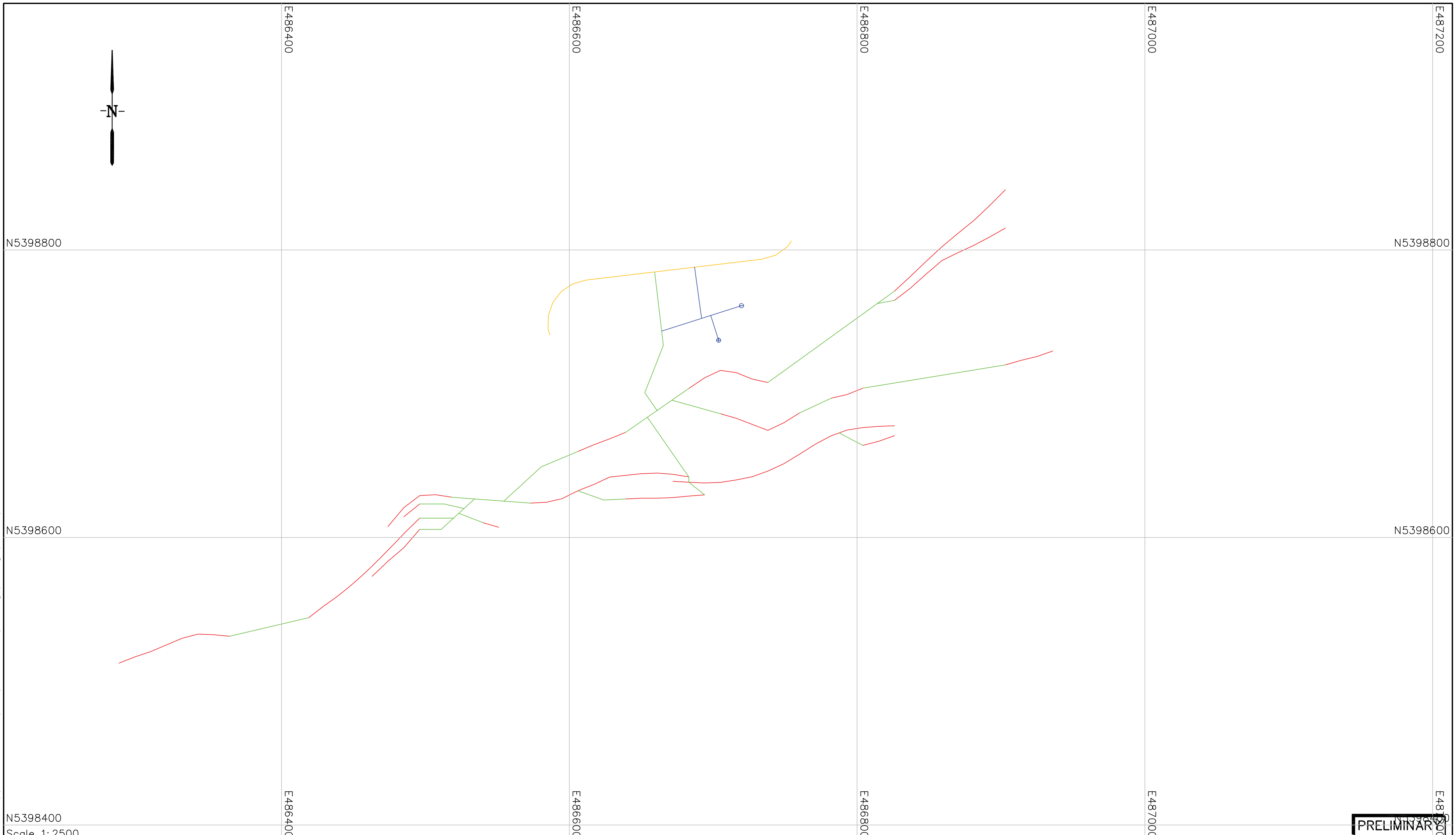
Jun 15, 2015 – 1:05pm



REV No	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
A	---	---	A	PRELIMINARY –		

PROJ No 169514568 DWG No 0000000

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NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE-FEASIBILITY
225 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE: SCALE | PROJ No 169514568 | DWG No 0000000 | REV No A

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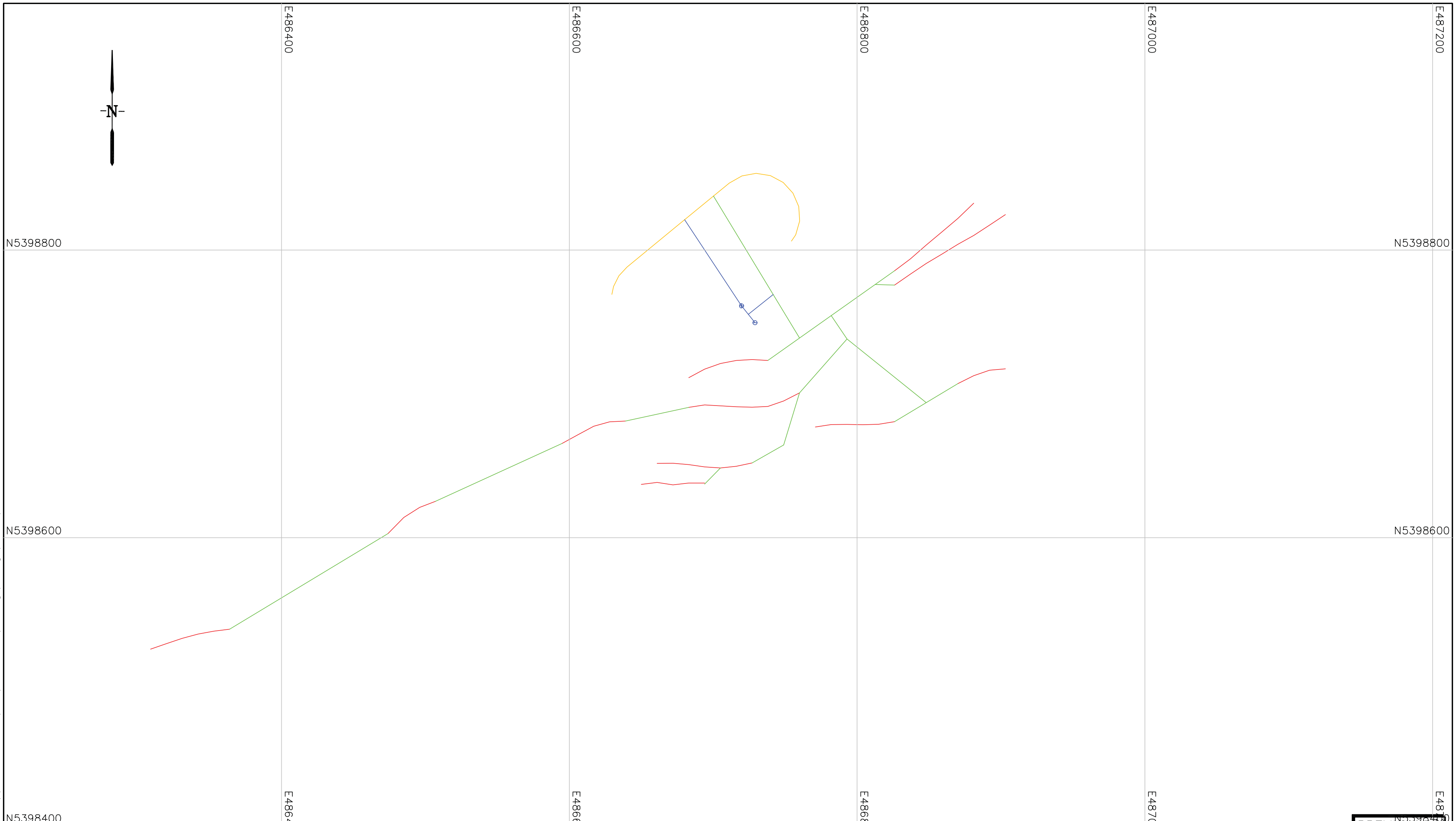
Jun 15, 2015 – 1:05pm



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REV No			REV	DESCRIPTION	DATE	BY
A	---	---	A	PRELIMINARY –		
PROJ No 169514568 DWG No 0000000	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED	YY.MM.DD	BY

NOT FOR CONSTRUCTION

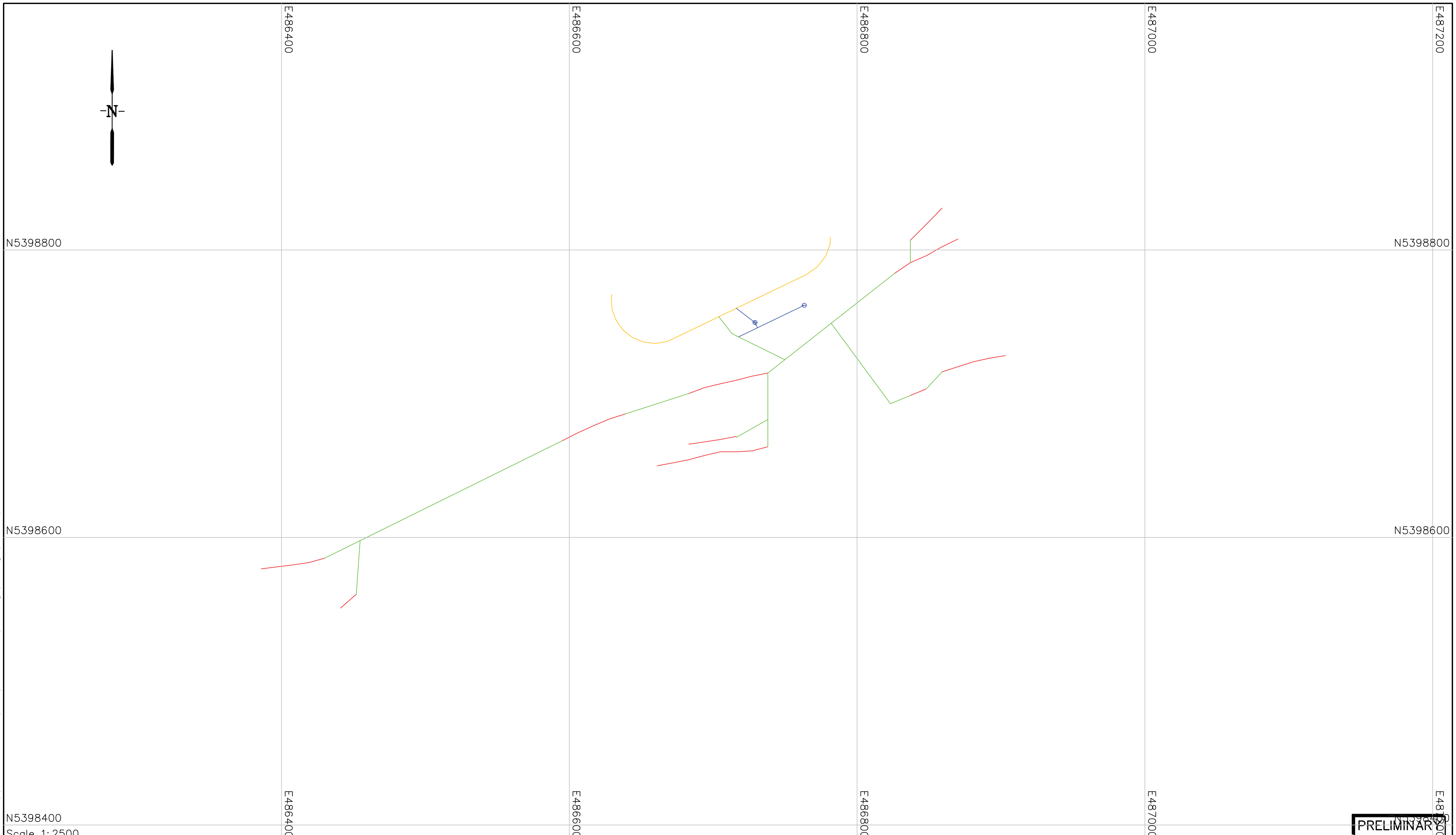
PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
255 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A



REV No								
A	---	---	A	PRELIMINARY –				
PROJ No 169514568 DWG No 0000000	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED	YY.MM.DD	BY	DSGN / CHKD / ENG / CHECK / PR MGR	

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

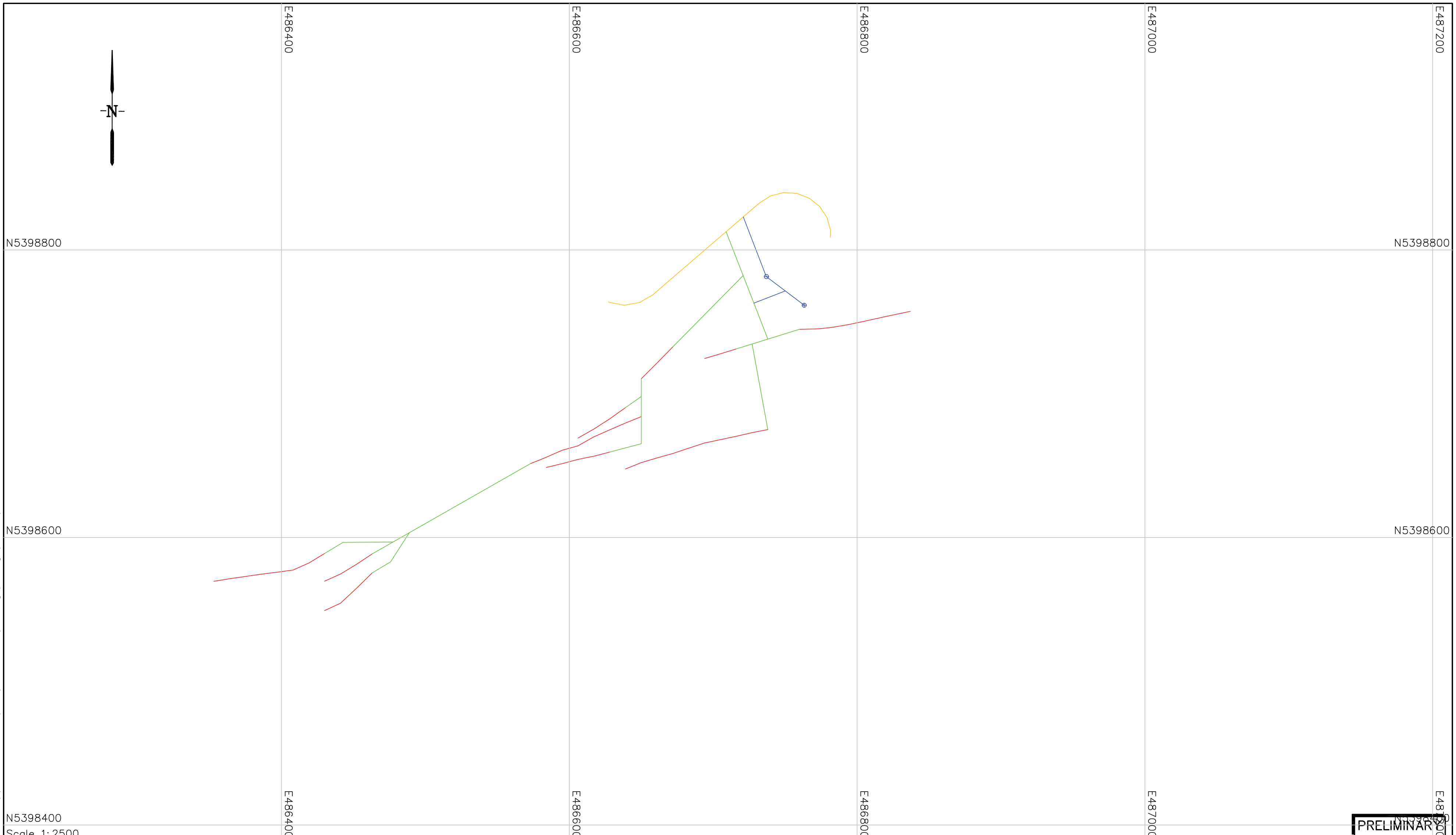
ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
285 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

Jun 15, 2015 – 1:07pm \\CD1211–N02S:\212\169514568\drawing\mining_prelim\520U0010-0001 – June 6–2015.dxf



Scale 1:2500

REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY
A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE-FEASIBILITY
315 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

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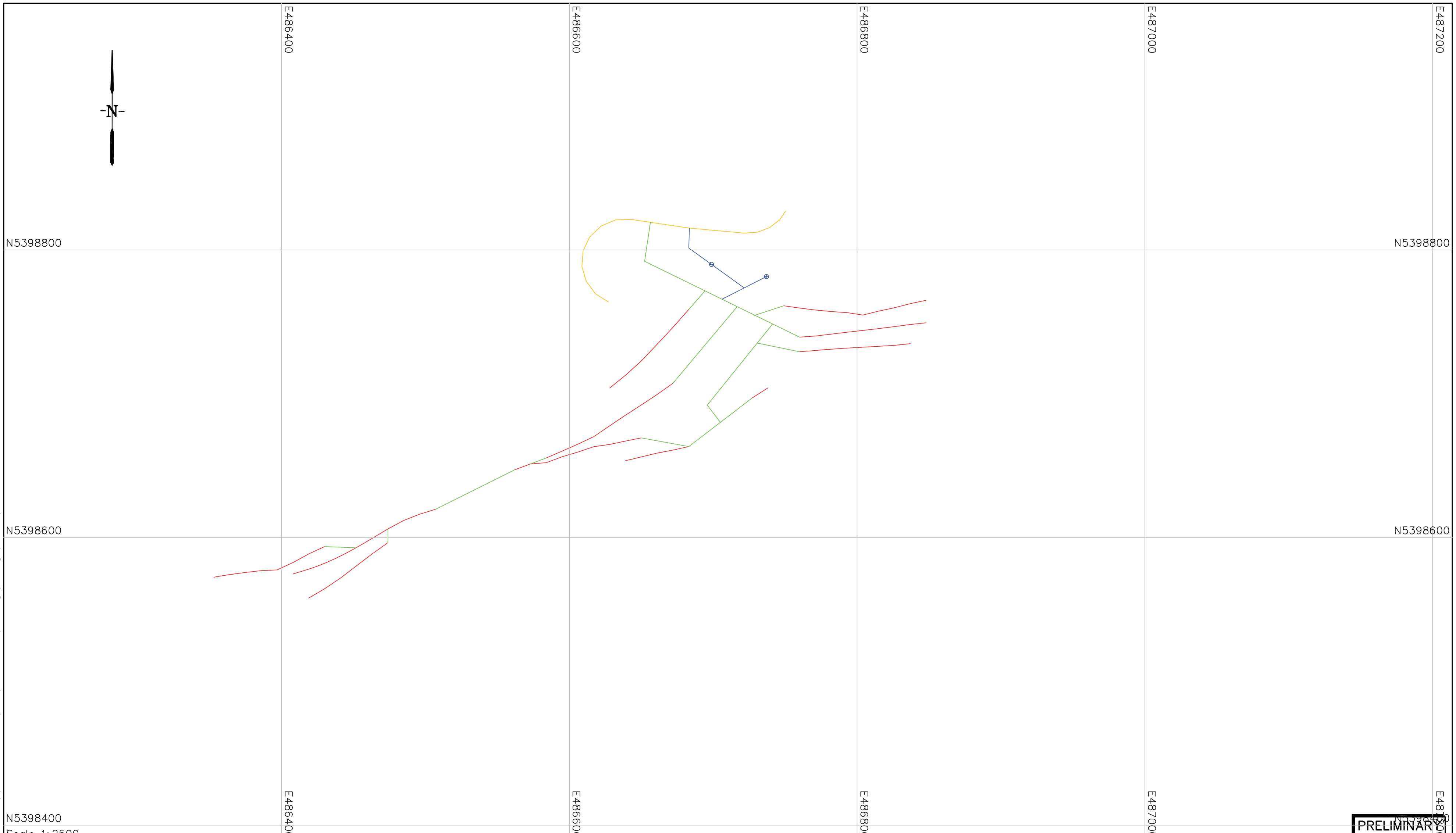
Jun 15, 2015 – 1:08pm



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PRELIMINARY

REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY
A	PRELIMINARY –		

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DRWN DSGN CHECK ENG PR MGR

NOT FOR CONSTRUCTION

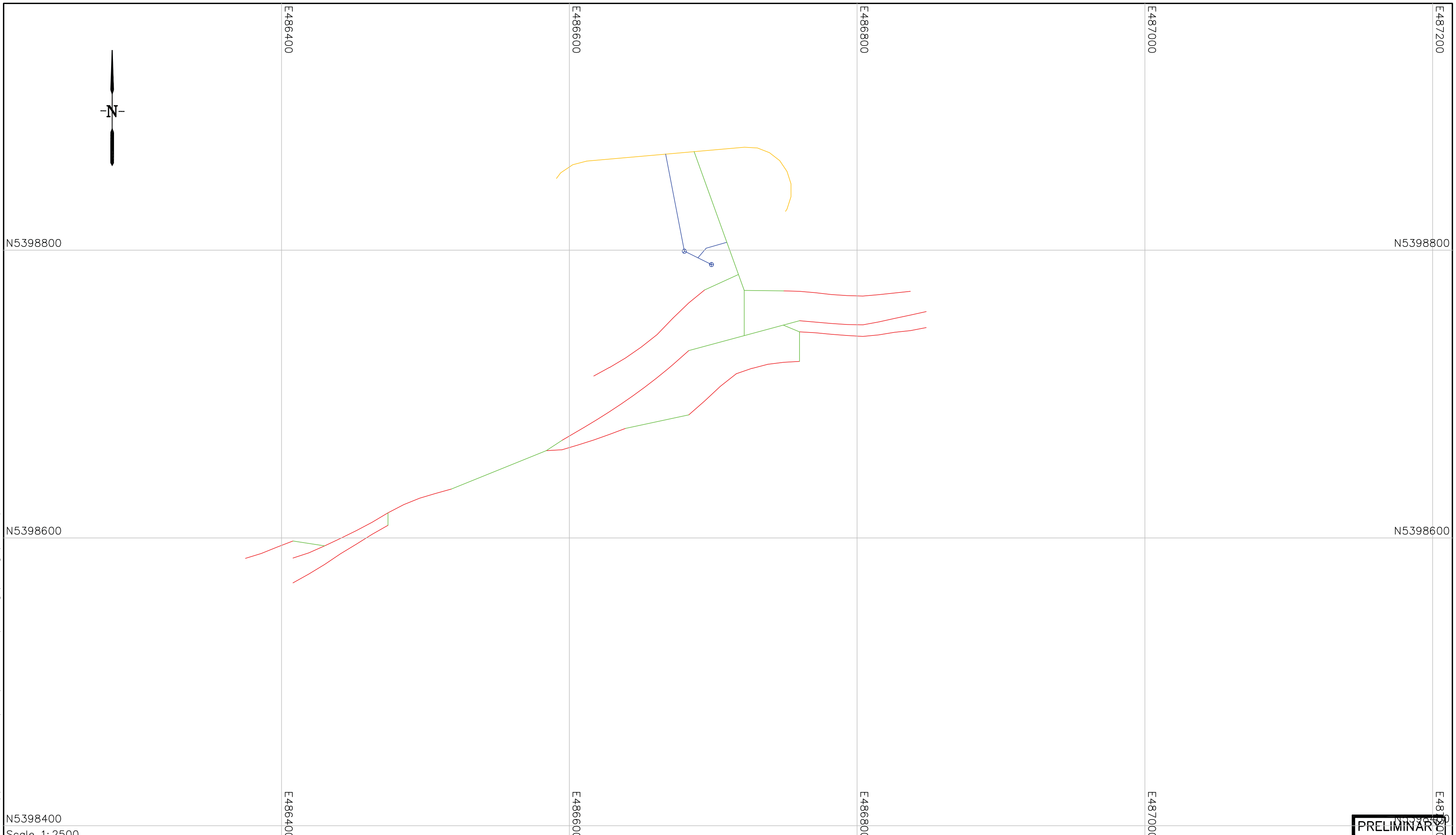
PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
345 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A



REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY	CHKD DSGN	DSGN ENG	CHECK	PR MGR
A	PRELIMINARY –						

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

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PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

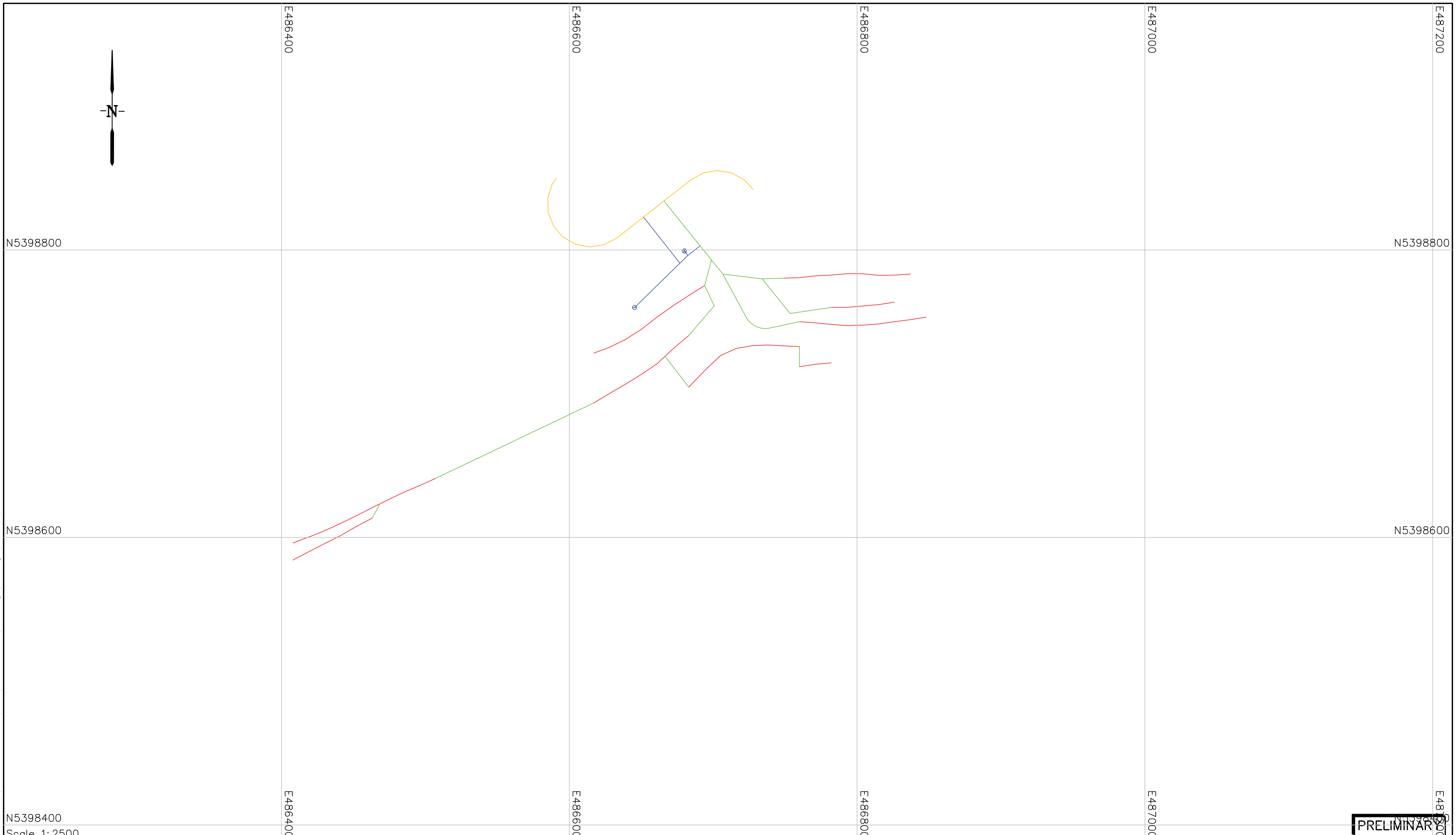
ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
375 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

Jun 15, 2015 – 1:09pm \\CD1211–N02S:\212\169514568\drawing\mining_prelim\520U0010-0001 – June 6–2015.dxf



REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY
A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
405 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE: SCALE
PROJ No: 169514568
DWG No: 0000000
REV No: A

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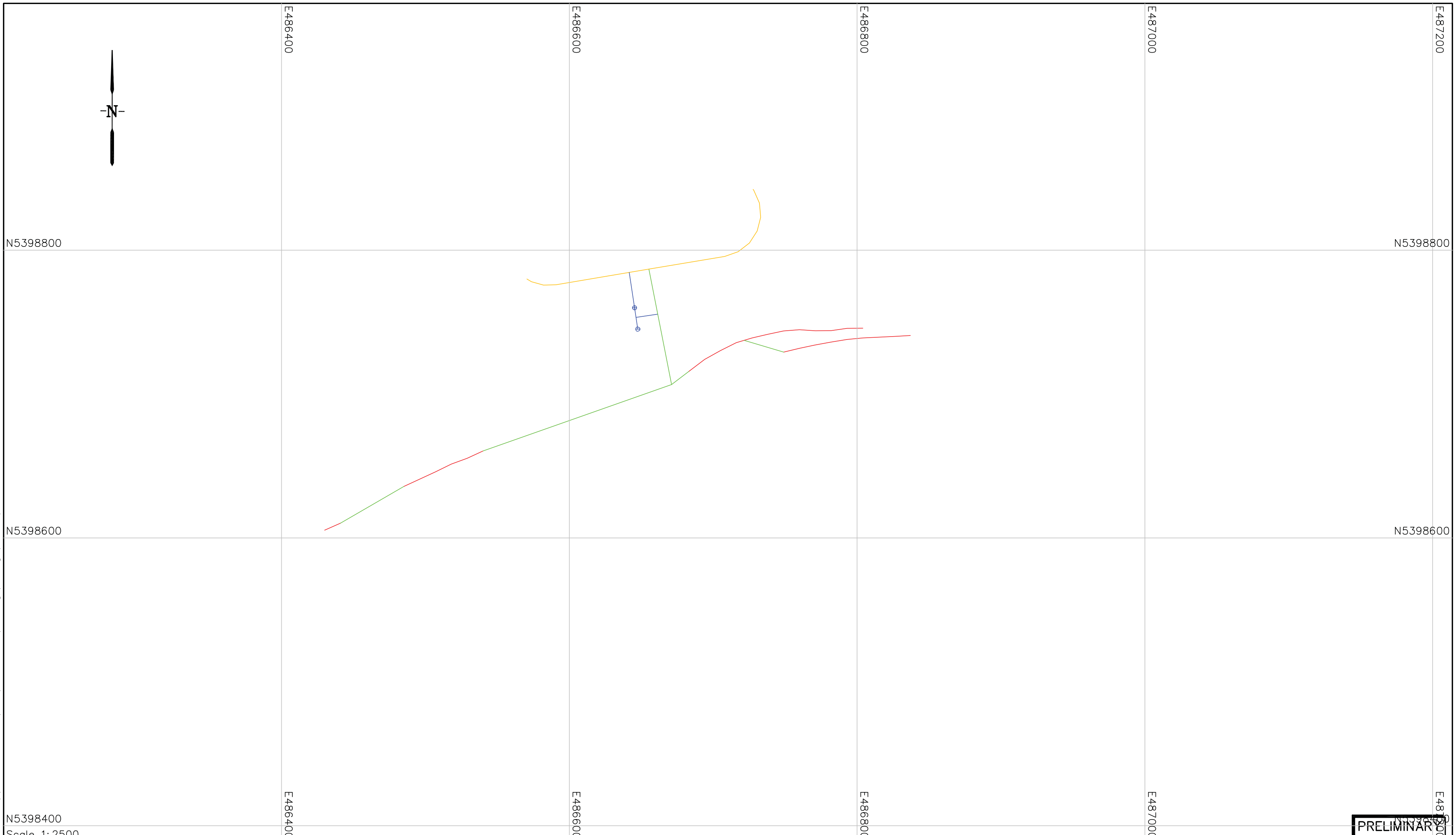
Jun 15, 2015 – 1:09pm



REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY
A	PRELIMINARY –		

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CHKD DSGN CHECK DSGN ENG PM MGR

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
435 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

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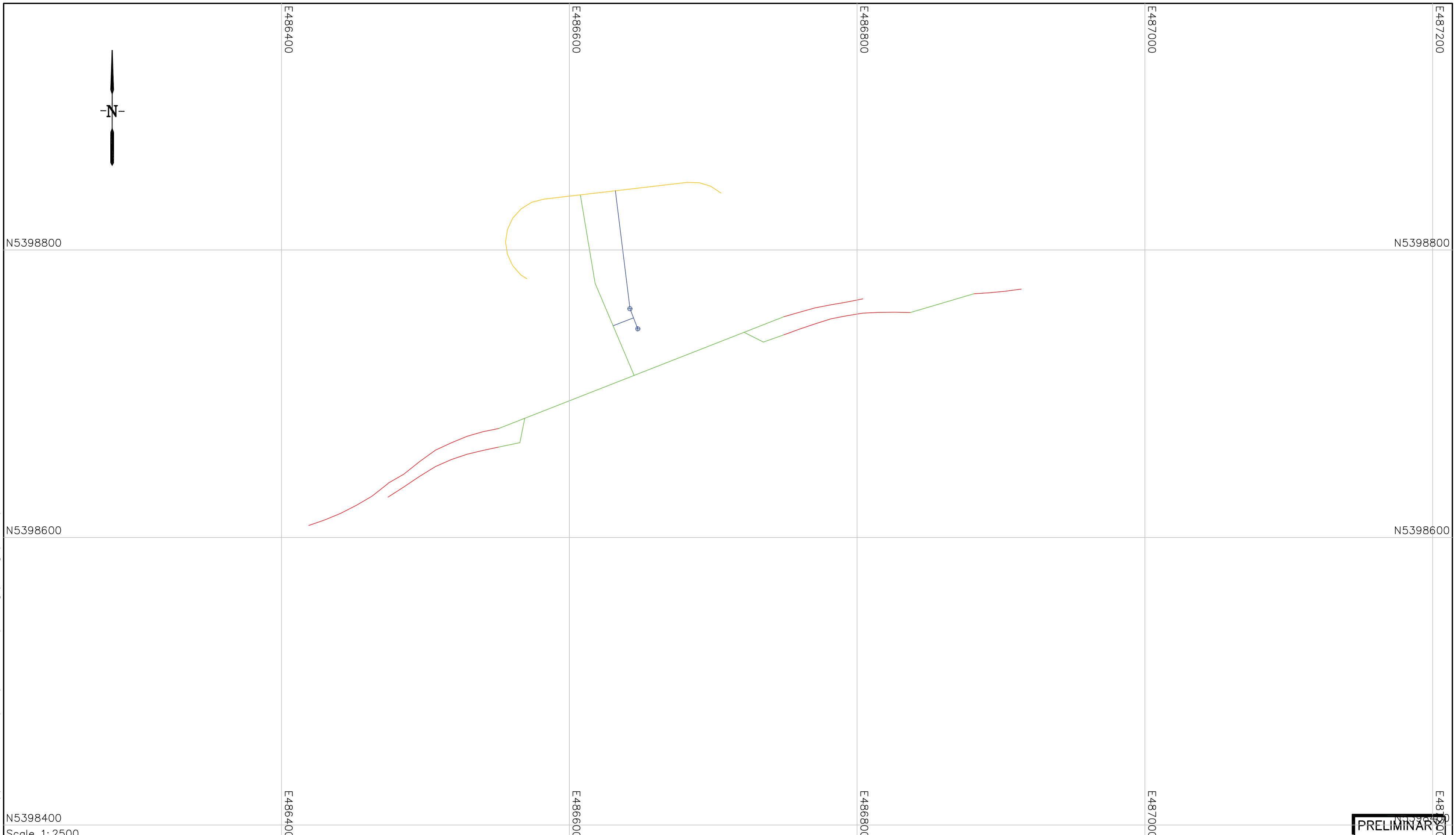
Jun 15, 2015 – 1:09pm



REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
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THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

CHKD DSGN / DSGN ENG / CHECK / PR MGR

PROJ No 169514568 DWG No 0000000 REV No A

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX | DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
465 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE:	PROJ No	DWG No	REV No
SCALE	169514568	0000000	A

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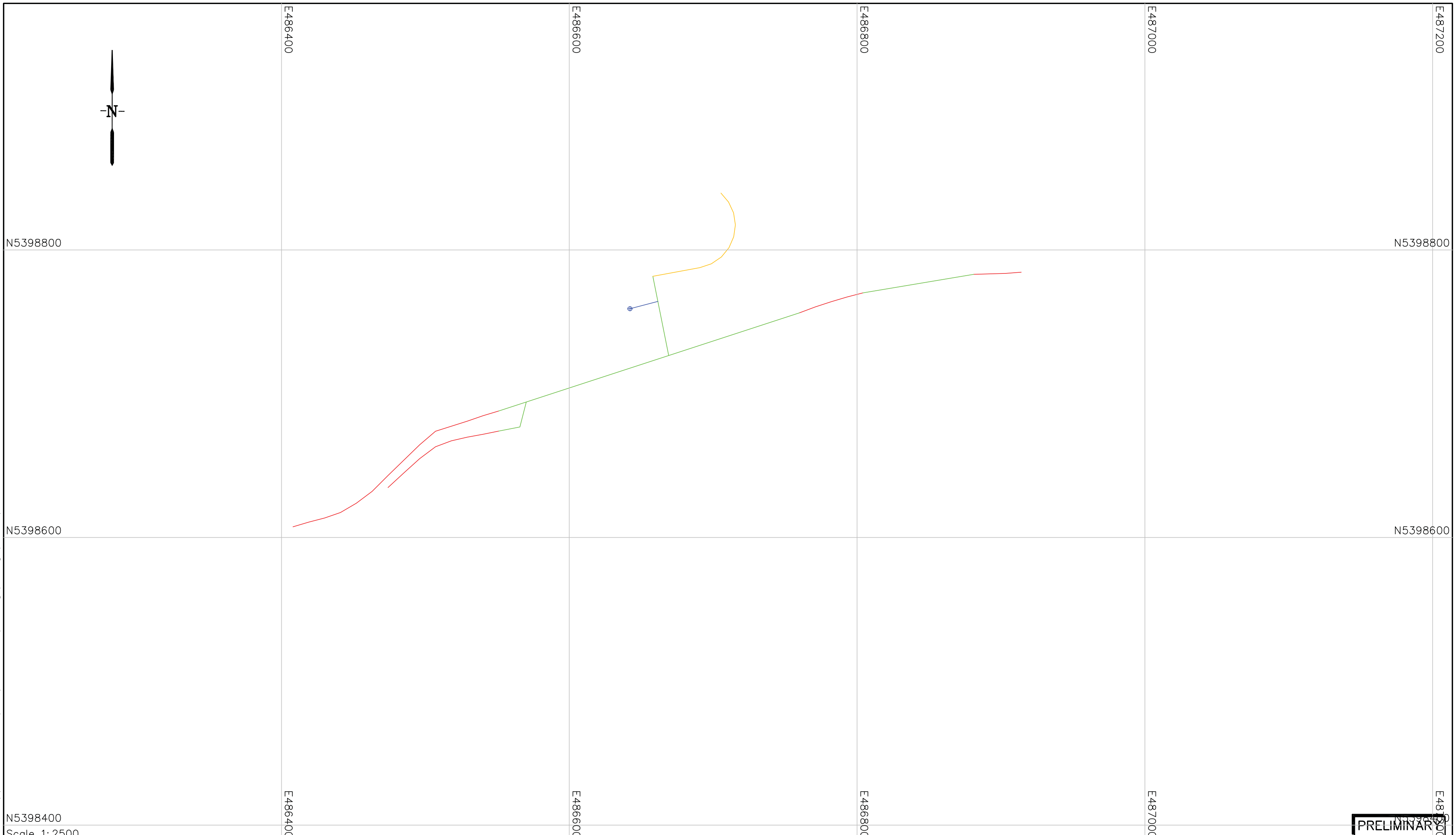
Jun 15, 2015 – 1:10pm



REV No	PROJ No	DWG No	REF DWG No	REFERENCE DRAWING TITLE
A	169514568	0000000	---	---

REV	DESCRIPTION	YY.MM.DD	BY
A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: XXX
DATE: YYYY.MM.DD

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY
495 LEVEL – PLAN
TITLE LINE 2
TITLE LINE 3

SCALE: SCALE | PROJ No 169514568 | DWG No 0000000 | REV No A

Jun 15, 2015 – 1:10pm \\CD1211–N02S:\212\169514568\drawing\mining_prelim\520U0010-0001 – June 6–2015.dxf

Indicators	ID	Name	Duration	Rate	Start	Finish	CREW	LEVEL	Meters	Mined Tonnes	Au
	6	RAMP RAMP	76.0d	5.0m/d	Feb 28, 17	May 15, 17			380.00	0.00	0.00
	16	RAMP RAMP	15.1d	7.0m/d	May 15, 17	May 30, 17			106.00	0.00	0.00
	9	LVL_ACC CRITICAL - BS	9.0d	7.0m/d	May 30, 17	Jun 08, 17			63.00	0.00	0.00
	10	MIX DEV CRITICAL - BS	50.1d	7.0m/d	Jun 08, 17	Jul 28, 17			351.03	16,232.00	1.29
	11	ORE_DEV CRITICAL - BS	21.2d	7.0m/d	Jul 28, 17	Aug 18, 17			148.57	6,870.00	4.08
	24	RAMP RAMP	34.6d	7.0m/d	Dec 31, 17	Feb 03, 18			242.00	0.00	0.00
	21	LVL_ACC CRITICAL	12.7d	7.0m/d	Feb 03, 18	Feb 16, 18			89.00	0.00	0.00
	19	FA_DEV CRITICAL	6.6d	7.0m/d	Feb 16, 18	Feb 22, 18			46.00	0.00	0.00
	32	RAMP RAMP	28.9d	7.0m/d	Feb 22, 18	Mar 23, 18			202.00	0.00	0.00
	29	LVL_ACC CRITICAL	10.7d	7.0m/d	Mar 23, 18	Apr 03, 18			75.00	0.00	0.00
	27	FA_DEV CRITICAL	10.6d	7.0m/d	Apr 03, 18	Apr 14, 18			74.00	0.00	0.00
	40	RAMP RAMP	28.9d	7.0m/d	Apr 14, 18	May 12, 18			202.00	0.00	0.00
	37	LVL_ACC CRITICAL	14.9d	7.0m/d	May 12, 18	May 27, 18			104.00	0.00	0.00
	35	FA_DEV CRITICAL	10.1d	7.0m/d	May 27, 18	Jun 06, 18			71.00	0.00	0.00
	30	MIX DEV CRITICAL	38.7d	7.0m/d	Jun 06, 18	Jul 15, 18			270.57	12,511.00	1.11
	31	ORE_DEV CRITICAL	14.8d	7.0m/d	Jul 15, 18	Jul 30, 18			103.93	4,806.00	4.04
	38	MIX DEV CRITICAL	29.2d	7.0m/d	Jul 30, 18	Aug 28, 18			204.54	9,458.00	0.98
	39	ORE_DEV CRITICAL	12.7d	7.0m/d	Aug 28, 18	Sep 10, 18			88.78	4,105.00	3.55
	48	RAMP RAMP	28.7d	7.0m/d	Sep 10, 18	Oct 09, 18			201.00	0.00	0.00
	45	LVL_ACC CRITICAL	18.0d	7.0m/d	Oct 09, 18	Oct 27, 18			126.00	0.00	0.00
	43	FA_DEV CRITICAL	14.4d	7.0m/d	Oct 27, 18	Nov 10, 18			101.00	0.00	0.00
	56	RAMP RAMP	29.0d	7.0m/d	Nov 10, 18	Dec 09, 18			203.00	0.00	0.00
	53	LVL_ACC CRITICAL	16.1d	7.0m/d	Dec 09, 18	Dec 25, 18			113.00	0.00	0.00
	51	FA_DEV CRITICAL	16.0d	7.0m/d	Dec 25, 18	Jan 10, 19			112.00	0.00	0.00
	64	RAMP RAMP	28.9d	7.0m/d	Jan 10, 19	Feb 08, 19			202.00	0.00	0.00
	61	LVL_ACC CRITICAL	16.4d	7.0m/d	Feb 08, 19	Feb 24, 19			115.00	0.00	0.00
	59	FA_DEV CRITICAL	15.6d	7.0m/d	Feb 24, 19	Mar 12, 19			109.00	0.00	0.00
	62	MIX DEV CRITICAL	72.9d	7.0m/d	Mar 12, 19	May 24, 19			510.02	23,583.00	1.06
	63	ORE_DEV CRITICAL	31.1d	7.0m/d	May 24, 19	Jun 24, 19			217.95	10,078.00	5.01
	72	RAMP RAMP	28.9d	7.0m/d	Jun 24, 19	Jul 23, 19			202.00	0.00	0.00
	69	LVL_ACC CRITICAL	8.0d	7.0m/d	Jul 23, 19	Jul 31, 19			56.00	0.00	0.00
	67	FA_DEV CRITICAL	10.1d	7.0m/d	Jul 31, 19	Aug 10, 19			71.00	0.00	0.00
	70	MIX DEV CRITICAL	29.4d	7.0m/d	Aug 10, 19	Sep 08, 19			205.57	9,505.00	1.30
	78	RAMP RAMP	28.9d	7.0m/d	Sep 08, 19	Oct 07, 19			202.00	0.00	0.00
	77	LVL_ACC CRITICAL	11.4d	7.0m/d	Oct 07, 19	Oct 19, 19			80.00	0.00	0.00
	75	FA_DEV CRITICAL	14.4d	7.0m/d	Oct 19, 19	Nov 02, 19			101.00	0.00	0.00
	86	RAMP RAMP	28.9d	7.0m/d	Nov 02, 19	Dec 01, 19			202.00	0.00	0.00
	83	LVL_ACC CRITICAL	20.6d	7.0m/d	Dec 01, 19	Dec 21, 19			144.00	0.00	0.00
	81	FA_DEV CRITICAL	13.7d	7.0m/d	Dec 21, 19	Jan 04, 20			96.00	0.00	0.00
	84	MIX DEV CRITICAL	36.9d	7.0m/d	Jan 04, 20	Feb 10, 20			258.20	11,939.00	1.71
	85	ORE_DEV CRITICAL	27.2d	7.0m/d	Feb 10, 20	Mar 08, 20			190.27	8,798.00	3.12
	94	RAMP RAMP	28.9d	7.0m/d	Mar 08, 20	Apr 06, 20			202.00	0.00	0.00
	91	LVL_ACC CRITICAL	18.9d	7.0m/d	Apr 06, 20	Apr 25, 20			132.00	0.00	0.00
	89	FA_DEV CRITICAL	16.3d	7.0m/d	Apr 25, 20	May 11, 20			114.00	0.00	0.00
	92	MIX DEV CRITICAL	29.9d	7.0m/d	May 11, 20	Jun 10, 20			209.36	9,681.00	1.17
	93	ORE_DEV CRITICAL	45.3d	7.0m/d	Jun 10, 20	Jul 25, 20			316.80	14,649.00	4.14
	102	RAMP RAMP	32.9d	7.0m/d	Jul 25, 20	Aug 27, 20			230.00	0.00	0.00
	99	LVL_ACC CRITICAL	9.4d	7.0m/d	Aug 27, 20	Sep 06, 20			66.00	0.00	0.00
	97	FA_DEV CRITICAL	15.3d	7.0m/d	Sep 06, 20	Sep 21, 20			107.00	0.00	0.00
	100	MIX DEV CRITICAL	72.0d	7.0m/d	Sep 21, 20	Dec 02, 20			503.76	23,294.00	0.71
	110	RAMP RAMP	26.9d	7.0m/d	Dec 02, 20	Dec 29, 20			188.00	0.00	0.00
	107	LVL_ACC CRITICAL	11.7d	7.0m/d	Dec 29, 20	Jan 10, 21			82.00	0.00	0.00
	105	FA_DEV CRITICAL	7.9d	7.0m/d	Jan 10, 21	Jan 17, 21			55.00	0.00	0.00
	108	MIX DEV CRITICAL	39.9d	7.0m/d	Jan 17, 21	Feb 26, 21			279.20	12,910.00	0.75
	109	ORE_DEV CRITICAL	18.3d	7.0m/d	Feb 26, 21	Mar 17, 21			127.91	5,915.00	3.07
	118	RAMP RAMP	27.0d	7.0m/d	Mar 17, 21	Apr 13, 21			189.00	0.00	0.00
	115	LVL_ACC CRITICAL	20.1d	7.0m/d	Apr 13, 21	May 03, 21			141.00	0.00	0.00
	113	FA_DEV CRITICAL	16.1d	7.0m/d	May 03, 21	May 19, 21			113.00	0.00	0.00
	126	RAMP RAMP	28.9d	7.0m/d	May 19, 21	Jun 17, 21			202.00	0.00	0.00
	123	LVL_ACC CRITICAL	8.0d	7.0m/d	Jun 17, 21	Jun 25, 21			56.00	0.00	0.00
	121	FA_DEV CRITICAL	2.9d	7.0m/d	Jun 25, 21	Jun 28, 21			20.00	0.00	0.00
	124	MIX DEV CRITICAL	54.5d	7.0m/d	Jun 25, 21	Aug 18, 21			381.73	17,651.00	1.13
	125	ORE_DEV CRITICAL	24.0d	7.0m/d	Aug 18, 21	Sep 11, 21			167.90	7,764.00	4.59
	116	MIX DEV CRITICAL	46.5d	7.0m/d	Sep 11, 21	Oct 28, 21			325.21	15,038.00	1.00
	117	ORE_DEV CRITICAL	12.9d	7.0m/d	Oct 28, 21	Nov 10, 21			90.21	4,171.00	2.92
	127	MIX DEV REMAINING	4.8d	7.0m/d	Nov 10, 21	Nov 14, 21			33.37	1,543.00	0.97
	128	ORE_DEV REMAINING	12.7d	7.0m/d	Nov 14, 21	Nov 27, 21			89.03	4,117.00	3.44
	33	MIX DEV REMAINING	101.6d	7.0m/d	Nov 27, 21	Mar 09, 22			711.00	32,877.00	1.30
	34	ORE_DEV REMAINING	11.8d	7.0m/d	Mar 09, 22	Mar 20, 22			82.67	3,823.00	4.75
	7	MIX DEV REMAINING	116.7d	7.0m/d	Mar 20, 22	Jul 15, 22			816.58	37,759.00	1.38
	8	ORE_DEV REMAINING	32.6d	7.0m/d	Jul 15, 22	Aug 17, 22			228.44	10,563.00	4.61
	2	LVL_ACC CRITICAL - BS	7.9d	7.0m/d	May 15, 17	May 22, 17			55.00	0.00	0.00
	3	MIX DEV CRITICAL - BS	34.4d	7.0m/d	May 22, 17	Jun 26, 17			240.90	11,139.00	1.16
	4	ORE_DEV CRITICAL - BS	13.8d	7.0m/d	Jun 26, 17	Jul 10, 17			96.54	4,464.00	7.75
	12	FA_DEV CRITICAL	6.4d	7.0m/d	Dec 31, 17	Jan 06, 18			45.00	0.00	0.00
	5	FA_DEV CRITICAL	9.4d	7.0m/d	Jan 06, 18	Jan 15, 18			66.00	0.00	0.00
	14	MIX DEV CRITICAL	36.1d	7.0m/d	Jan 15, 18	Feb 21, 18			252.93	11,696.00	1.37
	15	ORE_DEV CRITICAL	34.0d	7.0m/d	Feb 21, 18	Mar 27, 18			238.32	11,020.00	3.81
	22	MIX DEV CRITICAL	104.4d	7.0m/d	Mar 27, 18	Jul 09, 18			730.75	33,869.00	1.56
	23	ORE_DEV CRITICAL	18.1d	7.0m/d	Jul 09, 18	Jul 27, 18			126.66	5,857.00	3.43
	17	MIX DEV REMAINING	102.2d	7.0m/d	Jul 27, 18	Nov 06, 18			715.07	33,138.00	1.37
	18	ORE_DEV REMAINING	18.1d	7.0m/d	Nov 06, 18	Nov 24, 18			126.35	5,842.00	3.19
	46	MIX DEV CRITICAL	38.5d	7.0m/d	Nov 24, 18	Jan 02, 19			269.80	12,475.00	1.64
	54	MIX DEV CRITICAL	71.2d	7.0m/d	Jan 02, 19	Mar 14, 19			498.14	23,034.00	1.30
	55	ORE_DEV CRITICAL	26.4d	7.0m/d	Mar 14, 19	Apr 09, 19			184.49	8,531.00	3.04
	25	MIX DEV REMAINING	69.9d	7.0m/d	Apr 09, 19	Jun 18, 19			489.13	22,617.00	1.46
	26	ORE_DEV REMAINING	16.7d	7.0m/d	Jun 18, 19	Jul 05, 19			116.76	5,399.00	4.04
	49	MIX DEV REMAINING	124.7d	7.0m/d	Jul 05, 19	Nov 07, 19			873.01	40,368.00	1.46
	50	ORE_DEV REMAINING	42.6d	7.0m/d	Nov 07, 19	Dec 19, 19			298.24	13,790.00	4.66
	57	MIX DEV REMAINING	116.3d	7.0m/d	Dec 19, 19	Apr 14, 20			814.17	37,647.00	1.21
	58	ORE_DEV REMAINING	67.6d	7.0m/d	Apr 14, 20	Jun 20, 20			472.91	21,867.00	4.61
	73	MIX DEV REMAINING	89.7d	7.0m/d	Jun 20, 20	Sep 18, 20			627.75	29,027.00	0.89
	74	ORE_DEV REMAINING	24.6d	7.0m/d	Sep 18, 20	Oct 12, 20			171.93	7,950.00	5.42
	41	MIX DEV REMAINING	124.5d	7.0m/d	Oct 12, 20	Feb 14, 21			871.67	40,306.00	1.25
	42	ORE_DEV REMAINING	27.1d	7.0m/d	Feb 14, 21	Mar 13, 21			189.47	8,761.00	4.14
	65	MIX DEV REMAINING	49.9d	7.0m/d	Mar 13, 21	May 02, 21			349.43	16,158.00	1.22
	66	ORE_DEV REMAINING	34.4d	7.0m/d	May 02, 21	Jun 05, 21			240.57	11,124.00	3.71
	79	MIX DEV REMAINING	122.2d	7.0m/d	Jun 05, 21	Oct 05, 21			855.55	39,561.00	1.22
	80	ORE_DEV REMAINING	26.2d	7.0m/d	Oct 05, 21	Nov 01, 21			183.39	8,480.00	4.09
	87	MIX DEV REMAINING	83.6d	7.0m/d	Nov 01, 21	Jan 23, 22			585.07	27,054.00	1.63
	88	ORE_DEV REMAINING	36.6d	7.0m/d	Jan 23, 22	Mar 01, 22			256.41	11,857.00	3.47
	95	MIX DEV REMAINING	45.9d	7.0m/d	Mar 01, 22	Apr 16, 22			321.45	14,864.00	1.74
	96	ORE_DEV REMAINING	48.6d	7.0m/d	Apr 16, 22	Jun 03, 22			340.41	15,740.00	4.04
	103	MIX DEV REMAINING	74.3d	7.0m/d	Jun 03, 22	Aug 17, 22			520.09	24,055.00	1.46
	104	ORE_DEV REMAINING	8.9d	7.0m/d	Aug 17, 22	Aug 26, 22			62.24	2,878.00	7.26
	111	MIX DEV REMAINING	16.8d	7.0m/d	Aug 26, 22	Sep 11, 22			117.68	5,441.00	2.28
	119	MIX DEV REMAINING	25.4d	7.0m/d	Sep 11, 22	Oct 07, 22			177.50	8,208.00	2.13
	120	ORE_DEV REMAINING	20.5d	7.0m/d	Oct 07, 22	Oct 27, 22			143.51	6,636.00	4.79
	130	POD 2	36.1d	200.0Tonnes/d	Aug 18, 17	Sep 23, 17			0.00	7,214.00	5.08
	132	POD 4	134.5d	200.0Tonnes/d	Aug 18, 17	Dec 31, 17			0.00	26,906.00	6.54
	134	POD 6	27.2d	200.0Tonnes/d	Sep 23, 17	Oct 20, 17			0.00	5,442.00	2.96
	145	POD 17	135.8d	200.0Tonnes/d	Jul 27, 18	Dec 10, 18			0.00	27,156.00	5.16
	131	POD 3	120.4d	200.0Tonnes/d	Sep 10, 18	Jan 08, 19			0.00	24,075.00	5.89
	168	POD 40	45.5d	200.0Tonnes/d	Jan 08, 19	Feb 23, 19			0.00	9,099.00	5.96
	133	POD 5	226.3d	200.0Tonnes/d	Apr 09, 19	Nov 22, 19			0.00	45,254.00	4.45
	146	POD 18	246.8d	200.0Tonnes/d	Jun 24, 19	Feb 26, 20			0.00	49,361.00	3.90
	129	POD 1	135.9d	200.0Tonnes/d	Jul 05, 19	Nov 18, 19			0.00	27,181.00	3.19
	151	POD 23	525.9d	200.0Tonnes/d	Sep 08, 19	Feb 15, 21			0.00	105,179.00	6.84
	158	POD 30	28.8d	200.0Tonnes/d	Jun 20, 20	Jul 19, 20			0.00	5,759.00	3.73
	136	POD 8	326.9d	200.0Tonnes/d	Jul 19, 20	Jun 11, 21			0.00	65,376.00	5.42
	144	POD 16	333.9d	200.0Tonnes/d	Jul 19, 20	Jun 18, 21			0.00	66,779.00	3.62
	141	POD 13	131.6d	200.0Tonnes/d	Jul 25, 20	Dec 04, 20			0.00	26,322.00	4.31
	150	POD 22	132.8d	200.0Tonnes/d	Dec 02, 20	Apr 14, 21			0.00	26,564.00	3.84
	156	POD 28	31.5d	200.0Tonnes/d	Mar 13, 21	Apr 13, 21			0.00	6,304.00	6.64
	161	POD 33	4.1d	200.0Tonnes/d	Mar 13, 21	Mar 17, 21			0.00	815.00	2.87
	149	POD 21	458.5d	200.0Tonnes/d	Apr 14, 21	Jul 16, 22			0.00	91,700.00	4.34
	153	POD 25	60.3d	200.0Tonnes/d	May 02, 21	Jul 01, 21			0.00	12,060.00	4.23
	137	POD 9	85.7d	200.0Tonnes/d	Jun 18, 21	Sep 11, 21			0.00	17,131.00	3.06
	155	POD 27	61.9d	200.0Tonnes/d	Jun 18, 21	Aug 19, 21			0.00	12,384.00	9.72
	163	POD 35	13.2d	200.0Tonnes/d	Jun 18, 21	Jul 01, 21			0.00	2,631.00	3.37
	152	POD 24	40.4d	200.0Tonnes/d	Jul 01, 21	Aug 10, 21			0.00	8,075.00	3.21
	165	POD 37	136.4d	200.0Tonnes/d	Sep 11, 21	Jan 26, 22			0.00	27,279.00	4.74
	138	POD 10	745.7d	200.0Tonnes/d	Nov 10, 21	Nov 25, 23			0.00	149,149.00	4.66
	139	POD 11	179.6d	200.0Tonnes/d	Nov 10, 21	May 08, 22			0.00	35,910.00	3.76
	160	POD 32	21.8d	200.0Tonnes/d	Jan 26, 22	Feb 17, 22			0.00	4,350.00	3.25
	164	POD 36	279.9d	200.0Tonnes/d	Jun 29, 22	Apr 05, 23			0.00	55,965.00	5.24
	140	POD 12	374.4d	200.0Tonnes/d	Oct 27, 22	Nov 06, 23			0.00	74,883.00	3.43
	142	POD 14	221.3d	200.0Tonnes/d	Mar 30, 23	Nov 08, 23			0.00	44,269.00	7.97
	157	POD 29	144.8d	200.0Tonnes/d	Apr 05, 23	Aug 28, 23			0.00	28,964.00	4.43
	166	POD 38	236.5d	200.0Tonnes/d	Apr 05, 23	Nov 27, 23			0.00	47,294.00	4.78
	148	POD 20	445.1d	200.0Tonnes/d	Nov 06, 23	Jan 24, 25			0.00	89,017.00	5.29
	143	POD 15	282.4d	200.0Tonnes/d	Nov 06, 23	Aug 15, 24			0.00	56,479.00	5.04
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	162	POD 34	69.5d	200.0Tonnes/d	Dec 01, 23	Feb 09, 24			0.00	13,902.00	3.79
	159	POD 31	313.6d	200.0Tonnes/d	Feb 09, 24	Dec 18, 24			0.00	62,713.00	4.08
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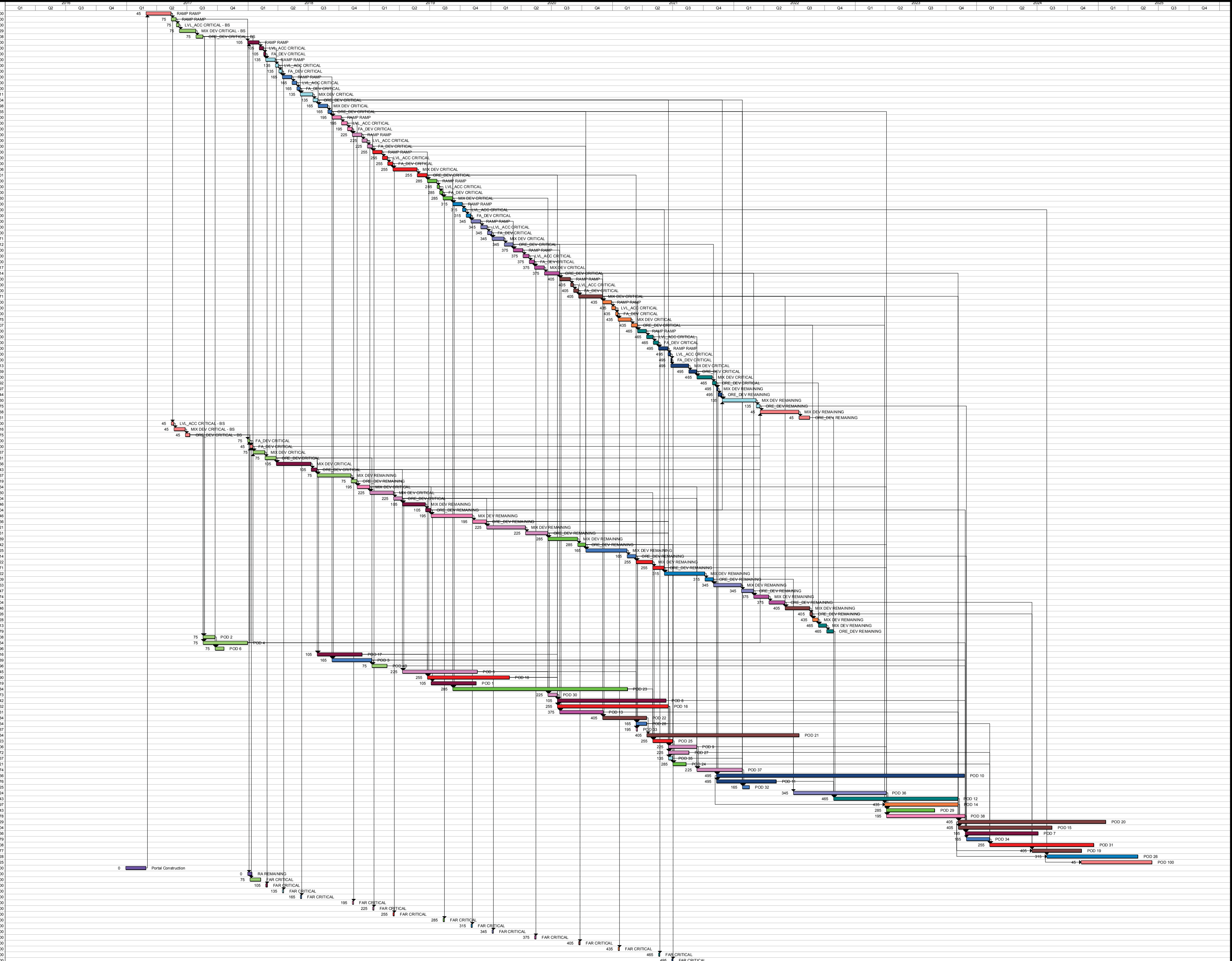
	154	POD 26	274.0d	200.0Tonnes/d	Jul 30, 24	Apr 30, 25			0.00	54,798.00	5.28
	167	POD 100	212.8d	300.0Tonnes/d	Nov 10, 24	Jun 11, 25			0.00	63,840.00	6.25
	169	Portal Construction	61.0d	0.0Tonnes/d	Dec 29, 16	Feb 28, 17			0.00	0.00	0.00
	1	RA REMAINING	13.0d	2.0m/d	Dec 31, 17	Jan 13, 18			26.00	0.00	0.00
	13	FAR CRITICAL	33.0d	2.0m/d	Jan 08, 18	Feb 08, 18			66.00	0.00	0.00
	20	FAR CRITICAL	5.6d	7.0m/d	Feb 22, 18	Feb 28, 18			39.00	0.00	0.00
	28	FAR CRITICAL	4.3d	7.0m/d	Apr 14, 18	Apr 18, 18			30.00	0.00	0.00
	36	FAR CRITICAL	4.3d	7.0m/d	Jun 06, 18	Jun 11, 18			30.00	0.00	0.00
	44	FAR CRITICAL	4.3d	7.0m/d	Nov 10, 18	Nov 14, 18			30.00	0.00	0.00
	52	FAR CRITICAL	4.3d	7.0m/d	Jan 10, 19	Jan 14, 19			30.00	0.00	0.00
	60	FAR CRITICAL	4.3d	7.0m/d	Mar 12, 19	Mar 16, 19			30.00	0.00	0.00
	68	FAR CRITICAL	4.3d	7.0m/d	Aug 10, 19	Aug 14, 19			30.00	0.00	0.00
	76	FAR CRITICAL	4.3d	7.0m/d	Nov 02, 19	Nov 06, 19			30.00	0.00	0.00
	82	FAR CRITICAL	4.3d	7.0m/d	Jan 04, 20	Jan 08, 20			30.00	0.00	0.00
	90	FAR CRITICAL	4.3d	7.0m/d	May 11, 20	May 16, 20			30.00	0.00	0.00
	98	FAR CRITICAL	4.6d	7.0m/d	Sep 21, 20	Sep 26, 20			32.00	0.00	0.00
	106	FAR CRITICAL	4.3d	7.0m/d	Jan 17, 21	Jan 22, 21			30.00	0.00	0.00
	114	FAR CRITICAL	4.1d	7.0m/d	May 19, 21	May 23, 21			29.00	0.00	0.00
	122	FAR CRITICAL	4.3d	7.0m/d	Jun 28, 21	Jul 02, 21			30.00	0.00	0.00

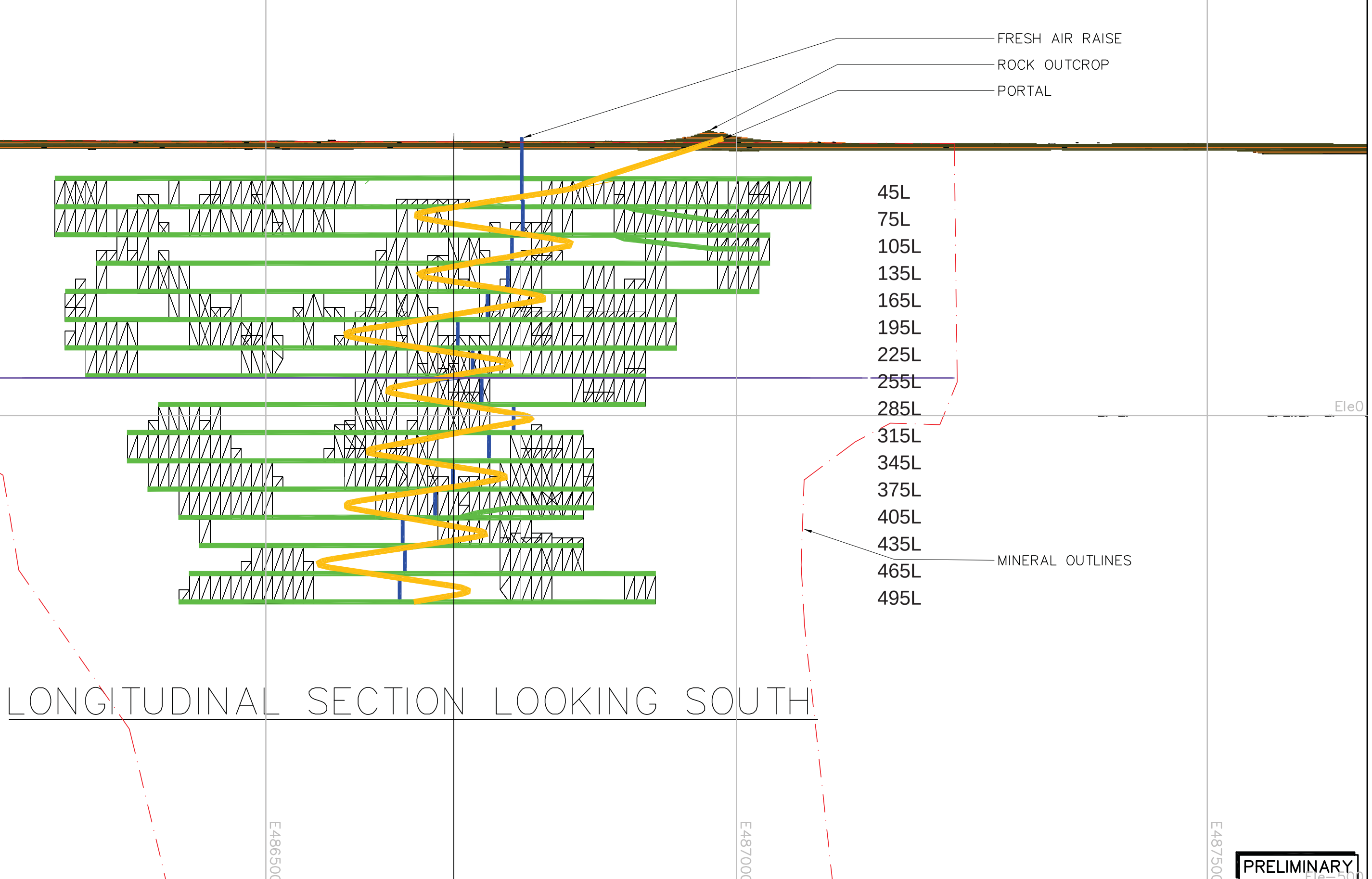


Indicators	Name	Description	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
	RAMP_METERS			493	1,043	808	620	391				
	LVL_ACC_METERS			118	507	395	217	260				
	MIX_DEV_METERS			592	2,434	2,671	2,888	3,202	2,582			
	ORE_DEV_METERS			245	684	817	1,152	1,088	1,114			
	FA_DEV_LAT			7	441	418	247	188				
	TOTAL METERS			1,454	5,110	5,110	5,124	5,130	3,695			
	Crew 1 metres			1,055	2,555	2,555	2,562	2,575	1,597			
	crew 2 metres			399	2,555	2,555	2,562	2,555	2,098			
	ORE_DEV_TONNES			11,334	31,630	37,798	53,264	50,332	51,497			
	PRODUCTION_TONNES			39,562	49,695	144,019	188,695	250,448	197,468	302,046	302,550	76,957
	TOTAL_TONNES			50,896	81,325	181,817	241,959	300,780	248,965	302,046	302,550	76,957
	ORE_DEV_AU			5.52	3.63	4.30	4.36	3.82	4.35			
	PRODUCTION_AU			5.78	5.49	4.56	5.31	4.64	4.48	4.95	4.88	5.89
	TOTAL_GRADE			5.72	4.77	4.50	5.10	4.50	4.45	4.95	4.88	5.89
	FAR				195	120	92	89				
	RAR			2	24							

0	465	435	405	375	345	315	285	255	225	195	165	135	105	45	495	75

Project:

Level	Total Level Tonnes	Level Grade	Au oz
45	78,867	6.12	15,525
75	162,059	5.02	26,165
105	124,030	4.73	18,877
135	71,204	4.30	9,851
165	79,627	4.27	10,940
195	109,470	4.27	15,038
225	192,326	5.15	31,836
255	149,336	5.19	24,924
285	89,796	5.55	16,020
315	56,773	4.33	7,901
345	102,862	4.38	14,490
375	172,389	4.94	27,381
405	127,431	5.36	21,952
435	38,644	3.54	4,393
465	102,644	3.90	12,883
495	130,337	4.59	19,244
Totals	1,787,794	4.83	277,419



LONGITUDINAL SECTION LOOKING SOUTH

PRELIMINARY

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DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.03.09

Stantec

CLIENT: GOWEST GOLD LTD.
PROPERTY: BRADSHAW GOLD DEPOSIT
PROJECT: PRE–FEASIBILITY LONG SECTION

SCALE: SCALE = NTS
PROJ No 169514568
DWG No 510U1015–BA1
REV No A

PROJ No 169514568 DWG No 510U1015 REV No A

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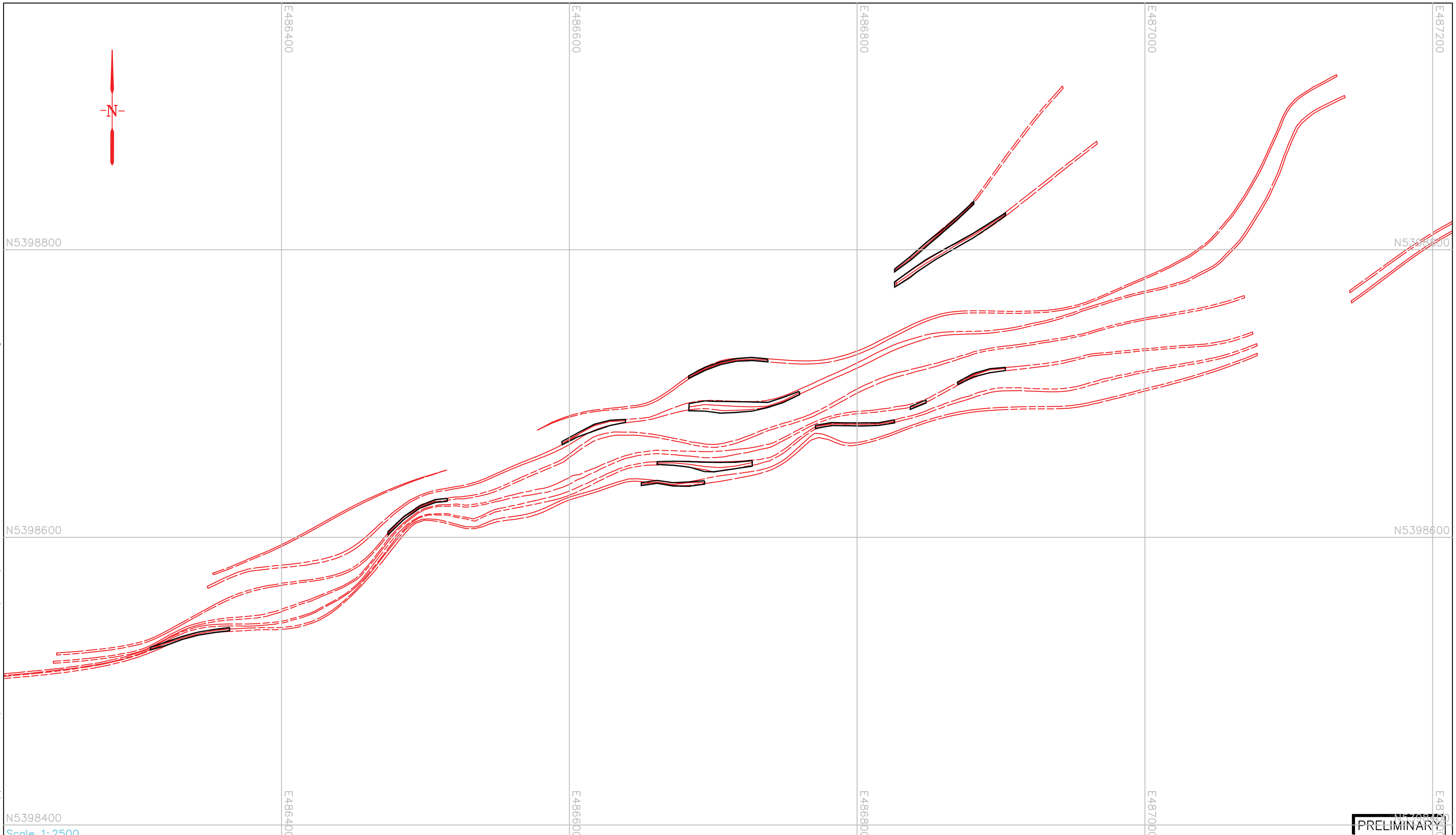
APPENDIX F
MINE DRAWINGS



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BY: XXX

ORIGINAL DRAWN BY: | DATE:

Stantec

CLIENT:
GOWEST GOLD LTD.
PROPERTY:
PROJECT:

SCALE: | PROJ No | DWG No | REV No

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Stantec

CLIENT:
GOWEST GOLD LTD.
PROPERTY:
PROJECT:

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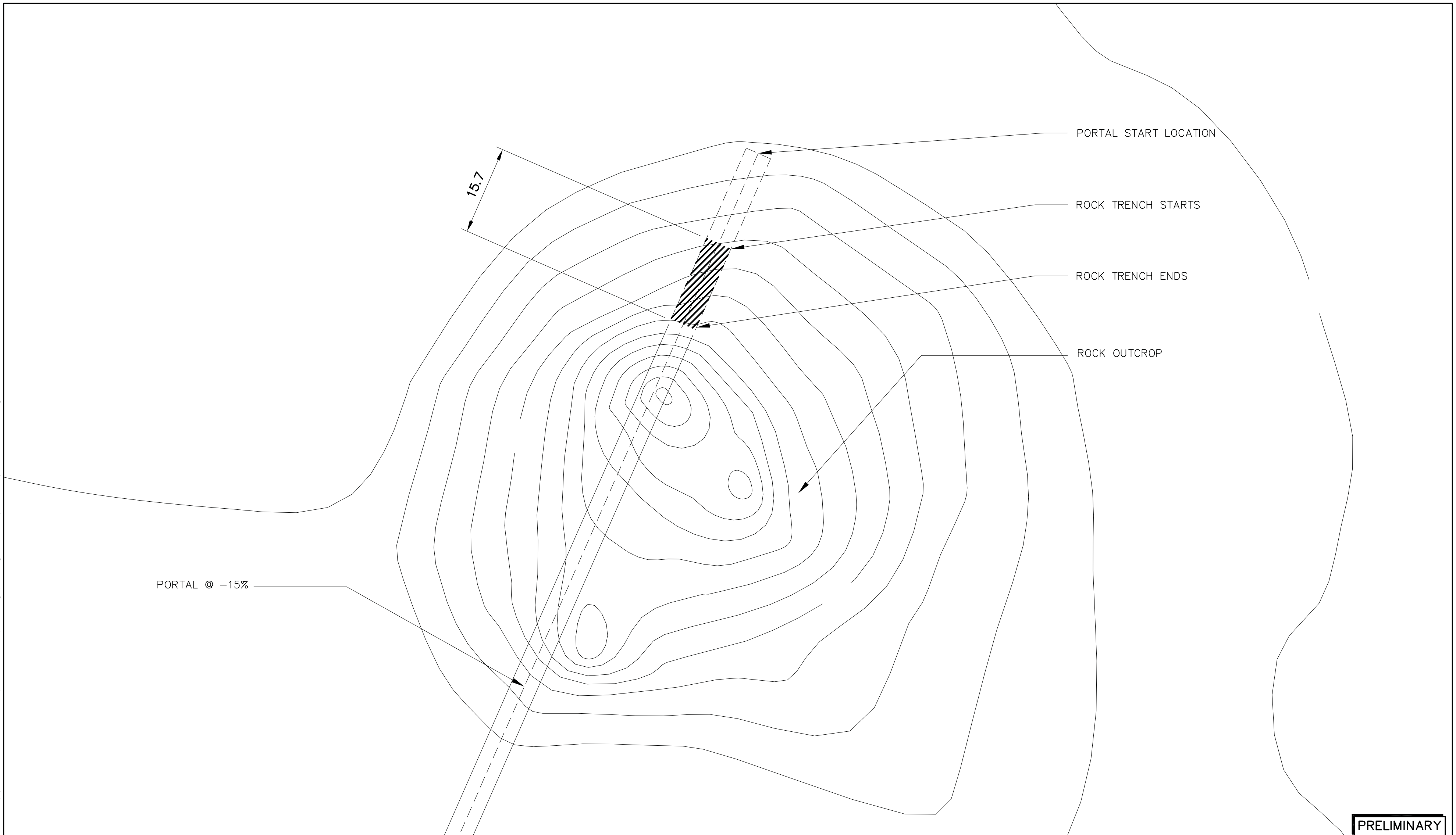

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BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.01.22

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit

SCALE: SCALE = NTS
PROJ No: 169514568
DWG No:
REV No: A

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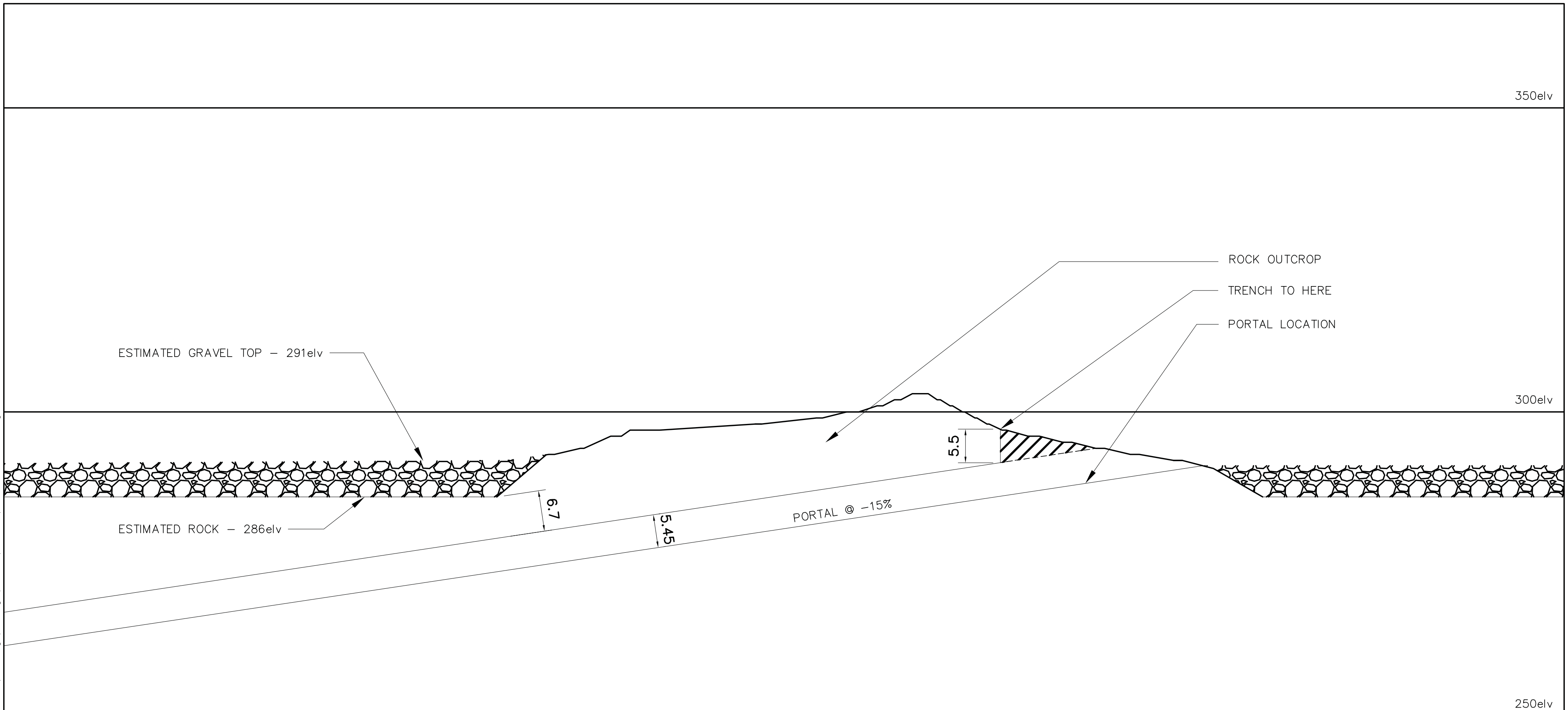

PRELIMINARY

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REF DWG No	REFERENCE DRAWING TITLE
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PROJ No 169514568 — DWG No — REV No A

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PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.01.22

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit
SCALE: SCALE = 30:1
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DWG No
REV No A

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APPENDIX G
MINE DESIGN

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BY: XXX

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PROJECT:

SCALE: PROJ No DWG No REV No

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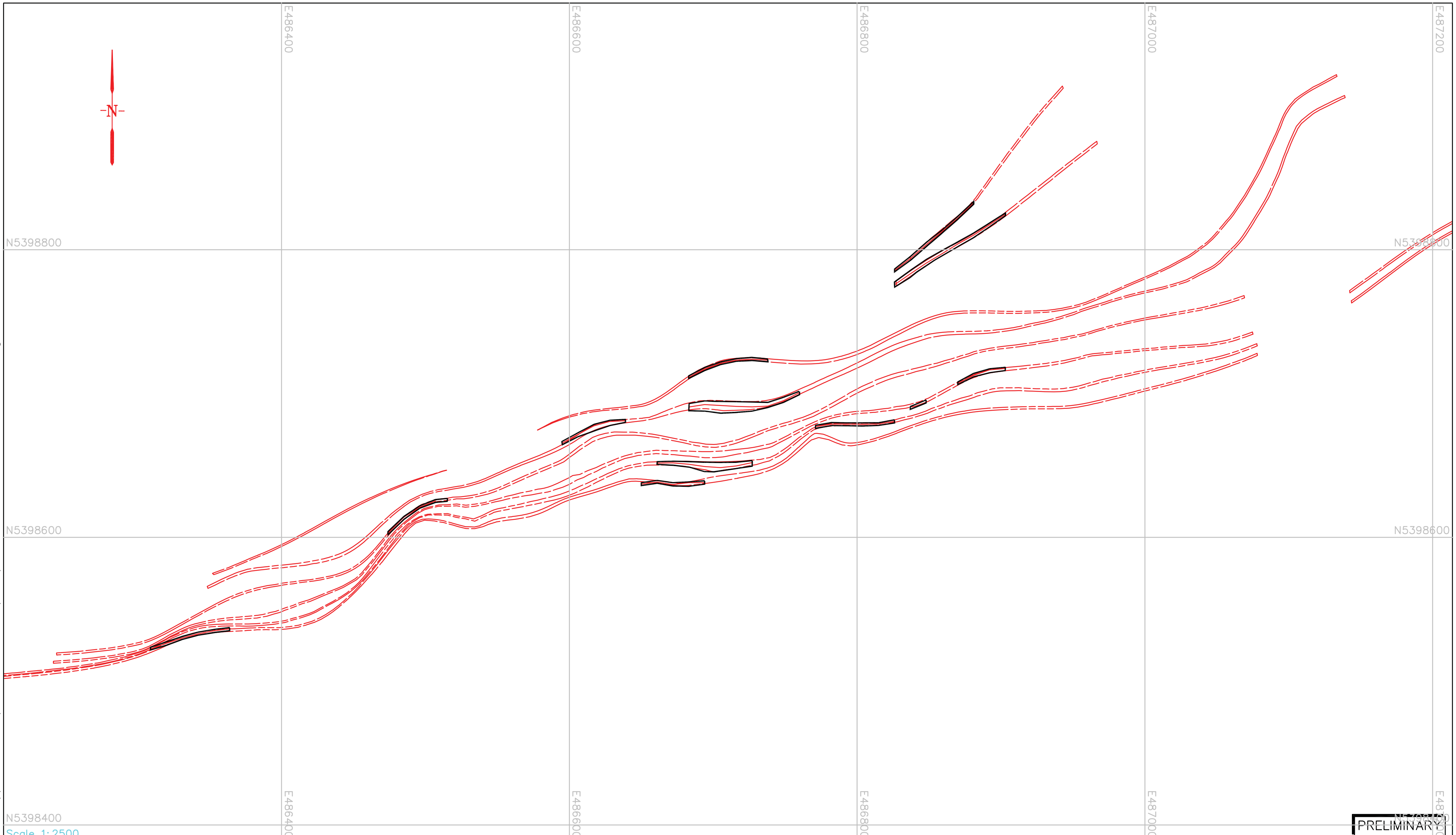

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Stantec

CLIENT:
GOWEST GOLD LTD.

PROPERTY:

PROJECT:

SCALE: PROJ No DWG No REV No

Project No. 169514568 06 February 2015

PREPARED FOR:

Gowest Gold Limited

CONCERNING:

Bradshaw Gold Deposit Pre-Feasibility Study

Mine Design Parameters

PREPARED BY:
Stantec Consulting Ltd.
1760 Regent Street
Sudbury, Ontario P3E 3Z8
Canada

REVISION NOTES

Revision	Date	Description	Name
A	19-Jan-2015	Document Initiated	M. St-Laurent
A1	05-Feb-2015	Internal Review	N. Del Bel Belluz
B	09-Feb-2015	Issued for Review	N. Del Bel Belluz
C		Issued as Final	N. Del Bel Belluz

REVISION APPROVAL

Revision	Designation	Reviewer	Date	Signature
A	Originator	M. St-Laurent	21-Jan-2015	
B	Issue for Review and Comment	Project Manager N. Del Bel Belluz	05-Feb-2015	
C	Issue as Final	Project Manager N. Del Bel Belluz		

Table of Contents

Appendices

1.0 INTRODUCTION

This Mine Design Parameters Report is a compilation of information with input from mining, ventilation, geomechanical, and infrastructure disciplines. The information is the basis for the mine design components of the study schedule and cost estimate including lateral development, vertical development, and stoping.

2.0 ROCK PROPERTIES

Table 2.1 lists the rock properties used to determine typical tonnages in development advance and stopes.

Table 2.1: Rock Properties

Category	Design Item	Value
Specific Gravity	Average ore (in-situ)	2.89 tonnes/cubic metre
	Average ore (broken)	1.93 tonnes/cubic metre
	Average waste (in-situ)	2.75 tonnes/cubic metre
	Average waste (broken)	1.83 tonnes/cubic metre
	Average cement slurry (1 water:1 cement ratio)	1.5 tonnes cubic metre
	Average CRF	1.5 tonnes/cubic metre
Swell Factor	Swell factor	50%

3.0 PROPOSED MINING LEVELS

Table 3.1 provides a list of the maximum interval between the proposed mining levels using 65 mm production drill holes. These level intervals are currently being used at other similar operations.

Table 3.1: Maximum Level Intervals

Mining Method	Max interval
Uppers drilling with uphole slot	15 metres
Down hole drilling	20 metres
Down hole and up hole combination	30 metres
Cut and fill	25 metres

4.0 DEVELOPMENT PARAMETERS

The following tables list design parameters for development activities. This information is used to generate typical performances and cost per metre for development activities in the study estimate. Table 4.1 to Table 4.4, list parameters for the development headings, including dimensions, gradient, ground support, electrical installations, and ventilation installations. Refer to Appendix A for a typical ramp and level cross section. Table 4.5 lists secondary support requirements (application within the study cost estimate to be determined). Table 4.6 lists parameters for ventilation raises, egress raises, and ore/rock passes.

Table 4.1: Main Ramp Development

Category	Design Item	Value
Dimensions	Width	5 metres
	Height	5 metres
Gradient	Max Design Gradient	15%
Turn Radius	Turn Radius	30 metres
Offset	From Ore Body	less than 20 metres
	From Dykes/Faults	Greater than 30 metres
Ground Support	Bolting Pattern	1.2 metre x 0.8 metre (4.0 foot x 2.5 foot) staggered, 1.5 metre from floor
	Bolt Size/Type - Back and Walls	2.4 metre (8 foot) rebar back, 1.8 metre (6 foot) rebar walls
	Welded Wire Mesh Size	6 gauge, galvanized
	Shotcrete Thickness (allowance if required)	75mm (3 inch) nominal (5%)
Explosives	Explosive	Anfo with stick in wet conditions
	Detonator	Nonel
Piping	Compressed Air	6 inch Schedule 40
	Process Water	6 inch Schedule 40
	Dewatering	6 inch Schedule 80
Cables	Electrical	Double 600V
	Communication	Leaky feeder
Ventilation	Auxiliary Ducting	Double 60 inch steel
Ballast	Depth	30 cm (1.0 foot)
	Material	Crushed waste rock

Table 4.2: Sub Level Development*

Category	Design Item	Value
Dimensions	Width	4 metres
	Height	4 metres
Gradient	Design Gradient	Flat
	Actual Typical Gradient	3% minimum
Offset	Drift to Stope	greater than 10 metres
Ground Support	Bolting Pattern	1.2 metre x 0.8 metre (4.0 foot x 2.5 foot) staggered, 1.5 metre from floor
	Bolt Size/Type - Back	1.8 metre (6 foot) rebar bolt
	Bolt Size/Type - Walls	1.7 metre (5 foot 6 inch) friction bolt
	Welded Wire Mesh Size	6 gauge, non-galvanized
	Shotcrete Thickness (allowance if required)	75mm (3 inch) nominal (5%)
Explosives	Explosive	Anfo with stick in wet conditions
	Detonator	Nonel
Piping	Compressed Air	4 inch Schedule 40
	Process Water	2 inch Schedule 40
	Dewatering	4 inch Schedule 80
	Backfill	4 inch Schedule 80
Cables	Electrical	Double 600V
	Communication	Leaky feeder, 12 core fibre optic
Ventilation	Auxiliary Ducting	Double 42 inch steel
Ballast	Depth	30 cm (1.0 foot)
	Material	Crushed waste rock

*Note: Includes level access drifts, infrastructure drifts (equivalent metres), ventilation access drifts, etc.

Table 4.3: Ore Sill Development

Category	Design Item	Value
Dimensions	Minimum Width - Ore	3 metres
	Maximum Width - Ore	4 metres
	Height	4 metres
Gradient	Design Gradient	Flat
	Actual Typical Gradient	3% minimum
Ground Support	Bolting Pattern	1.2 metre x 0.8 metre (4.0 foot x 2.5 foot) staggered, 1.5 metre from floor
	Bolt Size/Type - Back	1.8 metre (6 foot) rebar bolt
	Bolt Size/Type - Walls	1.7 metre (5 foot 6 inch) friction bolt
	Welded Wire Mesh Size	6 gauge, non-galvanized
	Shotcrete Thickness – Ore (if required)	75mm (3 inch) nominal, back and shoulders
Explosives	Explosive	Anfo with stick in wet conditions
	Detonator	Nonel
Piping	Compressed Air	4 inch Schedule 40
	Process Water	2 inch Schedule 40
	Backfill	4 inch Schedule 80
Cables	Communication	Leaky feeder
	Electrical - Waste	Single 600V
Ventilation	Auxiliary Ducting	Double 42 inch flexible

Table 4.4: Cut and Fill Access Ramp Development

Category	Design Item	Value
Dimensions	Width	4 metres
	Height	4 metres
Gradient	Design Gradient	18% maximum
	Actual Typical Gradient	18% maximum
Ground Support	Bolting Pattern	1.2 metre x 0.8 metre (4.0 foot x 2.5 foot) staggered, 1.5 metre from floor
	Bolt Size/Type - Back	1.8 metre (6 foot) mechanical bolt
	Bolt Size/Type - Walls	1.8 metre (5 foot 6 inch) friction bolt
	Welded Wire Mesh Size	6 gauge, non-galvanized
	Shotcrete Thickness (allowance if required)	75 mm (3 inch) nominal (5%)
Explosives	Explosive	Anfo with stick in wet conditions
	Detonator	Nonel
Piping	Process Water	2 inch Schedule 10

Category	Design Item	Value
	Dewatering	4 inch Schedule 80
	Backfill	4 inch Schedule 80
Cables	Electrical	Double 600V
	Communication	Leaky feeder
Ventilation	Auxiliary Ducting	Double 42 inch flexible

Table 4.5: Intersection Ground Support

Category	Design Item	Value
Cable Bolt	Larger than 10m dia	Double cables:1/3 length of span
Rebar Support	8m to 10m dia Intersection	
	Bolt Size/Type	3.0 metre (10 foot) fully grouted rebar
	Bolt Pattern	1.2 metre x 1.5 metre
Rebar Support	6m to 7m dia Intersection	
	Bolt Size/Type	2.4 metre (8 foot) fully grouted rebar
	Bolt Pattern	1.2 metre x 1.5 metre

Table 4.6: Vertical Excavations

Description	Dimensions	Length	Method	Ground Support	Installations
Surface Fresh Air	3.0 metres x 3.0 metres	Varies	Alimak	1.5 metres x 1.0 metres, 1.2 metres fully grouted rebar, chain link	Timber Escapeway
Internal Fresh Air	3.0 metres x 3.0 metres	Varies	Alimak	1.5 metres x 1.0 metres, 1.2 metres fully grouted rebar, chain link	Timber Escapeway

5.0 PRODUCTION PARAMETERS

The following tables list design parameters for production activities. The parameters listed are for typical stopes and during execution will be modified stope to stope as additional information is gained on rock properties and geology. However, this information is used to generate typical performances and cost per tonne for production activities in the study estimate.

5.1 Cut and Fill Mining

Table 5.1 lists design parameters for a typical cut and fill mining method. Table 5.2 contains miscellaneous cut and fill information regarding development to slash ratios/slot to cross cut ratios, and access ramp and backfill details. Refer to Appendix A for a schematic of post-pillar cut and fill mining.

Table 5.1: Cut and Fill Development

Category	Design Item	Value
Dimensions	Width	9 metres max
	Height	5 metres
Gradient	Design Gradient	Flat
Ground Support	Bolting Pattern	1.2 metre x 0.8 metre (4.0 foot x 2.5 foot) staggered, 2.5 metre from floor
	Bolt Size/Type - Back	1.8 metre (6 foot) mechanical bolt if span is less than 6m. 2.4 metre (8 foot) mechanical bolt if span if greater than 6m.
	Bolt Size/Type - Walls	1.8 metre (5 foot 6 inch) friction bolt
	Welded Wire Mesh Size	6 gauge
	Shotcrete Thickness (allowance if required)	75 mm (3 inch) nominal (5%)
	Face Bolting (if required)	1.2 metre x 1.2 metre staggered pattern of friction bolts to 3 metres from floor
Explosives	Explosive	Anfo with stick in wet conditions
	Detonator	Nonel
Piping	Process Water (reused for Backfill)	4 inch Schedule 10
Cables	Communication	Leaky feeder
Ventilation	Auxiliary Ducting	Double 42 inch flexible

Table 5.2: Cut and Fill Mining Miscellaneous

Category	Design Item	Value
Access Ramp	Cuts per Level	5
	Level Spacing	50 metres
	Access Ramp Gradient	18% max
	Equivalent Access Ramp Development per Cut	80 metres
Backfill	Backfill Replacement Factor (weight percentage to ore)	50%
	Backfill Recipe on Remaining Cuts	Waste rock as required
	Backfill Barricade	Rock berm as required

5.2 Bulk Mining

Table 5.3 to 5.5 lists the design parameters for slot slash and uppers retreat mining, respectively. Table 5.6 and 5.7 contains miscellaneous bulk mining information. Refer to Appendix A for schematics of slot slash and uppers retreat mining.

Table 5.3: Slot Slash 20 Metre High (downholes)

Category	Design Item	Value*
Dimensions	Width	2 metres Minimum
	Length	11 metres
	Height	20 metres
Mucking	Tonnage (excluding bottom sill)	1017 tonnes (2m wide) 1526 tonnes (3m wide) 2543 tonnes (5m wide)
Drilling	Hole Diameter	64mm (2.5 inch) dia
	Drill Spacing	1.2 metres
	Drill Burden	1.2 metres
	Overdrill per Hole (past contact)	1.5 metres
	Number of Drill Rings	9
	Holes per Drill Ring	2
	Footage per Drill Ring	32 metres
	Redrill/Cleaning Allowance	10%
	Total Footage per Stope	317 metres
	Drill Factor	4.8 tonnes/metre
Raise	Raise	11-holes 1.2 metre x 1.2 metre drop raise
	Length of Raise	16 metres
	Footage in Drop Raise (including reaming and 10% redrill allowance)	194 metres
Blasting	Explosive	Anfo and stick in wet conditions
	Maximum 2 Rings Full Height	390 tons
	Detonator	Nonel in all blasts
	Number of Blasts	4
Backfill	Tonnage	50% of mined tonnage

* Note: Actual quantities vary stope by stope. Typical quantities are shown here for estimating purposes to determine the cost per ton.

Table 5.4: Slot Slash 30 Metre High (upholes and downholes)

Category	Design Item	Value*
Dimensions	Width	2 metres Minimum
	Length	11 metres
	Height	30 metres
Mucking	Tonnage (excluding bottom sill)	1653 tonnes (2m wide) 2480 tonnes (3m wide) 4133 tonnes (5m wide)
Drilling	Hole Diameter	64mm (2.5 inch) dia
	Drill Spacing	1.2 metres
	Drill Burden	1.2 metres
	Overlap per Hole (past stope mid-point)	1.5 metres
	Number of Drill Rings	9
	Holes per Drill Ring	2
	Footage per Drill Ring	54 metres
	Redrill/Cleaning Allowance	10%
	Total Footage per Stope	535 metres
	Drill Factor	4.6 tonnes/metre
Raise	Raise	11-holes 1.2 metre x 1.2 metre drop raise
	Length of Raise	26 metres
	Footage in Drop Raise (including reaming and 10% redrill allowance)	315 metres
Blasting	Explosive	Anfo and stick in wet conditions
	Maximum 2 Rings Full Height	20 feet
	Detonator	Nonel in all blasts.
	Number of Blasts	6
Backfill	Tonnage	50% of mined tonnage

* Note: Actual quantities vary stope by stope. Typical quantities are shown here for estimating purposes to determine the cost per ton.

Table 5.5: Slot Slash 15 Metre High (uppers retreat)

Category	Design Item	Value*
Dimensions	Width	2 metres Minimum
	Length	11 metres
	Height	15 metres
Mucking	Tonnage (excluding bottom sill)	954 tonnes (2m wide) 1430 tonnes (3m wide) 2384 tonnes (5m wide)
Drilling	Hole Diameter	64mm (2.5 inch) dia
	Drill Spacing	1.2 metres
	Drill Burden	1.2 metres
	Number of Drill Rings	9
	Holes per Drill Ring	2
	Footage per Drill Ring	30 metres
	Redrill/Cleaning Allowance	10%
	Total Footage per Stope	297 metres
	Drill Factor	4.8 tonnes/metre
Raise	Raise	11-holes 1.2 metre x 1.2 metre uphole raise
	Length of Raise	15 metres max
	Footage in Drop Raise (including reaming and 10% redrill allowance)	180 metres
Blasting	Explosive	Anfo and stick in wet conditions
	Maximum 2 Rings Full Height	20 feet
	Detonator	Nonel in all blasts.
	Number of Blasts	6
Backfill	Tonnage	50% of mined tonnage

* Note: Actual quantities vary stope by stope. Typical quantities are shown here for estimating purposes to determine the cost per ton.

Table 5.6: Bulk Mining Miscellaneous

Category	Design Item	Value
Minimum Mining Width	Stope Width	2 metre minimum
Longitudinal Design	Stope Width	Less than 16 metres
Transverse Design	Stope Width	Greater than 16 metres
Remote Stand	Installation Frequency	100%
Brow Support	Shotcrete (if required)	75 mm (3 inch) nominal
Dump Wall	Installation Frequency	25%
Backfill	Backfill Replacement Factor (weight percentage to ore)	50%
	Backfill Barricade	Rock Berm

Table 5.7: Bulk Mining Stope Sizes

Stope Height (m)	Strike Length in Meta-volcanic (m)	Strike Length in Ultramafic (m)
30	13	10
20	15	11
15	20	12

6.0 MINEABILITY AND DILUTION

Table 6.1 to 6.3 lists the mineability and dilution factors that are applied to the tonnage and grades of the stope shapes designed and evaluated in GijimaAST Mine2-4D software. Bulk stope shapes were designed on a stope by stope basis with the aid of Mineable Stope Optimizer (MSO) shapes, typically with two strings per stope with six nodes per string. Cut and fill stopes were designed on a cut by cut basis with the aid of MSO and MRO (Mineable Reserve Optimizer) shapes.

Note, bottom sill development is not included in the bulk mining mineability.

Table 6.1: Typical Industry Mineability and Dilution

Mining Method	Mineability	Dilution
Cut and Fill, sill cut and remaining cuts	95%	10%
Slot Slash	85%	40%
Uppers Retreat	70%	40%

Table 6.2: Average Mining Recovery

Mining Method	Development Recovery	Stope Recovery	Operational Losses	Sill Pillar Losses	Rib Pillar Losses	Total Mining Recovery*
Longitudinal Mining-Pillars and No Fill	17%	83%	5%	8%	23%	**71%**
Longitudinal Mining-No Pillars and Cemented Fill	17%	83%	5%	0%	0%	**95%**
Longitudinal Mining with Uncemented Backfill	17%	83%	15%	0%	0%	**85%**
Cut and Fill	0%	100%	5%	0%	0%	**95%**

Note: Total mining recovery is calculated as development recovery + (stope recovery – operational losses) x pillar losses.

Table 6.3: Average Dilution

Mining Method	Planned Dilution	Unplanned Dilution (Overbreak)	Backfill Dilution	Total Dilution
Longitudinal Bulk Mining with Pillars	18%	8%	4%	**30%**
Longitudinal Bulk Mining No Pillars	18%	8%	8%	**34%**
Cut and Fill Mining	18%	0%	4%	**22%**

7.0 PERFORMANCES

Table 7.1: Performance by Crew

Crew	Description	Performance
Development – Ramp, Haulge Drift, Level and Miscellaneous Development	Single Heading	4.5 metres/day
	Double Heading	5.0 metres/day
	Multiple Heading	6.0 metres/day
Development – Longitudinal Sills (4x4 equivalent)	Single Heading	4.5 metres/day
	Double Heading	5.0 metres/day
	Multiple Heading	6.0 metres/day
Alimak Raise	Bald Raise	2.0 metres/day
	With Ladderway	1.5 metres/day
Production	Small Longitudinal Mining 20 m stope-3.5vs 6 yrd scoop	130-170 tonnes/day
	Large Longitudinal Mining 30 m stope-3.5 vs 6 yrd scoop	150-200 tonnes/day
	Cut and Fill Mining (3.5 vs 6 yrd scoop)	130-160 tonnes/day

Table 7.2: Performance by Task

Task	Equipment	Performance	Comment
Development	Jumbo (1-boom)	1.5 rounds per shift	Additional consideration to be given to geography
	Jumbo (2-boom)	1.5 rounds per shift	
	Scissor Lift	0.75 rounds per shift	
Mucking	LHD (3 yard)	400 tonnes per day	Assumed 100% rehandle on ore and 200% rehandle on waste
	LHD (6 yard)	600 tonnes per day	
Haulage	Haulage Truck (30 tonne)	325 to 650 tonnes per day	Varies with depth
Production Drilling	Top Hammer Drill (large diameter)	140 metres per day	Additional consideration to be given to geography
	Top Hammer Drill (small diameter)	160 metres per day	
Backfilling	Rock Backfill	150 tonnes per day	

8.0 MOBILE EQUIPMENT

The following tables describe the mobile equipment included in the Bradshaw gold deposit recommended/endorsed by Stantec Consulting Ltd., which will be used in developing production capacities, and determining ventilation requirements. Total ventilation requirements will be dependent on production rates, development crews and production faces.

Table 8.1: Equipment List

Equipment	Make	Model	Capacity	Ventilation Prescription (100cfm/hp)
Jumbo	RDH	Drillmaster 100	1-boom	-
Jumbo	RDH	Drillmaster 200	2-boom	11,600 cfm
Scissor Lift	RDH	Liftmaster 500N	-	11,600 cfm
LHD	Cat	R1300G	3 yard	16,500 cfm
LHD	Cat	R1600H	6 yard	27,900 cfm
Haulage Truck	Cat	AD30	30 tonne	40,000 cfm
Top Hammer Drill	Boart	StopeMate	63 mm diameter	-
Small Backhoe/Forklift	Kubota	R520	-	4,600 cfm
Small Personnel Carrier	Toyota	Mancarrier	-	13,400 cfm
Boom Truck	RDH	Loadmaster 600	-	11,600 cfm
Grader	-	-	-	-
Fuel Truck	-	-	-	-

9.0 BACKFILL

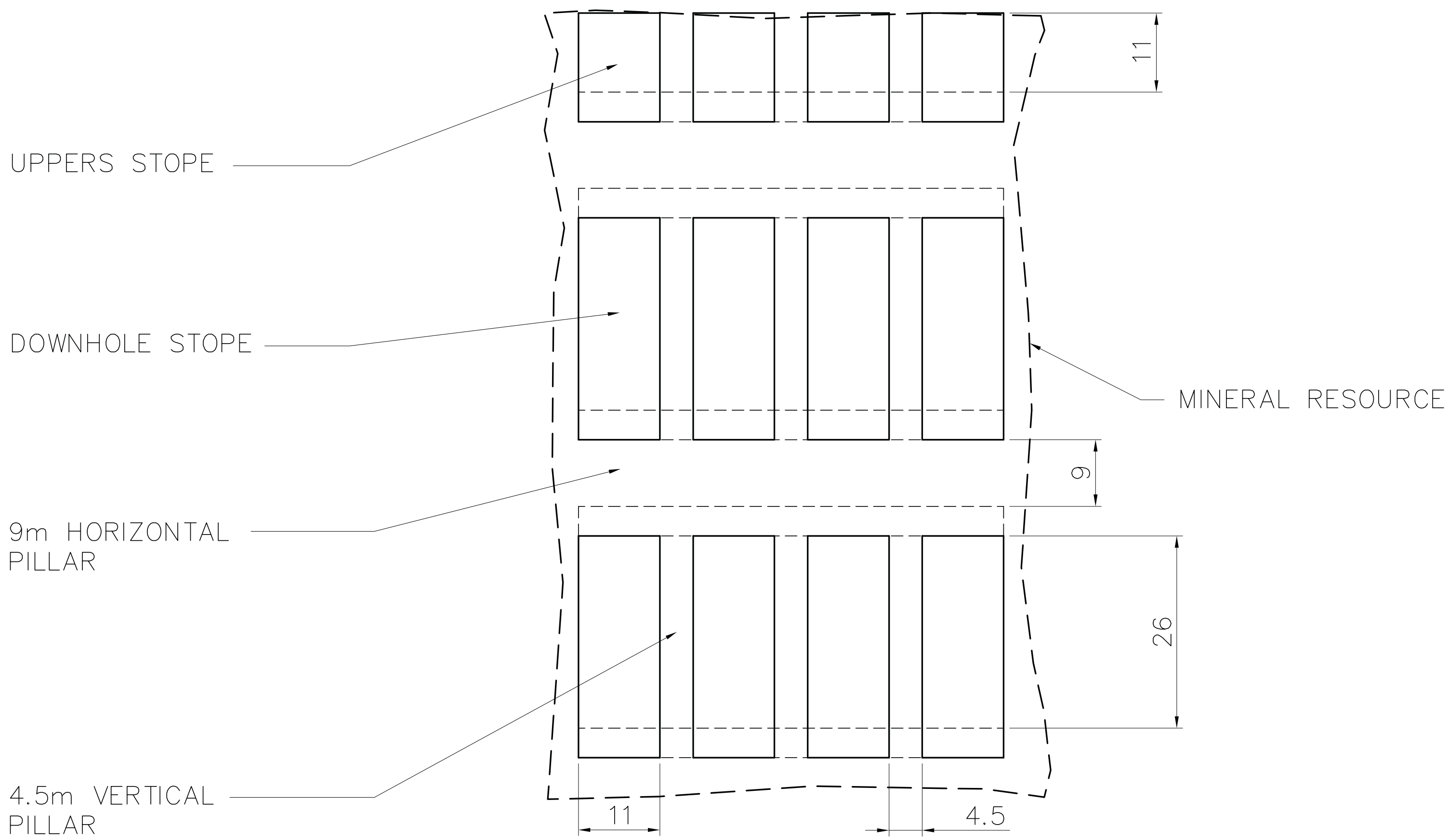
Production rates need to be determined before we can finalize the backfill design for the Bradshaw deposit.

Table 9.1: Backfill Design Criteria

Parameter	Value	Unit
Annual mining rate	TBD	t/y
Monthly mining rate t/mth	TBD	
Mining days d/y	365	
Avg daily mining rate t/d	TBD	
Ore S.G.	2.89	t/m^3
Rockfill bulk density	1.83	t/m^3
Ore to fill density ratio	1.47	
Annual fill rate	TBD	t/y
Fill Utilization	67.9%	
Design backfill rate (dry rock)	TBD	t/d
Design slurry production rate	TBD	t/d
Portion of fill that is cemented	TBD	
Average binder content - CRF	5.0%	dry wt%
Annual binder consumption	TBD	t/y
Annual binder storage	TBD	m3
Design binder consumption	TBD	t/d
Design slurry production rate	TBD	t/d
Annual retardant consumption	TBD	L/y
Annual retardant storage	TBD	m^3
Batch Recipe		
Water	3.00	tonnes
Binder	4.50	tonnes
Initial slurry batch size	4.43	m^3
Initial slurry density	1,694	kg/m^3
Flush water	1.50	tonnes
Total slurry batch size	5.93	m^3
Final slurry density	1,518	kg/m^3
Water Storage Tank	5.0	m^3
Fill pipe for mix tank	6	inch
Mix tank size	5.0	m^3
Transfer pipe to U/G	3	inch
U/G tank size	10.0	m^3
Spray bar pipe	3	inch

APPENDIX A

SKETCHES



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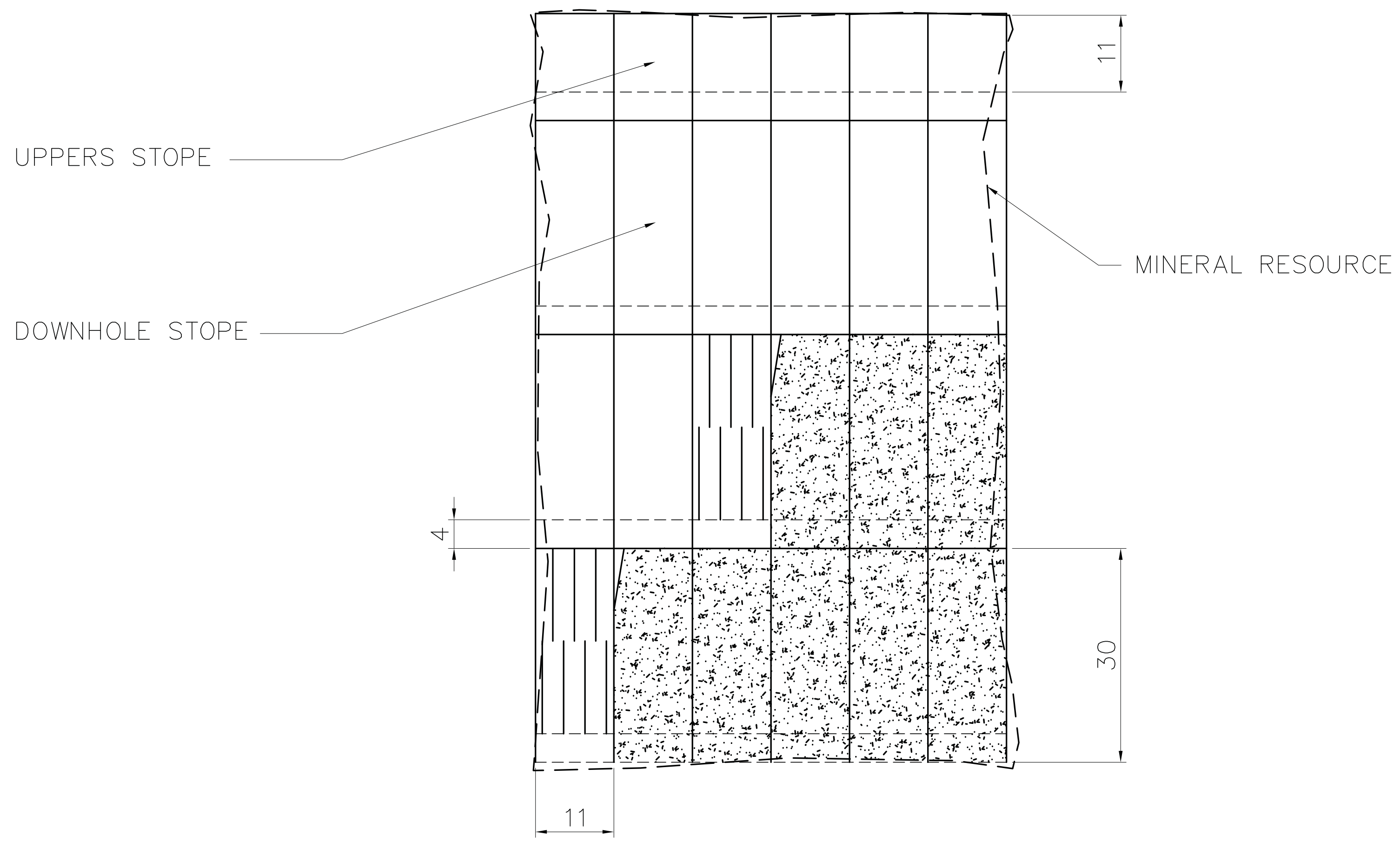

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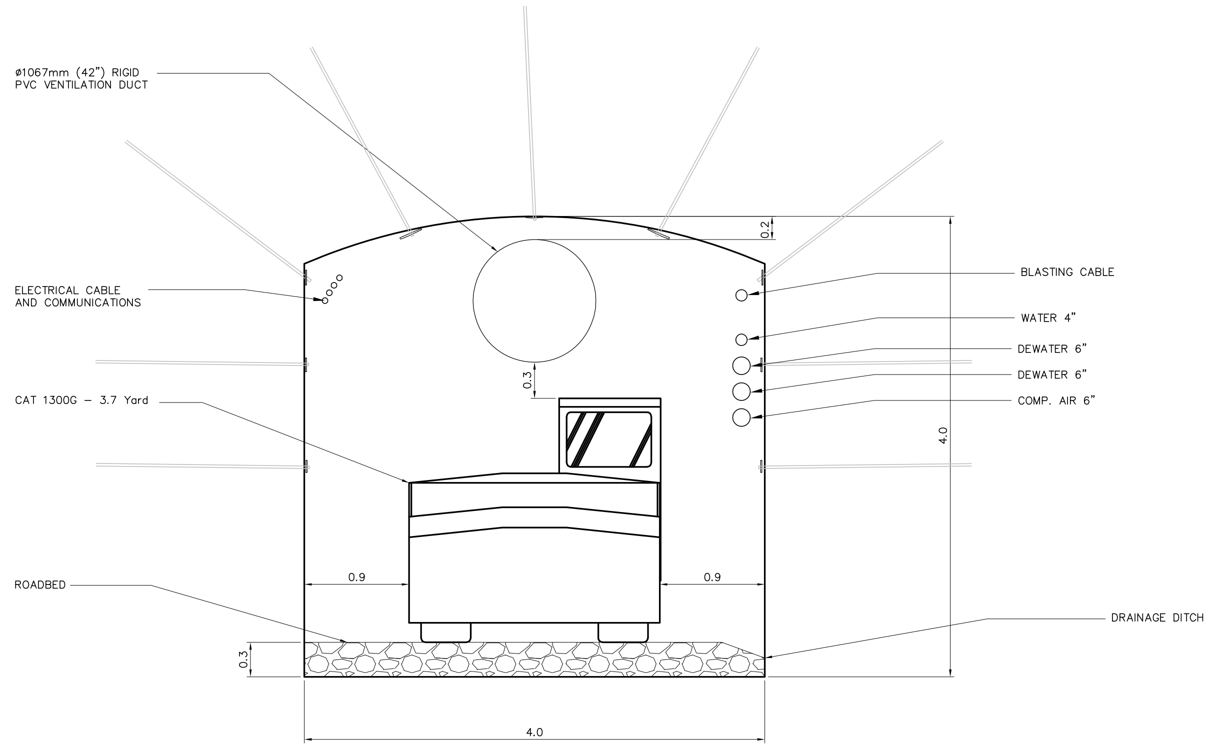

DRIFT PROFILE WITH VENTILATION DUCTING & 3.7 YARD LHD
4.0m WIDE x 4.0m HIGH ARCH BACK DRIFT

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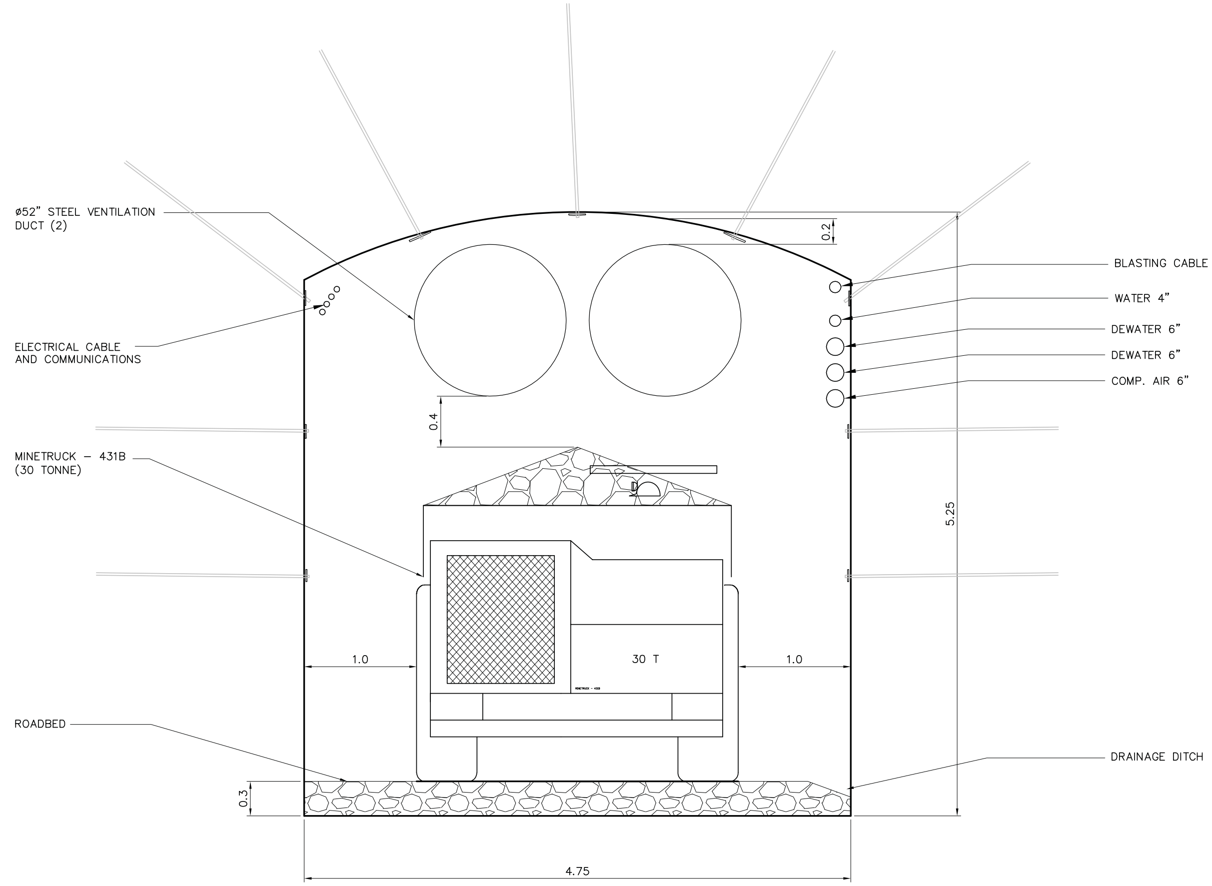

DRIFT PROFILE WITH VENTILATION DUCTING & 30 TONNE TRUCK
4.75m WIDE x 5.25m HIGH ARCH BACK DRIFT

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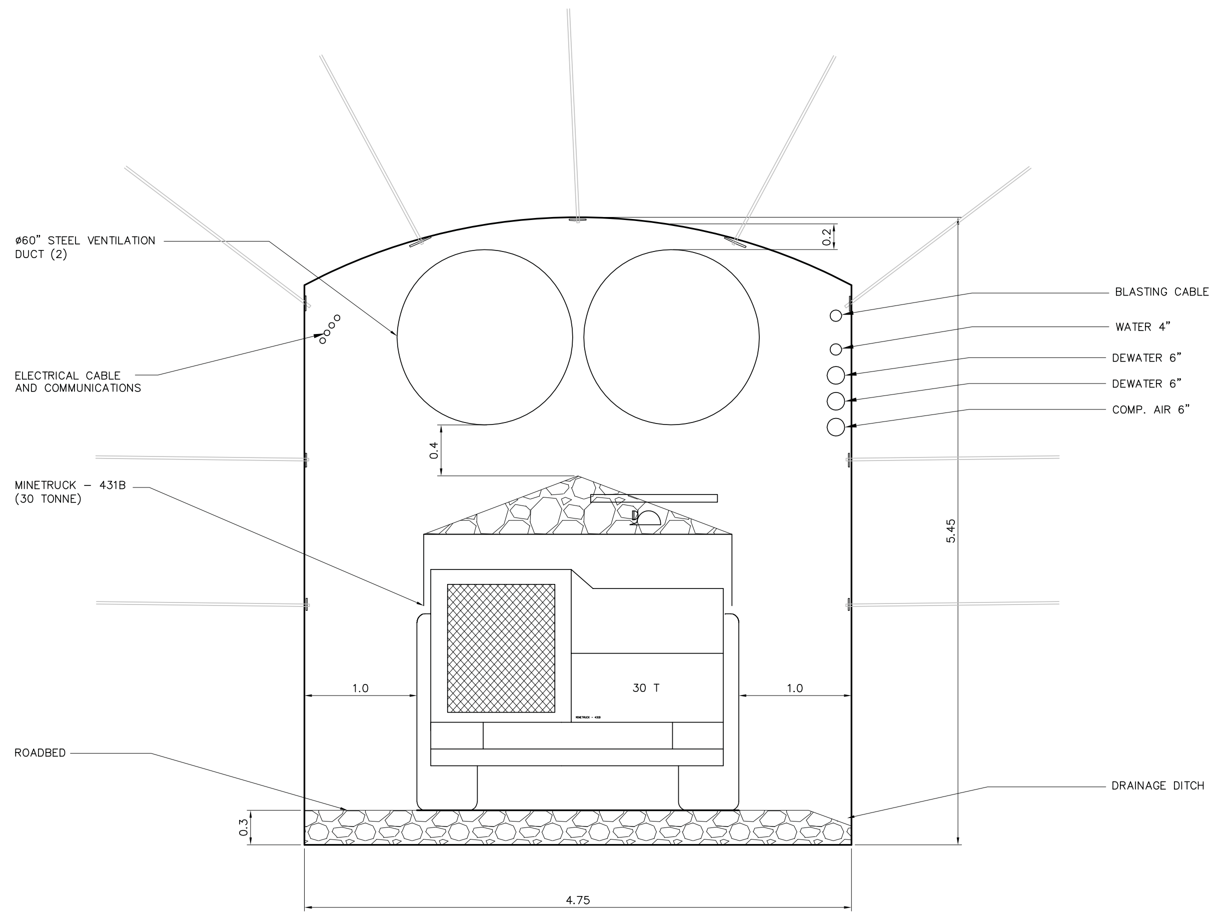

DRIFT PROFILE WITH VENTILATION DUCTING & 30 TONNE TRUCK
4.75m WIDE x 5.25m HIGH ARCH BACK DRIFT

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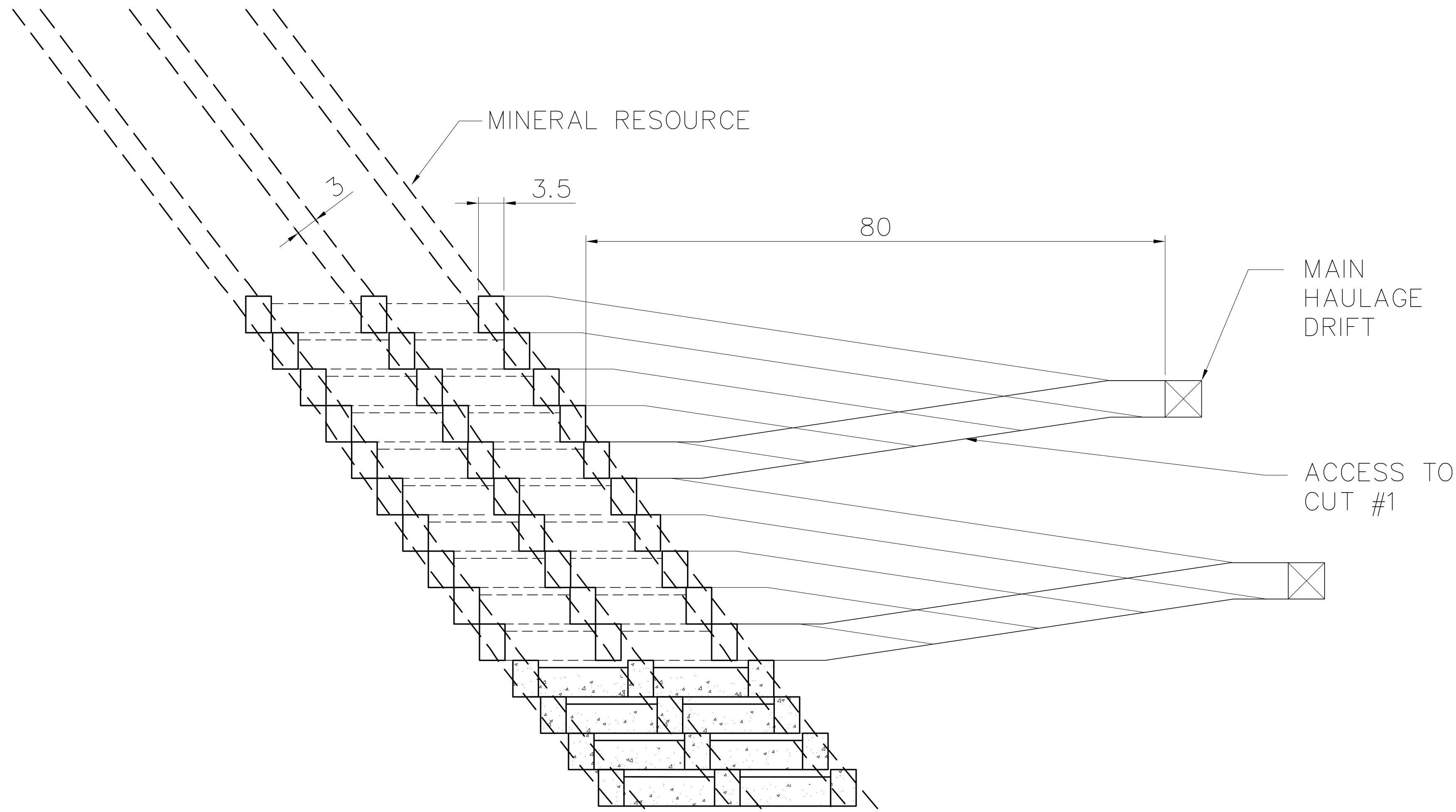

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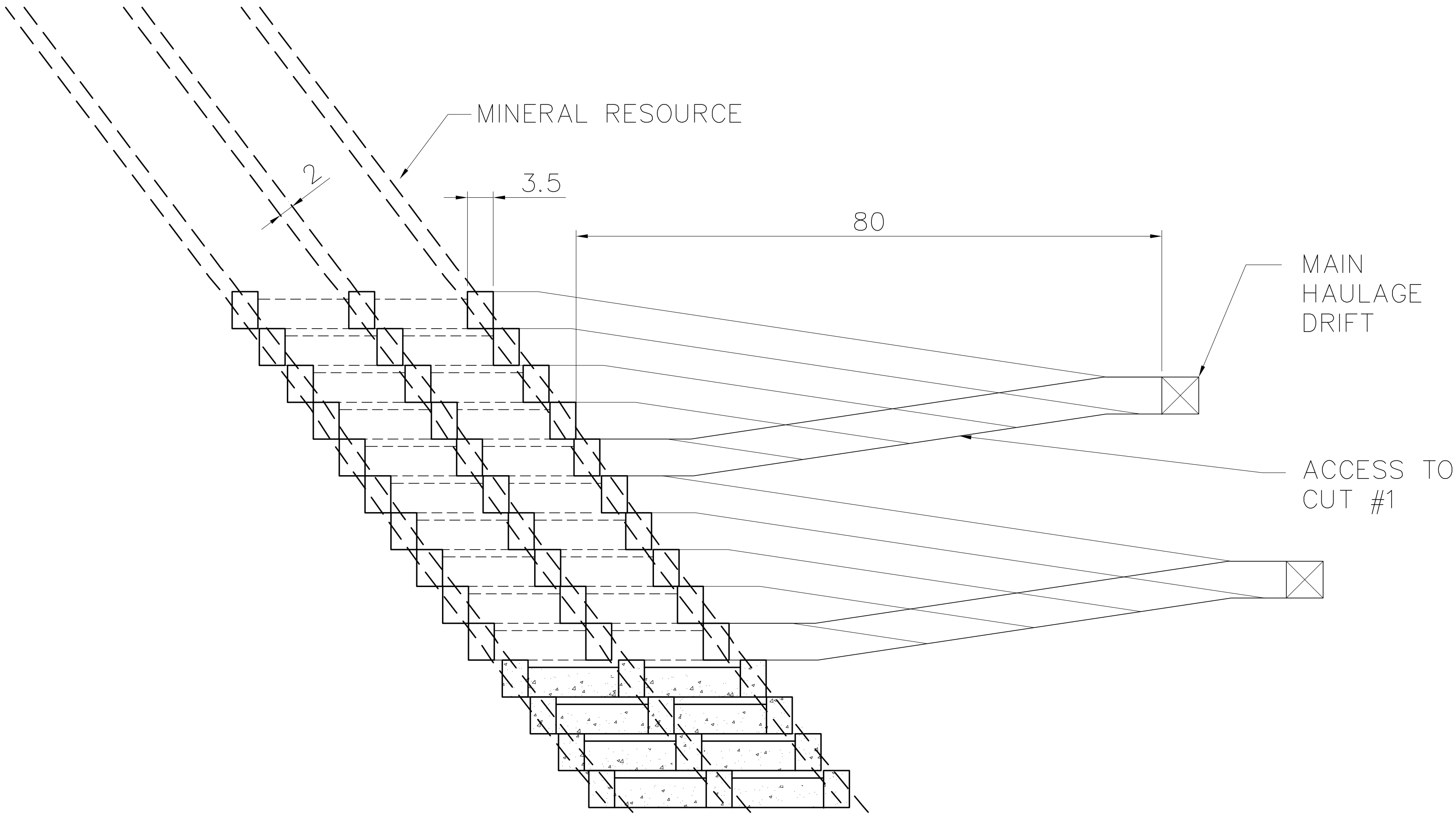

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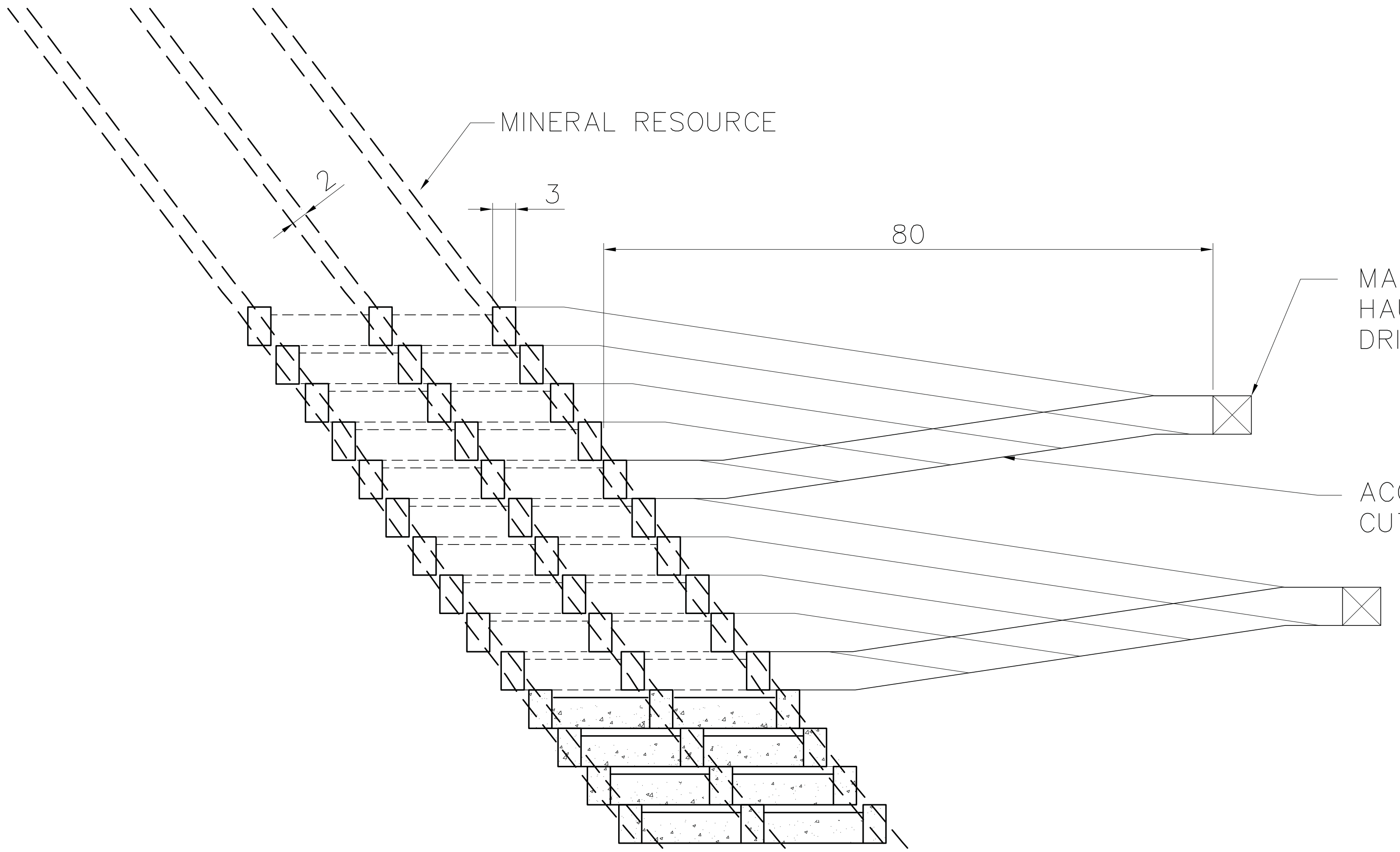

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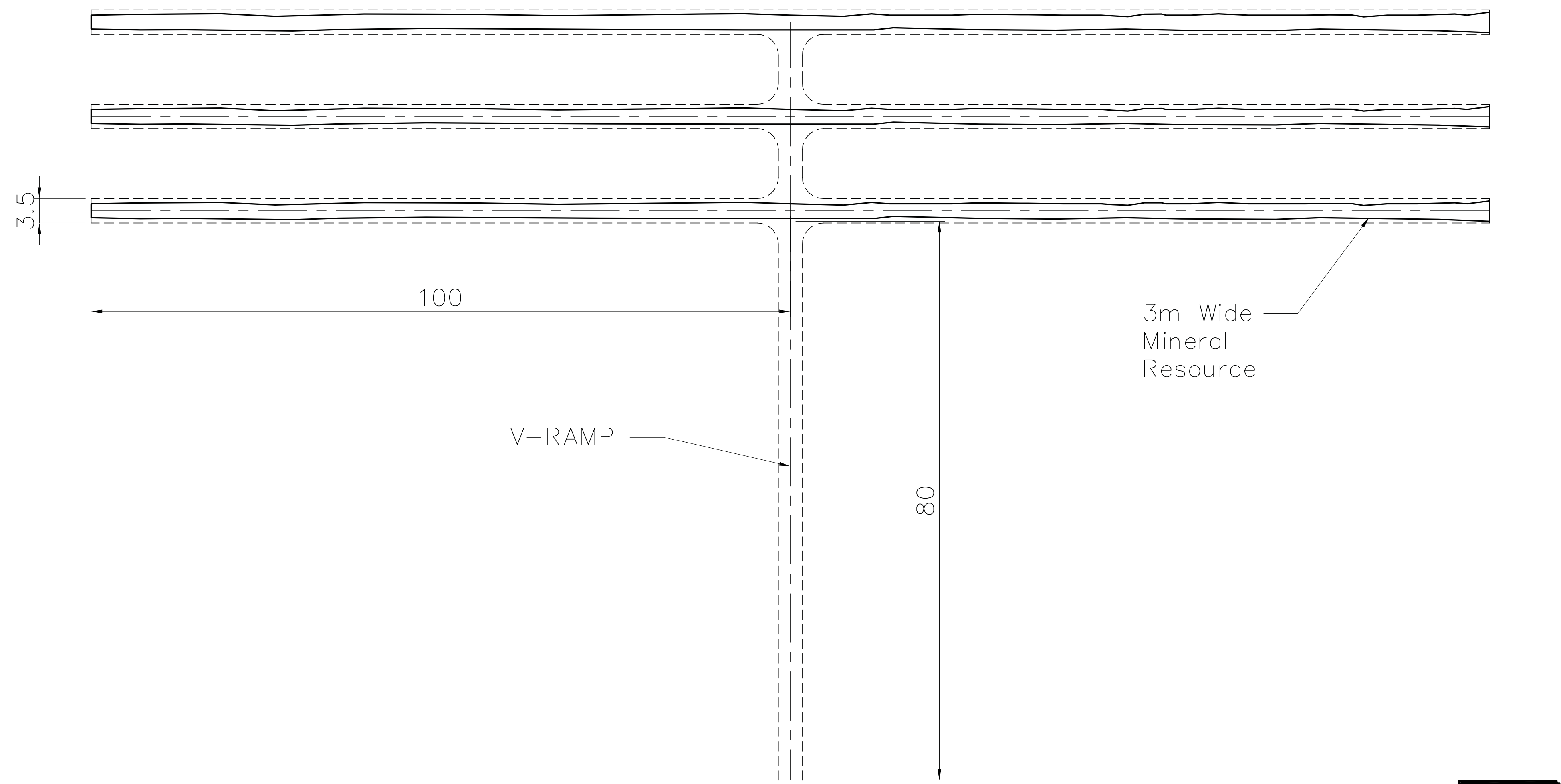

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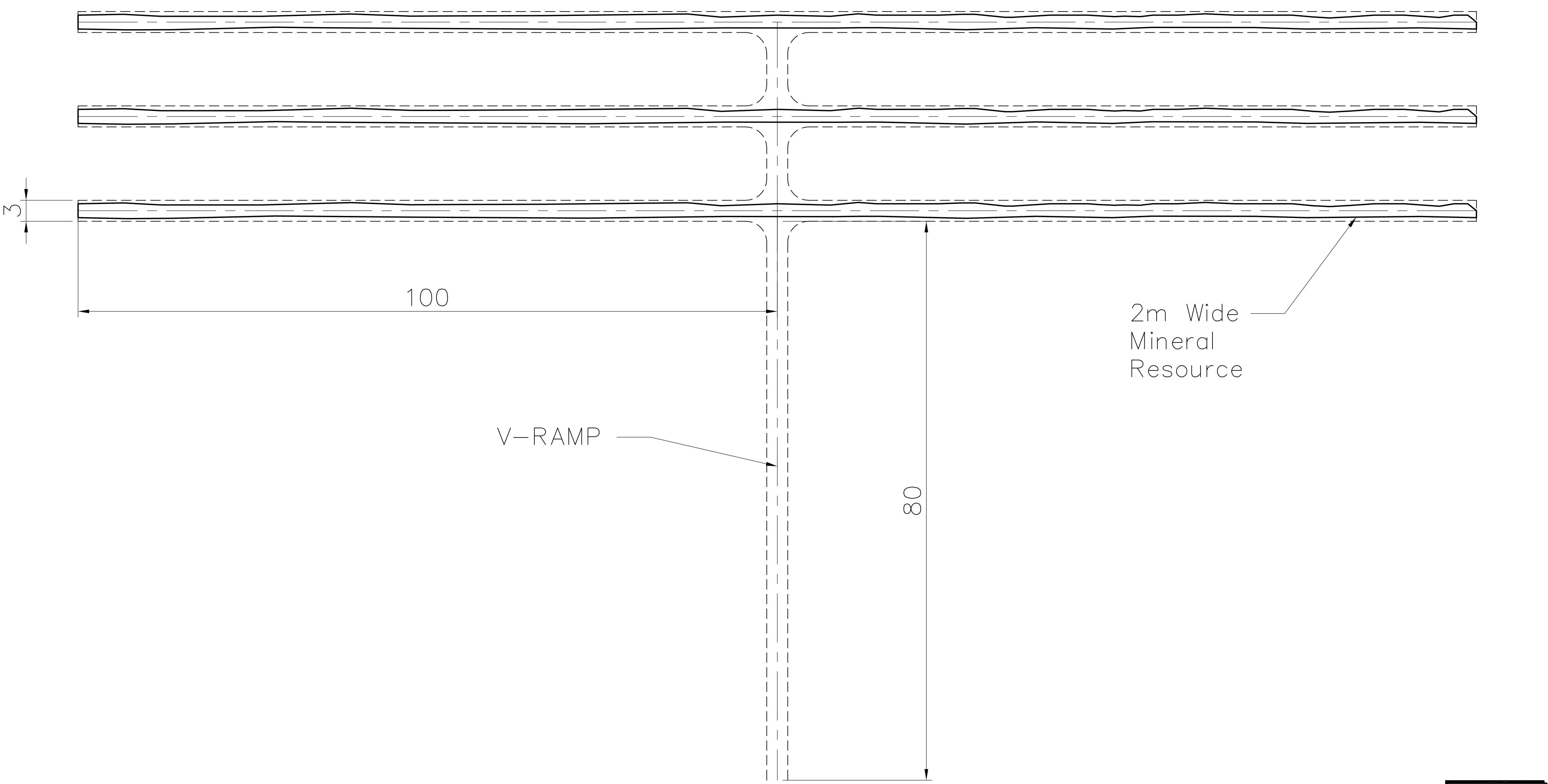

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THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

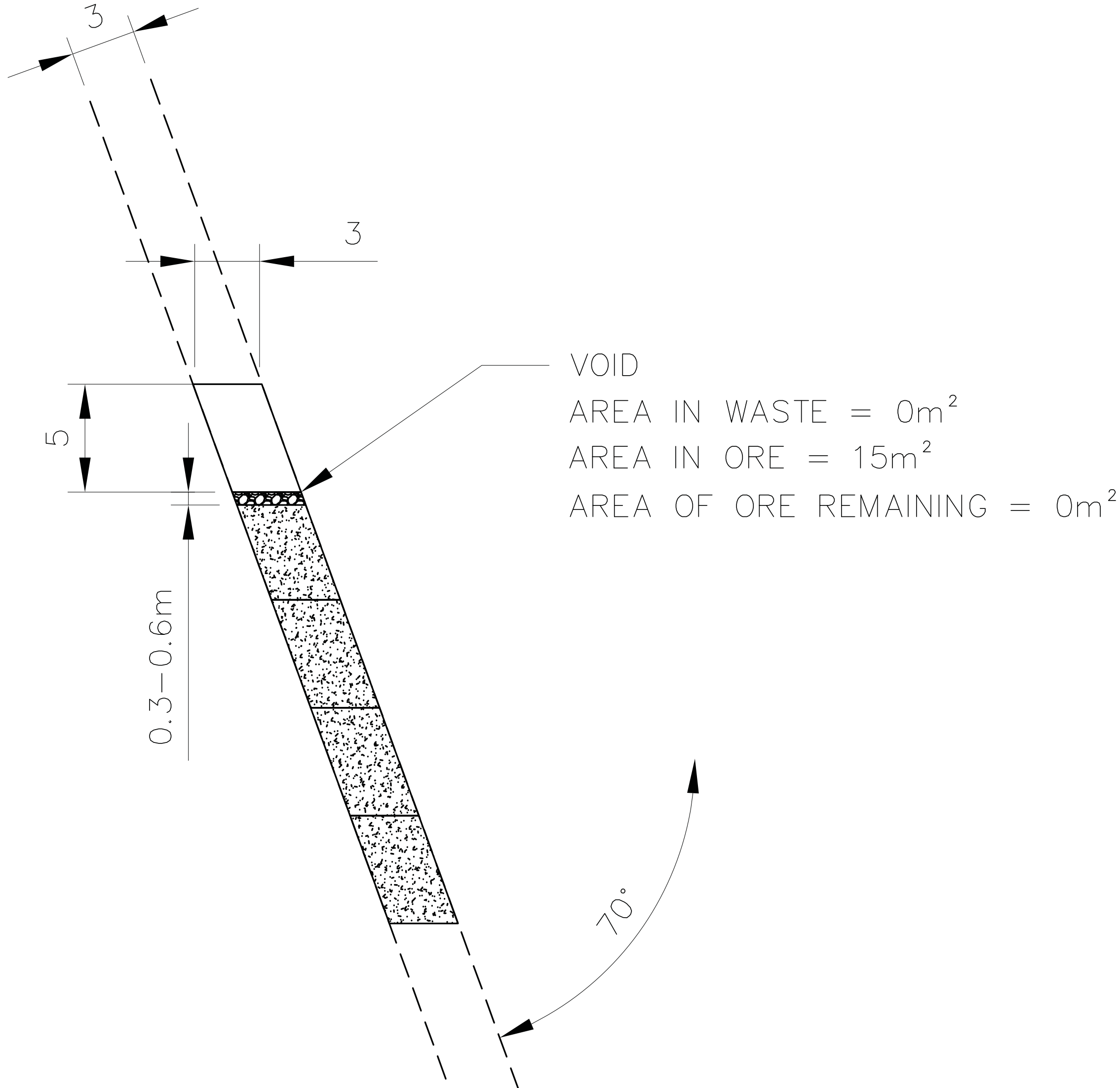
ORIGINAL DRAWN BY: RJB
DATE: 2015.01.24

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit

SCALE: NTS
PROJ No 000000000
DWG No 0000000
REV No A

Jan 30, 2015 – 12:24pm \\CD1211–N02C:\GoWest – Gold 2015\Cut and Fill Profile – Gowest – 3m drift – 2m ore – Plan.dwg



NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.01.24

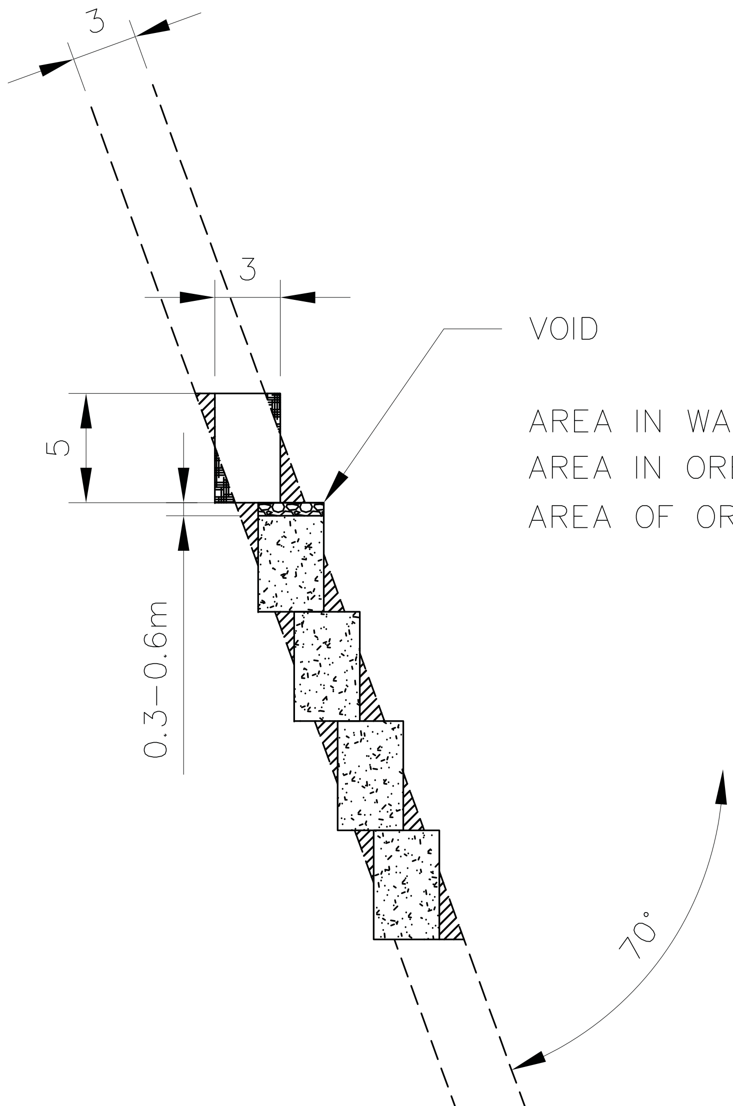
Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit
SCALE: NTS
PROJ No 000000000
DWG No 0000000
REV No A

PROJ No	DWG No	REV No	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
000000000	0000000	A	---	---	A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

Jan 29, 2015 – 3:01pm \\CD1211-N02C:\GoWest – Gold 2015\Cut and Fill Profile – Gowest – 3m drift – 3m ore Dilution – 70d Walls.dwg



AREA IN WASTE = $1.86m^2$

AREA IN ORE = $13.14m^2$

AREA OF ORE REMAINING = $2.82m^2$

PRELIMINARY

PROJ No	DWG No	REV No	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
000000000	0000000	A	---	---	A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

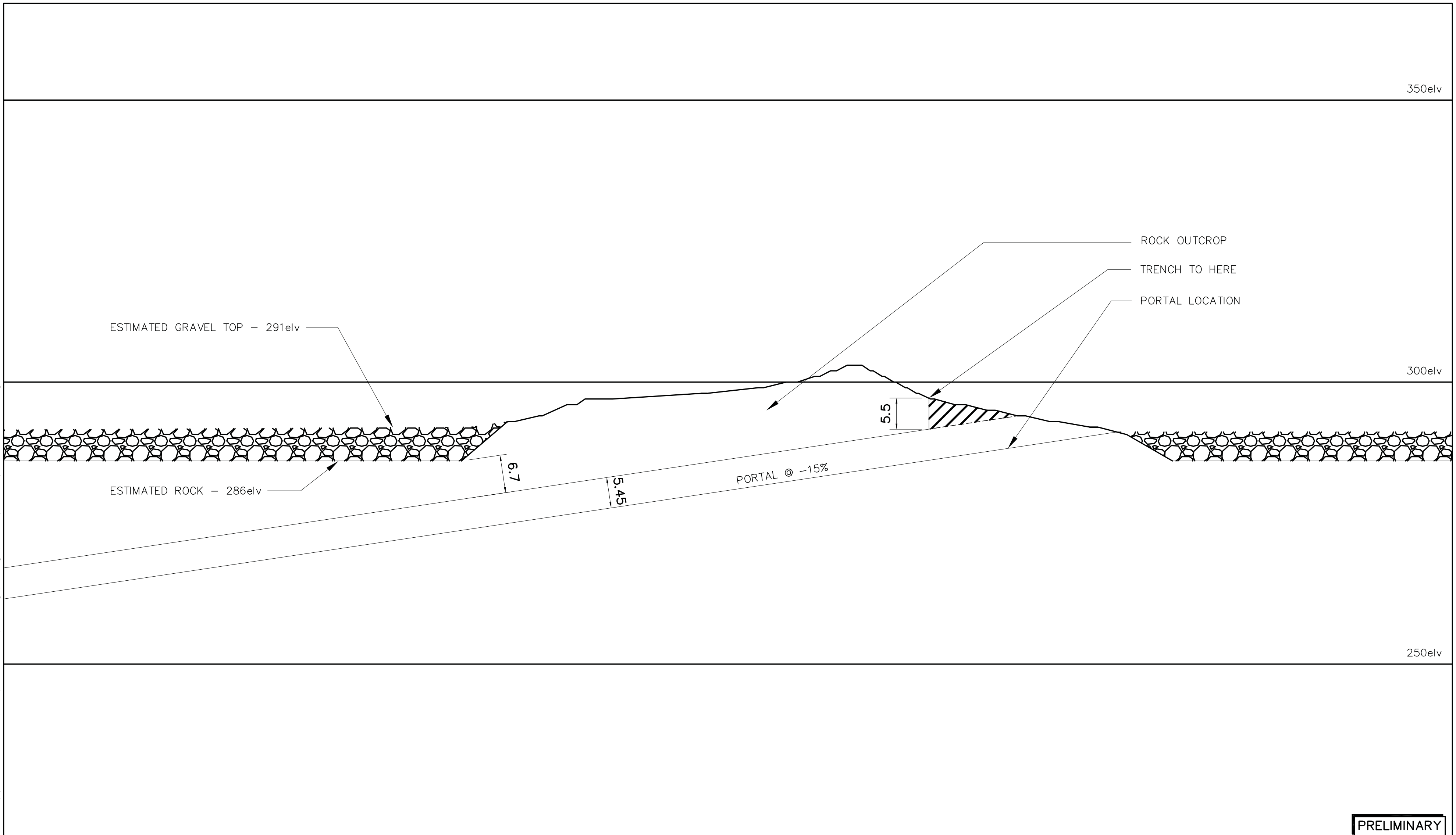
PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.01.24

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit
SCALE: NTS
PROJ No 000000000
DWG No 0000000
REV No A

Jan 29, 2015 – 2:45pm \\CD1211-N02C:\GoWest – Gold 2015\Cut and Fill Profile – Gowest – 3m drift – 3m ore Dilution.dwg



REV No	A
PROJ No DWG No	169514568

REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
---	---	A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

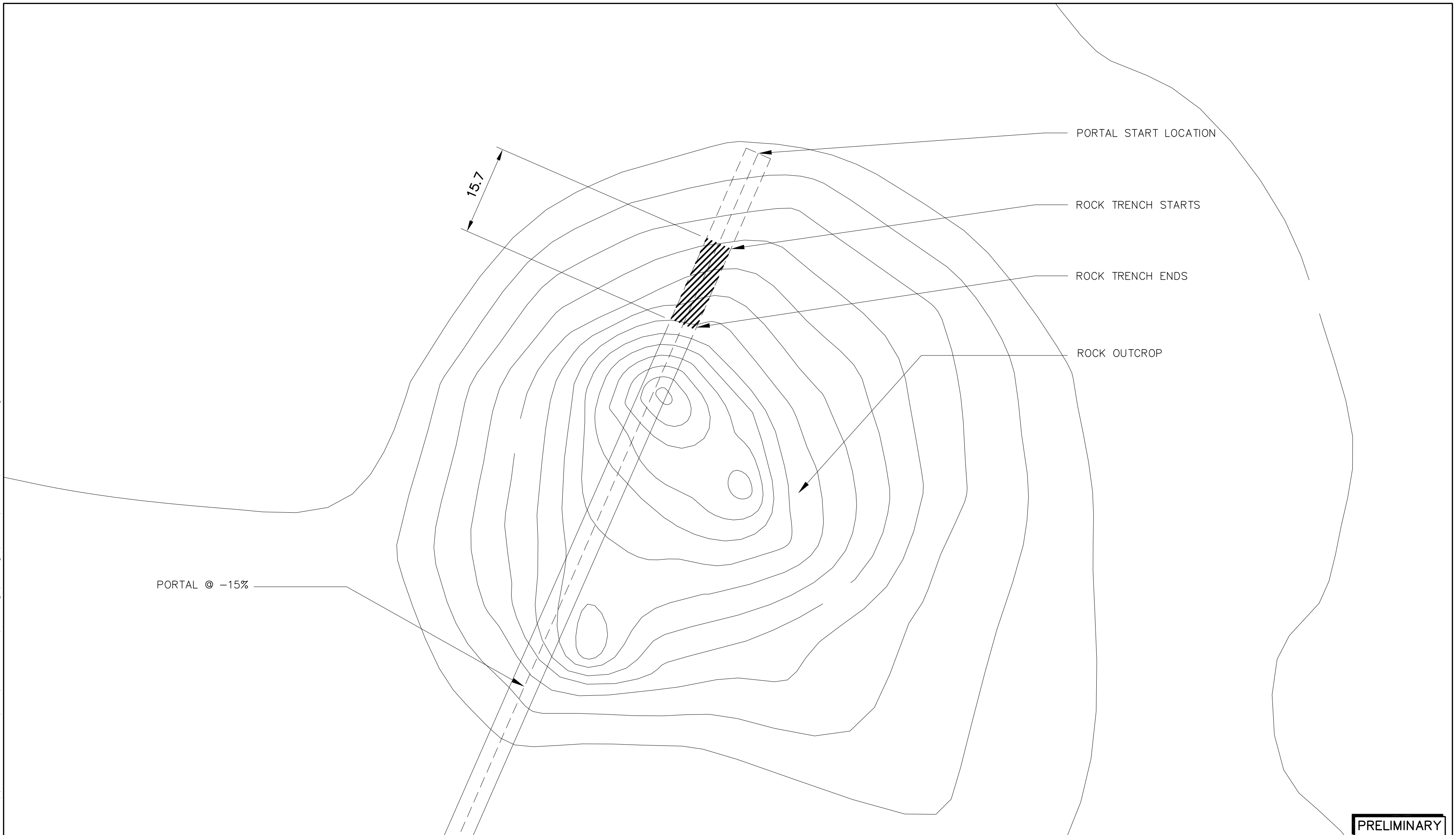
ORIGINAL DRAWN BY: RJB
DATE: 2015.01.22

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit

SCALE: SCALE = 30:1 | PROJ No 169514568 | DWG No | REV No A

Feb 09, 2015 – 3:01pm \\CD1211-N02S:\212\169514568\drawing\mining_prelim\sketch\Portal Section.dwg



REV No	REF DWG No	REFERENCE DRAWING TITLE	REV	DESCRIPTION	YY.MM.DD	BY
A	---	---	A	PRELIMINARY –		

THIS DRAWING IS NOT VALID UNLESS THE LAST REVISION IS HAND SIGNED

PROJ No DWG No 169514568

NOT FOR CONSTRUCTION

PROGRESS PRINT
DWG UPDATED: YY.MM.DD
BY: XXX

ORIGINAL DRAWN BY: RJB
DATE: 2015.01.22

Stantec

CLIENT: Gowest
PROPERTY: Timmins
PROJECT: Bradshaw Deposit
SCALE: SCALE = NTS
PROJ No 169514568
DWG No
REV No A

Feb 09, 2015 – 2:57pm \\CD1211-N02S:\212\169514568\drawing\mining_prelim\sketch\Portal Plan.dwg

APPENDIX H
AIR AND WATER

Stantec
1760 Regent St.
Sudbury, Ontario
P3E 3Z8
Telephone: 1-705-566-6891
Fax: 1-705-566-5589

DESIGN CALCULATIONS

PROJECT NUMBER:	**169514568**
TITLE:	**BRADSHAW PRE-FEASIBILITY PROJECT**
CLIENT:	**GOWEST GOLD LIMITED**
LOCATION:	**ONTARIO**
DOCUMENT No:	**CAL_COMP_AIR_CONSUMP**
SUBTITLE:	**COMPRESSED AIR CONSUMPTION CALCULATION**
REVISION:	**B – FINAL PRELIMINARY CALCULATION**
DEPARTMENT:	**MECHANICAL ENGINEERING**
BY:	**KIMBERLY NADON**
DATE:	**MARCH 10, 2015**
PROJECT MGR:	**NORIS DEL BEL BELLUZ**
CHECKED BY:	**JESSICA KERANEN, RON FLEMING**

MECHANICAL CHECK	**YES (X)**	**NO ( )**
INDEPENDENT REVIEW	**YES ( )**	**NO ( )**
COMPARATIVE REVIEW	**YES ( )**	**NO ( )**

COMMENTS

These calculations provide estimates for compressed air requirements for the Bradshaw Pre-Feasibility study. The calculation includes:

- **Compressed air requirements**
- **Compressor sizing**

Stantec			
Client: Gowest Gold Limited	Rev.: B	Date: 10-Mar-15
Project: 169514568	Made By: KAN	Date: 27-Feb-15
Subject: Compressed Air	Checked By: JMK	Date: 09-Mar-15
Title: Compressed Air Consumption and Compressor Sizing		

Notes:

1) The purpose of this spreadsheet is to estimate the compressed air consumption for the Gowest Gold Bradshaw Pre-Feasibility project.

Assumptions:

1) Leakage is estimated at 20%.
2) Average utilization is based on typical work sequences during the mining cycle (drill, blast and muck).
3) Analysis assumes all compressed air users will be operating at maximum rated consumption (conservative assumption).
4) Hand tool air consumption is based on 3/4" to 1-1/4" impact wrench at 20 CFM.
5) Two production crews will be working simultaneously, outfitted with the following equipment:
   - Two blow pipes (one per crew) for removing loose rock.
   - Two Boart Longyear StopeMaster drills (one per crew).
6) There will be two development crews working simultaneously, outfitted with the following equipment:
   - Two 2-boom jumbos (Drillmaster 200), for face drilling (one per crew).
   - Two stopers/jacklegs (one per crew).
   - Two blow pipes for removing loose rock (one per crew).
   - Two pneumatic pumps (one per crew) for dewatering at the face.
7) Two diamond drills will be operating at one time, with one dewatering pump operating on location.
8) Two construction crews will be required to install infrastructure, outfitted with the following equipment:
   - Two dry shotcrete sprayers (one per crew).
   - Two stopers/jacklegs (one per crew).
   Note: only 2 activities will occur simultaneously, it will be assumed that one shotcrete sprayer and one stoper/jackleg will be operating at the same time.
9) Refuge stations will only consume air in emergency situations when no other equipment is in use, therefore will not contribute to the daily average or peak consumption.

Legend: Input Data: XXXX

Calculation:

Compressed Air Requirement

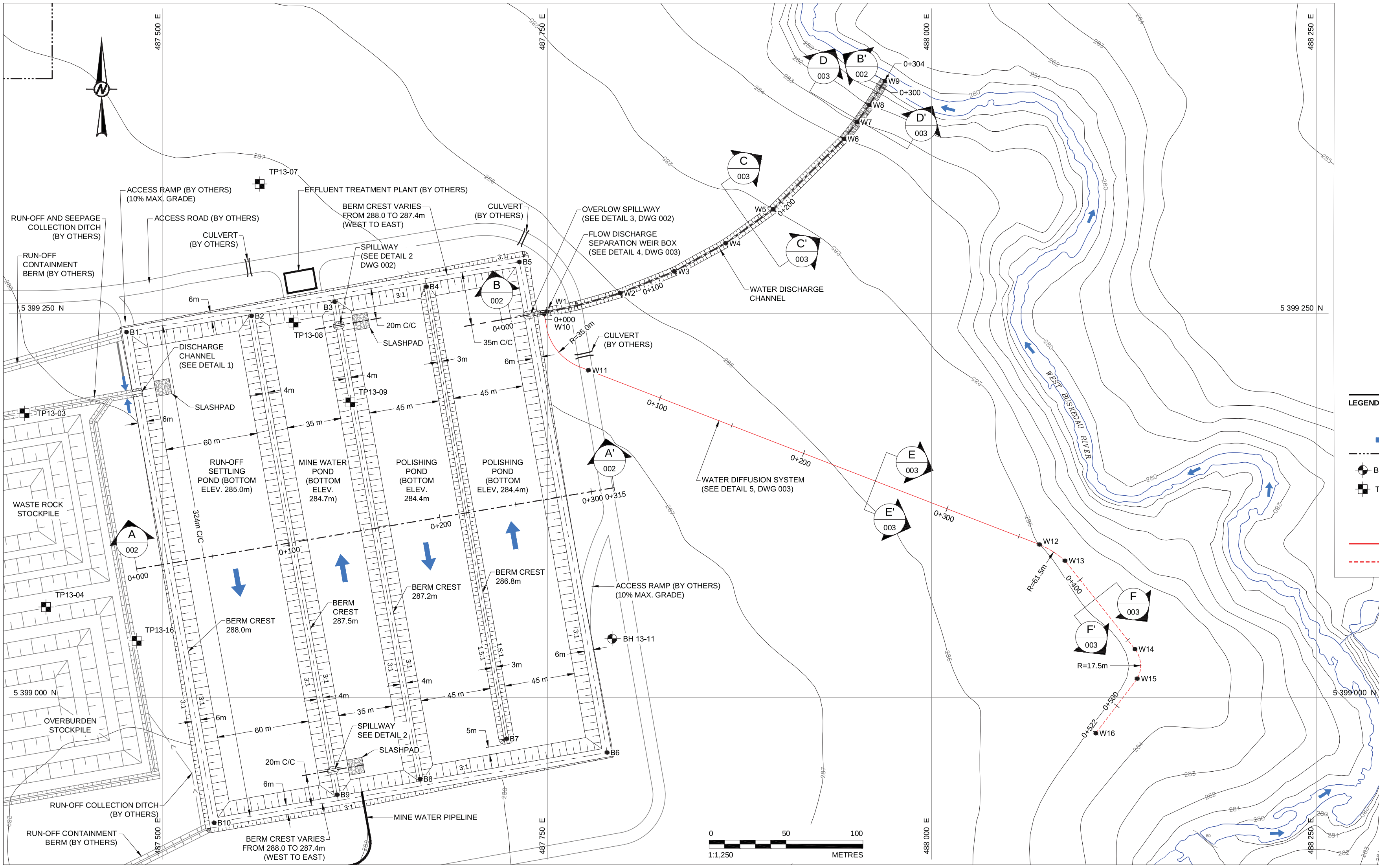
Area	Crew	Equipment Model	Operation	Unit Rated Consumption (CFM)	No. of Units	Total Operating Consumption (CFM)	Average Utilization per Shift (CFM)	Operating Time (Hours per Shift)	Cumulative Consumption per Shift (CFM)	Possible Consumption Scenario During Typical Shift (CFM) 6:00	7:00	8:00	9:00	10:00	11:00	12:00	13:00	14:00	15:00	16:00	17:00	Comments
Underground	Production Crew (2)	Jumbo (2 boom) - StopeMaster	Production Drilling	5	2	10	40%	6	1,440			10	10	10			10	10	10			
		Blowpipe (2" Copper)	Removing Loose Rock	500	2	1000	5%	2	6,000						1000					1000		
	Development Crews (2)	Blowpipe (1/4" Copper)	Removing Loose Rock	100	2	200	5%	3	1,800			200				200				200		
		Jumbo (2 boom) - DrillMaster 200	Face Drilling	10	2	20	40%	8	3,840			20	20	20	20		20	20	20	20		
		Pneumatic Pump	Dewatering	40	2	80	40%	9	17,280			80	80	80	80	80	80	80	80	80		
		Jackleg/Stoper Drills	Drilling	175	2	350	40%	4	33,600			350	350				350	350				
	Diamond Drilling	Pneumatic Pump	Dewatering	40	2	80	50%	9	21,600			80	80	80	80	80	80	80	80	80		
	Construction Crew (1)	Shotcrete Sprayer	Bulkhead Construction	750	1	750	40%	4	72,000					750	750				750	750		
		Jackleg/Stoper Drills	Rock Anchor Drilling	175	1	175	40%	4	16,800			175	175				175	175				
	Underground Infrastructure	Refuge Station (4)	Misc.	20	4	80	0%	0	0													Assumption 9
									Total:	0	0	915	715	940	1930	360	715	715	940	2130	0	
									20% Leakage:	0	0	183	143	188	386	72	143	143	188	426	0	Assumption 1
									Total Consumption for Underground:	0	0	1098	858	1128	2316	432	858	858	1128	2556	0	
Surface	Garage	Hand Tool (3)	Misc.	20	3	60	5%	4	720			60		60				60		60		Assumption 4
	Water Treatment Plant	Hand Tool (1)	Misc.	20	1	20	5%	4	240		20		20					20		20		Assumption 5
	Dry/Offices	Hand Tool (1)	Misc.	20	1	20	5%	4	240	20					20	20					20	Assumption 6
									Total:	20	20	60	20	60	20	20	0	80	0	80	20	
									20% Leakage:	4	4	12	4	12	4	4	0	16	0	16	4	Assumption 1
									Total Consumption for Surface:	24	24	72	24	72	24	24	0	96	0	96	24	
								Totals:	174,360 CFM	20	20	975	735	1,000	1,950	380	715	795	940	2,210	20	
								Leakage:	17,436 CFM	4	4	195	147	200	390	76	143	159	188	442	4	
								Totals:	191,796 CFM	24	24	1,170	882	1,200	2,340	456	858	954	1,128	2,652	24	
								Peak:	2,660 CFM													
								Average:	270 CFM													

Compressor Sizing:

No. of Compressors	2	(2 duty, 1 spare)
Total Peak Capacity	2,660 CFM	
Each compressor Requirement	1,400 CFM	

Compressor Type	Manufacturer	Model	Working Pressure (psig)	Power Requirements HP	Power Requirements kW	Capacity CFM	Capacity L/s	Comments
Electric - Rotary Screw	Ingersoll Rand	EPE 300-2S	125	300	225	1,476	697	Oil free, 2-stage compressor
Portable - Diesel	Atlas Copco	XAS 746	125	440	328	1,500	709	Optimized fuel consumption, 796L fuel tank capacity
Electric - Rotary Screw	Atlas Copco	ZT 315 VSD	150	400	298	1,501	709	Oil free, rotary screw compressor
Electric - Centrifugal	Ingersoll Rand	C400	50 - 125	350 - 500	260 - 400	1,600 - 2,350	755 - 1,110	High efficiency, 100% oil-free
Electric	Atlas Copco	ZH 350+	87 - 130	470	350	1,483 - 2,330	700 - 1,100	High efficiency, 100% oil-free

Conclusion: The plant will require 2 duty compressors, and one spare compressor, each with a minimum 1,400 CFM capacity. The expected horsepower for each compressor is approximately 400 hp.



WATER MANAGEMENT POND SETTING OUT POINTS			
POINT No.	EASTING	NORTHING	CREST ELEV.
B1	487476.38	5399237.71	288.00
B2	487557.69	5399248.22	287.50
B3	487611.69	5399257.38	287.20
B4	487671.37	5399267.28	286.80
B5	487731.86	5399283.39	287.40
B6	487788.93	5398964.43	287.40
B7	487723.41	5398973.46	286.80
B8	487667.31	5398947.21	287.20
B9	487613.36	5398937.05	287.50
B10	487533.38	5398918.76	288.00

WATER DISCHARGE SYSTEM SETTING OUT POINTS			
POINT No.	EASTING	NORTHING	INVERT ELEV.
W1	487749.94	5399251.07	285.80
W2	487797.43	5399262.98	285.40
W3	487832.66	5399277.16	285.00
W4	487865.95	5399295.43	284.70
W5	487896.86	5399317.54	284.40
W6	487942.86	5399363.30	283.40
W7	487951.42	5399374.23	282.40
W8	487959.36	5399385.33	281.40
W9	487969.38	5399400.93	279.00
W10	487748.02	5399249.56	285.00
W11	487776.75	5399212.91	285.74
W12	488070.10	5399099.63	284.53
W13	488086.69	5399089.13	284.97
W14	488132.12	5399031.68	284.88
W15	488133.71	5399012.46	284.85
W16	488106.66	5398976.96	284.80

LEGEND

	SPILLWAY
	FLOW DIRECTION
	PROPERTY BOUNDARY
BH 13-11	BOREHOLE (GOLDER)
TP13-03	TESTPIT (GOLDER)
	CULVERT
C/C	CENTRE TO CENTRE
	16 INCH SOLD HDPE PIPE
	16 INCH PERFORATED HDPE PIPE

NOTES

1. REFER TO THE MOST RECENT DESIGN DOCUMENT ENTITLED "APPENDIX G - POND AND DISCHARGE SYSTEM DESIGN DETAILS" TO UNDERSTAND DESIGN INTENT.
2. ACTUAL LOCATION OF PERFORATED PIPE MAY VARY TO ACHIEVE 0.13% SLOPE.

Rev.	YYYY-MM-DD	DESCRIPTION	PREPARED	DESIGN	REVIEW	APPROVED
A	2015-03-06	DRAFT	MF	IM/AC	IM	KAB

SEAL

NOT FOR CONSTRUCTION

DRAFT

CLIENT
GOWEST GOLD LTD.
BRADSHAW PROJECT
TIMMINS, ONTARIO

CONSULTANT

Golder Associates

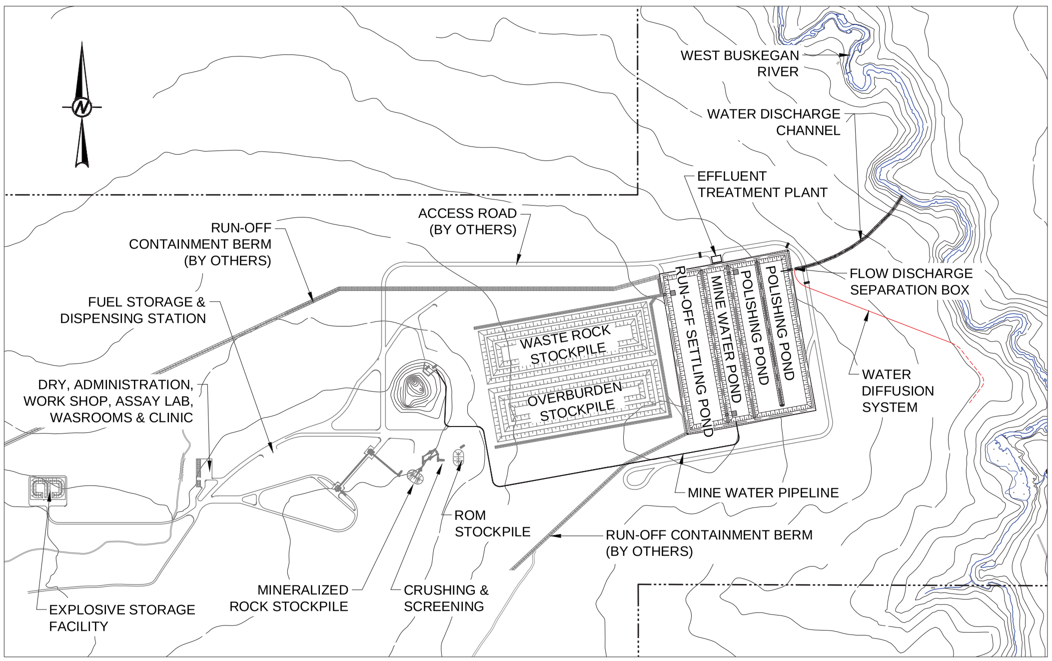
MISSISSAUGA
6925 CENTURY AVE, SUITE #100
MISSISSAUGA, ONTARIO
CANADA
[+1] (905) 567 4444
www.golder.com

PROJECT
BRADSHAW PROJECT

TITLE
GENERAL ARRANGEMENT PLAN OF WATER MANAGEMENT PONDS AND DISCHARGE SYSTEMS

PROJECT No.	PHASE	Rev.	of	DRAWING
09-1118-6010	14000	A		001

GOWEST GOLD LTD.
BRADSHAW PROJECT
WATER MANAGEMENT PONDS AND DISCHARGE SYSTEMS



SITE LOCATION PLAN

0 200 400
1:5,000 METRES

INDEX OF DRAWINGS		
DRAWING NO.	DRAWING SHEET TITLE	REVISION NO.
000	COVER PAGE	A
001	PLAN OF WATER MANAGEMENT PONDS AND DISCHARGE SYSTEMS	A
002	CROSS-SECTION A-A', PROFILE B-B', AND DETAILS 1 TO 4	A
003	CROSS-SECTIONS AND DETAILS	A

INDEX OF SPECIFICATIONS	
SECTION NO.	TITLE
1	SCOPE OF WORK
2	EXCAVATION
3	EARTHWORKS
4	FILTERED TAILINGS
5	GEOTEXTILE
6	HDPE PIPES
7	CARE OF WATER
8	SURVEY SERVICES
9	QA/QC PLAN

GENERAL NOTES:

1. THESE NOTES APPLY TO ALL DRAWINGS UNLESS OTHERWISE NOTED.
2. ALL DIMENSIONS ARE IN METRES (m) UNLESS OTHERWISE NOTED.
3. GROUND SURFACE ELEVATIONS SHOWN ON THE DRAWINGS SHOULD BE CONSIDERED APPROXIMATE AND CONFIRMED IN THE FIELD.
4. ELEVATION ARE IN METRES REFERRED TO GEODETIC DATUM.
5. DIMENSIONS SHOULD NOT BE SCALED FROM THE DRAWINGS ONLY WRITTEN DIMENSIONS AND ELEVATIONS SHOULD BE USED.
6. THE DRAWINGS SHALL BE READ IN CONJUNCTION WITH THE MOST RECENT REVISION OF THE SPECIFICATIONS (GEOTECHNICAL INVESTIGATION INFORMATION IS ATTACHED TO THEM).

REFERENCES:

1. BASE DATA - MNR NRVIS, OBTAINED 2004, CANMAP V2008.4
2. SITE FEATURES - PROVIDED BY GOWEST, NOVEMBER 2009 (SITE FEATURES.DXF)
3. PROJECTION: TRANSVERSE MERCATOR, DATUM: NAD 83, COORDINATE SYSTEM: UTM ZONE 17N.

Rev.	YYYY-MM-DD	DESCRIPTION	PREPARED	DESIGN	REVIEW	APPROVED
A	2015-03-06	DRAFT	MF	IM	KH	KAB

SEAL

NOT FOR CONSTRUCTION

DRAFT

CLIENT
GOWEST GOLD LTD.
BRADSHAW PROJECT
TIMMINS, ONTARIO

CONSULTANT
Golder Associates
PROJECT OFFICE
ADDRESS
LOCATION
COUNTRY
[+#] (#) ### ####
www.golder.com

PROJECT

BRADSHAW PROJECT

TITLE
COVER PAGE

PROJECT No.	PHASE	Rev.	of	DRAWING
09-1118-6010	14000	A		000

1760 Regent St.
Sudbury, Ontario
P3E 3Z8
Telephone: 1-705-566-6891
Fax: 1-705-566-5589

DESIGN CALCULATIONS

PROJECT NUMBER:	**169514568**
TITLE:	**BRADSHAW PRE-FEASIBILITY PROJECT**
CLIENT:	**GOWEST GOLD LIMITED**
LOCATION:	**ONTARIO**
DOCUMENT No:	**CAL_WATER_BALANCE**
SUBTITLE:	**WATER BALANCE**
REVISION:	**B – FINAL PRELIMINARY CALCULATION**
DEPARTMENT:	**MECHANICAL ENGINEERING**
BY:	**KIMBERLY NADON**
DATE:	**MARCH 10, 2015**
PROJECT MGR:	**NORIS DEL BEL BELLUZ**
CHECKED BY:	**JESSICA KERANEN / RON FLEMING**

MECHANICAL CHECK	**YES (X)**	**NO ( )**
INDEPENDENT REVIEW	**YES ( )**	**NO ( )**
COMPARATIVE REVIEW	**YES ( )**	**NO ( )**

COMMENTS

These calculations provide estimates for process water supply and for mine dewatering for the Bradshaw Pre-Feasibility study. The calculation includes:

- **Water balance calculation.**
- **Sizing of Typical Level Sump, pump and discharge line.**
- **Sizing of 40 Level, 130 Level, 220 Level and 310 Level Sumps, pumps and discharge lines.**
- **Sizing of Surface Settling Pond.**

Client: Gowest Gold Limited	Rev.: B	Date: 10-Mar-15
Project: 169514568	Made By: KAN	Date: 20-Feb-15
Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15
Title: Process Water and Dewatering Requirements		

Notes: 1) The purpose of this spreadsheet is to estimate the underground process water consumption rate on average and at peak, and to establish the expected total water reporting to the Level Sumps and Surface Settling Pond.
2) Peak flow is used for sizing delivery lines.
3) Average flow is used for calculating total process water delivery requirements and for calculating mine dewatering requirements.

References: 1) Technical Memorandum - Groundwater Inflow Summary - Bradshaw Gold Project, Timmins, Ontario by Golder Associates.

Revisions: A) Initial Calculation

Legend: Input Data / Calculated Data

Process Water Consumption Calculation:

Assumptions:
1) Two production crews will be working simultaneously, outfitted with the following equipment:
- Two hoses (one per crew) for dust control
- Two Boart Longyear StopeMaster drills (one per crew);

2) Two development crews will be working simultaneously, outfitted with the following equipment:
- Two 2-boom jumbos (Drillmaster 200), for face drilling and cable bolting (one per crew); only one task will be perfomed at a time, either face drilling or cable bolting.
- Two stopers/jacklegs (one per crew).
- Two hoses and nozzles for dust control (one per crew).

3) Two diamond drill crews will be operating at one time.
4) Two construction crews are required to install infrastructure, outfitted with the following equipment:
- Two cement mixers (Aliva 252).
- Two dry shotcrete sprayers.
- Two stopers/jacklegs (one per crew).

5) Leakage is estimated at 10%.
6) Backfill decant is assumed to occur over 24 hours, at a rate of 10 US gpm (based on 34 US gpm per ton of ore, 400 tpd, at 63% dilution ratio).

Crew	Equipment Model	Operation	Unit Operating Consumption (US gpm)	No. of Units	Total Operating Consumption (US gpm)	Utilization (Per Shift)	Operating Hours per Shift	Cumulative Consumption per Shift (US gal)	Possible Consumption Scenario During 12 Hour Shift (US gpm)												Comments and Possible Operating Scenarios
									6:00	7:00	8:00	9:00	10:00	11:00	12:00	13:00	14:00	15:00	16:00	17:00	
Production Drilling Crews (2)	Jumbo (2 boom) - StopeMaster	Drilling	10	2	20	75%	6	5,400			20	20	20			20	20	20			Full shift production drilling available
	Hose and Nozzle	Dust Control	4	2	8	75%	8	2,880			8	8	8	8		8	8	8	8		LHD mucking and re-mucking
Diamond Drilling Crews (2)	Diamond Drill - BQ Rod	Water Flushing	8	2	16	50%	9	4,320			16	16	16	16	16	16	16	16			
Development Crews (2)	Jumbo (2 boom) - Drillmaster 200	Face Drilling	27.7	1	27.7	50%	8	6,648			27.7	27.7	27.7	27.7		27.7	27.7	27.7	27.7		Assumption 2 - Assume 2 headings available
		Cable Bolting	27.7	1	27.7	50%	8	6,648			27.7	27.7	27.7	27.7		27.7	27.7	27.7	27.7		Assumption 2 - Assume 2 headings available
	Hose and Nozzle	Dust Control	4	2	8	100%	8	3,840		8	8	8	8	8	8	8	8				LHD mucking and re-mucking
	Jackleg/Stoper Drills	Drilling	5	2	10	25%	4	600			10	10				10	10				
Roadways & Draw Points	Hose and Nozzle	Dust Control	4	1	4	100%	10	2,400		4	4	4	4	4	4	4	4	4	4		Hose nozzle cracked open, various locations
Construction Crews (2)	Jackleg/Stoper Drills	Drilling	5	2	10	80%	2	960			10				10						Bulkhead construction - wall pins, backfill lines
	Cement Mixer (Aliva 252)	Cement Mixing	1	2	2	75%	4	360			2	2	2	2							Assume 5 tonnes/shift, form work complete
		Cement Mixer Flushing	10	2	20	10%	1	120							20						Machine wash (immediately after use)
	Dry Shotcrete Sprayer	Wall Preparation	10	2	20	100%	1	1,200		20											Surface wetting for better adhesion
		Dust Control	4	2	8	15%	8	576			8	8	8	8		8	8	8	8		Dust Control
		Shotcrete Mixing	1	2	2	75%	8	720			2	2	2	2		2	2	2	2		Assume 10 tonnes/shift
		Sprayer Flushing	10	2	20	10%	2	240							20					20	Machine wash (immediately after use)
Misc.	Hose and Nozzle	End of Shift Gear Cleanup	10	1	10	100%	2	1,200										10	10		End of shift cleanup, sump cleaning, etc.
							Subtotal:	38,112 USgal	0	32	143	133	123	103	78	131	131	123	87	20	
							Leakage:	3,811 USgal	0	3	14	13	12	10	8	13	13	12	9	2	Assumption 5
							Totals:	41,923 USgal	0	35	158	147	136	114	86	145	145	136	96	22	
							Peak =	158 US gal													
							Average =	58 USgpm													

Dewatering Requirement:

Assumptions:
1) Long term daily groundwater inflow rate is 3,000 m³/day (Ref 1).
2) Peak daily groundwater inflow rate is 4,000 m³/day (calculated with 33% contingency).
3) Backfill decant is assumed to occur over a period of 24 hours, at a rate of 10 US gpm (based on 34 US gpm per ton of ore, 400 tpd, at 63% dilution ratio).

Operation	Maximum Expected 24-Hour Water Inflow to Sumps		Comments
Ground Water Inflow	4,000 m³/day	734 US gpm	Reference 1 - 3,000 m³/day plus 33% contingency for peak long term flow
Backfill Decant water	55 m³/day	10 US gpm	Assumption 6
Backfill Flush water	190 m³/day	35 US gpm	Assuming 2 pours per day, at 900 US gpm
Process Water	317 m³/day	58 US gpm	Dewatering of process water based on 24-hour average inflow rate.
Total	190 m³/h	837 US gpm	

	Client: Gowest Gold Limited	Rev.: B	Date: 04-Mar-15
Stantec	Project: 169514568	Made By: KAN	Date: 20-Feb-15
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	Title: Dewatering - Summary of Level Sumps and Pumps		

General Assumptions

1) All settling of solids will be done on surface.
2) A cascading system of level sumps located every third level will deliver all minewater and groundwater to a surface settling pond.
3) Short term flow rates of up to 14,200 m³/day from newly opened stopes or tunnels, however a long term peak flow rate of 3,000 m³/day will be used.
4) Dewatering pumps will be submersible slurry pumps.

Summary

The following sumps installations will be required for the dewatering system.

	Location(s)	Flowrate		Sump Size	Sump Length	TDH	Equiv. Pressure	Pump Power		Line Size
40 Level Sump	40L	300 m³/h	1,320 US gpm	432 m³	36 m	54 m	575 kPa	80 kW	107 hp	DN200
130 Level Sump	130L	270 m³/h	1,180 US gpm	336 m³	28 m	116 m	1,235 kPa	154 kW	207 hp	DN200
220 Level Sump	220L	270 m³/h	1,180 US gpm	240 m³	20 m	116 m	1,235 kPa	154 kW	207 hp	DN200
310 Level Sump	310L	300 m³/h	1,320 US gpm	432 m³	12 m	120 m	1,279 kPa	83 kW	111 hp	DN150
Typical Level Sump	400L, 490L	80 m³/h	310 US gpm	64 m³	8 m	157 m	1,679 kPa	62 kW	83 hp	DN100

Client: Gowest Gold Limited	Rev.: B	Date:	04-Mar-15
Project: 169514568	Made By: KAN	Date:	20-Feb-15
Subject: Water Balance	Checked By: JMK	Date:	09-Mar-15
Title: Dewatering - Typical Lower Level Sump and Pump Sizing Calculation			

Purpose: The purpose of this calculation is to size the Level Sump Pumps located on every third level below the 310 elevation (400 EL and 490 EL). All Level Sumps report to the next Level Sump above, up to the 310 Level Sump.

References: 1) Calculation: Process Water and Dewatering Requirements

Assumptions:
1) Minor losses estimated as 15% of total length
2) All ground water will be collected above 280 Level. As such, only process water, and backfill decant/flush water will report to Typical Lower Level Sumps.
3) Each lower level sump will be sized to collect water from all mining activities occuring simultaneously.
4) Development water has a specfic gravity of 1.09 (15% solids content)
5) Pipeline flows to be maintained below 3 m/s (10 ft/s)
6) Pump operating hours per day will be less than 16 hours per day.
7) Levels are located every 30 metres down to 490 metres from surface, and sumps are located in redundant remuck bays near the main ramp. Level sumps are located every third level.
8) Typical remuck bays are 4 metres wide.
9) Leakage is estimated at 0%.
10) Sump live height and length selected to provide between 2 and 3 hours of storage time.

Legend:
(shaded) Represents Input Data
XXXX Represents Calculated Data

Calculation
Sump Sizing:

	Metric		Imperial	
Sump Width:	4	metres	13.1	feet
Live Sump Depth:	2	metres	6.6	feet
Sump Length:	8	metres	26.2	feet
Live Sump Size:	64.0	m^3	2,260	ft^3

Pump Sizing - Pressure Due to Flow and Elevation (Hazen-Williams Formula):

	Metric		Imperial	
Minor Losses Estimated at:	15%	of Pipe Length	15%	of Pipe Length
Development Water S.G. (for 15% solids content)	1.09		1.09	
Development Water Density	1,089.8	kg/m^3		
Average Inflow (24 hrs)	23	m^3 / hr	103	gpm
Pump Operating Hours per Day	8.00	hour/day	8.00	hour/day
Required Total Pumping Flowrate	70	m^3 / hr	309	gpm
Number of Pumps Operating	1	pump	1	pump
Required Flowrate per Pump	80	m^3 / hr	310	gpm

Sump Fill Time:	2.7	hours
Sump Discharge Time:	1.1	hours
Full Cycle:	3.9	hours
Pump On/Off:	6.2	cycle/day

Stantec	Client: Gowest Gold Limited	Rev.: B	Date: 04-Mar-15		
	Project: 169514568	Made By: KAN	Date: 20-Feb-15		
	Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15		
	Title: Dewatering - Typical Lower Level Sump and Pump Sizing Calculation				

Typical Level Sump Pump

	Metric		Imperial		
Pipe Type	Steel - Schedule 80		Steel - Schedule 80		
Selected Pipe Diameter	DN100		4	inches	
Pipe Inner Diameter	9.718	cm	3.826	inches	
H-W "C" Factor	120	used steel	120	used steel	
Elevation Change Between Levels	90	metres	295	feet	
Ramp Grade	15%		15%		
Total Pipe Length Along Ramp	600	metres	1,968	feet	
Minor Loss	90	equiv. metres	295	equiv. feet	
Total Equivalent Length	690	metres	2,264	feet	
Design Flow Cap.	80	m^3 / hr	352	US gpm	
Design Flow Cap.	1,333	l / min			
Water speed	3.0	m / s	9.1	ft / s	
Calculated Friction Head:	67	m of head	220	ft of head	*(Equation 3)*
TDH for Each Section	157	m of head	515	ft of head	
	1,679	kPa	244	psi	*(Equation 6)*
Pump Motor Requires:	62	kW	83	hp	*(Equation 5)*
Pump Efficiency - Assume:	60%		60%		
Mechanical Efficiency - Assume:	100%		100%		
Class 300# Fittings Pressure Rating	8,335	kPa	1,209	psi	
6-inch Schedule 80 Pipe Pressure Rating	4,790	kPa	695	psi	
Estimated Hydrostatic Pressure	1,679	kPa	244	psi	

Equations:

Hazen & Williams Formula:

- Imperial equations:

$$h_f = 0.002083L\left(\frac{100Q}{C}\right)^{1.85}\left(\frac{1}{d^{4.8655}}\right)$$ Equation 1

$$p_{loss} = \rho gh = \left(1.94\frac{lb \cdot s^2}{ft^4}\right)\left(32.174\frac{ft}{s^2}\right)\left(\frac{ft^2}{144in^2}\right)h_f ft = 0.433h_f$$ Equation 2

- Convert to single metric equation:

$$h_f = \frac{162.03133LQ^{1.85}}{C^{1.85}d^{4.8655}}$$ Equation 3

Where:

h_f = friction loss (m of head)
p_{loss} = friction loss (kPa)
L = length of pipe (m)
Q = flowrate (cu.m per hour)
C = friction factor for Hazen-Williams
d = internal pipe diameter (inches)

Conversion from Head to Pressure:

$$P = \rho gh$$ Equation 6

Where:
P = Pressure (Pa)
h = head (m)
ρ = density of fluid (kg/m^3)
g = gravity (m/s^2)

Total Discharge Head Formula:

$$TDH = SH + FH$$ Equation 4

Where:

TDH = total discharge head (m)
SH = static head (m)
FH = friction head (m)

Pump Motor Requirements Formula:

Imperial

$$P = \frac{(TDH)(SG)(Q)}{3960(P.Eff.)(M.Eff.)}$$ Equation 5

Metric

$$P = \frac{(TDH)(SG)(Q)}{6128(P.Eff.)(M.Eff.)}$$ Equation 6

Where:

P = pump motor size (kW)
TDH = total discharge head (m)
SG = specific gravity of pumped fluid
Q = flowrate (cu.m per hour)
$P.Eff.$ = pump efficiency (%)
$M.Eff.$ = mechanical efficiency (%)
0.00272222 = conversion constant

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	Project: 169514568	Made By: KAN	Date: 20-Feb-15
	Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15
	Title: Dewatering - 310 Level Sump and Pump Sizing Calculation		

Purpose: The purpose of this calculation is to size the 310 Level Sump Pump. The water reporting to this level includes water inflow from the Typical Level Sumps below, as well as decant water from backfill operations.

References: 1) Calculation: Process Water and Dewatering Requirements

Assumptions:
1) Minor losses estimated as 15% of total length
2) Approximately 25% of groundwater inflow will be collected in this sump. Other inflows will include water inflow from lower level sumps and backfill decant water.
3) Development water has a specfic gravity of 1.09 (15% solids content)
4) Pipeline flows to be maintained below 3 m/s (10 ft/s)
5) Assume pumps operate up to 16 hours per day.
6) Levels are located every 30 metres down to 490 metres from surface, and sumps are located in redundant remuck bays near the main ramp.
7) Level sumps are located every third level.
8) Typical remuck bays are 4 metres wide.
9) Leakage is estimated at 0%.
10) Sump live height and length selected to provide between 2 and 3 hours of storage time.

Legend:
(shaded) Represents Input Data
XXXX Represents Calculated Data

Calculation

Sump Sizing:

	Metric		Imperial	
Sump Width:	4	metres	13.1	feet
Live Sump Depth:	3	metres	9.8	feet
Sump Length:	12	metres	39.4	feet
Live Sump Size:	144.0	m^3	5,085	ft^3

Pump Sizing - Pressure Due to Flow and Elevation (Hazen-Williams Formula):

	Metric		Imperial	
Minor Losses Estimated at:	15%	of Pipe Length	15%	of Pipe Length
Development Water S.G. (for 15% solids content)	1.09		1.09	
Development Water Density	1,089.8	kg/m^3		
Average Inflow (24 hrs)	67	m^3 / hr	297	gpm
Pump Operating Hours per Day	12.00	hour/day	12.00	hour/day
Required Total Pumping Flowrate	135	m^3 / hr	593	gpm
Number of Pumps Operating	1	pump	1	pump
Required Flowrate per Pump	140	m^3 / hr	600	gpm

Sump Fill Time:	2.1	hours
Sump Discharge Time:	2.0	hours
Full Cycle:	4.1	hours
Pump On/Off:	5.8	cycle/day

Stantec	Client: Gowest Gold Limited	Rev.: B		Date:	04-Mar-15
	Project: 169514568	Made By: KAN		Date:	20-Feb-15
	Subject: Water Balance	Checked By: JMK		Date:	09-Mar-15
	Title: Dewatering - 310 Level Sump and Pump Sizing Calculation				

Typical Level Sump Pump

	Metric		Imperial		
Pipe Type	Steel - Schedule 80		Steel - Schedule 80		
Selected Pipe Diameter	DN150		6	inches	
Pipe Inner Diameter	14.633	cm	5.761	inches	
H-W "C" Factor	120	used steel	120	used steel	
Elevation Change Between Levels	90	metres	295	feet	
Ramp Grade	15%		15%		
Total Pipe Length Along Ramp	600	metres	1,968	feet	
Minor Loss	90	equiv. metres	295	equiv. feet	
Total Equivalent Length	690	metres	2,264	feet	
Design Flow Cap.	140	m^3 / hr	616	US gpm	
Design Flow Cap.	2,333	l / min			
Water speed	2.3	m / s	7.0	ft / s	
Calculated Friction Head:	30	m of head	97	ft of head	*(Equation 3)*
TDH for Each Section	120	m of head	393	ft of head	
	1,279	kPa	186	psi	*(Equation 6)*
Pump Motor Requires:	83	kW	111	hp	*(Equation 5)*
Pump Efficiency - Assume:	60%		60%		
Mechanical Efficiency - Assume:	100%		100%		
Class 300# Fittings Pressure Rating	8,335	kPa	1,209	psi	
6-inch Schedule 80 Pipe Pressure Rating	4,790	kPa	695	psi	
Estimated Hydrostatic Pressure	1,279	kPa	186	psi	

<u>Equations:</u>

<u>Hazen & Williams Formula:</u>

- Imperial equations:

$$h_f = 0.002083L\left(\frac{100Q}{C}\right)^{1.85}\left(\frac{1}{d^{4.8655}}\right)$$ Equation 1

$$p_{loss} = \rho g h = \left(1.94\frac{lb \cdot s^2}{ft^4}\right)\left(32.174\frac{ft}{s^2}\right)\left(\frac{ft^2}{144in^2}\right)h_f ft = 0.433h_f$$ Equation 2

- Convert to single metric equation:

$$h_f = \frac{162.03133LQ^{1.85}}{C^{1.85}d^{4.8655}}$$ Equation 3

Where:
h_f = friction loss (m of head)
p_{loss} = friction loss (kPa)
L = length of pipe (m)
Q = flowrate (cu.m per hour)
C = friction factor for Hazen-Williams
d = internal pipe diameter (inches)

<u>Conversion from Head to Pressure:</u>

$$P = \rho g h$$ Equation 6

Where:
P = Pressure (Pa)
h = head (m)
ρ = density of fluid (kg/m^3)
g = gravity (m/s^2)

<u>Total Discharge Head Formula:</u>

$$TDH = SH + FH$$ Equation 4

Where:
TDH = total discharge head (m)
SH = static head (m)
FH = friction head (m)

<u>Pump Motor Requirements Formula:</u>

Imperial

$$P = \frac{(TDH)(SG)(Q)}{3960(P.Eff.)(M.Eff.)}$$ Equation 5

Metric

$$P = \frac{(TDH)(SG)(Q)}{6128(P.Eff.)(M.Eff.)}$$ Equation 6

Where:
P = pump motor size (kW)
TDH = total discharge head (m)
SG = specific gravity of pumped fluid
Q = flowrate (cu.m per hour)
$P.Eff.$ = pump efficiency (%)
$M.Eff.$ = mechanical efficiency (%)
0.00272222 = conversion constant

	Client: Gowest Gold Limited	Rev.: B	Date: 04-Mar-15
Stantec	Project: 169514568	Made By: KAN	Date: 20-Feb-15
	Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15
	Title: Dewatering - 220 Level Sump and Pump Sizing Calculation		

Purpose: The purpose of this calculation is to size the 220 Level Sump Pump. The water reporting to this level includes water inflow from the 310 Level Sump, as well as decant water from backfill operations.

References: 1) Calculation: Process Water and Dewatering Requirements

Assumptions:
1) Minor losses estimated as 15% of total length
2) Approximately 25% of groundwater inflow will be collected in this sump. Other inflows will include water inflow from lower level sumps and backfill decant water.
3) Development water has a specfic gravity of 1.09 (15% solids content)
4) Pipeline flows to be maintained below 3 m/s (10 ft/s)
5) Assume pumps operate up to 16 hours per day.
6) Levels are located every 30 metres down to 490 metres from surface, and sumps are located in redundant remuck bays near the main ramp.
7) Level sumps are located every third level.
8) Typical remuck bays are 4 metres wide.
9) Leakage is estimated at 0%.
10) Sump live height and length selected to provide between 2 and 3 hours of storage time.

Legend:
(shaded) Represents Input Data
XXXX Represents Calculated Data

Calculation

Sump Sizing:

	Metric		Imperial	
Sump Width:	4	metres	13.1	feet
Live Sump Depth:	3	metres	9.8	feet
Sump Length:	20	metres	65.6	feet
Live Sump Size:	240.0	m^3	8,476	ft^3

Pump Sizing - Pressure Due to Flow and Elevation (Hazen-Williams Formula):

	Metric		Imperial	
Minor Losses Estimated at:	15%	of Pipe Length	15%	of Pipe Length
Development Water S.G. (for 15% solids content)	1.09		1.09	
Development Water Density	1,089.8	kg/m^3		
Average Inflow (24 hrs)	111	m^3 / hr	490	gpm
Pump Operating Hours per Day	10.00	hour/day	10.00	hour/day
Required Total Pumping Flowrate	267	m^3 / hr	1,176	gpm
Number of Pumps Operating	1	pump	1	pump
Required Flowrate per Pump	270	m^3 / hr	1,180	gpm

Sump Fill Time:	2.2	hours
Sump Discharge Time:	1.5	hours
Full Cycle:	3.7	hours
Pump On/Off:	6.5	cycle/day

Stantec						
Client:	Gowest Gold Limited	Rev.:	B	Date:	04-Mar-15
Project:	169514568	Made By:	KAN	Date:	20-Feb-15
Subject:	Water Balance	Checked By:	JMK	Date:	09-Mar-15
Title:	Dewatering - 220 Level Sump and Pump Sizing Calculation				

Typical Level Sump Pump

	Metric		Imperial		
Pipe Type	Steel - Schedule 80		Steel - Schedule 80		
Selected Pipe Diameter	DN200		8	inches	
Pipe Inner Diameter	19.368	cm	7.625	inches	
H-W "C" Factor	120	used steel	120	used steel	
Elevation Change Between Levels	90	metres	295	feet	
Ramp Grade	15%		15%		
Total Pipe Length Along Ramp	600	metres	1,968	feet	
Minor Loss	90	equiv. metres	295	equiv. feet	
Total Equivalent Length	690	metres	2,264	feet	
Design Flow Cap.	270	m^3 / hr	1,189	US gpm	
Design Flow Cap.	4,500	l / min			
Water speed	2.5	m / s	7.8	ft / s	
Calculated Friction Head:	26	m of head	84	ft of head	*(Equation 3)*
TDH for Each Section	116	m of head	379	ft of head	
	1,235	kPa	179	psi	*(Equation 6)*
Pump Motor Requires:	154	kW	207	hp	*(Equation 5)*
Pump Efficiency - Assume:	60%		60%		
Mechanical Efficiency - Assume:	100%		100%		
Class 300# Fittings Pressure Rating	8,335	kPa	1,209	psi	
6-inch Schedule 80 Pipe Pressure Rating	4,790	kPa	695	psi	
Estimated Hydrostatic Pressure	1,235	kPa	179	psi	

Equations:

Hazen & Williams Formula:

- Imperial equations:

$$h_f = 0.002083L\left(\frac{100Q}{C}\right)^{1.85}\left(\frac{1}{d^{4.8655}}\right)$$ Equation 1

$$p_{loss} = \rho g h = \left(1.94\frac{lb\cdot s^2}{ft^4}\right)\left(32.174\frac{ft}{s^2}\right)\left(\frac{ft^2}{144in^2}\right)h_f ft = 0.433h_f$$ Equation 2

- Convert to single metric equation:

$$h_f = \frac{162.03133LQ^{1.85}}{C^{1.85}d^{4.8655}}$$ Equation 3

Where:
h_f = friction loss (m of head)
p_{loss} = friction loss (kPa)
L = length of pipe (m)
Q = flowrate (cu.m per hour)
C = friction factor for Hazen-Williams
d = internal pipe diameter (inches)

Conversion from Head to Pressure:

$$P = \rho g h$$ Equation 6

Where:
P = Pressure (Pa)
h = head (m)
ρ = density of fluid (kg/m^3)
g = gravity (m/s^2)

Total Discharge Head Formula:

$$TDH = SH + FH$$ Equation 4

Where:
TDH = total discharge head (m)
SH = static head (m)
FH = friction head (m)

Pump Motor Requirements Formula:

Imperial

$$P = \frac{(TDH)(SG)(Q)}{3960(P.Eff.)(M.Eff.)}$$ Equation 5

Metric

$$P = \frac{(TDH)(SG)(Q)}{6128(P.Eff.)(M.Eff.)}$$ Equation 6

Where:
P = pump motor size (kW)
TDH = total discharge head (m)
SG = specific gravity of pumped fluid
Q = flowrate (cu.m per hour)
$P.Eff.$ = pump efficiency (%)
$M.Eff.$ = mechanical efficiency (%)
0.00272222 = conversion constant

Stantec	Client: Gowest Gold Limited	Rev.:	B	Date:	04-Mar-15
	Project: 169514568	Made By:	KAN	Date:	20-Feb-15
	Subject: Water Balance	Checked By:	JMK	Date:	09-Mar-15
	Title: Dewatering - 130 Level Sump and Pump Sizing Calculation				

Purpose: The purpose of this calculation is to size the 130 Level Sump Pump. The water reporting to this level includes water inflow from the 220 Level Sump, as well as decant water from backfill operations.

References: 1) Calculation: Process Water and Dewatering Requirements

Assumptions:
1) Minor losses estimated as 15% of total length
2) Approximately 25% of groundwater inflow will be collected in this sump. Other inflows will include water inflow from lower level sumps and backfill decant water.
3) Development water has a specfic gravity of 1.09 (15% solids content)
4) Pipeline flows to be maintained below 3 m/s (10 ft/s)
5) Assume pumps operate up to 16 hours per day.
6) Levels are located every 30 metres down to 490 metres from surface, and sumps are located in redundant remuck bays near the main ramp.
7) Level sumps are located every third level.
8) Typical remuck bays are 4 metres wide.
9) Leakage is estimated at 0%.
10) Sump live height and length selected to provide between 2 and 3 hours of storage time.

Legend:
(shaded) Represents Input Data
XXXX Represents Calculated Data

Calculation

Sump Sizing:

	Metric		Imperial	
Sump Width:	4	metres	13.1	feet
Live Sump Depth:	3	metres	9.8	feet
Sump Length:	28	metres	91.9	feet
Live Sump Size:	336.0	m^3	11,866	ft^3

Pump Sizing - Pressure Due to Flow and Elevation (Hazen-Williams Formula):

	Metric		Imperial	
Minor Losses Estimated at:	15%	of Pipe Length	15%	of Pipe Length
Development Water S.G. (for 15% solids content)	1.09		1.09	
Development Water Density	1,089.8	kg/m^3		
Average Inflow (24 hrs)	155	m^3 / hr	683	gpm
Pump Operating Hours per Day	14.00	hour/day	14.00	hour/day
Required Total Pumping Flowrate	266	m^3 / hr	1,172	gpm
Number of Pumps Operating	1	pump	1	pump
Required Flowrate per Pump	270	m^3 / hr	1,180	gpm

Sump Fill Time:	2.2	hours
Sump Discharge Time:	2.9	hours
Full Cycle:	5.1	hours
Pump On/Off:	4.7	cycle/day

Client: Gowest Gold Limited		Rev.: B		Date:	04-Mar-15
Project: 169514568		Made By: KAN		Date:	20-Feb-15
Subject: Water Balance		Checked By: JMK		Date:	09-Mar-15
Title: Dewatering - 130 Level Sump and Pump Sizing Calculation					

Typical Level Sump Pump

	Metric		Imperial		
Pipe Type	Steel - Schedule 80		Steel - Schedule 80		
Selected Pipe Diameter	DN200		8	inches	
Pipe Inner Diameter	19.368	cm	7.625	inches	
H-W "C" Factor	120	used steel	120	used steel	
Elevation Change Between Levels	90	metres	295	feet	
Ramp Grade	15%		15%		
Total Pipe Length Along Ramp	600	metres	1,968	feet	
Minor Loss	90	equiv. metres	295	equiv. feet	
Total Equivalent Length	690	metres	2,264	feet	
Design Flow Cap.	270	m^3 / hr	1,189	US gpm	
Design Flow Cap.	4,500	l / min			
Water speed	2.5	m / s	7.8	ft / s	
Calculated Friction Head:	26	m of head	84	ft of head	*(Equation 3)*
TDH for Each Section	116	m of head	379	ft of head	
	1,235	kPa	179	psi	*(Equation 6)*
Pump Motor Requires:	154	kW	207	hp	*(Equation 5)*
Pump Efficiency - Assume:	60%		60%		
Mechanical Efficiency - Assume:	100%		100%		
Class 300# Fittings Pressure Rating	8,335	kPa	1,209	psi	
6-inch Schedule 80 Pipe Pressure Rating	4,790	kPa	695	psi	
Estimated Hydrostatic Pressure	1,235	kPa	179	psi	

Equations:

Hazen & Williams Formula:

- Imperial equations:

$$h_f = 0.002083L\left(\frac{100Q}{C}\right)^{1.85}\left(\frac{1}{d^{4.8655}}\right) \quad \text{Equation 1}$$

$$p_{loss} = \rho g h = \left(1.94\frac{lb\cdot s^2}{ft^4}\right)\left(32.174\frac{ft}{s^2}\right)\left(\frac{ft^2}{144in^2}\right)h_f ft = 0.433h_f \quad \text{Equation 2}$$

- Convert to single metric equation:

$$h_f = \frac{162.03133LQ^{1.85}}{C^{1.85}d^{4.8655}} \quad \text{Equation 3}$$

Where:
- h_f = friction loss (m of head)
- p_{loss} = friction loss (kPa)
- L = length of pipe (m)
- Q = flowrate (cu.m per hour)
- C = friction factor for Hazen-Williams
- d = internal pipe diameter (inches)

Conversion from Head to Pressure:

$$P = \rho g h \quad \text{Equation 6}$$

Where:
- P = Pressure (Pa)
- h = head (m)
- ρ = density of fluid (kg/m^3)
- g = gravity (m/s^2)

Total Discharge Head Formula:

$$TDH = SH + FH \quad \text{Equation 4}$$

Where:
- TDH = total discharge head (m)
- SH = static head (m)
- FH = friction head (m)

Pump Motor Requirements Formula:

Imperial

$$P = \frac{(TDH)(SG)(Q)}{3960(P.Eff.)(M.Eff.)} \quad \text{Equation 5}$$

Metric

$$P = \frac{(TDH)(SG)(Q)}{6128(P.Eff.)(M.Eff.)} \quad \text{Equation 6}$$

Where:
- P = pump motor size (kW)
- TDH = total discharge head (m)
- SG = specific gravity of pumped fluid
- Q = flowrate (cu.m per hour)
- $P.Eff.$ = pump efficiency (%)
- $M.Eff.$ = mechanical efficiency (%)
- 0.00272222 = conversion constant

Stantec	**Client:** Gowest Gold Limited	**Rev.:** B		**Date:**	04-Mar-15
	Project: 169514568	**Made By:** KAN		**Date:**	20-Feb-15
	Subject: Water Balance	**Checked By:** JMK		**Date:**	09-Mar-15
	Title: Dewatering - 40 Level Sump and Pump Sizing Calculation				

Purpose: The purpose of this calculation is to size the 40 Level Sump Pump. The water reporting to this level includes water inflow from the 130 Level Sump, as well as decant water from backfill operations.

References: 1) Calculation: Process Water and Dewatering Requirements

Assumptions:
1) Minor losses estimated as 15% of total length
2) Approximately 25% of groundwater inflow will be collected in this sump. Other inflows will include water inflow from lower level sumps and backfill decant water.
3) Development water has a specfic gravity of 1.09 (15% solids content)
4) Pipeline flows to be maintained below 3 m/s (10 ft/s)
5) Assume pumps operate up to 16 hours per day.
6) Levels are located every 30 metres down to 490 metres from surface, and sumps are located in redundant remuck bays near the main ramp.
7) Level sumps are located every third level.
8) Typical remuck bays are 4 metres wide.
9) Leakage is estimated at 0%.
10) Sump live height and length selected to provide between 2 and 3 hours of storage time.

Legend:
(shaded) Represents Input Data
XXXX Represents Calculated Data

Calculation

Sump Sizing:

	Metric		Imperial	
Sump Width:	4	metres	13.1	feet
Live Sump Depth:	3	metres	9.8	feet
Sump Length:	36	metres	118.1	feet
Live Sump Size:	432.0	m^3	15,256	ft^3

Pump Sizing - Pressure Due to Flow and Elevation (Hazen-Williams Formula):

	Metric		Imperial	
Minor Losses Estimated at:	15%	of Pipe Length	15%	of Pipe Length
Development Water S.G. (for 15% solids content)	1.09		1.09	
Development Water Density	1,089.8	kg/m^3		
Average Inflow (24 hrs)	199	m^3 / hr	877	gpm
Pump Operating Hours per Day	16.00	hour/day	16.00	hour/day
Required Total Pumping Flowrate	299	m^3 / hr	1,315	gpm
Number of Pumps Operating	1	pump	1	pumps
Required Flowrate per Pump	300	m^3 / hr	1,320	gpm

Sump Fill Time:	2.2	hours
Sump Discharge Time:	4.3	hours
Full Cycle:	6.5	hours
Pump On/Off:	3.7	cycle/day

	Client: Gowest Gold Limited	Rev.: B	Date: 04-Mar-15
Stantec	Project: 169514568	Made By: KAN	Date: 20-Feb-15
	Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15
	Title: Dewatering - 40 Level Sump and Pump Sizing Calculation		

Typical Level Sump Pump

	Metric		Imperial		
Pipe Type	Steel - Schedule 80		Steel - Schedule 80		
Selected Pipe Diameter	DN200		8	inches	
Pipe Inner Diameter	19.368	cm	7.625	inches	
H-W "C" Factor	120	used steel	120	used steel	
Elevation Change Between Levels	40	metres	131	feet	
Ramp Grade	15%		15%		
Total Pipe Length Along Ramp	267	metres	875	feet	
Minor Loss	40	equiv. metres	131	equiv. feet	
Total Equivalent Length	307	metres	1,006	feet	
Design Flow Cap.	300	m^3 / hr	1,321	US gpm	
Design Flow Cap.	5,000	l / min			
Water speed	2.8	m / s	8.6	ft / s	
Calculated Friction Head:	14	m of head	45	ft of head	*(Equation 3)*
TDH for Each Section	54	m of head	177	ft of head	
	575	kPa	83	psi	*(Equation 6)*
Pump Motor Requires:	80	kW	107	hp	*(Equation 5)*
Pump Efficiency - Assume:	60%		60%		
Mechanical Efficiency - Assume:	100%		100%		
Class 300# Fittings Pressure Rating	8,335	kPa	1,209	psi	
6-inch Schedule 80 Pipe Pressure Rating	4,790	kPa	695	psi	
Estimated Hydrostatic Pressure	575	kPa	83	psi	

Equations:

Hazen & Williams Formula:

- Imperial equations:

$$h_f = 0.002083L\left(\frac{100Q}{C}\right)^{1.85}\left(\frac{1}{d^{4.8655}}\right) \quad \text{Equation 1}$$

$$p_{loss} = \rho g h = \left(1.94\frac{lb \cdot s^2}{ft^4}\right)\left(32.174\frac{ft}{s^2}\right)\left(\frac{ft^2}{144in^2}\right)h_f ft = 0.433h_f \quad \text{Equation 2}$$

- Convert to single metric equation:

$$h_f = \frac{162.03133LQ^{1.85}}{C^{1.85}d^{4.8655}} \quad \text{Equation 3}$$

Where:
- h_f = friction loss (m of head)
- p_{loss} = friction loss (kPa)
- L = length of pipe (m)
- Q = flowrate (cu.m per hour)
- C = friction factor for Hazen-Williams
- d = internal pipe diameter (inches)

Conversion from Head to Pressure:

$$P = \rho g h \quad \text{Equation 6}$$

Where:
- P = Pressure (Pa)
- h = head (m)
- ρ = density of fluid (kg/m^3)
- g = gravity (m/s^2)

Total Discharge Head Formula:

$$TDH = SH + FH \quad \text{Equation 4}$$

Where:
- TDH = total discharge head (m)
- SH = static head (m)
- FH = friction head (m)

Pump Motor Requirements Formula:

Imperial

$$P = \frac{(TDH)(SG)(Q)}{3960(P.Eff.)(M.Eff.)} \quad \text{Equation 5}$$

Metric

$$P = \frac{(TDH)(SG)(Q)}{6128(P.Eff.)(M.Eff.)} \quad \text{Equation 6}$$

Where:
- P = pump motor size (kW)
- TDH = total discharge head (m)
- SG = specific gravity of pumped fluid
- Q = flowrate (cu.m per hour)
- $P.Eff.$ = pump efficiency (%)
- $M.Eff.$ = mechanical efficiency (%)
- 0.00272222 = conversion constant

	Client: Gowest Gold Limited	Rev.: B	Date: 04-Mar-15
Stantec	Project: 169514568	Made By: KAN	Date: 20-Feb-15
	Subject: Water Balance	Checked By: JMK	Date: 09-Mar-15
	Title: Dewatering - Surface Settling Pond Area Sizing Calculation		

Purpose: This calculation is used to determine the minimum settling area required for settling slimes for theSurface Settling Pond. These slimes are generated from mine production and development activities and will be delivered to the Surface Settling Pond via a cascading pumping system from Level Sumps.

Minimum Pond Size for Settling Slimes

References:
1) Chemistry of Water Treatment, 2nd Edition
2) AWWA Water Treatment Plant Design, 3rd Edition

Assumptions:
1) The minewater delivered to the sump is relatively clean due to excessive quantity of inflow from groundwater.
2) The temperature of the water is ambient.

Enter the **estimated** calculated 24-hour peak flow inflow from all sources to the entire mine:	**199**	*m^3 / hr*
Enter Contingency for Inflow:	**5**	*%*
Calculate the inflow rate to the Surface Settling Pond:	**209**	*m^3 / hr*

General Parameters

Sedimentation loading rate for process water without floc. :	**1.1**	*m^2 per m^3/hr inflow (Reference 1)*
Sedimentation loading rate for warm process water with floc. :	**0.1**	*m^2 per m^3/hr inflow (Reference 1)*
Sedimentation loading rate for turbidity removal :	**0.5 to 0.7**	*m^2 per m^3/hr inflow (Reference 2)*
Flow Rate through a Rectangular Horizontal Tank (min.) :	**0.6**	*m / min (Reference 2)*
Flow Rate through a Rectangular Horizontal Tank (max.) :	**1.2**	*m / min (Reference 2)*
Maximum allowable slimes/sludge depth :	**1.2**	*m (For LHD bucket height)*
Minimum residence time for horizontal flow rectangular tanks :	**4.0**	*hours (Reference 1)*

Surface Area Calculation Based on Sedimentation Loading Rate

Enter assumed sedimentation loading rate :	**1.1**	*m^2 per m^3/hr inflow (sedimentation loading rate for water without flocculant)*
Enter Estimated Contingency for Surface Area:	**10.0**	*% (Allowance for variances of feed rates and constituents of minewater feed)*
Primary/Secondary Sump inflow rate :	**209**	*m^3 / hr*
Calculate Surface area requirement :	**251**	*m^2*

THEREFORE ASSUME THE REQUIRED AREA OF THE SURFACE SETTLING POND TO BE: **260** ***METRES²***

APPENDIX I
BULK SAMPLE

Gowest

Gowest Bulk Sample Options

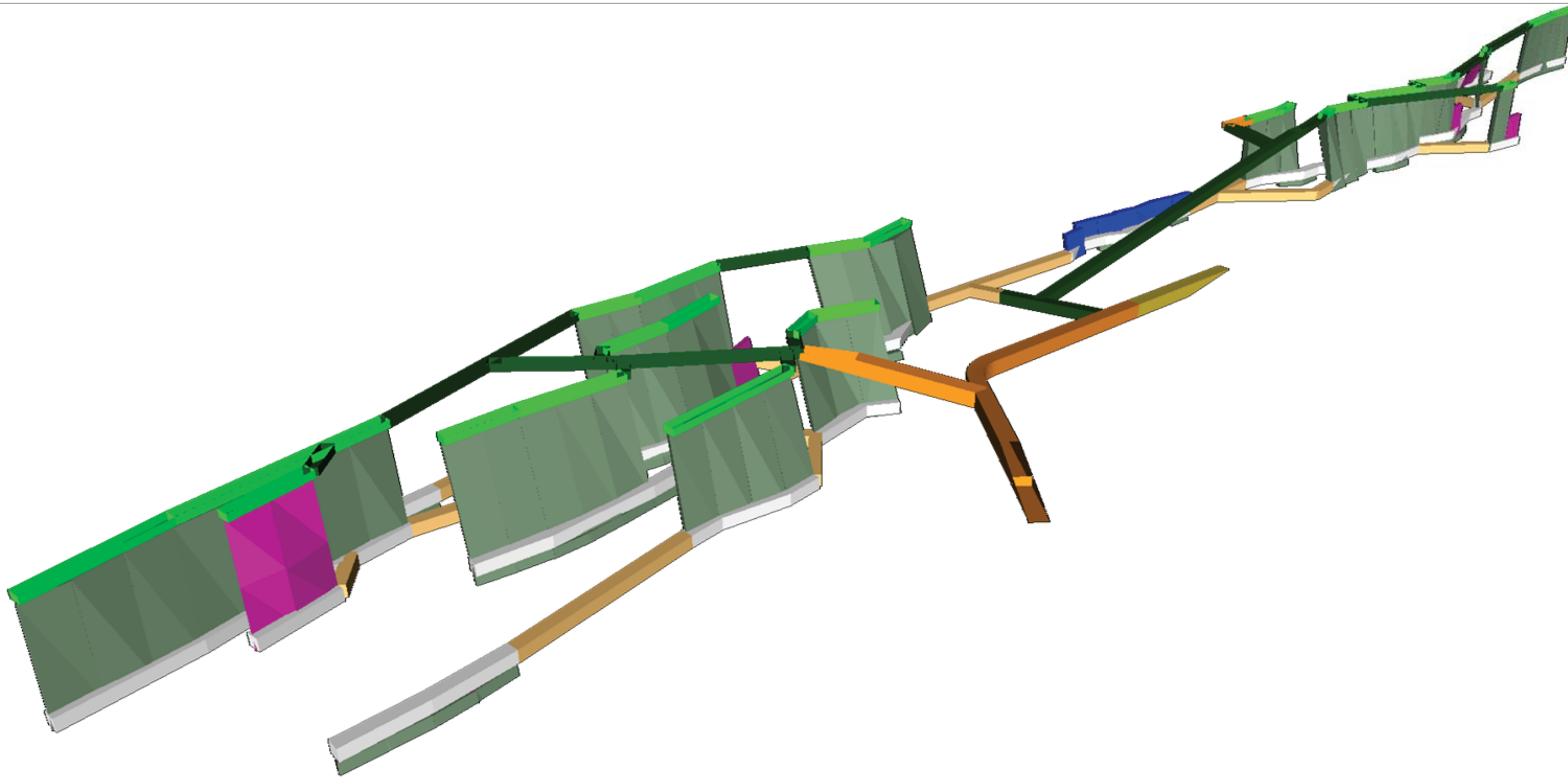
March 23, 2015

Reasons for an Underground Bulk Sample

- Confirmation of ore distribution and grades
- A larger sample than drill core for metallurgical testing
- Confirm geotechnical assumptions
- Confirm mine recovery and dilution factors for the bulk mining method
- Confirm ground water assumptions
- Provides the beginning of an underground exploration program

Bulk Sample Option 1 – 30,000 Tonnes

This option allows for sampling zones MZ1, MZ2, HW1, HW2, HW3, HW4 along the entire 45 Level (1st mining level, top sill or top cut)



Option 1

Pros

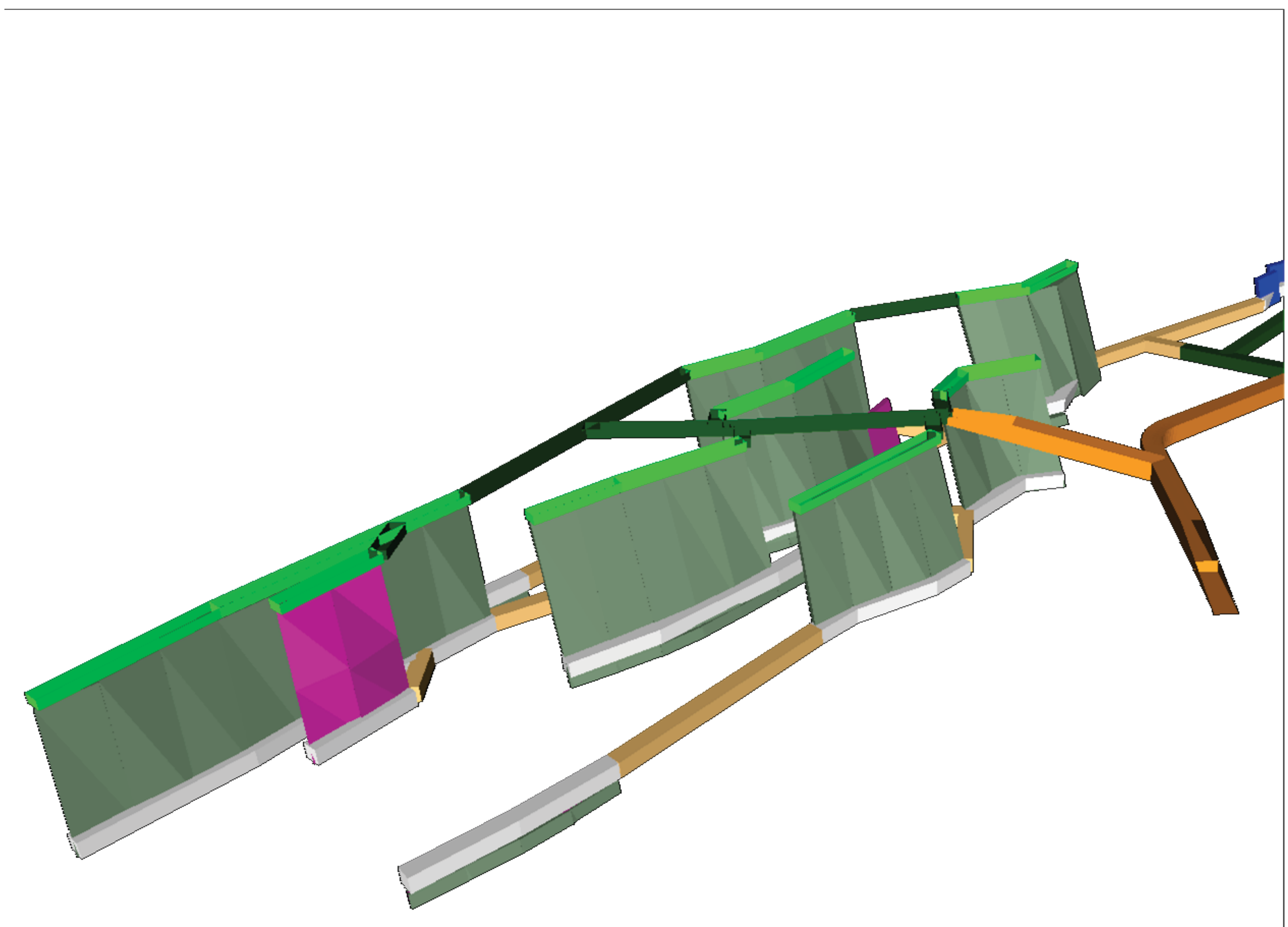
- Large 30,000 tonne bulk sample
- All mineralized zones on 45 level are sampled
- The recovered gold offsets some bulk sampling costs
- Some geotechnical and ground water data is gathered

Cons

- Only the grade of a 4mx4m x-sectional area is confirmed
- Highest cost to Gowest after Au credits
- Longer project period (11 mths)
- No gathering of mining method data

Bulk Sample Option 2 – 15,000 Tonnes

This option allows for sampling zones MZ2, HW1, HW2, HW3, HW4 only on the east side of 45 level



Option 2

Pros

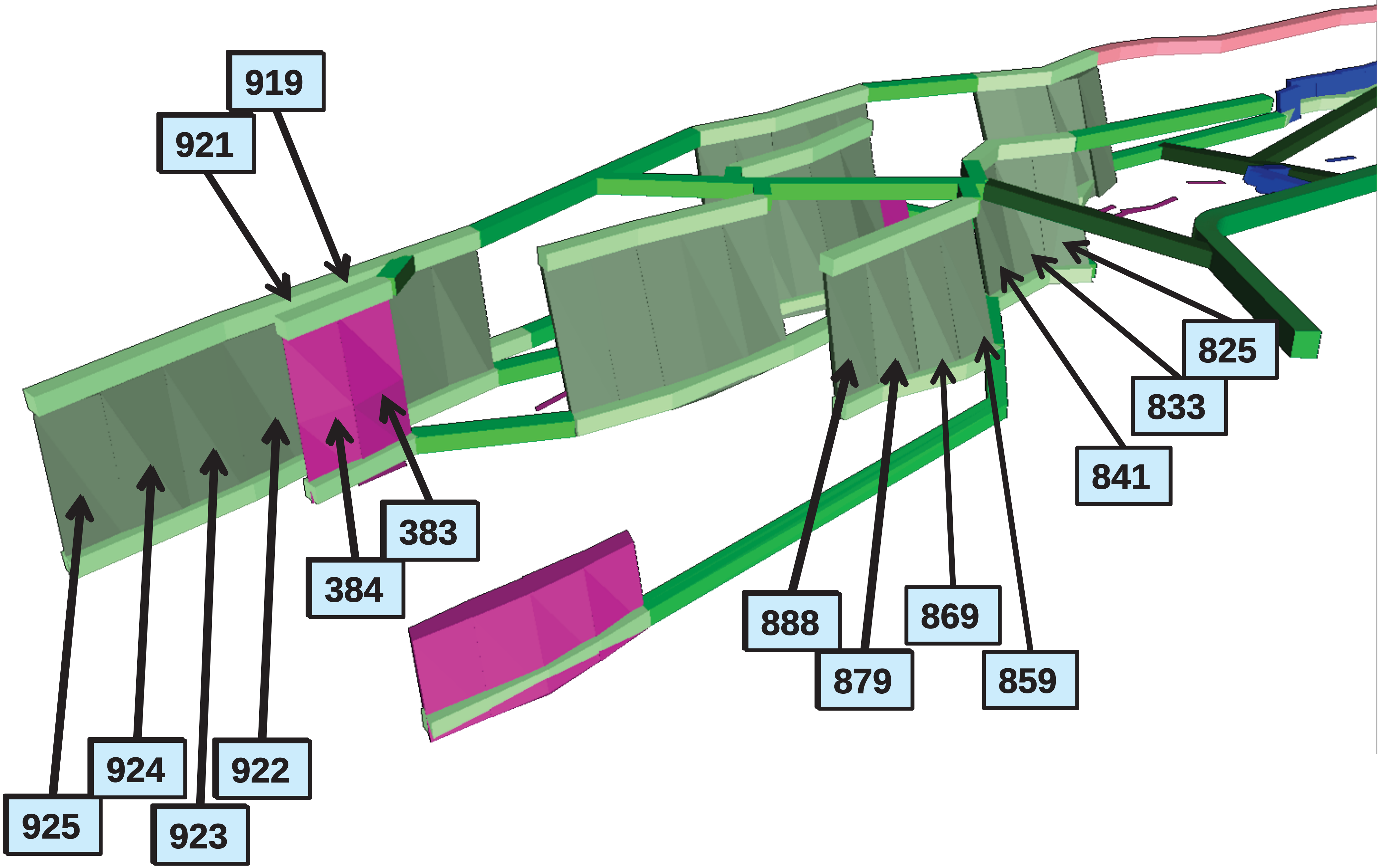
- Smaller 15,000 tonne bulk sample
- The recovered gold offsets some bulk sampling costs
- Some geotechnical and ground water data is gathered
- Lower cost to Gowest than option 1
- Shortest project period (9 mths)

Cons

- Only the grade of a 4mx4m x-sectional area is confirmed
- Lower amounts of recovered gold
- No gathering of mining method data
- Only mineralized zones on the east side of 45 level are sampled

Bulk Sample Option 3 – 30,000 Tonnes

This option allows for sampling zones MZ2, HW1, HW3, HW4 only on the east side of 45 level



Option 3

Pros

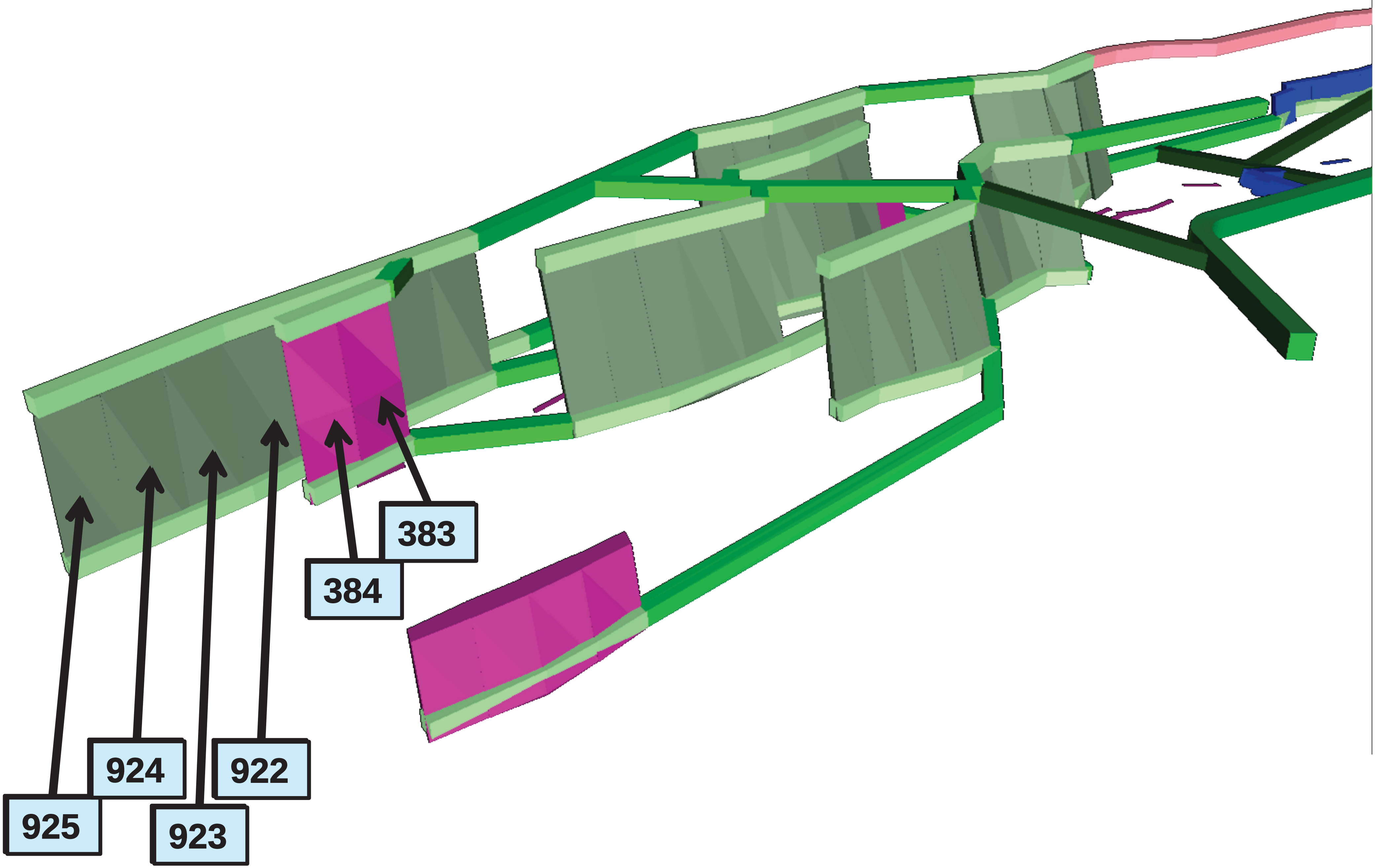
- Large bulk sample includes stope & development tonnes
- Better representative of grade along a 30m height
- Confirmation of geotechnical, ground water and bulk mining data
- Lowest cost to Gowest after Au credits

Cons

- Only mineralized zones on the east side of 45 level are sampled
- Longest project period (13 mths)
- Most expensive bulk sampling costs before Au credits

Bulk Sample Option 4 – 15,000 Tonnes

This option allows for sampling zones MZ2, HW1 on the east side of 45 level



Option 4

Pros

- Smaller bulk sample includes stope & development tonnes
- Better representative of grade along a 30m height
- Confirmation of geotechnical, ground water and bulk mining data
- Next lowest cost to Gowest after Au credits

Cons

- Less mineralized zones on the east side of 45 level are sampled
- Longer project period (12 mths)

Recommendations

The bulk sample decision should be based on geology and not on costs. However;

- The development & stoping options 3 & 4 provide the best scenarios for Au grade confirmation
- Option 2 has the shortest project period and the lowest bulk sampling costs prior to Au credits
- Option 3 has the longest project period but the lower cost to Gowest when Au credits are considered