TECHNICAL REPORT

ON THE

WINGDAM PROPERTY

Cariboo Mining District, British Columbia

Prepared for:

OMINECA MINING AND METALS INC.

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By

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1.0 SUMMARY

The Wingdam Property ("Wingdam" or the "Property") is made up of 467 hectares of placer title and 2,702 hectares of mineral title. The placer title encompasses the entire historic underground placer mine workings along Lightning Creek at Wingdam. Wingdam is located 36 linear kilometers east from the city of Quesnel in central British Columbia, Canada. The Property is directly accessible by driving 45 km or 35 minutes along Hwy 26 from Hwy 97 in North Quesnel.

CVG Mining Ltd. ("CVG") was actively exploring the auriferous Deep Lead Channel gravels at Wingdam for a 3 year period extending from 2009 to 2012. On October 16, 2013 Omineca Mining and Metals Ltd. ("Omineca" or the "Company") announced the successful completion of the acquisition of CVG Mining Ltd. through a reverse take-over ("RTO") transaction whereby Omineca acquired all issued and outstanding shares of CVG Mining Ltd including its 100% interest in the Wingdam Gold Project.

The Deep Lead Channel gravels are part of a reworked or modified fluvial paleochannel basement that pre-dates the last Pacific Cordillera glacial period called the Fraser Period (95,000 to 10,000 ybp). The channel occupies the deepest portion of the bedrock floor along the Lightning Creek valley that is buried from top to bottom by a sequence of postglacial alluvium, glacial till, and interglacial lacustrine sediments totalling 48.8 m thick.

Gold concentrations along the Deep Lead Channel basement are made up of native placer particles averaging 90.9% pure (909 fineness) and are efficiently recoverable by gravity separation methods that require no crushing, milling or leaching. The gold particles were liberated from lode sources at unknown bedrock locations surrounding the Wingdam area by a long period of deep Tertiary weathering. The liberated gold was transported and concentrated along the bedrock floor beneath the present location of Lightning Creek during the Pleistocene by a complex history of periodic interglacial streams. Stratigraphic evidence indicates that the auriferous gravel layer along the channel floor is a Sangamon (132,500-95,000 ybp) interglacial paleochannel remnant.

The Deep Lead Channel contains some of the highest placer gold concentrations historically reported in all of the Cariboo Mining District and perhaps throughout British Columbia that remain unmined. Parts of the channel were previously explored by drift methods and sampled along drilled fence lines during the 1910's, 30's and 60's. Compilation of the historic drill results indicates that various areas along the channel contain a gold-enriched zone with grades averaging 33.65 g/m^3 across a horizon that varies from 1.8 to 2.1 meters thick. This grade is equivalent to 13.74 g/tonne or 0.401 oz/ton. Historic and recent results from drilling and seismic surveying show that the channel floor width varies from 6 to 39 m wide and extends 2,430 m along the length of the Wingdam Property. The weighted average gold grade indicated by 8 fences of historical churn drill holes along the 2,400 m of channel controlled by the Company is 2.445 placer ounces per m^2.

Six attempts were made to mine the Deep Lead Channel during the 1930's and 60's by utilizing the *Australian deep-lead mining method*. The method involved raises or breakthrough locations that were accessed from a parallel bedrock drift driven along the length of the auriferous gravel layer occupying

the channel basement. The mining method proved to be unsuccessful through the unstable ground that directly overly the gold-enriched gravel layer. The unstable ground consists of water-saturated glaciolacustrine silt and sand layers (referred as *slum*) that easily cave and flow when undermined by a timber-supported raise.

CVG combined the *Australian deep-lead mining method* with a *ground-freeze* method in 2012 and successfully completed a breakthrough drift or crosscut (CC1) across the entire width (23.5 m) of the Deep Lead Channel. The 140 bank cubic meter sample extracted from Drift CC1 (2.44X2.44-meter) produced 173.5 ounces of raw placer gold (90.9% purity). The refined-equivalent gold grade across the channel width and throughout the 2.44-meter drift height averaged 34.55 g/m^3 (14.10 g/tonne or 0.411 oz/ton). The level of gold-enrichment identified across the channel exceeds historic grade estimates (33.65 g/m^3, 1.83-meter drift height).

Since acquiring control of the Wingdam project in late 2013, Omineca has focused on developing a viable, cost-effective approach to safely explore the Deep Lead Channel gravels based on the ground-freeze mining concept pioneered by CVG. The ability to physically sample and evaluate the Deep Lead Channel gravels using traditional exploration methods is limited. As the target gravels are located partially beneath Lightning Creek and the Barkerville Highway, and largely within a broad riparian wetland zone, access for drill testing is limited. In addition, the unstable nature of the overlying sediments makes it very difficult to complete drill holes and to recover meaningful samples. As well, the nuggety nature of the gold bearing strata does not lend itself to accurate correlation between drill hole intercepts. The depth of overburden (48.8m) and the location of the overlying creek does not make it feasible to test with trenching. Omineca believes that the only method to successfully evaluate and quantify the Deep Lead Channel gravels is with a bulk sample test.

In 2014, the Company systematically reviewed all historical work on the project available to it and submitted a Notice of Work to amend its permit to allow the Company to conduct additional drilling and geophysical work and test sample an initial 300m of the Wingdam paleo-channel. This amendment was granted in January 2015. The permit contemplates the production and processing of 20,000 cubic meters of gold bearing pay gravels per year.

In the spring of 2015 Omineca completed a field program of geotechnical drilling and seismic refraction geophysics. A 3D model of the gold bearing channel has been developed from this drill data and seismic interpretation. The model shows the potential mineable channel area approximately 60 percent greater than indicated by the historical channel profiles and also indicates a series of depressions along the channel which are postulated to be potential natural traps for the placer gold during deposition of the gravels. During this program the Company also recovered representative core samples which were analyzed to determine frozen material characteristics, the results of which have now been incorporated into a detailed ground-freezing plan.

Omineca has also thoroughly investigated other project parameters including power options, mining methods and equipment and ground water control strategies all culminating in the completion of a bulk sample plan. Based on this work, and with all required permits in place, Company management is confident that the Wingdam project development has reached a stage of readiness for the test sampling of an initial 300 meters of paleo-channel. The Company is proposing further work on the Wingdam

Property consisting of geotechnical analyses and engineering to finalize the parameters for underground freeze mining. The estimated cost for this work is $465,000.

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2.0 Introduction

The authors have been retained by Omineca Mining and Metals Inc. (“Omineca” or the Company”) to prepare a NI 43-101 compliant Technical Report for the Wingdam Property, which is located in the Cariboo Mining District in central British Columbia. This report will serve to update Omineca’s SEDAR disclosure record and was prepared under applicable NI 43-101 guidelines.

Omineca Mining and Metals Inc. is a publicly traded company listed on the TSX Venture Exchange (OMM.V). The Company was created in 2011 when Novagold Resources Ltd. acquired by way of a Plan of Arrangement all of the outstanding shares of Copper Canyon Resources Ltd. Under the terms of the arrangement, a spin out company was formed, Omineca Mining and Metals Inc. Two of Copper Canyon’s projects, Abo (Harrison) Gold and Kiwi, were transferred into Omineca and the other project, Copper Canyon, was transferred to Novagold.

On October 16, 2013 Omineca announced the successful completion of the acquisition of CVG Mining Ltd. through a reverse take-over (“RTO”) transaction. Shareholder approval for the arrangement was received at the company’s AGM on September 5th, 2013. The primary asset held by CVG was a 100% interest in the Wingdam Gold Project. Under terms of the Agreement, Omineca issued 47,471,548 common shares to CVG Mining Ltd. at a deemed price of $0.35/share, representing a purchase price of $16,615,041 and issued a $5,400,000 convertible debenture in connection with the assumption by Omineca of certain debt related to the property.

The main source of information contained in this report is from a report authored by Stephen Kocsis, P.Geo. titled Wingdam Property NI 43-101 Report; Deep Lead Channel Ground-Freeze Drift Sampling Program, dated October 02, 2012. This report was prepared for Omineca as part of the requirements for their acquisition of the Wingdam Project. Mr. Kocsis also supervised the entire phase of drifting, sampling and gold concentrate processing during the 2012 CVG bulk sample on the property.

One of the authors, Stephen Kenwood, P.Geo., visited the property on April 13, 2016 accompanied by Charles Downie, Director and President of Omineca. Given that none of the mineralization outcrops on surface and that underground access is not available, no representative or data verification samples were taken. Historical project infrastructure and numerous project drill sites were observed.

3.0 Reliance on Other Experts

The information contained in the Property Description and Location section, regarding property ownership and claim status was provided to the authors by Omineca. Mr. Kenwood has reviewed British Columbia’s Mineral Titles Online website and confirmed that CVG Mining Ltd., a wholly owned subsidiary of Omineca, is the registered owner of the placer and mineral claims as stated.

4.0 Property Description and Location

The Wingdam Property is located 35.7 km east from the city of Quesnel and is situated in the Cariboo Mining District, British Columbia, Canada (Figure 1). The Property is accessible from Quesnel by driving 45 km along Highway 26 a two-lane paved road that is open year-round. The Property is made up of and one mineral tenure block (Figure 2a) and two separate placer tenure groups (Figure 2b). All tenures are centrally located along Lightning Creek. The west placer group is located at Wingdam and the east group at Pinegrove. The central part of the Wingdam tenure group is at UTM NAD (83) Zone (10) coordinates 5877334N and 568979E.

The Property is made up of 11 placer claims, 2 placer leases, and 14 mineral claims (Tables 1a and 1b). The mineral claims occupy 2701.76 hectares (Figure 2a) and the placer tenures cover 467.21 hectares of land (Figures 2b). The area covered by the Property can be viewed on NTS map sheet number 093H4W or Trim base map number 093H001. Recent exploration on the property took place on Placer Lease Tenure 791222 which is part of the Wingdam tenure group.

All of the Wingdam placer and mineral tenures are held in the name of CVG Mining Ltd., a wholly-owned subsidiary of Omineca Mining and Metals Ltd.

The property currently has a reclamation bond in the amount of $25,000 in place related to British Columbia Ministry of Energy and Mines Permit # P-11-612. The author is not aware whether this bond is sufficient for any significant mining that may be contemplated by the Company.

In 2011, ground failure in one of the underground development headings caused rapid subsurface subsidence which created a sinkhole on surface between the Lightning Creek channel and the main highway. The area of the subsidence was reclaimed under the supervision of a Geotechnical Engineer and an Environmental company experienced in reclamation of stream banks. The site has been inspected by personnel from both the federal Department of Fisheries and Oceans and the provincial Ministry of Environment.

The author is not aware of any other environmental liabilities related to the Wingdam project.

In addition to British Columbia Ministry of Energy and Mines Permit #P-11-612, the Wingdam Project also has an Effluent Discharge Permit #11088 issued by British Columbia Ministry of Environment. There may be further permitting requirements as the project moves forward.

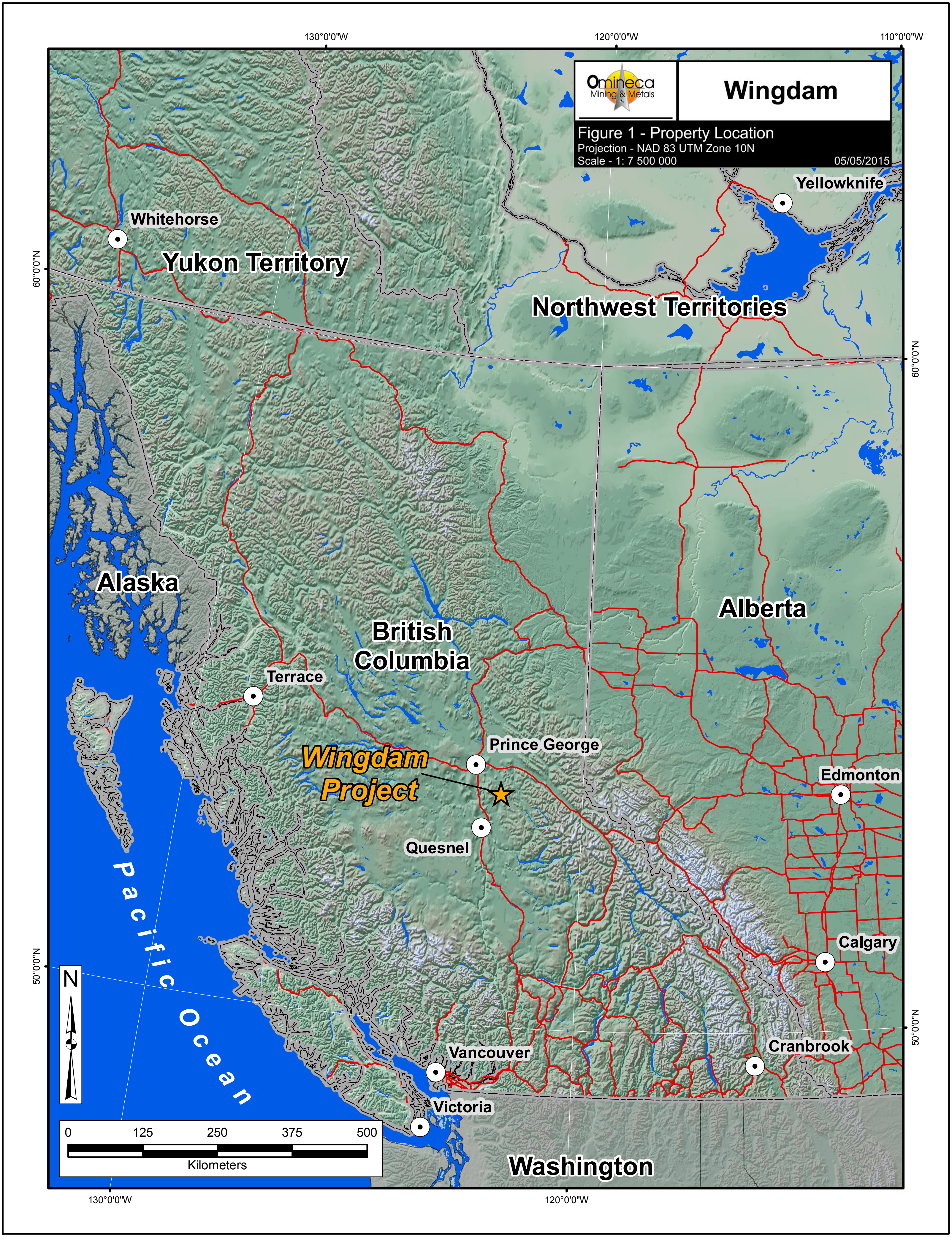
To the best knowledge of the author, there are no additional permits required to carry out any kind of production related work on the property, nor are there any significant factors and risks besides those noted that may affect access, title or the right or ability to perform work on the property.

Table 1a – Placer Tenure Summary

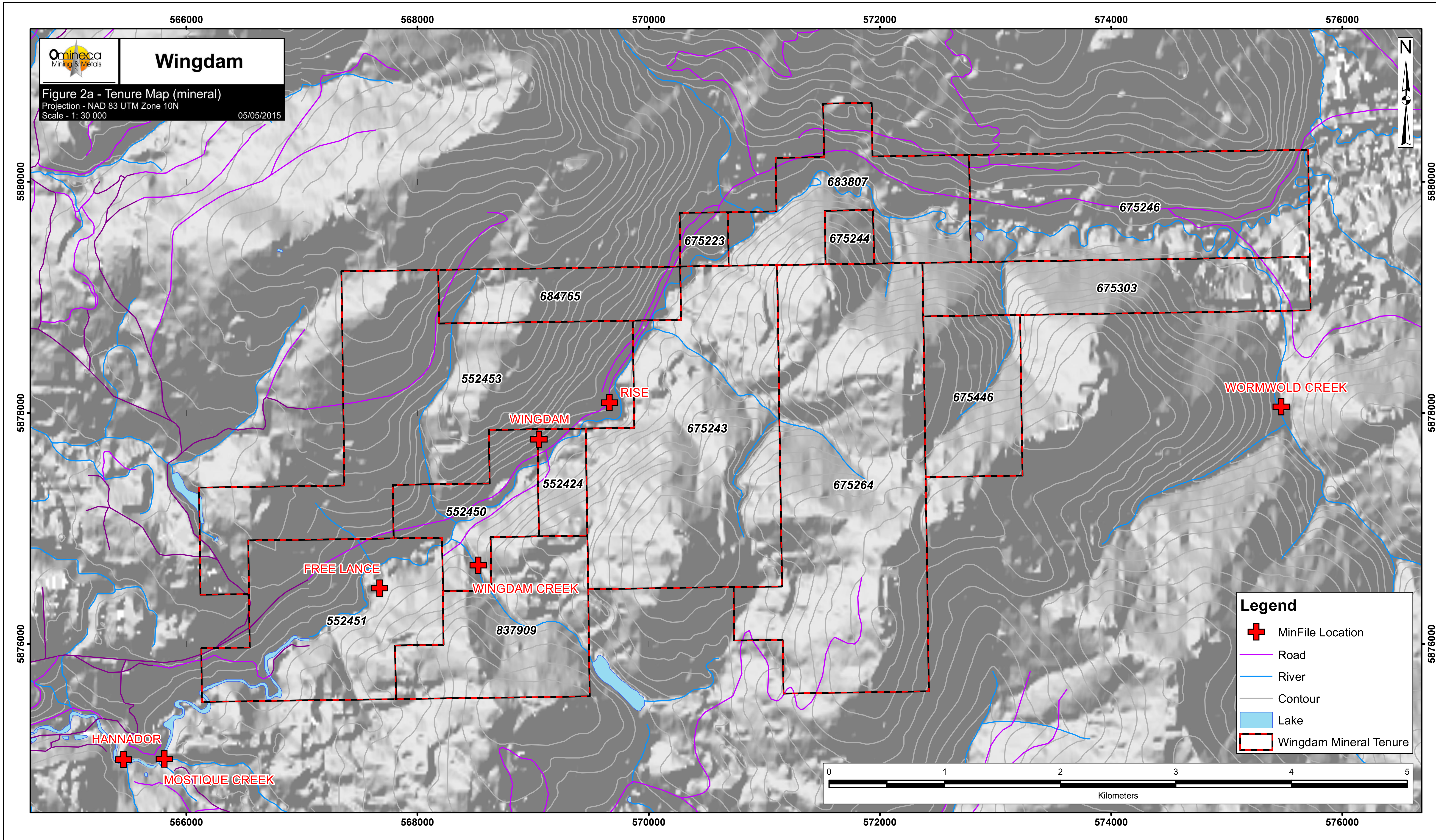
Tenure Number	Claim Name	Tenure Subtype	Map Number	Issue Date	Good to Date	Area (hectares)
564933	WINGDAM	Claim	093H	2007/aug/22	2018/may/10	19.44
579200	LIGHT 1	Claim	093H	2008/mar/26	2018/may/10	38.86
579203	LIGHT 2	Claim	093H	2008/mar/26	2018/may/10	38.86
579206	LIGHT 3	Claim	093H	2008/mar/26	2018/may/10	38.86
579207	LIGHT 4	Claim	093H	2008/mar/26	2018/may/10	19.43
600943	LIGHT 5	Claim	093H	2009/mar/12	2018/may/10	19.43
600944	LIGHT 6	Claim	093H	2009/mar/12	2018/may/10	19.43
600945	LIGHT 7	Claim	093H	2009/mar/12	2018/may/10	19.43
659603	SKI	Claim	093H	2009/oct/26	2018/may/10	19.43
659643	LIGHT 8	Claim	093H	2009/oct/26	2018/may/10	38.86
665744	WD 6	Claim	093H	2009/nov/06	2018/may/10	38.87
791222		Lease	093H	2010/jun/11	2017/july/11	49.62
791242		Lease	093H	2010/jun/11	2017/july/11	106.69
Total Area						**467.21**

Table 1b – Mineral Tenure Summary

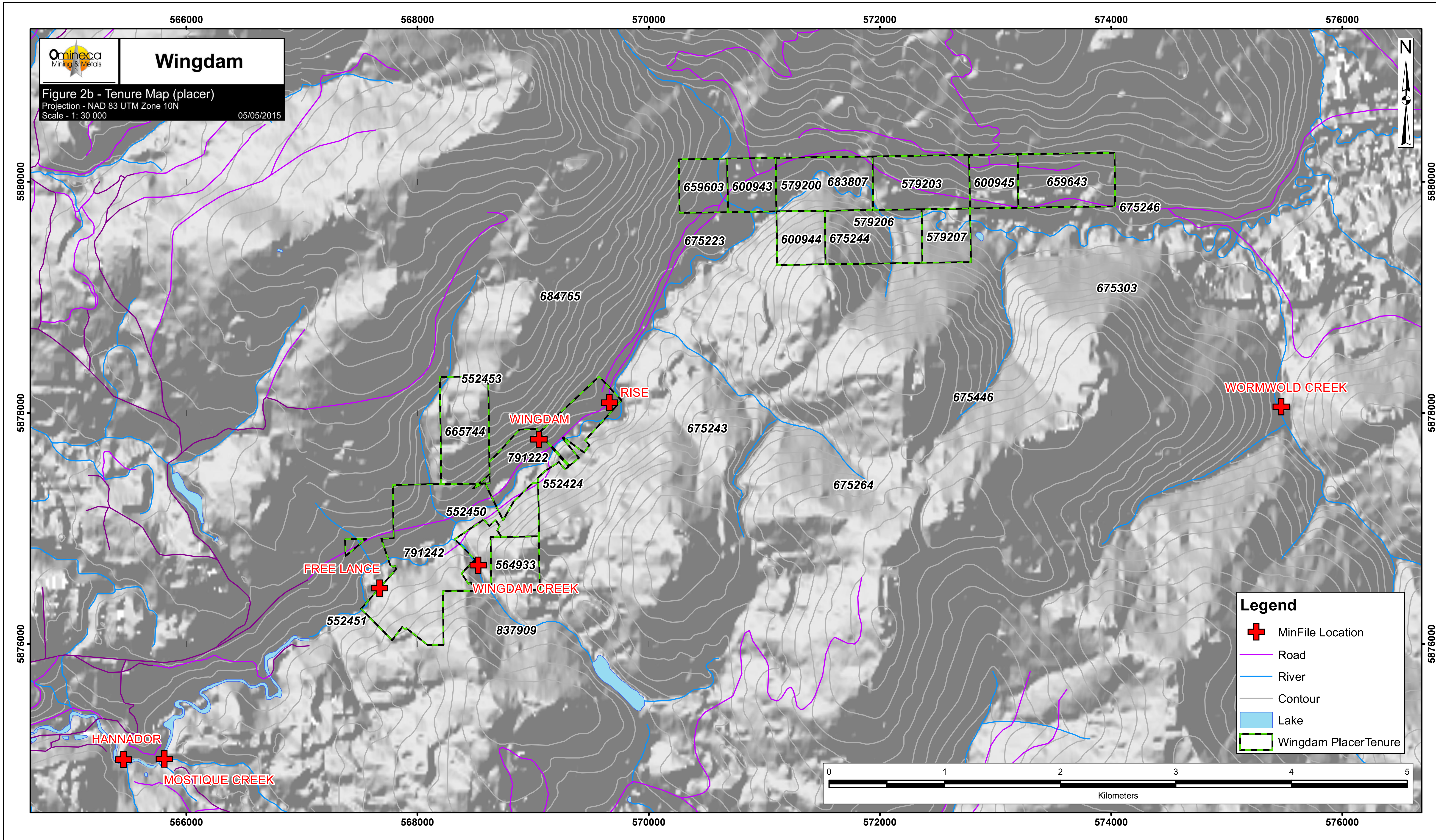
Tenure Number	Claim Name	Tenure Subtype	Map Number	Issue Date	Good to Date	Area (hectares)
552424	WINGDAM MINE	Claim	093H	2007/feb/20	2022/feb/19	38.88
552450	WD 2	Claim	093H	2007/feb/21	2022/feb/19	97.20
552451	WD 3	Claim	093H	2007/feb/21	2022/feb/19	233.33
552453	WD 4	Claim	093H	2007/feb/21	2022/feb/19	427.61
675223	Trailer Camp	Claim	093H	2009/nov/27	2022/feb/19	19.43
675243	WD-M	Claim	093H	2009/nov/27	2022/feb/19	388.76
675244	WD – M	Claim	093H	2009/nov/27	2022/feb/19	19.43
675246	LIGHTS ON	Claim	093H	2009/nov/27	2022/feb/19	272.02
675264	WD – M	Claim	093H	2009/nov/27	2022/feb/19	485.99
675303	WD –M	Claim	093H	2009/nov/27	2022/feb/19	155.46
675446	ULC	Claim	093H	2009/nov/27	2022/feb/19	116.62
683807	WD-M 5	Claim	093H	2009/dec/11	2022/feb/19	174.87
684765	WD-M 5	Claim	093H	2009/dec/14	2022/feb/19	97.16
837909	WD-M SE	Claim	093H	2009/nov/27	2022/feb/19	175.00
Total Area						**2701.76**



Wingdam
Figure 1 - Property Location
Projection - NAD 83 UTM Zone 10N
Scale - 1: 7 500 000
05/05/2015
Omineca Mining & Metals
130°0'0"W
120°0'0"W
110°0'0"W
60°0'0"N
50°0'0"N
Yellowknife
Whitehorse
Yukon Territory
Northwest Territories
Alaska
Alberta
British Columbia
Terrace
Prince George
Wingdam Project
Edmonton
Quesnel
Pacific Ocean
Calgary
Cranbrook
Vancouver
Victoria
Washington
N
0
125
250
375
500
Kilometers



Omineca Mining & Metals
Wingdam
Figure 2a - Tenure Map (mineral)
Projection - NAD 83 UTM Zone 10N
Scale - 1: 30 000
05/05/2015
566000
568000
570000
572000
574000
576000
5880000
5878000
5876000
683807
675246
675223
675244
684765
675303
552453
RISE
WINGDAM
675446
WORMWOLD CREEK
675243
552424
675264
552450
FREE LANCE
WINGDAM CREEK
552451
837909
HANNADOR
MOSTIQUE CREEK
N
Legend
MinFile Location
Road
River
Contour
Lake
Wingdam Mineral Tenure
0
1
2
3
4
5
Kilometers


Omineca Mining & Metals
Wingdam
Figure 2b - Tenure Map (placer)
Projection - NAD 83 UTM Zone 10N
Scale - 1: 30 000
05/05/2015
566000
568000
570000
572000
574000
576000
5880000
5878000
5876000
N
659603
600943
579200
683807
579203
600945
659643
675246
579206
675223
600944
675244
579207
684765
675303
552453
RISE
WINGDAM
665744
675243
675446
WORMWOLD CREEK
791222
552424
675264
552450
791242
564933
FREE LANCE
WINGDAM CREEK
552451
837909
HANNADOR
MOSTIQUE CREEK
Legend
MinFile Location
Road
River
Contour
Lake
Wingdam PlacerTenure
0
1
2
3
4
5
Kilometers

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The property is approximately 35.7 km east of the small city of Quesnel, which has a population of just over 20,000 and is accessible via Highway 26 that branches off Highway 97 adjacent to the Quesnel Airport. There are daily scheduled flights from Vancouver to Quesnel with flying time of about 1.5 hours. Gravel roads established during historic placer and lode gold mining activities provide access throughout the Property area.

The area is known as the “Interior Wet Belt” in British Columbia and is known as the Columbia Mountains and Highlands Ecoregion. The mean annual average temperature in the region is 5°C with a summer mean of 14°C and a winter mean of -5°C. Average annual precipitation is up to 48 inches; cumulated snow depths of up to 6 feet are common. The climate would allow for and underground operation to be run year-round. The Property is forested with second-growth spruce, pine and fir, along with alders and other deciduous varieties.

The ground elevation gradually rises in an easterly direction across the width of the highland area from 1,500m to over 2,000m above sea level. Valley dissection becomes more prominent in the same direction. Mount Watt (2,520 m) and Mount Perseus (2,548 m) are two of the highest points in the region.

Lightning Creek locally follows a southwesterly wandering pattern through a moderately narrow valley across the Wingdam Property with hillside peaks reaching 1,572m to the north and 1,332m to the south. The creek elevation varies from 945m to 907m above sea level along a 232-degree downstream heading and falls 38m across the total length (2,900m) of the Wingdam Placer Tenure Block. The tenure block covers 100% of the Lightning Creek valley bottom across a 2,160m distance.

Quesnel is the major supply and service center for resource industries in the area and there is access to a workforce in the region. The closest regional hospital is in Quesnel. There is sufficient water available in the region to support a mining operation. Single phase power is available on the Property although it would be insufficient for any underground development of the Property. The Company has sufficient surface rights in the Property area for mineral exploration and development, including the use of timber and water, as generally conveyed in some of the rights given with the Company’s Crown-granted mineral claims or with certain permitted rights, including tailings and waste disposal areas.

6.0 History and Previous Work

Table 2 - Wingdam Mining History

Time Period	Descriptions
1861	Ned Campbell discovered placer gold on Lightning Creek.
1878	Prospectors John Boyde and Angus McPhail discovered a northwest-striking quartz vein along Lightning Creek at a location 7 km upstream or east from Cold Spring

	House. The vein was named the Lightning Creek Ledge and later called the Free Lance Vein. The prospectors staked 1500 feet of ground under company name Cold Spring Company and staked an additional 1500 feet for John Fleming who owned the Cottonwood Company. The two companies amalgamated the ground and formed the Big Bonanza Company (BBC).
1879	BBC constructed a Wingdam across Lightning Creek and sunk a 24-meter deep shaft through the sediments along the valley floor. Unstable ground and high volumes of groundwater prevented the workings from reaching bedrock. The shaft operation was discontinued after Spring flooding washed away part of the Wingdam.
1896	Lightning Creek Gold Gravels and Drainage Company (LCGG) purchased the claims held by BBC. The purchasing company was a subsidiary of Great Cariboo Gold Company with a main office in New York.
1898	LCGG commenced to construct a proposed 2400-meter long drainage tunnel at Wingdam Hill for dewatering and exploration purposes.
1899	The LCGG drainage tunnel was abandoned after a 460-meters length was completed. John Fleming staked 64.7 hectares (160 acres) of ground on Wingdam Flat. The ground becomes Crown Grant Lot 446 in 1904.
1900-1902	LCGG sank the Jones Shaft through bedrock near the site of the present-day Melvin Shaft. Two horizontal drifts were driven from the shaft with attempts to break through and explore buried gravels at 30.5 and 42.7-meter depths. The shaft was flooded with water-saturated sediments immediately after breakthrough during each attempt.
1904-1905	New LCGG company management commenced Keystone drilling and identified significant gold grades along two buried gravel horizons now called the Deep Lead and Sanderson deposits; each situated 48.8 and 36.6 m respectively below Lightning Creek.
1906	LCGG sank the No. 1 shaft through 31.4 m of sediments and 28.0 m of underlying bedrock totalling 59.4 meters. A drift was driven horizontally through 33.2 m of bedrock from the 50.3-meter shaft level towards the Deep Lead Channel.
1907	LCGG continued development in the No. 1 shaft. A 30.5-meter long drive was extended through bedrock in a direction parallel to the Deep Lead Channel. The workings flooded during a breakthrough attempt into the channel.
1908	LCGG sank the No. 2 shaft to explore the Sanderson deposit.
1909-1912	No records.
1913	LCGG commenced a second bedrock drive at a location 3 m lower in the No. 1 shaft to access the Deep Lead Channel. There are no records of the results.
1914-1916	Period of limited intermittent work and no underground development.
1917-1919	A keystone drilling program was completed along fence lines now called Sections C, D, E, F (holes 1-8) and on Section BB (holes 9-15).
1920	The No. 3 shaft was constructed in gravels over keystone drill-hole location F-3. High volumes of water forced the workings to cease before the Deep Lead Channel was reached.
1921-1928	No available work records.

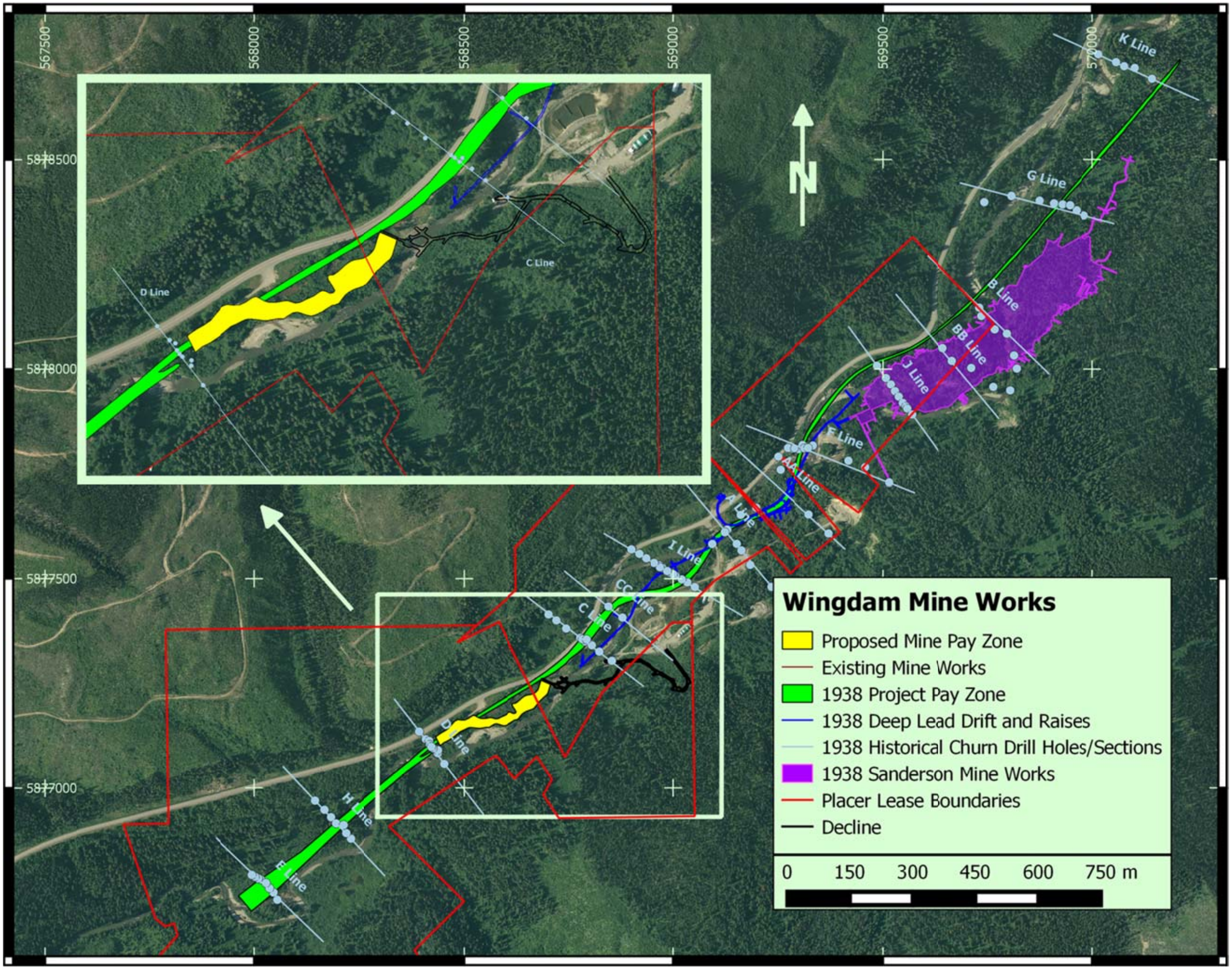
1929	Lightning Creek Gold Mines Ltd. acquired the Wingdam Property.
1930-1932	Consolidated Gold Alluvials of BC Ltd. (CGA) takes control of the Wingdam Property and neighboring placer claims up and down Lightning Creek along a 42 km distance. The company refurbished the Wingdam mine camp and plant. The No. 2 shaft was reconditioned and preparations were made to mine the 36.6-meter deep Sanderson deposit.
1933	The Sanderson Mine becomes the largest operation in the Cariboo District with 100 laborers employed. About 1 km of drifting was completed along the Sanderson deposit. The main drive through the deposit was reported to be in economic ground that extended 152 m upstream or east from the access shaft. The surface waters were naturally sealed off from the workings by an overlying layer of impermeable boulder clay or glacial lodgement till. CGA cased a 0.66-meter diameter hole into bedrock at a location 43.3 m south of the No. 1 shaft during the Fall season for the purpose of dewatering the Deep Lead channel. The lower 15-meter section of casing was perforated where it penetrated bedrock (3 m) and overlying layers of gravel, sand and silt totalling 12 m thick. The large quantities of sand and silt carried by the pumped slurry created a cavity and the area surrounding the shaft collapsed.
1934-1935	The Sanderson Mine continued to be a profitable operation. The workings measured over 300 m long in an upstream direction and 122 m wide. CGA management was replaced by British contactors who introduced the *Australian deep-lead mining method*. The Melvin Shaft was sunk through bedrock on the north side of Lightning Creek to 87 m to implement the mining method. From this point a bedrock drive was driven below and parallel to the Deep Lead Channel along a 460 m upstream and 460 m downstream direction.
1936	Operations continued at the Sanderson Mine and production for the year amounted to 36,528 m^3. CGA commenced a dewatering program by drilling 10 cm diameter drainage holes up into the Deep Lead Channel from locations along the Melvin bedrock drive.
1937	A second entry point or raise to the surface at the Sanderson Mine was constructed to facilitate higher production rates. The raise connected to the Melvin Shaft where the ore was hoisted to the surface. Production for the year amounted to 39,872 m^3. The total production (76,400 m^3) for the 1936-1937 time period yielded $464,300 in refined gold. This value is equivalent to 13,330 troy ounces by using the average gold price for the recorded time period ($34.83/oz). The first raise (No. 1 Upstream) was driven into the Deep Lead Channel after water pressures subsided in February. The up and downstream bedrock drives were extended to 979 meter total length. Two additional raises were driven into the channel. The gravels were described to be very rich; however caving and flooding in unstable ground limited the production to 555 m^3.
1938-1939	The Sanderson Mine remained in production during this time period and employed up to 90 laborers. Production for the time period amounted to 40,089 m^3. Operations discontinued at the Sanderson Mine on April 20, 1939. The up and downstream Deep Lead Channel drives were extended 68 m. The No. 1

	Downstream Raise was driven 41 m through the overlying channel gravels in a successful manner without incident. However, large volumes of water discharged from the Melvin washplant found its way from surface to the raise and flooded the workings. The water entered the raise via a drainage hole that connected the rock heading and the Melvin Shaft. The initial water flow caused the ground to break through to the creek and rapidly flood the entire Melvin workings. A concrete plug was placed in the raise that connected the Melvin shaft with the Sanderson workings to prevent floodwaters from reaching the neighboring mine operation.
1940-1961	There was no production at the Wingdam Property during this time period. The surface equipment was sold and the leases lapsed (1941-1944). Lightning Creek Gold Alluvials Ltd. acquired new leases in 1946. Wingdam & Lightning Creek Mining Company (WLCMC) acquired the property in 1961.
1961-1962	WLCMC commenced to dewater, clean out, and reopen the Melvin Shaft workings. The company reported that water from Lightning Creek continued to flow into the workings through the 1939 No. 1 Downstream Raise. The flow was stopped after a grout plug was successfully injected into the raise from the surface through a line of 8 drill holes. The sand and silt accumulation in the Melvin shaft station and sump was removed and two 125 hp pumps were installed.
1963	WLCMC cleaned out the Melvin up- and down-stream bedrock drives. A bulkhead was put in place to seal off the No. 1 Downstream Raise. The No. 2 and 3 Downstream raises were cleaned out and prepared for re-entry.
1964	A total of 154 oz of gold was recovered from 320 m^3 of gravel and bedrock extracted from the No.2 and 3 Downstream raises. The No. 4 Downstream Raise was driven into the Deep Lead Channel in September. This raise flooded and caved shortly after breakthrough into unstable ground and mining operations terminated.
1966-1967	Vigor Explorations optioned the Wingdam Property and other upstream and downstream placer titles from various claim owners.
1970	The historic buildings at Wingdam were removed and the Melvin shaft housing burnt down.
1974	Oriana Development Ltd. (ODL) optioned the Wingdam claims (leases 6685 and 6707) from B.G. Wilson of North Vancouver, BC. ODL completed a feasibility study that involved the consideration of open pit and underground mining methods. The company contracted Weir-Jones Engineering Consultants Ltd. to prepare a feasibility study involving ground-freeze methods across historic drill-section 'A' located near the No. 1 Shaft.
1975	Gold Channel Resources Ltd. optioned the Wingdam placer leases from ODL.
1976-1980	Bud Henning acquired the Wingdam placer leases and formed a company called Henning Mining and Milling Corporation (HMMC).
1981	Harvey Cohen Engineering Ltd. was contracted by HMMC to investigate the feasibility of dredging methods at Wingdam.
1986	Silver Ridge Resources (SRR) optioned the Wingdam leases (PL 747 and 743) from Bud Henning. Gold Ridge Resources Ltd. (GRR) optioned the Wingdam leases from SRR and took over as mine operator. Piteau Associates Engineering Ltd (PAEL) and Wright Engineering Ltd (WEL) conducted an exploration program involving four drill

	holes and the first phase of groundwater flow studies. WEL generated a mine feasibility report that included NI 43-101 non-compliant gold reserve estimates. PAEL completed the second phase of the hydrogeological study with the use of nine additional drill holes. Foundex Geophysics Inc. completed nine seismic refraction survey lines.
1987	PAEL performed a chloride-solution test to explore the possibilities of extracting placer gold concentrations along the Deep Lead Channel with in-situ leaching methods.
1989	GRR contracted Terry Garrow P.Eng. to outline a drift program including methods for extracting samples from the Deep Lead Channel.
1990-1992	GRR contracted Tonto Mining Contractors (TMC) to pursue underground mine developments for the purpose of sampling the Deep Lead Channel. The development work included site preparation, settling pond excavations, dewatering, and underground access to the channel. The channel was dewatered by pumping from the Melvin Shaft and new vertical holes drilled from surface and lined with perforated casing. The channel breakthrough location was accessed by driving a 520-meter long 180-degree-spiral decline to a main sump area and a 53-meter long incline drive. **Breakthrough Drive #1:** The incline and part of the decline flooded shortly after the channel was exposed by Breakthrough #1. GRR reported that the groundwater aquifer along the channel bedrock floor flowed into the breakthrough drive at the rate of 300 gpm. The water flow undercut the gravel, sand and silt layers and carried large volumes of sediments into the underground workings. The water-flow rate exceeded the capacity of the pumping station at the main sump. GRR discovered that the dewatering pumps were set too high. The dewatering process continued with success after the pumps were pulled, re-set and lowered to the proper subsurface elevation. The sediments were mucked throughout the entire workings and a new air fan with bagging was installed into an existing raise to improve ventilation. The initial breakthrough was sealed with Bulkhead #1. **Breakthrough Drive #2:** A second breakthrough into the channel was attempted at a location 14 m downstream or southwest from Drive #1. The drive broke through into a dry sand layer after advancing 3.7 m into bedrock. The sand layer and overlying sediments collapsed into the drive and all attempts to control the caving failed. The drive was sealed with Bulkhead #2.
1992	**GRR Pipe Drive:** GRR collared a 1.07-meter (42-inch) diameter horizontal pipe drive into the Deep Lead Channel at a location 14 m downstream from Drive #2. The pipe drive, totalling 19 m long, penetrated 7.2 meters of bedrock and 11.8 meters of alluvium across the channel. The drive was improperly collared at an elevation 0.46 meters above the channel bedrock floor where the majority of the gold was concentrated. The total amount of gold recovered from the drive was 13.9 grams. Most of this gold derived from a 2 m

	long section where the drive intersected rimrock along the channel. GRR accessed and sampled the bedrock floor by cutting through the pipe and recovered 54 grams of gold from a 0.60 m^2 area.
1993	GRR shut down operations at the Wingdam Property.
1998	The placer leases covering the Wingdam Property lapsed and the ground was staked by John Bot of Quesnel, BC.
2009-2012	CVG Mining Ltd. (CVG), formerly 1011136288 Saskatchewan Ltd., optioned and later purchased 100% of the Wingdam Property title from John Bot. The history of exploration work carried out on the Property by CVG and Omineca is given under the Exploration Section.

Figure 3 - Historical Mine Workings and Exploration Work



6.1 Historic Gold Production

The historic placer gold production reported to and recorded by the Cariboo District Gold Commissioner from areas along Lightning Creek at Wingdam amounts to 27,648 raw ounces (Holland, 1950). The results from nine historic fineness determinations indicate that the raw placer gold contains a fineness that ranges from 901 to 915 and averages 910.5. The gold was produced from two underground workings called the Sanderson and Melvin mines. The Sanderson Mine produced 25,474 ounces of gold from 136,753 loose cubic meters during a six year period extending from 1934 and 1939 (Table 3). Limited production at the Melvin Mine derived from six breakthrough attempts into the Deep Lead Channel during two time periods; 1937-38 and 1963-64 (Table 4). The total channel volume extracted from the breakthroughs amounted to 2,515.4 loose cubic meters and yielded 1,240 ounces of gold. The gold grades given in tables 3 and 4 represent loose volumes and refined-equivalent ounces.

Table 3 - Sanderson Mine Gold Production and Grades

Time Period	**Loose Volume (m^3)**	**Gold Recovered (oz)**	**Gold Grade**	
			(oz/m^3)	**(g/m^3)**
1934	1,681.3	312	0.186	5.77
1935	12,087.6	2,498	0.207	6.43
1936	32,695.5	6,800	0.208	6.47
1937	39,872.3	7,306	0.183	5.70
1938	40,165.9	6,627	0.165	5.13
1939	10,250.4	1,931	0.188	5.86
Total/Average	**136,753**	**25,474**	**0.186**	**5.79**

Table 4 - Melvin Mine Gold Production and Grades

Time Period	**Loose Volume (m^3)**	**Gold Recovered (oz)**	**Gold Grade**	
			(oz/m^3)	**(g/m^3)**
1937	1,393.0	667	0.479	14.89
1938	802.8	419	0.522	16.23
1963-64	319.6	154	0.482	14.99
Total/Average	**2,515.4**	**1,240**	**0.493**	**15.33**

The author has not been able to independently verify the methodology and results related to the historical production at the Sanderson and Melvin Mines. However, the author believes that the information is relevant.

6.2 WEL Historical Data Compilation (1986)

Wright Engineers Limited (WEL) was employed by Gold Ridge Resources Ltd. in 1986 to investigate past gold production and historic drill exploration results along the Deep Lead Channel (Leader, J. and Clarke, W. (1986): Preliminary Engineering Report on the Wingdam Placer Property).

WEL compiled drill results obtained from 15 churn holes and one 26-inch diameter rotary hole (Table 5). The holes were drilled along 7 fence lines (F, AA, A, I, CC, C and D) illustrated in Figure 4. The churn holes were drilled by Lightning Creek Gold Gravels and Drainage Company (LCG) in 1905 and during a period extending from 1915 to 1917. The rotary hole was drilled by Consolidated Gold Alluvials of BC Ltd. (CGA) in 1933. The drill data and gold grades presented in the CGA historical cross-sections is useful information, but little is known about the qualification and reliability of the person or persons who collected the samples and conducted the sample analyses.

Table 5 - Historical Drill Results

Drill Hole Number	Intersection (feet)	Width (feet)	Grade (ozs/yd)
F-5	6	22	1.314
F-8	6	19	0.252
A-5	6		5.633
26” J Well	6	50	0.079
I-5	9	45	0.122
I-12	6	30	0.206
CC-2	4	75	0.495
C-1 & C-9 (Average)	6	43	0.989
C-7	6	26	1.077
C-8	6	33	0.726
D-1	6	14	3.675
D-2	6	24	0.275
D-4	6	26	0.031
D-5	6	25	0.046
D-6	6	27	0.008

The historic drill-indicated gold grades are relatively comparable with the 2012 CVG drift grades identified across various parts of the Deep Lead Channel. The drift-indicated average gold grade at the CVG breakthrough location amounted to 34.55 g/m^3 across a 2.44-meter mine height. The average gold grade (33.65 g/m^3) used in the WEL calculations for a 1.83-meter mine height is equivalent to 25.24 g/m^3 for a diluted 2.44-meter mine height.

6.3 GRR Pipe Drive (1992)

Gold Ridge Resources Inc. (GRR) drove a 1.07-meter diameter pipe along a 19.1-meter horizontal distance through bedrock and the basal gravels of the Deep Lead Channel in 1992. The drive was orientated along a 294° direction that trends across the channel width. The drive penetrated 7.16 meters of bedrock (0-7.16 m) and intersected 11.86 meters of the channel (7.16-19.02 m).

GRR miscalculated the vertical location of the pipe drive entry point by about 0.91 meters. The horizontal location of the pipe was anywhere from 0.30 to 0.71 meters above the uneven channel bedrock floor where the gold-enrichment was primarily located. The sediments sampled along the pipe horizon were mainly fine-grained gravel, sand and silt layers that were barren or contained low concentrations of gold. The highest gold grades identified (2.50-8.45 g/m^3) were located along a 1.55-meter distance where the drive intersected rimrock along the south side of the channel (6.22-7.77 m).

GRR cut a 1.52-meter long by 0.76-meter wide hole along the base of the pipe to access and sample the bedrock at 12.65 meters along the drive. Bedrock was situated 0.55 m below the pipe drive at this location. The sample consisted of gravel (0.55 m) and about 0.15 meters of underlying bedrock totalling 0.70 meters thick. The area of bedrock sampled was reported to measure 1.16 m long and 0.52 m wide. A total of 54 grams of raw gold (900 fineness) was recovered from the sample). The total volume of the sample amounted to 0.422 bank cubic meters and the raw gold grade across the 0.70-meter thick horizon was 128.57 g/m^3 (Table 8).

Table 6 - GRR Pipe Drive Access Bedrock Sample

Sample	**Horizontal Distance** (m)		**Length** (m)	**Volume** (m^3)		**Gold** (g)	**Bank Grade** (g/m^3)
	Pipe Drive	Channel		Loose	Bank		
	12.83-13.99	5.64-6.80	1.16	n/a	0.422	54	128.57

The author has not been able to independently verify the methodology and results related to the historical GRR Pipe Drive sample. However, the author believes that the information is relevant

7.0 Geology

7.1 Bedrock Geology

The Wingdam Property is located along the western edge of the Omineca allochthonous superterrane Belt that makes up part of the Quesnel Highlands in central British Columbia. At a local scale, the eastern part of the property is underlain by older Hadrynian to Paleozoic Barkerville Terrane quartzites, phyllites, marbles, tuffs and a suite of intrusive rocks including diorite, rhyolite and rhyodacite. West of the Eureka Fault lie Upper Triassic phyllite, argillite, siltstone, limestone, quartzite, greenstone and tuff assigned to the Quesnel Terrane.

The rocks exposed along the Wingdam Mine portal and throughout the main decline consist of dark grey to black-colored siltite and phyllite belonging to the Harveys Ridge Succession. The section of bedrock exposed by the incline that extends to the Deep Lead Channel breakthrough point is made up of greyish tan-olive phyllite of the Ramos Succession. The bedrock across the channel floor changes back to dark grey siltite and phyllite layers of the Harveys Ridge Succession. The contact between the successions at this change is an unconformity controlled along a northeast-striking steeply-dipping fault that parallels the south channel rim. The fault forms a 3-meter wide vertical clay-filled gouge mainly composed of weathered felsic minerals and less quartz. Similar clay-filled gouges up to 15 cm wide have been identified in bedrock along underground drifts. The narrow gouges crosscut bedrock foliation strike along shallow angles and parallel dip. Bedrock foliation exposed along the underground workings strike northwest (308-degrees) and dip moderately to steeply towards the west.

7.2 Surficial Geology

The Lightning Creek valley at Wingdam is filled from top to bottom by a sequence of recent, postglacial, glacial and interglacial sediments. A detailed description of the Quaternary stratigraphy can be found in the 2012 Kocsis Wingdam Property NI 43-101Technical Report.

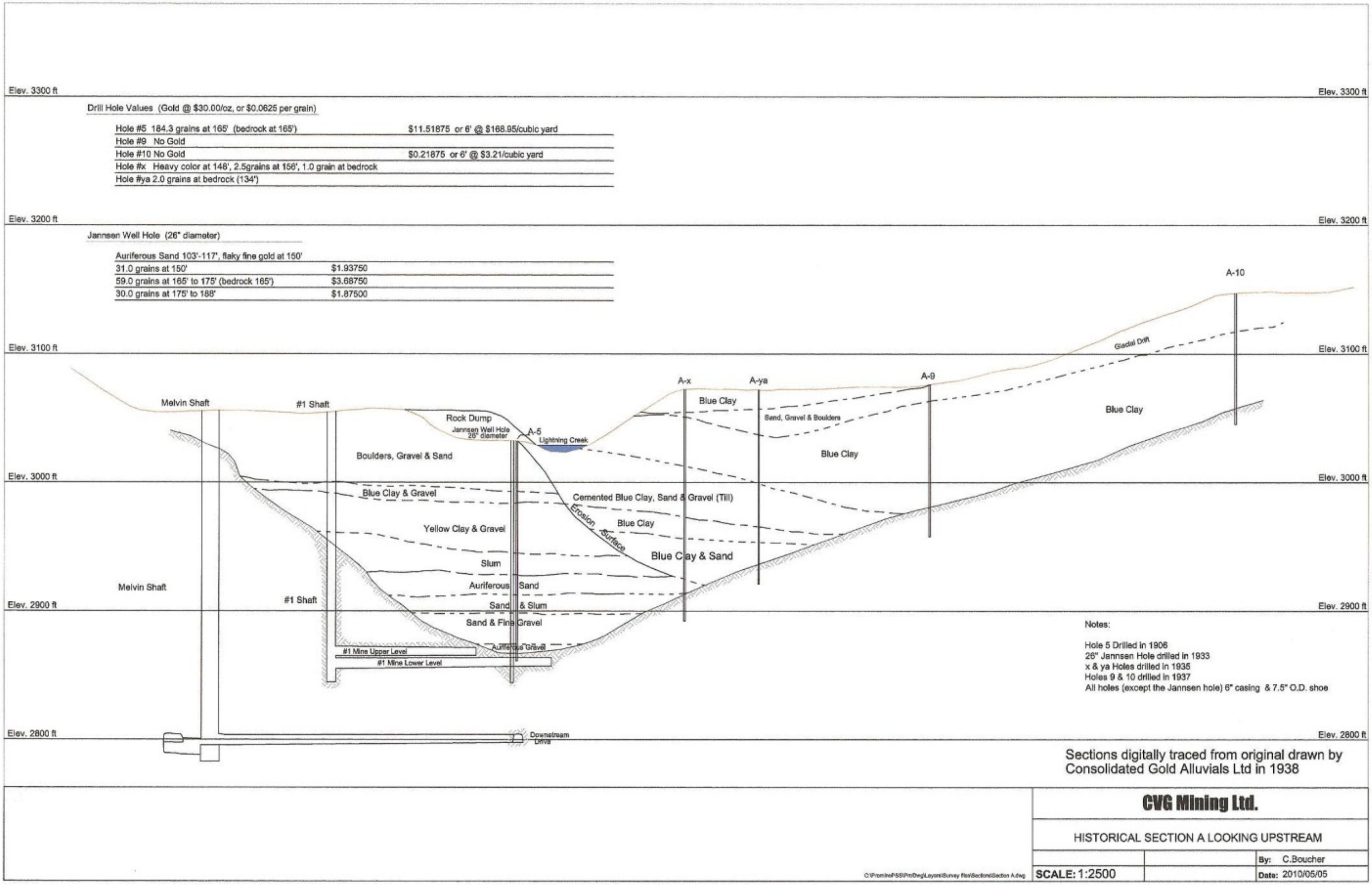
8.0 Deposit Type/Mineralization

The placer gold deposit at the Wingdam property consists of a buried paleochannel called the Deep Lead Channel. Significant gold concentrations along the channel are mainly confined to the gravel-bedrock interface situated 51 m below the Lightning Creek valley floor. The gravel and other alluvium overlying the bedrock were deposited by an ancient fluvial stream that predates the Fraser glacial period or 110,000 ybp. The gold-enriched zone exposed along Drift CC1 reaches up to 1.20 m thick. The zone consists of a boulder/cobble-rich fluvial gravel layer up to 0.90 m thick and 0.30 m of underlying fractured bedrock. Historic drill logs show that significant gold concentrations are confined to a bedrock-proximal zone measuring 1.83 m thick. Results from past drill programs and seismic surveys indicates that the channel bedrock floor varies from 6 to 39 meters wide. The lateral extent of the channel parallels Lightning Creek along a southwest paleoflow direction and is fully covered by a 2,430-meter distance along the length of the Property.

A geological model for the buried Deep Lead Channel (Auriferous Gravel) and the sequence of overlying sediments is illustrated in Figure 4. The sand and silt layers are referred as *slum* when water-saturated. The silt layer and underlying sediments are thawed and the overlying pebble/cobble-rich sand layers are frozen. All of the sediments overlying the coarse-grained gravel layer are barren or contain low gold concentrations (<1.0 g/m^3) along longitudinal cobble clusters.

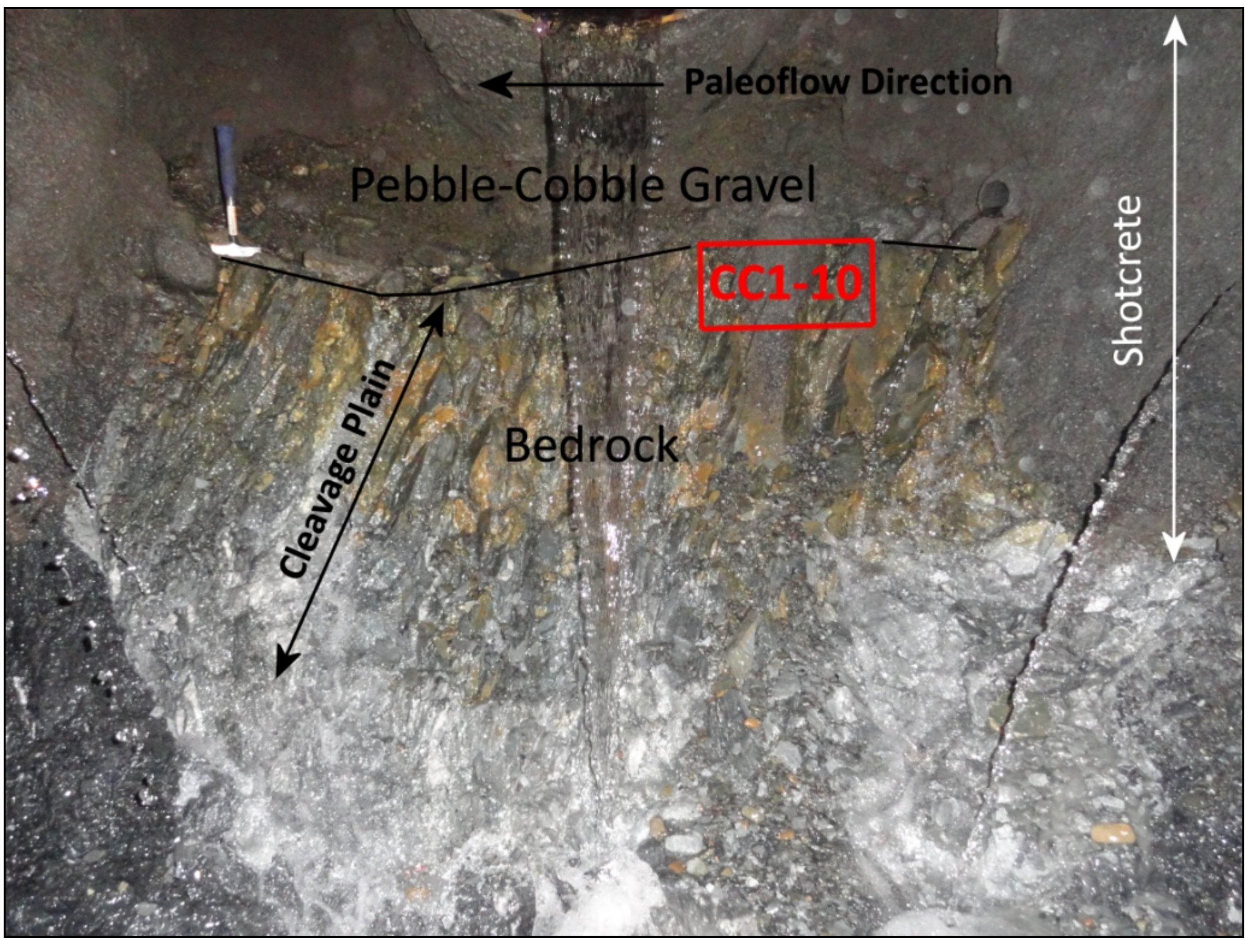
Figure 4 - Deep Lead Channel Profile

Historical Section "A" Looking Upstream



The bedrock along the Deep Lead Channel is made up of black phyllite and siltstone layers belonging to the Harveys Ridge Succession and light grey to tan-colored phyllite of the Ramos Succession. The contact between the successions at this location is an unconformity and part of a 3 m wide fault gouge that parallels rimrock along the south side of the channel. The typical foliation pattern in bedrock underlying the auriferous gravel layer is shown on Plate 1 looking northwest across the width of the channel. The paleoflow direction of the overlying gravel layer trends southwest along the channel length. The gold concentrations identified along the channel bedrock floor are mainly confined to narrow clay-filled gouges and fractures that reach up to 15 cm wide.

Plate 1 Bedrock Foliation and Paleoflow Direction



9.0 Exploration

CVG Mining Ltd. and Omineca Mining and Metals Inc. carried out the following exploration programs since the property was acquired by CVG in April 2009:

1. Underground de-watering and mine rehabilitation program (2009-2012)(CVG).
2. Hydrological survey performed by Clifton Associates Ltd (2009) (CVG).
3. Seismic refraction and reflection survey and ground geophysical surveys that included induced polarization and magnetometer measurements completed by Frontier Geosciences Ltd (2009)(CVG).
4. Sonic core-recovery drill program (2010) involving 14 holes (see Section 10)(CVG).
5. Drift sampling across the width of the Deep Lead Channel (2012)(CVG).
6. Lidar and Orthophoto survey (2014)(OMM).
7. Seismic Refraction and Geotechnical Drilling (2015)(OMM).

The exploration programs are summarized in the following sections. Underground Rehabilitation and Dewatering Project
CVG began rehabilitating the most recent underground workings at Wingdam in 2009. The workings consist of the spiral bedrock drive constructed by Gold Ridge Resources Ltd. (GRR). Rehabilitation work consisted of rock bolting, screening, grouting and shoring along fault zones and other layered bedrock sections susceptible to fracturing.

The purpose of the rehabilitation work was to provide access for a breakthrough drift into the Deep Lead Channel floor. The location and direction of the breakthrough drift was contiguous with the GRR Pipe Drive. A total of 15 freeze holes were drilled into the northwest chamber wall and across the width of the channel along 44-meter distances. The freeze holes were drilled along an arched pattern at 1 meter intervals that surrounds the GRR Pipe Drive. The pattern encompasses a freeze face measuring 15 m wide and 10 m high. The dimensions of the freeze zone was designed to provide sufficient ground support for the proposed construction of a 2.44-meter high by 2.44-meter wide exploration drift across the width of the channel.

9.1 Hydrological Surveys

CVG contracted Clifton Associates Ltd (“Clifton”) of Saskatoon SK in November 2009 to conduct a hydrological study on the property. The results of this study characterize the bedrock as quartzite and phylite layers with fractures and joints acting as active pathways for fluid movement. The hydrogeological regime is interpreted as having bedrock and basal gravel aquifers controlled by overlying low-permeability layers of clay, silt (*slum*), and lodgement till. The basal gravel and bedrock layers are considered to be the active paths that would generate a significant amount of seepage into the current and future underground workings. The silt is an extensive discontinuous unit and further work is required to assess its three dimensional extent on the property. Clifton recommended various depressurization methods such as grouted cut-off walls, drift drainage, and dewatering wells to rectify the seepage issues.

Additional hydrological work was carried out by Omineca in 2014 as part of the current Mine Permit application. Lorax Environmental of Vancouver was contracted to review historic data and model the groundwater flow patterns. Lorax concluded that a series of depressurization wells in conjunction with the isolation of the Sanderson mine from the Wingdam underground will provide sufficient drawdown to effectively manage the impact of groundwater flow on the freeze envelope.

9.2 Ground Geophysical Surveys

CVG contracted Frontier Geosciences Ltd. to carry out various ground geophysical surveys at the Wingdam property from November 24 to December 4, 2009. The Wingdam surveys were tied into the 1938 Consolidated Gold Alluvials Limited (CGA) drill fence-lines. The survey consisted of 1,035 m of resistivity imaging and induced polarization, and seismic reflection lines totaling 640 m long. The work was completed along the three parallel Clifton hydrological survey lines spaced approximately 150 m apart. The CC fence line provided reference for direct comparisons of the ground geophysical survey results and the sedimentological units. This area was also previously explored by geological mapping, drilling, and seismic refraction surveys completed by Fronteer in 1986 and 1990.

In 2015, Omineca engaged Frontier to carry out a program of seismic refraction and downhole seismic focused on the first 300m of channel downstream from the CVG bulk sample drift (Figure 7). Five separate seismic refraction traverses were completed over 1050 meters of line, and 4 of the drillholes completed in 2015 were surveyed for a total of 188.1 meters. The data generated in 2015 was integrated with the historical seismic data collected by Fronteer to generate very accurate profiles of the channel along the gravel / bedrock interface.

The interpreted seismic data indicates that the channel profile does not match the historic outline of the Deep Lead Channel. As well as following a different center line or thalweg, it appears that the walls of the channel are likely steeper than originally thought (Figure 5). This is corroborated by the results from the bedrock intercepts of the 2015 drilling as well as historical drill data. The seismic profiles also show a series of depressions along the channel which are postulated to be potential natural traps for the placer gold during deposition of the gravels (Figure 6). This new interpretation of the channel profile has increased the overall volume of material identified as potentially mineable. In addition, the steeper channel walls should facilitate shorter freeze holes while increasing the stability of the freeze structure.

Figure 5 - Deep Lead Channel Outline

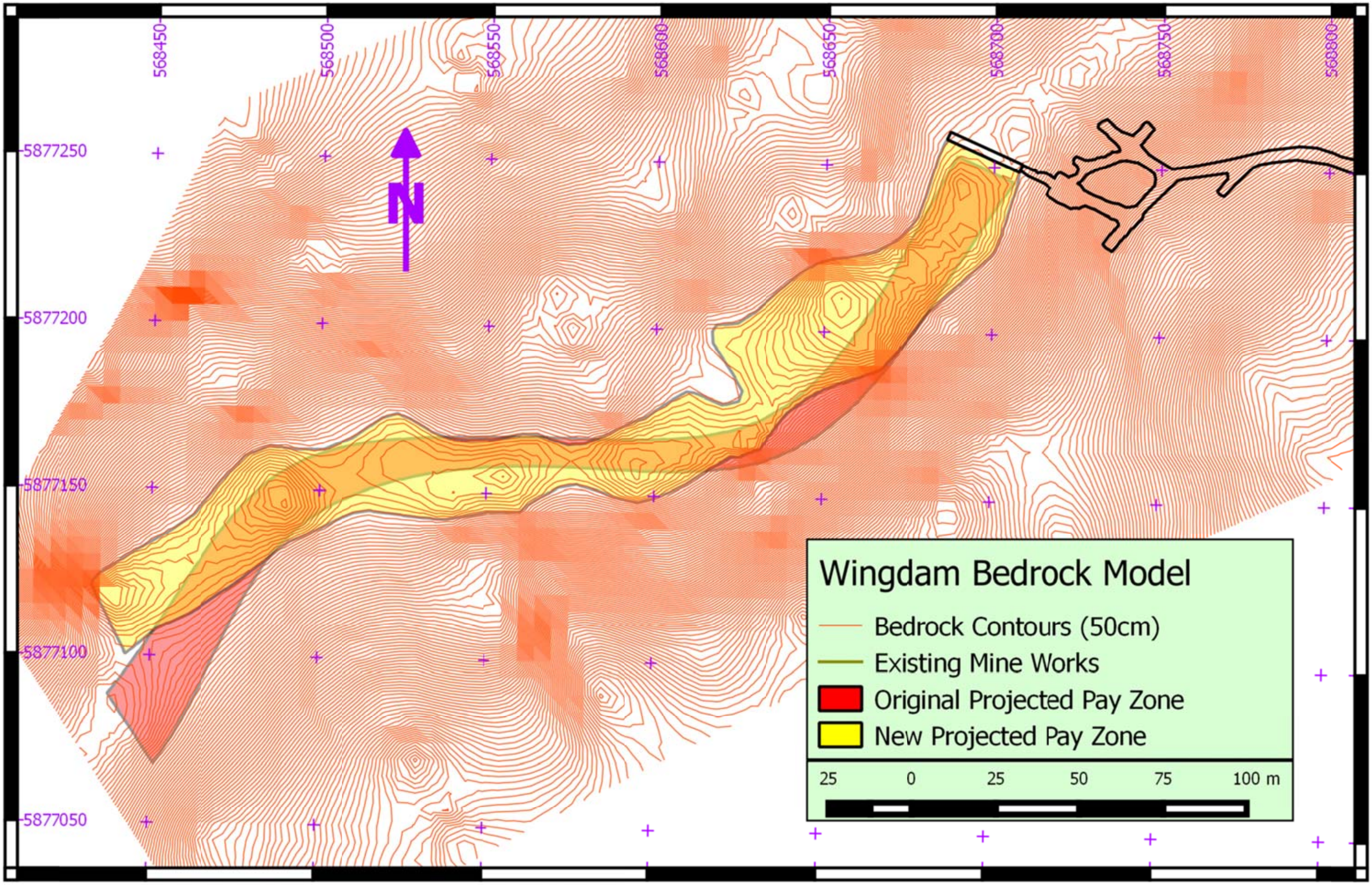
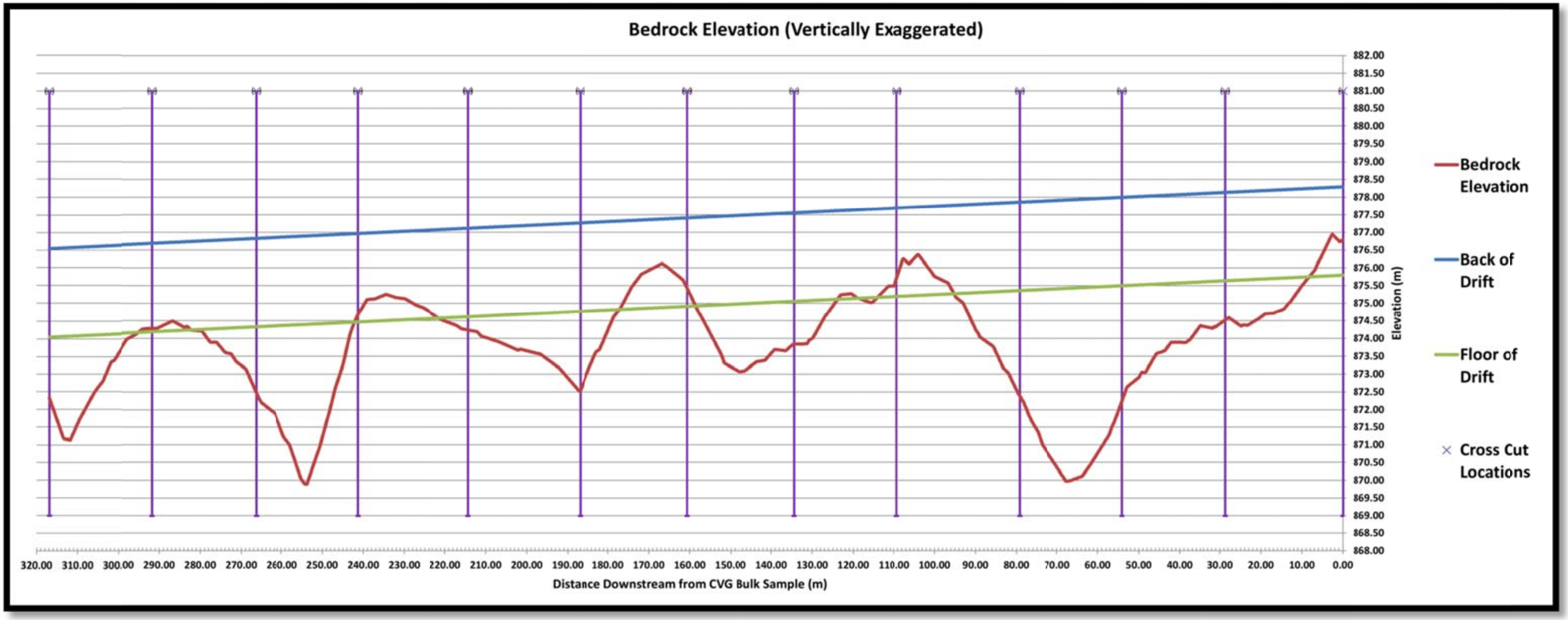


Figure 6 - Deep Lead Channel Section



9.3 CVG Sonic Drilling Program

CVG carried out a 720-meter sonic core-recovery drill program on the Wingdam Property in 2010. A total of 14 holes (101.6 mm diameter) were drilled vertically with a track-mounted Longyear SRO 74B drill rig to depths ranging from 38.4 to 62.2 m. The purpose of the drill program was to investigate the stratigraphy and sample basal gravels along a 956-meter length of the Deep Lead Channel. The CVG sonic holes (10 cm diameter) were drilled between historic CGA fence lines C and D (Figure 3). The fence lines were previously drilled by a cable-churn method (19 cm diameter) during the 1910’s and 30’s.

Three of the sonic drill-holes (03, 08 and 10) intercepted significant gold concentrations (6.56 to 32.67 g/m^3) along the Deep Lead Channel bedrock floor. These grades are comparable to the results obtained from the CVG drift-sampling program. It is uncertain why the remaining 10 holes failed to yield similar grades. Drill logs indicate that the basal gravels along the channel floor are continuous and exhibit no disruption or alteration by any type of geological event such as glacial erosion or scouring. There are three reasons that may contribute to the unreliability of the sonic drill-rig used by CVG:

1. The majority of the gold concentrated across the width of the Deep Lead Channel is confined to a narrow corridor and easily missed by drilling.
2. The gold along the enriched corridor and the remaining channel width is erratically distributed or confined to narrow bedrock fractures and gouges that are commonly separated by distances exceeding the drill-hole diameter (10 cm).
3. The geologist who logged the core reported that the majority of the bedrock recovered by the core-sleeve was severely broken-up and pulverised. This commonly happens when a drill-bit or core sleeve is jammed after penetrating competent cobbles and boulders occupying a gold-enriched channel bedrock floor. This effect could drastically inhibit the recovery of gold particles that are mainly concentrated at the gravel-bedrock interface.

The CVG drift program showed that the highest grades across the width of the channel (23.5 m), averaging 120.89 g/m^3, are confined to a narrow corridor measuring 5.3 m wide. Up to 70% of the total raw gold (173.5 ounces) recovered from the drift derived from this enriched corridor. This corridor appears to be represented by churn-drill D-1. The churn-holes (D-4 to 6) drilled outside of the corridor or along rimrock returned grades (0.33 to 1.26 g/m^3) similar to low grades (0.05 to 1.86 g/m^3) identified by sonic-holes 01, 01a, 02, 04-07, 09 and 11-13.

The overall results obtained from the sonic drill program were inconclusive. As a result CVG made the decision to test the grade of the Deep Lead Channel using a bulk sample drift (see Section 2012 Deep Lead Channel Drift Sampling Program).

9.4 2015 Geotechnical Drilling

In 2015, Omineca contracted Valiant Drilling to carry out a geotechnical drilling program in the area of the first proposed mine block downstream from the CVG bulk sample drift (Figure 7). Valiant provided a Fraste MultiDrill trackmount system capable of sonic, reverse circulation and diamond drilling. The program had a number of objectives. One was to establish the elevation of the gravel / bedrock interface at select locations along the inferred channel rim. Another important aspect of the program was to collect in situ samples of slum which were then sent to Weir Jones Engineering for specialized geotechnical testing. In addition to these objectives, samples of bedrock were collected for geotechnical testing at Weir Jones. Selected holes were also completed with PVC to allow for downhole seismic refraction surveying. 7 vertical and one angle hole were completed for a total of 392 meters. A total of twenty six 1.5 meter length samples of slum and six samples of bedrock were collected for geotechnical analyses.

9.4.1 2015 Geotechnical Drilling Results

The samples of slum were collected in clear Lexan tubes and capped and sealed onsite at Wingdam. The samples were delivered directly to the Weir-Jones testing facility in Vancouver by Omineca personnel. Weir-Jones analysed the samples for ultimate compressive strength, Compressive Creep and Flexural Creep to generate data to help refine the final freeze model.(Weir-Jones Engineering Consultants Ltd, Compression and Creep Test, May 2016)

The purpose of the testing was to provide a basis for designing a structurally competent and impermeable frozen roof beam and cut-off pillars. These structures will be formed from the weathered fragments of phyllitic bedrock which has formed a layer of saturated micaceous sediment interspersed with sand and gravel lenses immediately above the pay zone. This is the so called “slum layer”. The frozen structure is intended to facilitate underground gold mining in the Wingdam area. These tests provide an estimate of the ultimate compressive strength (UCS) and creep characteristics of frozen samples of core material obtained from drill holes provided by Omineca. The freeze wall will have a temperature gradient from the centre near the freeze pipes to the outside edge of the frozen zone, and therefore testing took take place at a range of temperatures from -10°C to -30°C. ASTM standards D7300, D5520, and C78/C78M are used as a reference for running these tests.

The core samples were taken from nine drill holes located along the anticipated extent of the buried channel which is assumed to contain the auriferous gravels. The drill samples provided consist of a range of soil types from micaceous silts and clay to sand and gravel with pebble size up to a maximum of about one centimeter in diameter. The samples were collected at depths from 2 to 4 m above the bedrock contact.

Thirty-six total samples were tested for compressive strength. The resulting average ultimate compressive strengths were as follows:

- 8.9 MPa @ -10°C
- 13.3 MPa @ -20°C
- 20.4 MPa @ -30°C

All of the samples returned values greater than 7 Mpa which was the base case strength determination used in the previous mine design prepared by Minetech International Limited in 2012. The testing also indicates that a ground freeze temperature of -10°C to -15°C will provide the necessary freezing as compared to temperatures of -20°C to -25°C used in earlier mine designs. This lower temperature is expected to result in a significant saving in terms of freeze energy costs and the higher strength factor may allow for more flexibility in designing the size of the mining blocks. A wide variance was shown across each set of tests which is indicative of the varying soil types.

Further testing will have to be conducted in order to obtain more representative values for the *in situ* frozen soil properties for detailed design. The additional soil samples should be carefully categorized by type and tests conducted on batches of similar soil types.

There is not enough data to have a high degree of confidence in strain rate predictions due to the widely spread results. The average strain rates observed are:

- 0.066%/hr @ -10°C
- 0.043%/hr @ -20°C
- 0.068%/hr @ -30°C

The flexural creep test results show that the bending of fine grained samples exhibited a very low amount of bending with some samples fracturing near the beginning of the 72 hour test and some near the end. The average displacement rates are:

- 0.246 mils/hr @ -20°C
- 0.155 mils/hr @ -30°C

As the sample size generated by the 2015 Geotechnical Program was relatively small, Weir-Jones recommends further testing in order statistically validate and confirm the results of their engineering study.

Figure 7 - 2015 Geotechnical Program

9.5 *2012 Deep Lead Channel Drift Sampling Program*

CVG Mining Ltd. constructed a large Freeze Staging Chamber (Figure 8) in bedrock centered around the pipe drive installed by Gold Ridge Resources Ltd. in 1992. A total of 15 horizontal freeze holes were drilled into the northwest chamber wall and across the width of the Deep Lead Channel along 44-meter distances. The freeze holes were collared in an arched pattern at 1 meter intervals that surrounds the GRR pipe drive (Figure 9). The refrigeration pattern produced a freeze face measuring 7.5 m wide and 7.5 m high. The holes were collared through 7.2 m of bedrock and terminated in 13.3 m of bedrock along the north side of the channel. The freeze zone was designed to provide sufficient ground support to drive a 2.85-meter high by 2.44-meter wide drift or crosscut (CC1) across the entire channel width (23.5 m). The drift height was corrected at breakthrough to 2.44 m after identifying the elevation of the gold-enriched zone occupying the bedrock floor across the channel.

Figure 8 - Freeze Staging Chamber

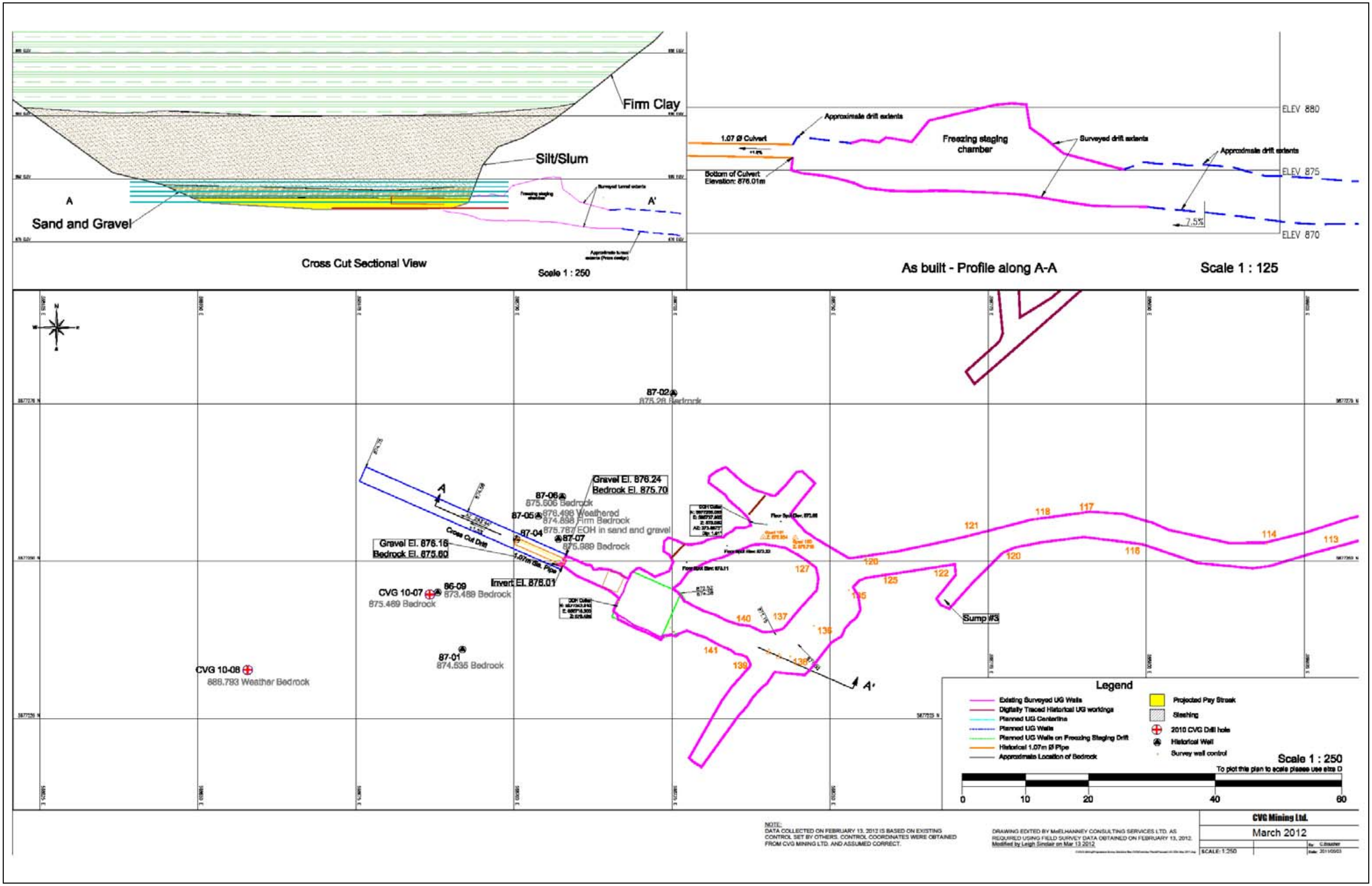
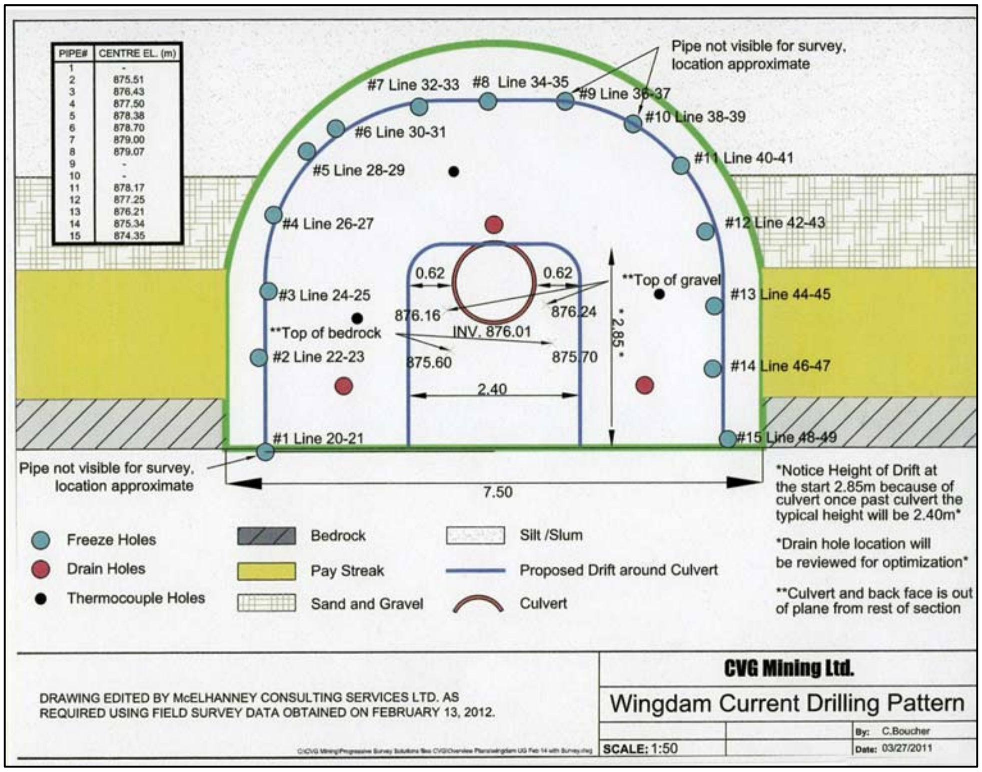


Figure 9 - Freeze Hole Drill Pattern



9.5.1 Drift Sample Preparation, Analyses and Security

The total volume of alluvial and bedrock material exposed and sampled along the drift location amounted to 139.9 bank cubic meters. This volume is equivalent to the dimensions of the channel exposed along the drift (2.44X2.44 m) length (23.5 m). The discrete or small samples totalling 1.34 m^3 were collected from 33 locations to define the distribution of gold concentrations across the channel. The washplant samples, totalling 131.27 m^3, consisted of remaining drift muck extracted by a scooptram and processed by a trommel installed in an underground chamber. The remaining material (7.29 m^3) was previously extracted and processed from the pipe drive installed by Gold Ridge Resources in 1992.

<u>Discrete Samples</u>

A total of 33 discrete samples made up of gravel and underlying bedrock were collected in 28-liter labelled pails. The samples were selected from various parts of the 23.5-meter Deep Lead Channel width exposed along Drift CC1.

The samples were transported to the surface by a scooptram and stored in a secured assay office located at the mine site. The inside and outside locations of the assay office were monitored by three video surveillance cameras throughout the duration of the sample analyses time period. Access to the office was limited to the Chief Geologist who performed all of the sampling and analytical procedures. Other authorized company personal or mine managers only entered the assay lab when accompanied by the Chief Geologist.

Samples 1 to 18 were thoroughly scrubbed in water by hand and sieved through two Tyler screens measuring 4-mesh and 12-mesh. Samples 18 to 33 were processed by a portable minus 0.5-inch mesh screening washplant. The screened concentrate recovered by the washplant was sieved in the same manner as described for samples 1 to 18. The resulting +4 mesh fraction produced from each sample was visually examined for gold nuggets. The screened -4 to +12 mesh and -12 mesh fractions were processed separately across a custom-built concentrating or shaker table similar to a Deister Table. Each gold fraction recovered from the table was weighed, photographed and stored in separate labelled containers. Individual gold fractions were weighed with an Ohaus electronic scale accurate to plus or minus 5 milligrams. The gold samples were collected by company management on a regular basis after analysed and transferred to a bank safety deposit box for safekeeping.

Washplant Samples

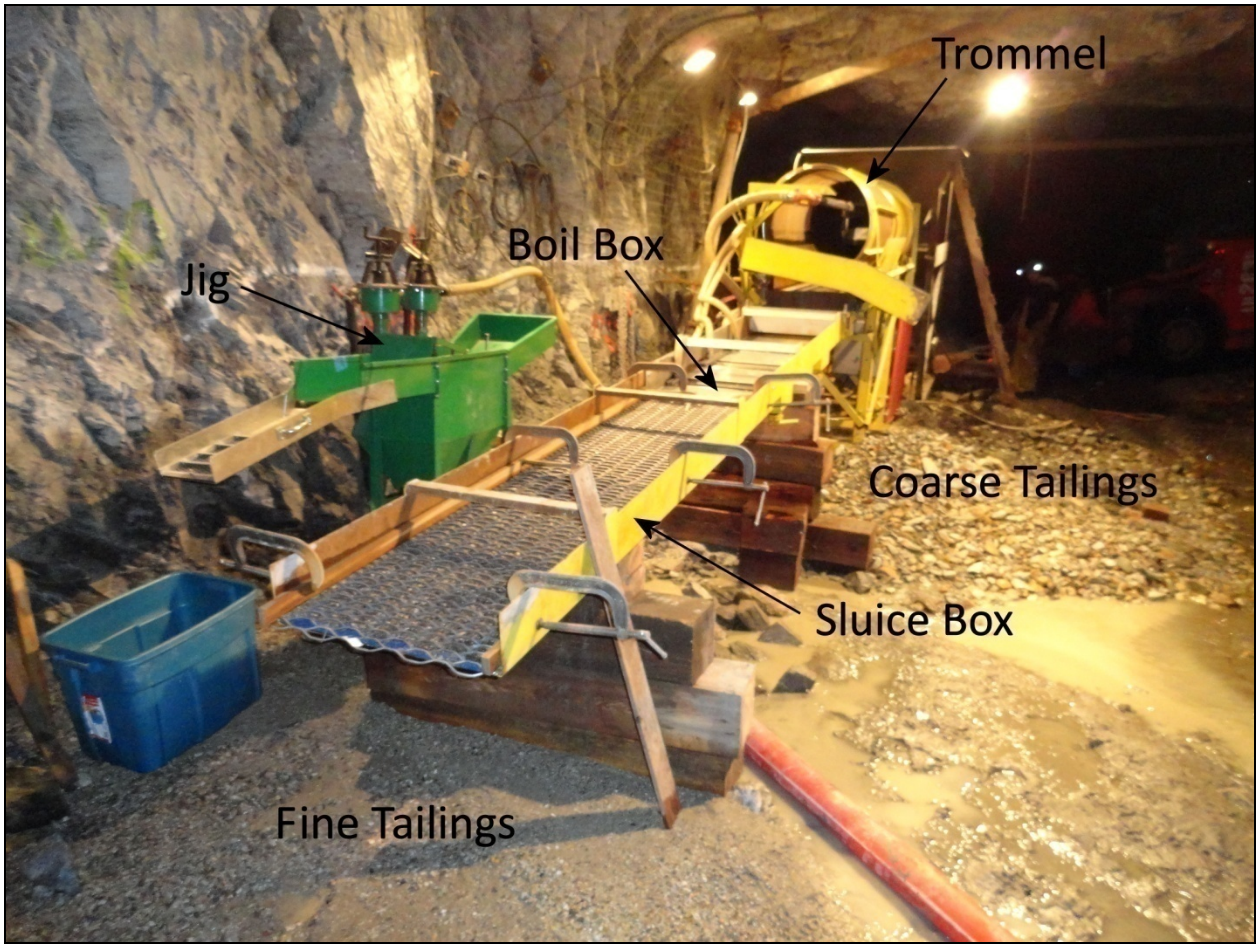
The Deep Lead Channel samples were extracted from a 30.7-meter long horizontal drift (Drift CC1) measuring 2.44 meters wide. The drift commenced at the Freeze Staging Chamber and entered the channel after penetrating 7.2 meters of bedrock. The drift height along this distance amounted to 3.66 m. The height along the remaining 23.5-meter distance that intersected the channel was adjusted to 2.44 m by reducing the sill depth. The height adjustment was selected after verifying the location of the gold-enriched zone along the channel floor. The total length of the drift was driven in increments involving horizontal drill and blast rounds that varied from 0.9 to 2.1 m long.

The 1.07-meter diameter GRR pipe drive exposed along part of the drift was cut into sections and removed before each drill and blast round. The pipe extended from 0 to 19 m along the drift and intersected the channel along an 11.9 m distance. Gold Ridge Resources Ltd. extracted all of the bedrock and sediments exposed by the pipe drive in 1992. Records of the gold (2.18 oz) recovered along the pipe drive are part of the total amount of gold recovered from Drift CC1.

The initial 3.8-meter intersection across the channel width was extracted and placed into a stockpile. This stockpile was processed by the washplant before the drift was advanced further across the channel. The final intersection across the channel width measuring 19.7 m was extracted and processed separately during a later time period. The muck including gold particles that remained along the drift sill after scraped by the scooptram bucket was thoroughly cleaned and collected by hand with the use of a high-pressure airline and shovel.

The bulk sample was washed, screened and sluiced by a trommel washplant installed underground in a gold recovery room located adjacent to the remuck station (Plate 2). The trommel consisted of a hopper and a 3.5-foot diameter by 8-foot long rotating steel drum. The drum contained a 4-foot long scrubbing area and a 4-foot long screening area. A high-pressure spray bar ran through the entire length of the drum interior and another was located in the hopper. The screened slurry made up of minus 0.5-inch mesh aggregate and water entered a 2-foot wide by 8-foot long steel flume containing 3 boil boxes. The slurry also passed through an attached 2-foot wide by 8-foot long sluice box lined with nomad carpet and sheets of raised expanded metal. The trommel was fed by conveyer at a controlled rate averaging about 1 m^3 of loose sample material per hour. The sample was screened by a grizzly containing upright 4-inch spaced steel flat bar prior to entering the conveyer.

Plate 2 Gold Recovery Plant



A total of 188.2 loose cubic meters of sample material was processed by the washplant during 31 separate days. The concentrate that accumulated in the first and second boil boxes (20 liters) was collected at the end of each processing day for security reasons. Over 99% of the gold produced by the washplant was recovered in the first and second boil boxes. The first boil box on average contained 93.5% of the total produced gold. The concentrate that accumulated in the third boil box and in the sluice box was collected after the duration of various multiple-day processing periods. The concentrate collected each day was transported to the surface and stored in a secured assay office located at the mine site.

The daily batch of concentrate collected from the first boil box was thoroughly scrubbed in water by hand and sieved through two Tyler screens measuring 4-mesh and 12-mesh. The resulting +4 mesh fraction produced from each sample was visually examined for gold nuggets. The screened -4 to +12 mesh and -12 mesh fractions were processed separately across a custom-built concentrating table similar to a Deister Table. Each gold fraction recovered from the table was weighed, photographed and stored in separate labelled containers. The concentrate collected from the second and third boil boxes, and the sluice box was processed separately with a 12-inch wide double-celled jig. The resulting jig concentrate (1 liter) was processed by the shaker table and the final gold product was measured and stored in the same manner as described above.

Individual gold fractions were weighed with an Ohaus electronic scale accurate to plus or minus 5 milligrams. The gold samples were collected by company management on a regular basis after analysed and transferred to a bank safety deposit box for safekeeping.

Analytical Results

The 33 discrete samples yielded a total of 97.7 grams or 2.513 ounces of raw gold.

The washplant samples extracted from the 23.5-meter intersection across the Deep Lead Channel entire width yielded 168.284 raw ounces of placer gold (Plate 3).

Plate 3 Total Gold Recovered From Washplant Samples (168.234 raw ounces)

Sampling Interpretation and Conclusions

The total amount of raw gold extracted from the 23.5-meter long drift-intersection across the total width of the Deep Lead Channel was 173.5 ounces. This amount includes the gold recovered from the washplant samples (CU1-32), 33 discrete samples (CC1-1 to 33), and the 1992 Gold Ridge Resources Inc. pipe drive and bedrock samples. Records of the gold recovered from the 0-3.8 m and 3.8-23.5 m intersections and across the total channel width (0-23.5 m) are given in Table 9.

Table 7 - CC1 Drift Gold Recovery Record

Sample	Volume (m^3)		Swell Factor	Raw Gold	
	Loose	Bank		grams	ounces
Gold Record (0-3.8 meters)					
CU1-8	28.83	20.08	1.44	588.99	18.936
CC1-1 to 10	0.32	0.27	1.20	19.54	0.628
GRR PD	3.83	2.27	1.69	9.20	0.296
Total	**33.81**	**22.62**	**1.49**	**617.73**	**19.860**
Gold Record (3.8-23.5 meters)					
CU9-32	154.34	111.19	1.39	4645.16	149.348
CC1-11 to 33	1.30	1.08	1.20	78.16	2.513
GRR PD	5.74	4.60	1.25	1.20	0.039
GRR BR	0.53	0.42	1.25	54.00	1.736
Total	**161.91**	**117.29**	**1.38**	**4778.52**	**153.635**
Total Gold Record (0-23.5 meters)					
Total	**195.72**	**139.91**	**1.40**	**5396.25**	**173.495**

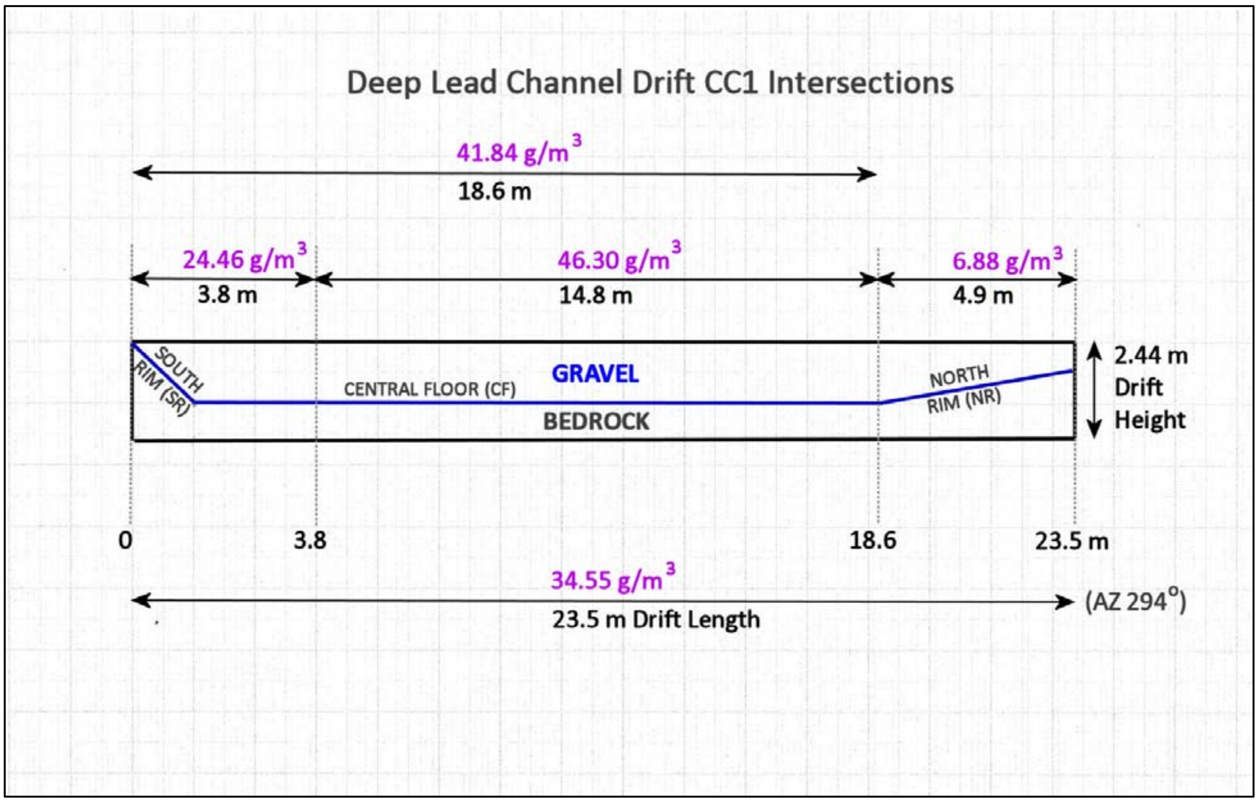
The refined-equivalent or pure gold grades for various intersections across the channel width are given in Table 22. The intersections and corresponding grades are illustrated on Figure 10. The grades were calculated by using 0.8957 as a multiplication factor to convert the raw gold to refined-equivalent (Ref-Eq) values. The volumetric grade (g/m^3) was converted to mass grade (g/tonne) by using 2.45 g/cc for the specific gravity. The grades represent refined-equivalent gold concentrations across the entire thickness or height of the drift (2.44 m).

Washplant samples CU26 and 27 were extracted from the north channel rim (18.6-23.5 m) where bedrock slopes upwards and gold concentrations significantly decrease. The samples produced 108.71 refined-equivalent grams of gold from 15.8 bank cubic meters. The gold grade across this 4.9-meter rimrock intersection amounts to 6.88 g/m^3. This grade is similar to the gold concentrations (7.34 g/m^3) identified in the largest and most representing of three discrete samples (CC1-33) collected along the north rim. The total amount of raw gold recovered along the north rim is estimated within reasonable accuracy to be 200 grams. The gold grades along intersections 0-3.8, 0-18.6, and 3.8-18.6 meters exclude the north rim.

Table 8 - Deep Lead Channel Gold Grades

Horizontal Intersection (m)	Distance (m)	Bank Volume (m^3)	Gold Recovered (oz)		Refined-Equivalent Gold Grade		
			Raw	Ref-Eq	g/m^3	g/tonne	oz/ton
0-3.8	3.8	22.62	19.860	17.789	24.46	9.98	0.291
0-18.6	18.6	110.74	166.296	148.951	41.84	17.08	0.498
0-23.5	23.5	139.91	173.496	155.400	34.55	14.10	0.411
3.8-18.6	14.8	88.11	146.435	131.162	46.30	18.90	0.551
18.6-23.5	4.9	29.17	7.201	6.450	6.88	2.81	0.082

Figure 10 - Deep Lead Channel Drift CC1 Intersections



The discrete samples collected from intersection 6.6 to 11.9 m contained the highest gold concentrations (71.35 to 185.62 g/m^3) measured across the channel width. Abundant gold flakes were visible on the channel bedrock floor across this 5.3-meter wide enriched corridor. The majority of the flakes were visible along bedrock gouges and narrow fractures that were flushed and exposed by groundwater flow.

Washplant samples CU9-11 were collected and processed from various parts of the gold-enriched corridor (6.6-11.9 m). The 12.31 loose or estimated 8.86 bank cubic meters processed from the three samples produced 38.264 ounces of raw gold. The refined-equivalent bank gold grade for the washplant samples collected across the enriched corridor amounts to 3.868 oz/m^3 or 120.32 g/m^3. The bank volume for the 5.3-meter wide high-grade intersection amounts to 31.8 m^3 and represents 22.7% of the 140 m^3 total sample volume extracted from the channel. Grades measured in the discrete and washplant samples indicate that over 65% of the gold recovered from the entire width of the channel (23.5 m) derived from this 5.3-meter wide central corridor.

9.5.2 Data Verification

A sample measuring 200 liters was collected from the accumulation of fine tailings discharged by the washplant during the first sluicing day to determine the efficiency of the boil boxes and sluice box. The sample was processed by the jig and the resulting concentrate was processed along the shaker table. There was no visible gold identified in the concentrate and for this reason it is believed that the washplant was operating more than 99% efficient.

There were no processing factors or deleterious elements identified in the gold recovery system that could have a significant effect on potential economic extraction. The gold particles concentrated along the Deep Lead Channel are relatively coarse-grained or greater than 32 mesh Tyler (0.50 mm) and easily recoverable by a trommel washplant and conventional sluicing system.

9.5.3 Mineral Processing and Metallurgical Testing

6 Deep Lead Channel samples were sent to ALS Minerals (ALS) in North Vancouver BC for analyses. The analytical results for the Au and Ag fineness of the raw placer gold particles are listed in Table 11. The ALS QC certificates are located in Appendix II.

Table 9 - Placer Gold Fineness Measurement

Sample ID	ALS Prep Code	ALS Analytical Code		QC Certificate Number
		ME-GRA24 Au Fineness	ME-GRA Ag Fineness	
CC1-PG-1	WEI-21, LOG 24	909	82	VA12114777
CC1-PG-1 (D)*	WEI-21, LOG 24	909	82	VA12114777
Average		**909**	**82**	

A raw gold sample weighing 164.5 ounces was smelted by Technic Canada (TC) in Richmond BC for refining purposes. The TC settlement weight or weight of the bullion bar (Plate 20) produced from the sample was 162.7 oz. The TC bullion bar assay or fineness determination was 905.6 Au and 80.2 Ag.

All of the foreign particles including black sand, sulfides and lead were thoroughly removed from placer gold sample CC1-PG-1. The ALS fineness determination (909 Au) for this sample represents the average purity of the placer gold extracted from the Deep Lead Channel.

The 164.5-ounce sample sent to TC for refining contained some foreign fine-grained particles or impurities that included lead. The non-base metal impurities removed by the smelting process amounted to 1.09% of the total sample weight. The lead derived from fragmented blasting caps and could not be separated by the shaker table. The lead blended into the gold bar and for this reason the TC fineness determination (905.6) was lower than the fineness measured in the ALS sample.

The author believes that the 2012 bulk sample program conducted by CVG Mining Ltd. verifies that the placer gold grades indicated by historical pre-1930's churn drilling are accurate and also demonstrates that the use of the freeze mining technique pioneered by CVG is a viable approach to mining under Lightning Creek. Although the author has not been able to independently verify the methodology and results from the 2012 CVG Drift Sampling Program, the work was carried out under the supervision of Steve Kocsis, P.Geo who is considered a Qualified Person under NI 43-101. The author believes that the work was carried out and documented in a professional manner and has no reason to doubt either the methodology or the accuracy of the results.

10.0 Phased Wingdam Bulk Sample Plan

10.1 Introduction -The Setting and Its Mining Challenges

The ability to physically sample and evaluate the Deep Lead Channel gravels using traditional exploration methods is limited. As the target gravels are located partially beneath a large creek and the Barkerville Highway, access for drilling vertical exploratory holes is limited. In addition the unstable nature of the overlying sediments makes it very difficult to complete drill holes and to recover meaningful samples. Finally, the nuggety nature of the gold bearing zone does not lend itself to accurate correlation between drill holes intercepts. The depth of overburden ~50m, and the location of the overlying creek does not make it feasible to test with trenching. Omineca believes that the only method to successfully evaluate and quantify the Deep Lead Channel gravels is to continue and expand bulk sample testing.

The Wingdam placer gold deposit lies in an ancient paleo-channel in the Lightning Creek valley covered by approximately 50-55 meters of glacial-fluvial, lacustrine, and more recently deposited alluvial gravel, silts and sands. The bedrock is typically an upturned phyllite cut by occasional quartz veins. The upturned phyllite bedding is typically oriented perpendicular to the general downstream direction of the channel. Mechanical testing of bedrock samples from drilled cores in 2015 has shown that the Uniaxial Compressive Strength of the bedrock material is highly variable ranging from a maximum of 180 MPa perpendicular to the foliation planes and with secondary mineralization, down to 35 MPa parallel to the geological bedding plane. The mining methods will be a combination of mechanical combination miner for the softer materials, and drill, blast and muck for the quartz veins sections.

The placer gold is concentrated at the contact between the weathered bedrock and the unconsolidated overburden, but may extend up to 30cm below in erosional fissures created by in the upturned phyllites, and for as much as 90cm above in the cobble, gravel and sands overlying the interface. The gravel and sands overlying the bedrock interface are typically rounded and vary in compositions from local phyllite material to much harder granites transported from the surrounding area over extended periods of glacial and alluvial deposition.

The placer gold is coarse, typically the shape and size of a "flax seed". Although larger nuggets are frequently encountered, very little, if any, fine gold exists making the gold readily recoverable by conventional gravity circuits. The Wingdam placer gold is bright, shows little or no rust staining and recent laboratory analysis conducted by ALS Minerals in Vancouver, B.C. shows the gold in its natural state has a fineness of 909 Au and 82 Ag.

The placer bearing gravels are typically immediately overlain by a lacustrine silt and fine sand layer known locally as "Cariboo Slum". This material is water saturated and under pressure. It appears to be thixotropic and will flow rapidly into areas of lower pressure if breached by mining activity. The existence of the Cariboo Slum is the main reason historical attempts to mine the Wingdam paleo-channel have failed.

Hydrologically the channel consists of two aquifers: a confined lower aquifer containing the placer gold deposit and an unconfined upper aquifer recharged by the surface waters of Lighting Creek. A thick clay layer separates the two aquifers. The lower aquifer is naturally recharged by groundwater flow through fissures in the bedrock and is under higher pressures. Wells drilled to the lower aquifer will typically be artesian, with widely varying head pressures ranging from minimal to ~5m above surface elevation in the vicinity of the mine, the flow rate and pressure are dependent upon the season. Similarly, ground water flow velocity can vary widely in the lower aquifer from 0.3 to 1.5 m/d (Piteau 1986).

Historical mining activity connected the lower aquifer to the upper one in the vicinity of the abandoned Sanderson mine workings. It has been estimated that these workings contain 750,000 m^3 (198 million US gallons) of water. Any effort to dewater the lower aquifer requires the dewatering of these extensive mine works which has the effect of extending the effective drawdown curve to over a kilometer upstream. Accordingly, historical attempts to mine the Wingdam have been burdened with the requirement of sustained dewatering using large pumps with resulting high dewatering costs.

The two main prerequisites to successful underground mining of the Wingdam deposit are:

1. Stabilize the Cariboo Slum to prevent it from flowing into the mine workings
2. Reduce dewatering volumes and thereby reduce ongoing pumping costs

In developing its' bulk sample plan, CVG Mining has developed strategies to overcome these challenges. A key strategy of the plan is to employ ground freezing, this was originally proposed in 1974 REF. This technique was successfully implemented by CVG Mining Ltd. when the company bulk sampled the channel in 2012 (Kocsis, 2012). CVG plans to establish a 4m thick frozen structure to encapsulate each mining panel. The structure will not only stabilize the Cariboo Slum but has the added benefit of forming a hydraulic seal over the mining panel. This greatly reduces the volume of water to be drained from the unfrozen material contained within the mining panel.

However, employing ground freezing introduces two additional challenges:

1. It is energy intensive and Wingdam mine lacks access to the 3 phase power grid; and

2. It requires a strategy to control groundwater velocity in order to prevent the formation of windows ("holes") in the frozen structure

CVG Mining has explored several energy options to power the Wingdam mine including diesel, wood pellets, Compressed Natural Gas (CNG), Liquefied Natural Gas (LNG) and propane and has determined that the most cost and energy-effective source is LNG. CVG Mining has identified a reliable supplier who can deliver LNG to Wingdam at a cost of approximately $14.00/GJ over the life of the mine and plans to employ LNG together with heat recovery strategies to economically address the energy requirements of ground freezing and for electrical generation. CVG Mining also intends to use an absorption chiller system as opposed to the conventional screw compressor type freeze plant used by CVG during the bulk sample program. The use of LNG and absorption chillers with heat recovery significantly reduces the overall cost of ground freezing.

To control groundwater velocity, CVG Mining plans to establish pumping wells upstream of the active workings, these will be monitored with piezometers over the active mine area. The wells will be pumped at a rate necessary to equalize the hydraulic gradient along the channel being mined thereby reducing the groundwater velocity to near as zero as possible. A hydrological study (Lorax, 2014) shows that a seasonal pumping rate of 950 to 1100m^3 per day are expected to be sufficient to control groundwater flow to acceptable levels.

CVG Mining has developed potentially viable strategies which address the main challenges at Wingdam. The author is confident that the Deep Lead Channel gravels can be safely explored and sampled by implementing these strategies.

10.2 Phased Mining Approach

CVG Mining controls approximately 2400 meters of paleo-channel and has developed a plan to mine the channel in two or more phases. The first phase involves the bulk sampling of channel that, to the company's knowledge, is undisturbed from previous underground mining activity. This section extends 300m in a downstream (westerly) direction from the location of the CVG bulk sample. CVG has obtained all permitting necessary to complete this first 300m Phase 1 test bulk sample with the goal of demonstrating and implementing its planned mining methods and strategies.

Phase 1 Bulk Sample

The first 300m phase of the mine plan involves:

1. Installing a grout plug to isolate the decline from historic mine works and minimize water inflow
2. Dewatering, re-entering and rehabilitating the existing decline
3. Establishing a new ventilation raise for mine services and secondary egress
4. Drilling and installing two 12 inch wells to the lower (basal) aquifer and begin pumping to minimize the downstream hydraulic gradient, and reduce groundwater velocity to less than 0.5 m/day

5. Extending the existing footwall drift parallel to the paleo channel using mechanized mining equipment
6. Hydraulically and mechanically isolating mining blocks of pay gravels through ground freezing
7. Mining and backfilling the mining blocks using mechanized mining
8. Structurally and hydraulically sealing the mined out blocks with engineered bulkheads
9. Allowing the mined out block to thaw
10. Extracting the freeze pipes
11. Sequentially repeating steps 4-8 above to advance the mine in a downstream direction
12. Bulk screen pay gravels underground using a trommel and sluice boxes to produce a concentrate – using the reject material to backfill the production panels
13. Process the resulting concentrate on surface using a combination of gravity techniques to recover the placer gold

CVG expects that it will require up to 8 months to mobilize equipment, dewater, rehabilitate and initially develop the mine infrastructure to a point where ground freezing of the first scheduled mining block is complete and placer gold production can begin and an additional approximately 8 months to complete exploration and bulk sampling of the first 300 meters of the Deep Lead Channel.

10.3 Grout Plug and Decline Rehabilitation

CVG Mining will access its underground workings by dewatering, re-entering and where required rehabilitating the existing decline. In 1992, when Goldridge Resources was driving the decline they broke into a previously existing vent raise (reportedly installed in the 1960's) connecting the historical "Australian Deep Lead Drift" to surface. This resulted in a direct connection between the decline and the old mine works extending as far upstream as the historic Sanderson Mine.

All previous mining attempts have involved dewatering the pay gravels without significantly modifying the hydrological environment. CVG does not intend to dewater the pay gravels until they have been isolated. In addition, sealing off of the hydraulic connection between the footwall drift and the existing Melvin shaft will avoid the significant initial and ongoing costs of dewatering the channel and old mine works prior to dewatering the decline.

To eliminate the connection, the current mine plan proposes to install a temporary grout plug and an engineered bulkhead between the raise and the decline drift that intersected the old vent raise. Installation of the grout plug will be conducted from surface and will involve a program of drilling, filling, and grouting using concrete, aggregate and cement grout. The grout plug design requires drilling four 6" holes (see Figure 11). Low slump concrete will be pumped into holes A and C to create two piles separated by a central depression. This depression will then be filled with aggregate and grouted from the invert to crown from Hole B. Hole C will be equipped with a camera system to monitor the grouting process. This will create a 10-12 m long plug capable of holding back the estimated ~520 KPa of hydrostatic head.

Once the grout plug is established, the decline will be dewatered down to a level below the grout plug at which point a permanent engineered bulkhead (see Figure 12) will be installed and the decline will continue to be dewatered down to the mining level. A more detailed description of the grout plug and concrete bulkhead design and installation is contained in the WJEL Grout Report.

Figure 11 - Location of Proposed Grout Plug Holes

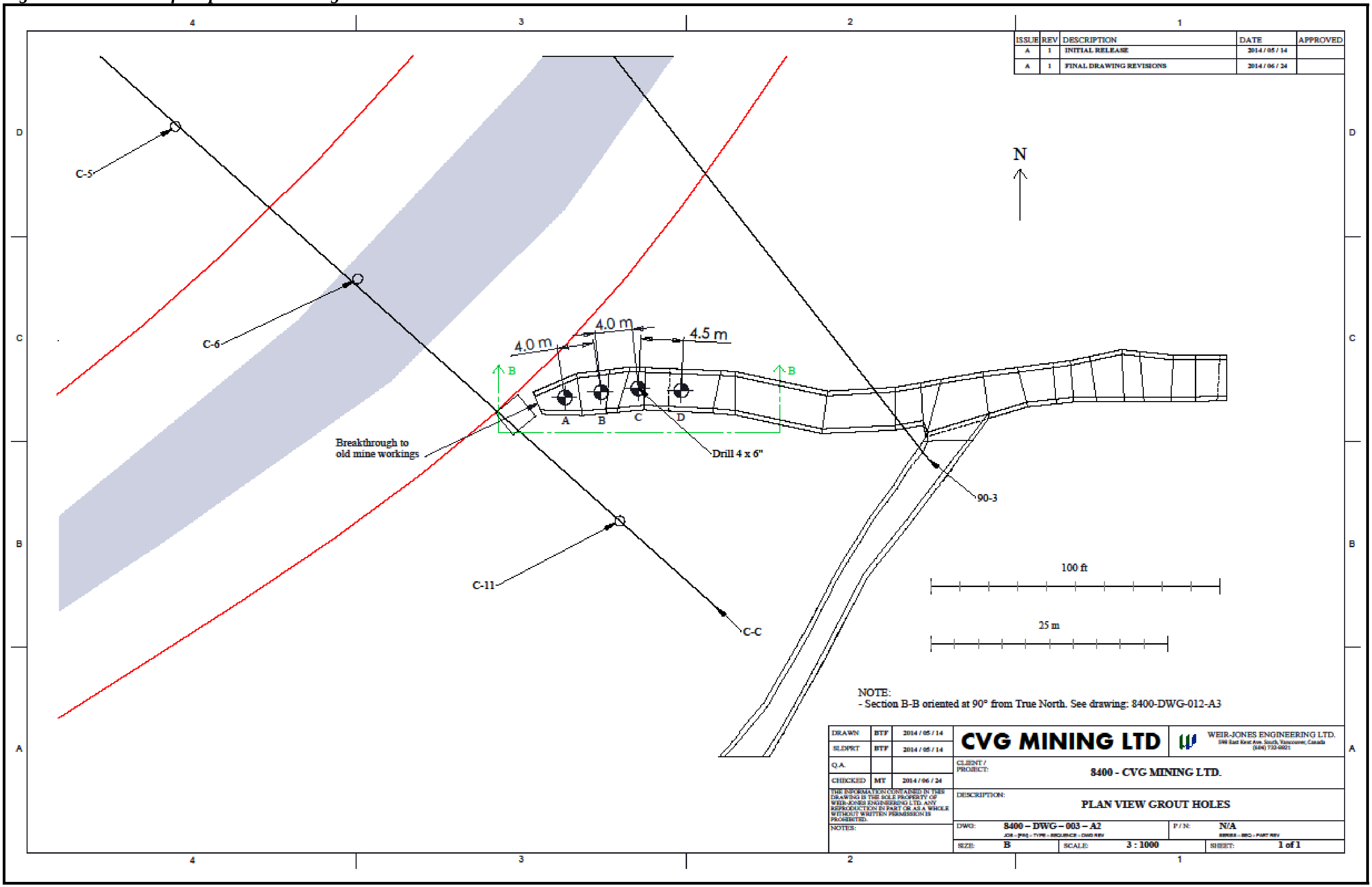
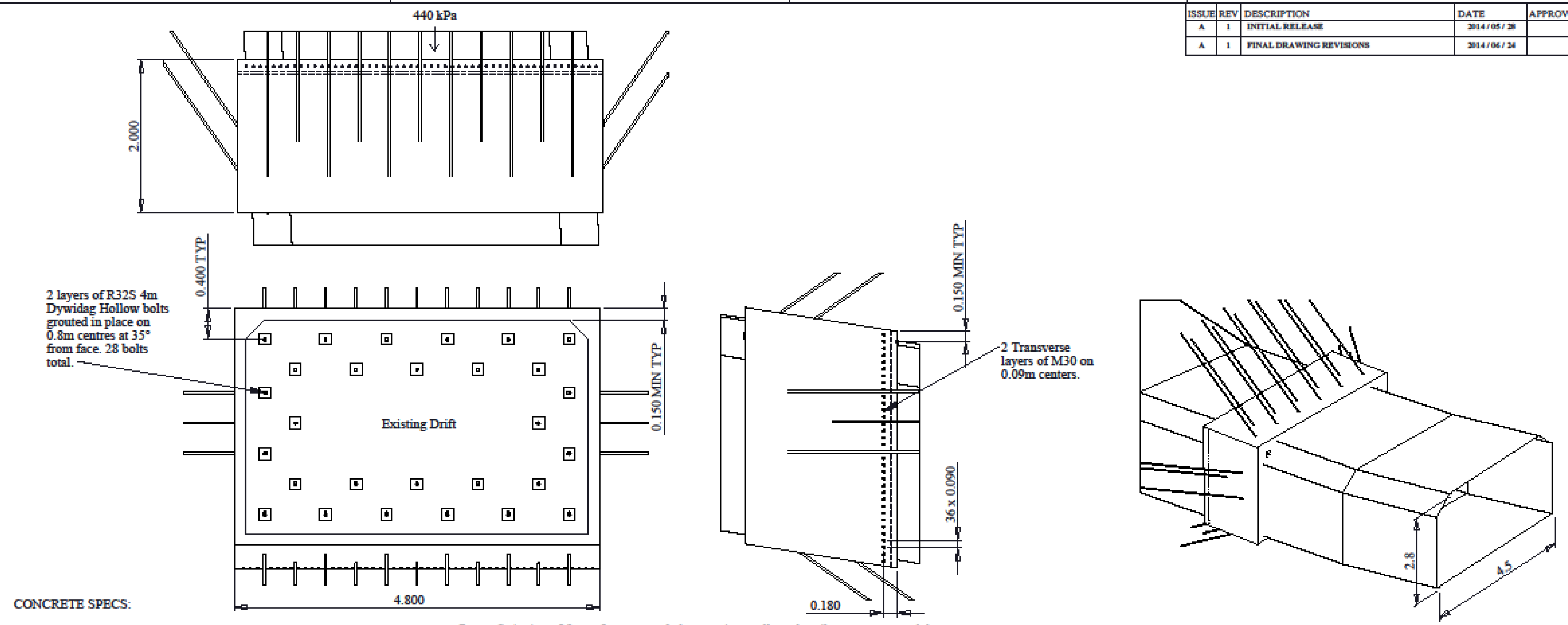


Figure 12 - Engineered Bulkhead to Permanently Seal Off Connection to Old Works

ISSUE	REV	DESCRIPTION	DATE	APPROVED
A	1	INITIAL RELEASE	2014 / 05 / 28	
A	1	FINAL DRAWING REVISIONS	2014 / 06 / 24	



CONCRETE SPECS:

1. All concrete shall conform to the requirements of CAN/CSA-A23.1-01. Cement shall be Type GU Portland Cement unless noted otherwise. Normal weight concrete for various purposes shall be as follows:
   1. Min 28 day strength MPa (psi): 35 (5000)
   2. Max slump mm (in): 75 (3")
   3. Max size aggregate mm (in): 20 (3/4")
   4. Air content %: 4 +/- 1
2. The General Contractor and Supplier shall determine slump and aggregate size to meet placement, finishing requirements without segregation, and Owner's specs
3. The unit weight of concrete shall be 24 kN/m3 unless noted otherwise
4. Submit mix designs to the Engineer and testing agency for review and approval prior to placement. Mix design submittals shall identify the elements for which they are intended.
5. Perform all work in accordance with CAN/CSA-A23.1-04 including the following:
   - Construction tolerances
   - Fabrication and placement of reinforcing
   - Placement of concrete, including proper vibration and curing
6. All hot and cold weather concrete work shall be carried out in accordance with CAN/CSA=A23.1, latest edition. Ensure ground temperature is above 5°C.
7. Stripping of forms for structural elements is not allowed until concrete strength has reached 50% of specified compressive strength
8. The General Contractor, in conjunction with the concrete supplier, shall meet the placing and finishing site requirements as well as the Owner'sspecified performance requirements for plastic and hardened mix properties. The General Contractor shall meet all documentation and quality control requirements as per the "performance" alternate of Table of CSA/CAN-A23.1-04
9. The Supplier shall be certified and meet all documentation requirements as per the "performance" alternate of Table 5 of CAN/CSA-A23.1-04
10. The Supplier shall provide test results for each proposed mix design at the request of the Owner. Test results shall meet the requirements specified for strength, durability, and shrinkage
11. The Supplier shall provide accelerated strength test data or alternative acceptable documentation for each proposed mix design for 56 day strength specs at the Owner's request. Test results or alternative documentation shall be used to evaluate anticipated 56 day strenght of the mix as placed on site within 14 days of placement
12. Take measures to minimize shrinkage cracking including covering and dampening concrete
13. The maximum water/cement ratio and air content shall be in accordance with Tables 2, 4, and 20 of CAN/CSA-A23.1-04

NOTE:
- ALL DIMENSIONS ARE IN METRES UNLESS OTHERWISE SPECIFIED
- 36pc rebar 30mm x 4.8m
- 52pc rebar 30mm x 3.1m
- 28pc rock bolts 32mm x 4.0m
- Volume of concrete: 29.5 m^3

DRAWN	BTF	2014 / 05 / 28	CVG MINING LTD
SLDPRT	BTF	2014 / 05 / 28	WEIR-JONES ENGINEERING LTD.
Q.A.			CLIENT / PROJECT: 8400 - CVG MINING LTD.
CHECKED	MT	2014 / 06 / 24	DESCRIPTION: PROPOSED BULKHEAD DESIGN
THE INFORMATION CONTAINED IN THIS DRAWING IS THE SOLE PROPERTY OF WEIR-JONES ENGINEERING LTD. ANY REPRODUCTION IN PART OR AS A WHOLE WITHOUT WRITTEN PERMISSION IS PROHIBITED.			DWG: 8400 – DWG – 013 – A2; P / N: N/A
NOTES:			SIZE: B; SCALE: 1 : 50; SHEET: 1 of 1

As the decline is being dewatered, it will be inspected and rehabilitated with additional ground support as needed. Work in the decline will be supported with what is anticipated to be a 75HP ventilation fan and ducting established temporarily at the portal (see Section “Ventilation”). Temporary power, compressed air and other services, as needed, will also be installed in the decline from surface to support these initial activities.

10.4 Groundwater Management

The Wingdam Mine is currently flooded. Piteau (1986) described the hydrological setting at Wingdam as consisting of two aquifers: a confined lower aquifer in the basal gravels and overlying slum materials and an unconfined upper aquifer in alluvial gravels connected to Lightning Creek. The aquifers are separated by a thick layer of relatively impervious clay and glacial till. Piteau estimated steady-state natural ground water flow velocities through the basal aquifer to be on the order of 0.3 to 1.5 m/d based on drilling, pump tests, and push/pull tracer tests. The groundwater temperature in the basal aquifer is approximately 6° C +/- 2°C depending on the season. For this temperature range, effective ground freezing requires that ground water velocities be maintained at <0.5 m/day. Groundwater velocities higher than this could result in unfrozen “windows” forming in the ice structure potentially reducing its structural integrity.

One method of reducing/controlling the ground water velocity is to fully dewater the mine horizon. Piteau determined in their 1986 report that the lower aquifer is hydraulically connected to the old mine works consisting of the Melvin shaft, Australian Deep Lead drift and Sanderson mine. Previous experience on site indicates that approximately 750,000 m^3 of groundwater is required to be pumped from the lower aquifer and connected network of old mine workings in order to draw down the local ground water table to the proposed mining horizon. Furthermore, when the water reaches this level approximately 3,300 m^3/d (600 USgpm) needs to be pumped on average to maintain a dry mine.

Not only is dewatering the mine horizon energy intensive and costly, it has undesirable potential side effects as follows:

1. Many of the historic monitoring wells drilled into the basal aquifer channel are artesian and free flowing. As they were historically not capped, in some cases this artesian ground water flow has enhanced the surface waters in the area of around a well, creating a wetland (see “Surface Wetlands”). Dewatering the basal aquifer leads to a temporary reduction of the hydraulic head in these wells, which in turn results in temporary cessation of the artesian flow from these wells; and,

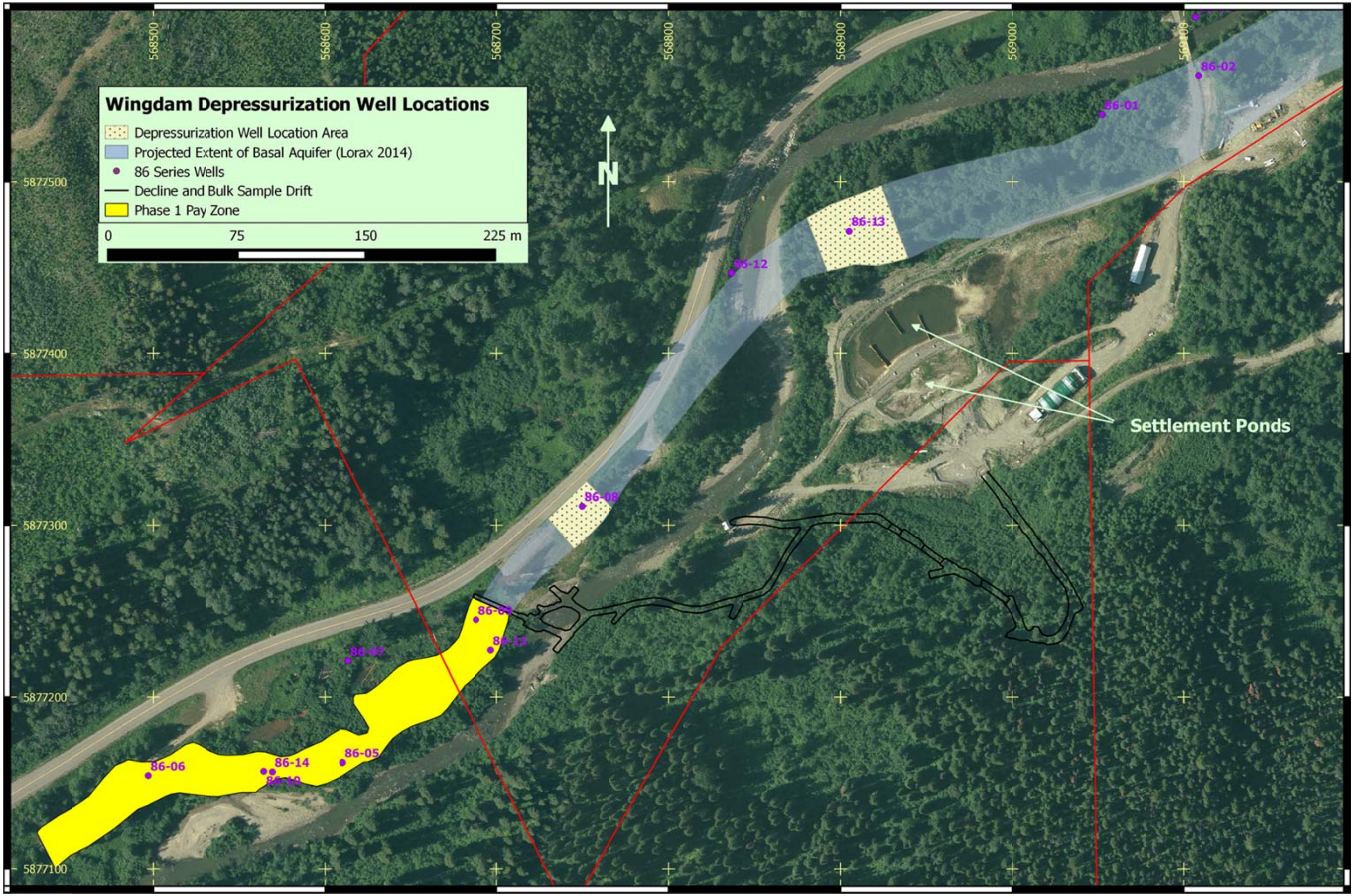
2. By dewatering the mine horizon, the hydraulic head in the area to be mined and upstream to the Sanderson mine is reduced by up to 55 meters (~540 KPa). This reduction in hydraulic head could cause ground instability leading to possible ground movement at surface (see “Surface Disturbance”).

3. Hydraulic gradient steepened in the vicinity of the proposed workings leading to pole line ground freezing problems.

CVG engaged the services of Lorax to assist it in identifying and developing a viable water management plan that would maximize the efficacy of the freeze plan, enhance mine safety, minimize any potential for ground movement at surface and minimize environmental impacts. The results of this work are detailed in the Lorax (2014) report. Lorax first analyzed CVG's well monitoring data during the period the company was conducting its bulk sample program and actively pumping to dewater the mine between 2010 and 2012. Lorax was able to confirm, based on this data analysis, that the lower basal aquifer is confined and has no direct connection to the upper unconfined aquifer of Lighting Creek under normal conditions. One exception is when a direct connection was established between the surface aquifer and the mine workings in May 2011 through the remobilization of a pre-existing sink hole in the vicinity of the mine workings. However, this issue was addressed by CVG's surface remediation efforts described as set out in the Clifton (2011) report and the construction of three engineered underground bulkheads. Following this event, CVG installed two monitoring wells in the surface aquifer (SA-1 and SA-2). The mine was maintained in a dewatered state for approximately one year after the sinkhole remobilization (until July 2012), and monitoring data to January 2014 indicate that the surface and basal aquifers have remained hydraulically isolated.

Using this information and a 3D model of the paleo-channel developed by CVG and WJEL, Lorax developed a computerized groundwater model to demonstrate the feasibility of depressurizing rather than dewatering the basal aquifer sufficient to maintain a zero hydraulic gradient and therefore minimize the groundwater flow velocity over the mine area. To depressurize the confined basal aquifer a pumping well will be drilled and completed from surface to the basal aquifer upstream of the mine area in the vicinity of the settlement ponds near existing monitoring well 86-13 (see Figure 14). Results of the model show that pumping this well at between 950 and 1100 m^3/day (175 and 200 USgpm) depending on season, will sufficiently depressurize the basal aquifer to maintain a <0.5m hydraulic gradient and therefore near zero ground water velocity over the mine area.

Figure 13 - Location of Depressurization Wells



It is important to note that Piteau (1987) reported having to locally minimize the hydraulic gradient in the basal aquifer in order to complete their push/pull tracer tests in October 1987. They accomplished this by controlling the flow rate from well 87-02 until it was within a few millimeters of the water levels at downstream wells (87-01-P2, 87-04, 87-06, and 87-07). Piteau reports that “[a] flow rate of 1.4 L/s (120 m^3/d) from the interception well was required to effectively reduce the natural hydraulic gradient across the site to zero.” Although this is significantly lower than the flow rates predicted by the Lorax groundwater model it supports the conservative results of the model. Furthermore, it demonstrates that this approach can be used to establish and maintain a near-zero gradient to minimize groundwater velocity over the proposed Mine Area.

Although a 6 inch diameter 10-15 hp pump installed in an 8 inch well casing would be capable of delivering the required flow rates, CVG plans to drill and complete a 12 inch well to accommodate higher pumping rates if needed. Discharge from this well will report to the existing settlement ponds in accordance with *Environmental Management Act* Permit No. 11088.

Although this strategy of reducing the hydraulic gradient is not expected to have any significant effect on surface ground stability, it is possible that the artesian flow from the historical wells could be temporarily reduced or eliminated during depressurization of the basal aquifer (see “Surface Wetlands”). Omineca Management has discussed this situation with the Department of Fisheries and Oceans (“DFO”) and has developed a plan to mitigate any reduction in artesian groundwater flow to the wetlands by diverting a portion of the groundwater being pumped to reduce the hydraulic gradient to a suitable location(s) in the wetlands.

To facilitate replacing the artesian groundwater flow into the wetlands, CVG Mining intends to drill and complete a second pumping well immediately upstream of the mine area (see Figure 14) and pump the “make-up artesian water” volumes needed to maintain water levels in the wetlands. The balance of pumping needed to depressurize the lower aquifer will be pumped from the upstream main well and discharged into the existing permitted settlement ponds.

As described below in Section “Gold Processing and Production” CVG Mining intends to conduct primary screening of the pay gravels into two size fractions underground and then produce a gravity concentrate. This concentrate will be sent to surface for secondary processing to extract the gold. CVG Mining plans to use approximately 550 – 850 m^3/day (100-150 US gpm) of water produced from mining operations in its underground primary screening operations. The wash water from this screening operation that requires settling will be pumped out of the mine and discharged into the settlement ponds. Secondary processing of the gravity concentrate at surface will source water from, and discharge water to, the settlement ponds.

CVG Mining’s Ministry of Environment Permit No 11088 allows the Wingdam operation to discharge up to 11,000 m^3/d, through the existing settling pond system when actively mining, which is reduced to 6000 m^3 /day, during fish spawning season (defined as August 1 to April 15) when pay gravel is being processed to separate gold. As described above water discharged from the Wingdam mining operations will be primarily through the combination of depressurization pumping and mine/process water increased by mine site surface drainage during period of precipitation/snowmelt. In total, discharge from the Wingdam mine operations is expected to average between 1,500 to 2,000 m^3/day (275-370 US gpm) and typically not exceed 3,000 m^3/day (550 US gpm).

The existing water handling system on surface consists of two settlement ponds as shown in Figure 14 which also shows the surface infrastructure. Curtains are installed in the second larger pond to increase residence time which previous experience has shown to be adequate to treat the threshold volumes allowed under the permit. Other than groundwater pumped to maintain the wetlands, all water whether surface discharged from the disturbed area of the mine or pumped from the underground mine works will report to the permitted settlement pond for treatment. Pumping rates and water quality discharge limitations will be monitored and maintained at rates and amounts allowed on the Ministry of Environment Permit No. 11088.

10.5 Groundwater Monitoring

Although all dewatering activities are expected to fall well below the existing permitted values CVG will install flow meters and totalizers to routinely monitor all discharges from the mine on a daily basis. Samples of the discharge and receiving waters will be routinely taken, analyzed and reported in accordance with the mine permit.

Additionally, CVG Mining intends to install piezometers and data loggers in an array of approximately 10-12 wells over the downstream length of mine area to record groundwater levels at a minimum frequency of 15 minutes. The data loggers will be downloaded on a regular basis and results monitored to ensure that the hydraulic gradient across the mine site is being maintained at <0.5m. A quasi real-time record of piezometric data will be maintained which will be correlated to flow data from the various pump discharges.

10.6 Footwall Drift

From the base of the existing decline, a 5m wide x 6m high footwall drift will be extended within bedrock downstream parallel to the paleo-channel to provide access to the pay gravels and create a drilling gallery from which two rows of horizontal freeze pipes will be installed across the channel (see "Ground Freezing"). The ~6 m height of the drift is required to allow sufficient room for the installation of the freeze system, movement of equipment, installation of hung ventilation tubing and other mine services. The drift will be advanced in two stages: initially a 4m high drift will be advanced leaving a "bench" to support the freeze hole drills. When these are completed 2m of rock will be removed from the invert.

Mechanical testing (Weir-Jones 2014) of the surface and cored bedrock samples has shown the uniaxial compressive strength ("UCS") of the bedrock material to be highly variable. The average of the tests on surface material conducted on three sample ranged from between 15.4 and 43.5 MPa with a high of 72.2 MPa and an overall average of all tests of 31.9 MPa perpendicular to the geological bedding plane; and, between 7.8 and 16.5 MPa with a high of 25.8 MPa and an overall average of all tests of 12.4 MPa parallel to the geological bedding plane. The relatively low UCS values and typical orientation of the geological bedding of the of the bedrock material make it suitable for mining with small/medium sized mechanized mining/tunneling equipment which typically are capable of working in rock having an UCS of up to 60 – 70 MPa.

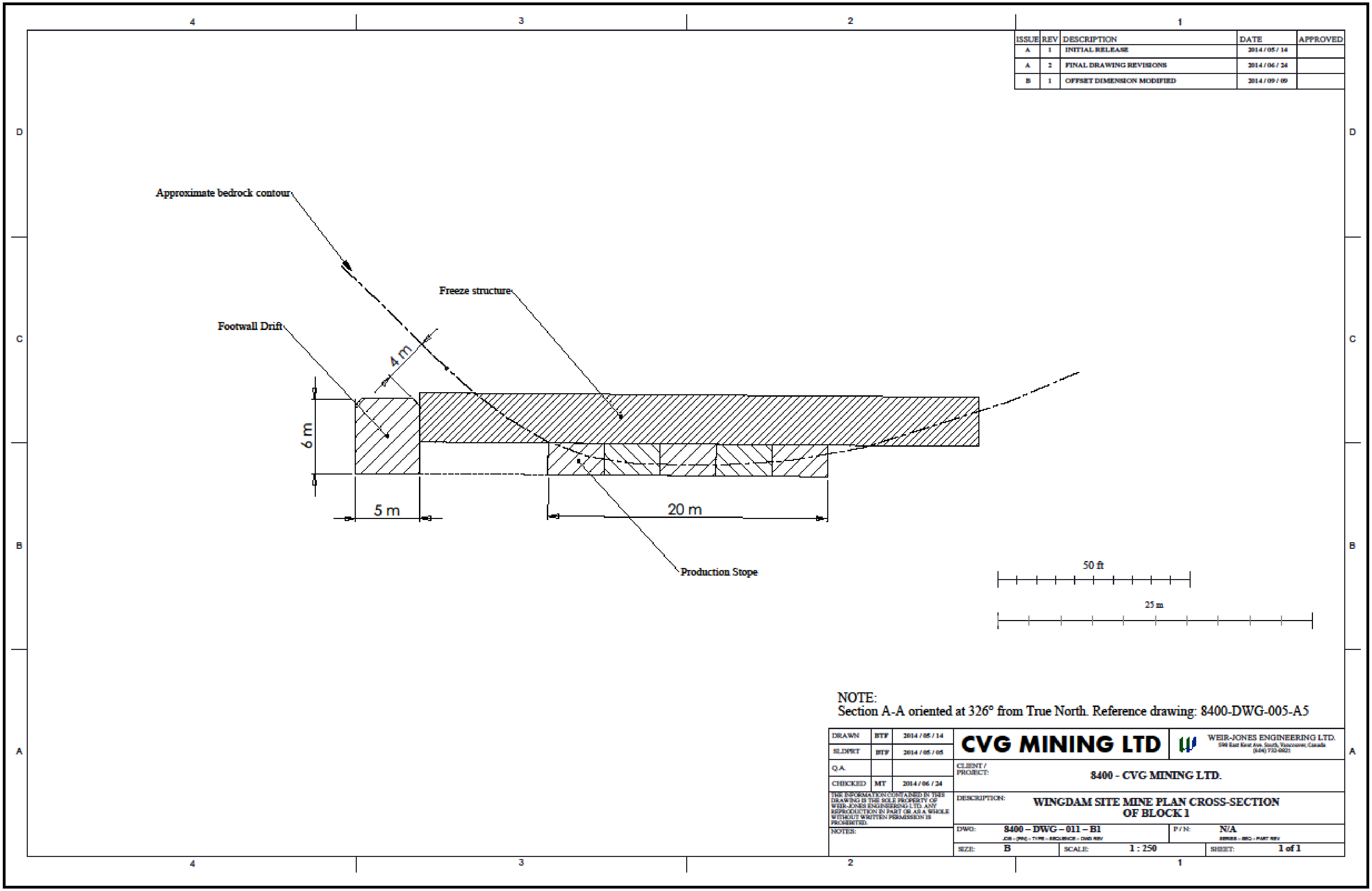
The UCS values obtained from testing three core samples were significantly higher than those of the surficial material, with values as high as 180 MPa perpendicular to the mineralized foliation planes, and 45 MPa parallel to them. The higher strength sections may require the adoption of a conventional drill and blast tunnelling procedure.

The advance rates for the footwall drift will clearly be faster in softer material and slower at the upper end of the UCS range; however, on average, management is projecting an advance rate of the footwall drift of $6m^3$ per hour. The mechanized approach to mining may be augmented with conventional drill/blast/muck techniques when stronger ground conditions are encountered.

During advance, the final floor elevation of the footwall drift will be maintained a distance of ~ 1 meter below the keel (lowest point) of the adjacent paleo-channel. For safety, probe holes will be drilled with extension steel in advance of drifting at frequent intervals to ensure that the footwall drift is maintained a minimum distance of 4m from the channel gravel/bedrock contact (see Figure 15). On a regular basis, a probe hole will be core drilled and the cores will be reviewed by the Mine Engineer and geologist to determine if any potential ground control or waste rock environmental issues exist ahead of drifting. The footwall drift will remain open during active mining and accordingly conventional ground support will be installed (see “Ground Control”).

The entire approximately 300m length of the footwall drift is expected to produce a mined volume of 10,200 BCM (27,500 tonnes at a 2.7 S.G) of waste rock which will be removed to surface and spoiled on the project’s permitted waste rock dump. Note that this estimate is based upon the concept of a 2° decline angle from the pre 2015 seismic interpretations. Recent seismic has shown that the footwall drift will need to follow the channel profile where possible and therefore the footwall drift invert elevation will need to follow the elevation changes of the paleo channel –See Recommendation 1 & 4.

Figure 14 - Typical Cross Section of Mined Channel Based on Seismic Information from 2014.



10.7 Ground-Freezing

In most instances ground-freezing systems consist of a closed system of pipes, valves and couplings through which a chilled brine is circulated to extract heat out of the surrounding materials to form a frozen underground structure capable of providing increased support strength, stabilizing the unconsolidated materials for mining and providing a hydraulic barrier to minimize groundwater flow into the mine. To be effective, groundwater flow around the frozen structure must be minimized or melting resulting from heat transfer from the groundwater may cause “windows” to form in the frozen structure compromising its strength and impermeability. A further key consideration in the design and implementation of a ground-freezing system is to minimize the risk of the circulating brine leaking from the system. Any leakage of brine into the surrounding materials could also cause “windows” to form in the freeze structure, again compromising its strength and impermeability.

CVG’s current ground-freezing plan was developed by Weir-Jones Engineering Ltd. (WJEL) and is based on previously gathered data and material strength assumptions. Details of the ground-freezing plan are set out in the WJEL Conceptual Mine Plan Report (2014). In developing the freeze plan, WJEL assumed a UCS value of 7 MPa and a Young’s Modulus of 369 MPa for the frozen slum based on earlier test work by MDH Engineered Solutions within the Fortis Report and has further assumed that groundwater velocity will not exceed 0.5m/day (see section “Groundwater Management”).

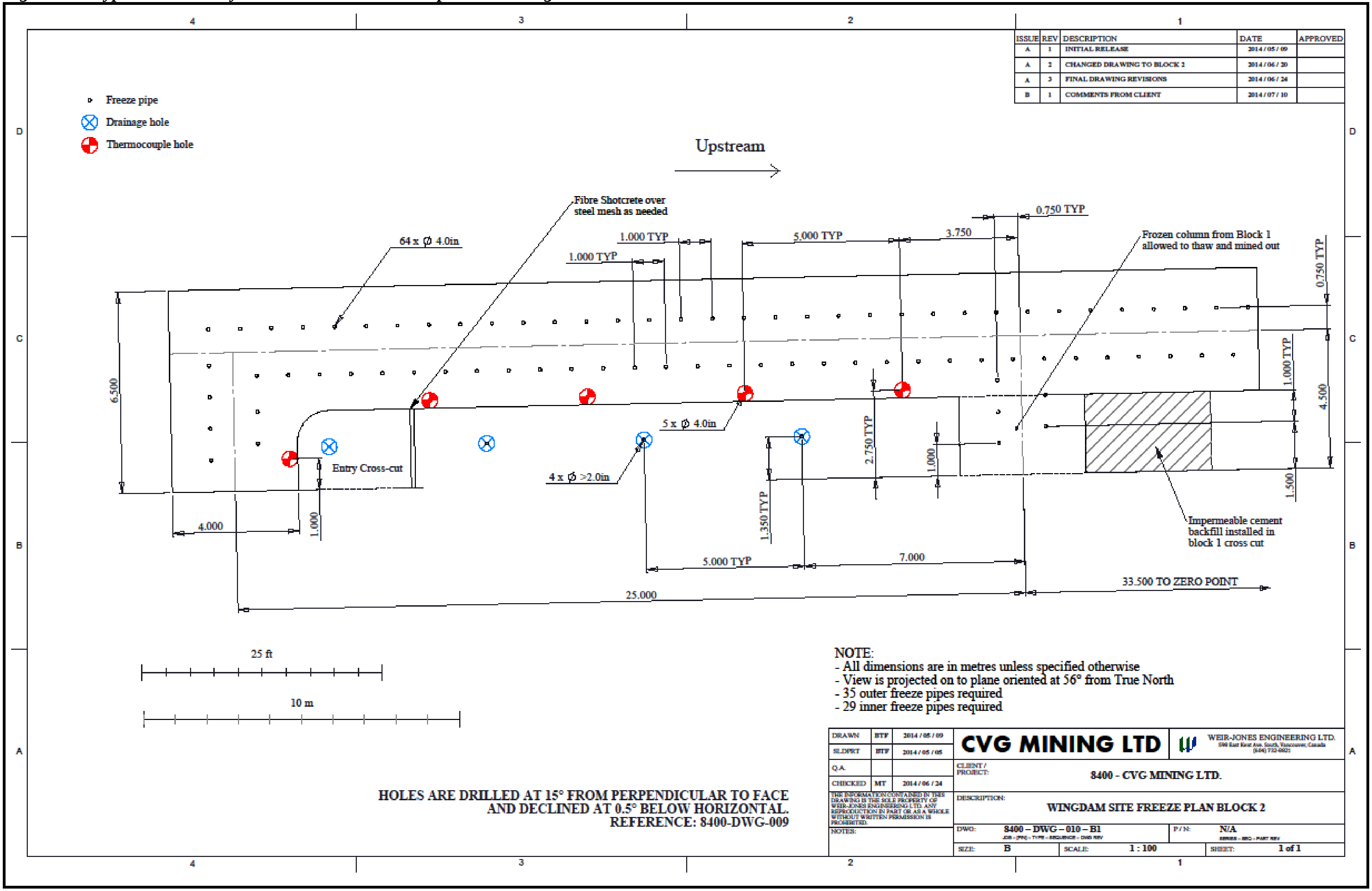
Samples collected from the drilling program in 2015, and subsequently frozen and tested, show a variability in material properties, but similar results. Whenever possible, new material from the reconnaissance drilling should be tested to ensure inconsistent material properties are identified as early as possible – See Recommendation 3 Further Testing.

Accordingly, the final spacing of the freeze pipes and size of frozen panels is subject to some change to incorporate the geophysical survey results from 2015, the 2015 laboratory testing programme, the recommended additional testing, and site conditions encountered as the footwall drift is mined.

The proposed freeze structure is 4m thick and is shaped like an extended, shallow inverted U designed to enclose and isolate a mining block beneath the freeze structure. The freeze structure is designed to support the weight of overburden and withstand the surrounding water pressure. The freezing system is designed to sufficiently freeze this structure enabling the mining block to be entered after approximately 30 days of freezing.

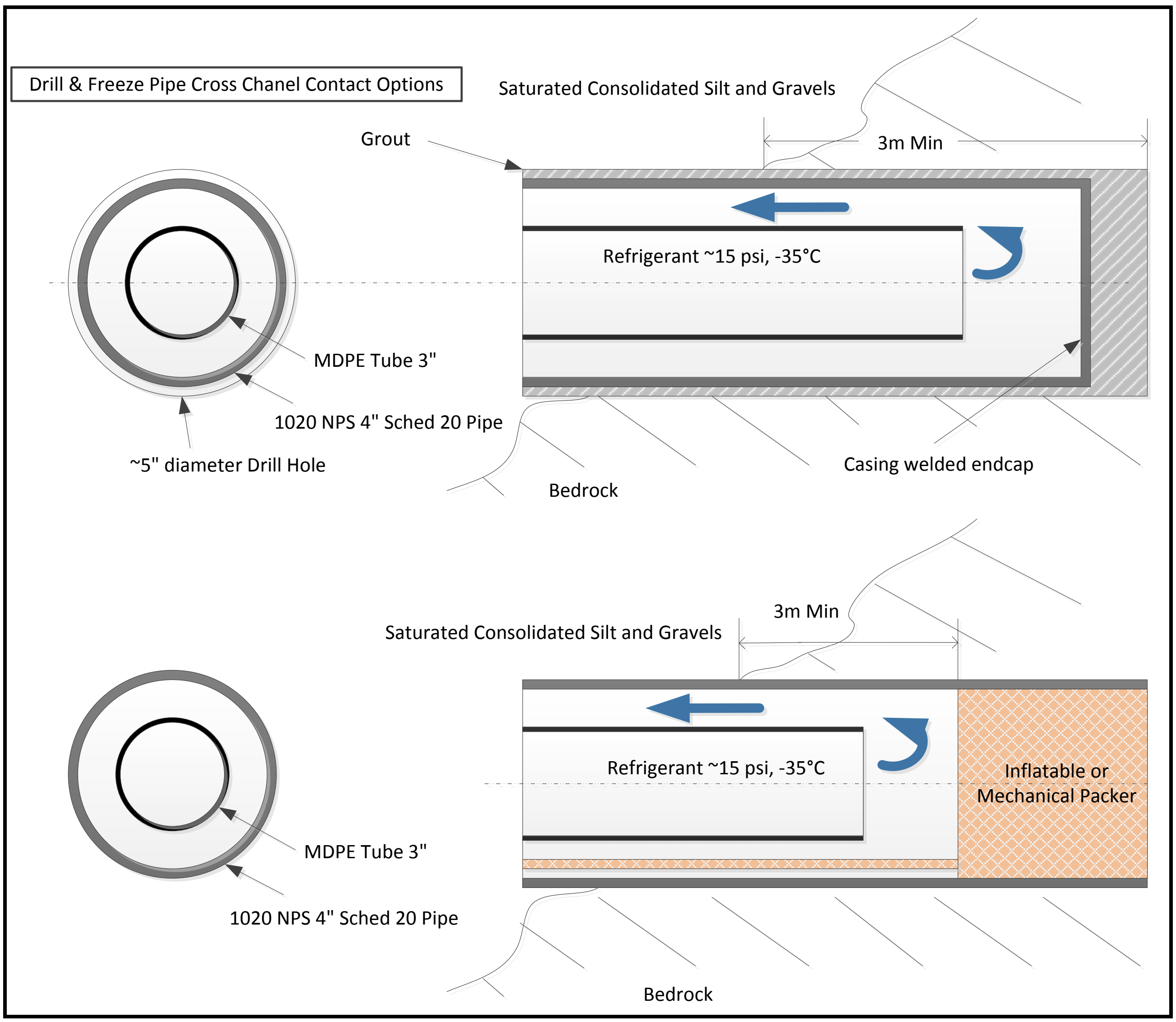
Once ground water velocity in the paleo-channel has been minimized CVG Mining will begin ground-freezing to stabilize the slum and materials overlying the pay gravels. Approximately ~70 - 4” diameter horizontal freeze pipes will be drilled and installed to freeze a structure over mining block. The freeze pipes will be located in 2 rows ~1m apart in the pattern shown in Figure 16. The freeze pipes will be collared in bedrock in the footwall drift and drilled horizontally across the channel to a distance of ~ 3m into competent bedrock in the opposite side of the channel. This will ensure that a hydraulic seal is created along the sidewall contacts between the freeze structure and the bedrock. All drifting will be done to maintain the orientation of the Pay Zone in the upper 1.5 m of the production drifts (see section “Mine Production”). Accordingly, the 2 freeze holes drilled in the bottom of each pillar of the freeze structure will be in 100% bedrock extending the hydraulic seal into the channel bottom bedrock (on the upstream and downstream edges of the freeze structure) to act as a barrier to possible groundwater flow through fractured bedrock into the mine workings.

Figure 15 - Typical Freeze Layout and Structure to Encapsulate Mining Block



The width of the footwall drift will allow the freeze pipes to be advanced in 3m sections. To reduce the risk of brine leakage from the freeze pipes, CVG Mining intends to advance the freeze pipes by welding new pipe sections on to the pipe being installed. An alternative method is to advance threaded pipe sections applying suitable sealant to the threads. Each freeze pipe will be sealed away from the footwall drift. CVG Mining intends to weld an "end cap" in place but an alternative method is to insert a mechanical packer after drilling and prior to installing the freeze pipes as shown in Figure 17. The efficacy and cost- effectiveness of each of the foregoing methods will be further explored with drilling contractors before a final method is chosen. Freeze pipe performance will be monitored during the freezing process through pressure gauges at the freeze pipe header. All freeze lines will be pressure tested prior to circulating brine.

Figure 16 - Freeze Pipe Drill Hole Detail



It is critical for the integrity and safety of the frozen barrier that the freeze pipes be drilled to maintain their relative spacing as they are drilled across the channel. Drill holes will need to be surveyed post drilling to locate their “as built” conditions and if too far out of acceptable variances, re-drilled. Once installed, freeze pipes will be tested to 60 PSI pressure for 12 hours to ensure that there are no leaks in the piping system.

Initial calculations (Figure 17) indicate that a freeze plant with a capacity of up to 1800kW$_R$ (500 Tons/hr) will be required to adequately freeze and maintain freezing during the first phase of mining. Daily freeze energy demand will vary between 800kW and 1700kW (230-480 Tons/hr) once the mining cycle is maximized. CVG Mining has determined that the most cost effective way to deliver this freeze energy requirement is through the use of ammonia absorption chillers (see section “Mine Energy Considerations”). CVG Mining intends to install two absorption chillers each with a capacity of 900 kW (250 Tons/hr) to provide redundancy and also operational efficiency and flexibility. These freeze plants will be installed at the surface a safe distance away from the mine ventilation system and portal as specified by the Mine Engineer. Chilled refrigerant will be piped underground either through a cased inclined borehole or down the new inclined access raise.

Figure 17 - Expected Daily Freeze Energy Requirement in Phase 1 Based on 2014 Seismic Interpretation

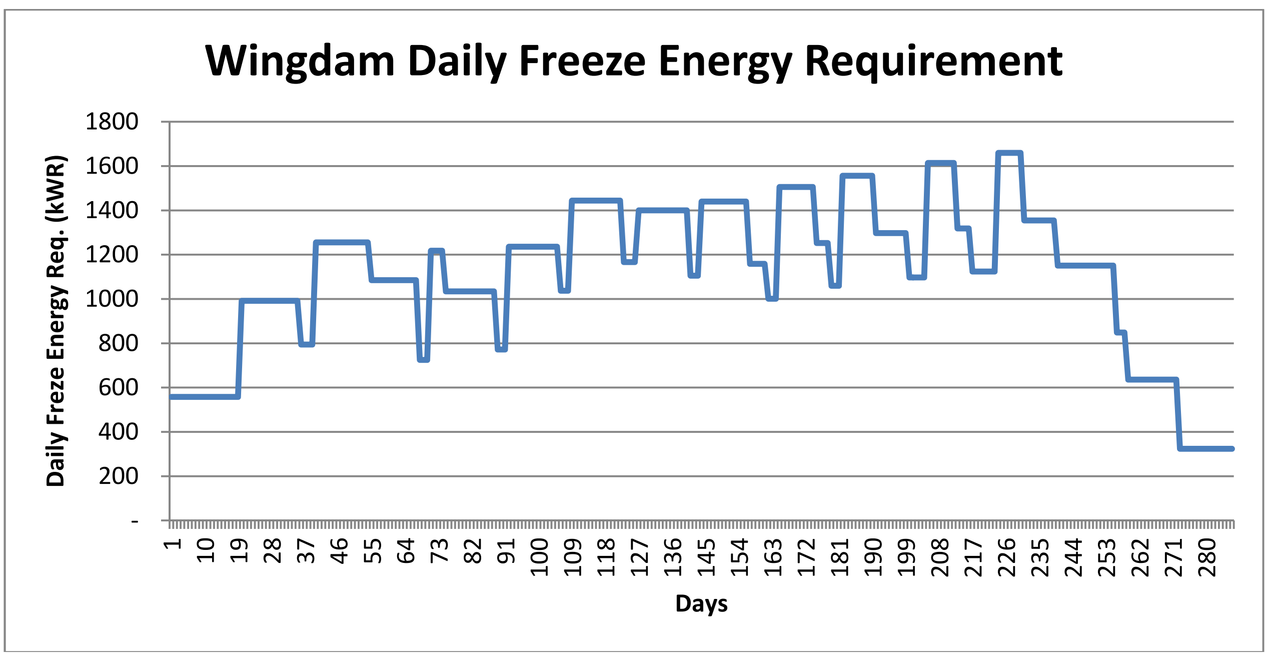
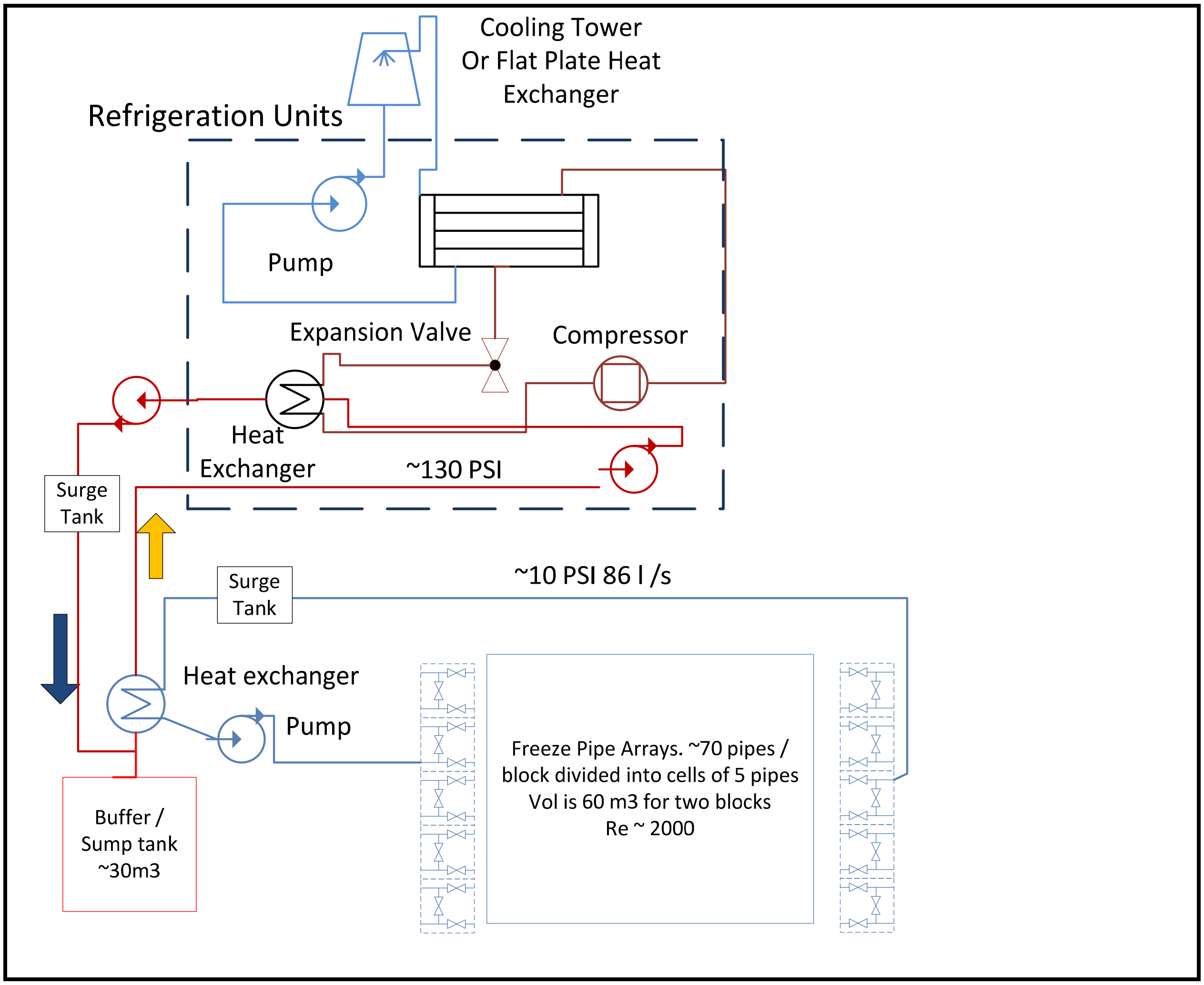


Figure 18 - Freeze Plan Schematic



Dynalene HC (a Potassium Formate mixture) is the intended refrigerant and has been selected because it is a biodegradable, non-flammable, water based, non-toxic material that is less corrosive than Calcium Chloride brine. It is used in the food and beverage industry for cooling and has been studied for ingestion toxicology. Weir-Jones has used it on other freezing projects.

The refrigeration circuit design shown in Figure 18 contemplates circulating the potassium formate solution at -35ºC to -40ºC in a closed-loop high pressure ~130 psi circuit from the freeze plant on surface to a heat exchanger located underground. In turn, the heat exchanger will chill a potassium formate solution circulating in a closed-loop low pressure ~10 psi circuit through the network of freeze pipes to freeze the ground.

The high- and low-pressure circuits will be installed with an insulated tank to mitigate any surge events in the piping system. As a further precaution, the high-pressure circuit will be installed with an emergency buffer tank to allow for rapid dumping and containment of chilled brine from the circuit in the event of an emergency.

Once tested, freeze pipes will be connected to the low-pressure refrigerant supply piping in groups of 5 with flexible Victaulic hoses and headers as shown in Figure 19. Safety shut-off valves will be installed at the boundary of each panel and at the entry and exit of each set of 5 freeze pipes to enable isolation of freeze pipes for the purposes of bleeding off air bubbles trapped within the pipe or in the case of a rupture or emergency. These valves will be inspected regularly to ensure that they do not become inoperative due to ice/frost buildup on their surface.

Pressure and temperature of the refrigerant will be continuously monitored at multiple points and connected to an alarm system to ensure that any leaks or air locks are detected with minimal delay. All headers, pipes and tanks will be wrapped in plastic and insulated using sprayed rigid foam where possible, and as approved by the Mine Engineer.

The low-pressure freeze pipe circuit contains many pipe connections and is therefore at a higher risk of developing a leak. Separating this circuit from the high pressure supply circuit and operating it at low pressure together with the ability to isolate a section on this circuit minimizes the risk of injury to personnel resulting from a possible leak in this circuit.

Per the 2014 mine plan, the footwall drift has a downstream gradient of ~2°; any refrigerant leakage will collect at the face away from the panel entrances. Any spilled refrigerant will be recovered, cleaned and reused in practice from these points via standard fluid totes. If the geophysical interpretation of 2015 is confirmed through further drilling, and as the mine plan is revised, intermediate collection locations may need to be considered (see Recommendation 10).

Temperature readings will also be taken using instrumentation holes installed between the freeze pipes to monitor the development of the freeze wall to ensure its integrity (see “Freeze Monitoring”). The roof beam deflection will be monitored to confirm that the rate of creep and beam deflection are in accordance with the design criteria. Once the freeze structure is established, holes will be drilled from the footwall drift as needed to depressurize and dewater the mining block.

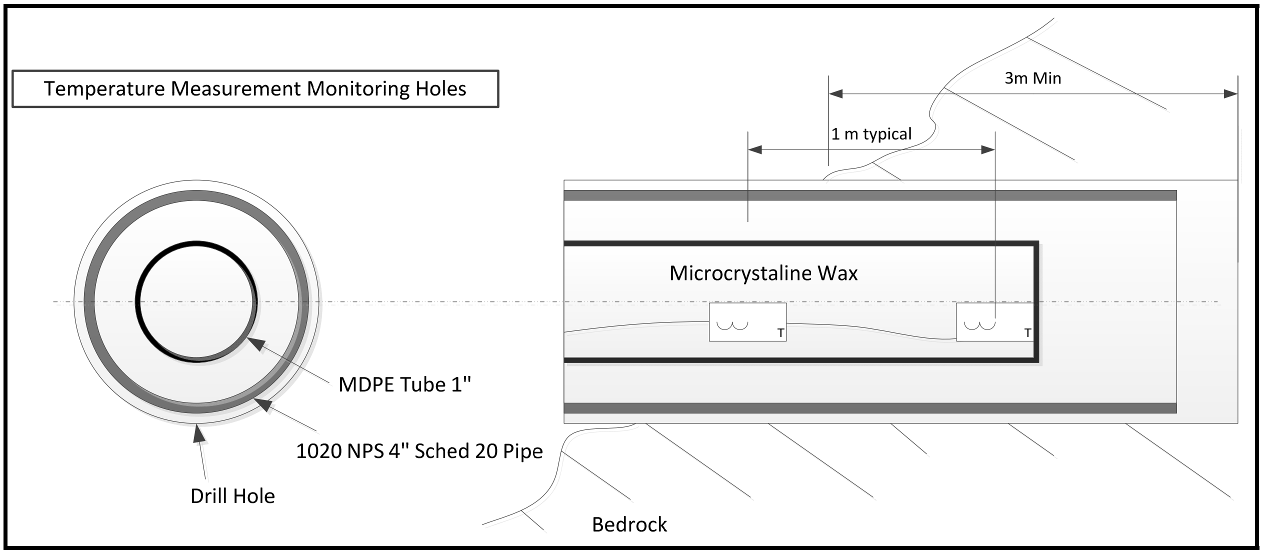
Before entering the mining block, the Mine Engineer must inspect the freeze plant operating logs, the temperature data, and the water flow rates from the drain holes to ensure that the freeze structure is sufficiently established and that there is no water recharge into the block occurring. The initial entry into each block also requires that the Mine engineer pick a suitable bulkhead location and fix the design.

As a final precaution, the mine plan is designed to limit the time the stopes remain open to mitigate the effects of creep in the frozen roof. The drift and fill process will ensure that only the production and the backfill stopes are open, keeping the unsupported freeze roof beam to ~ 4m.

10.8 Freeze Monitoring

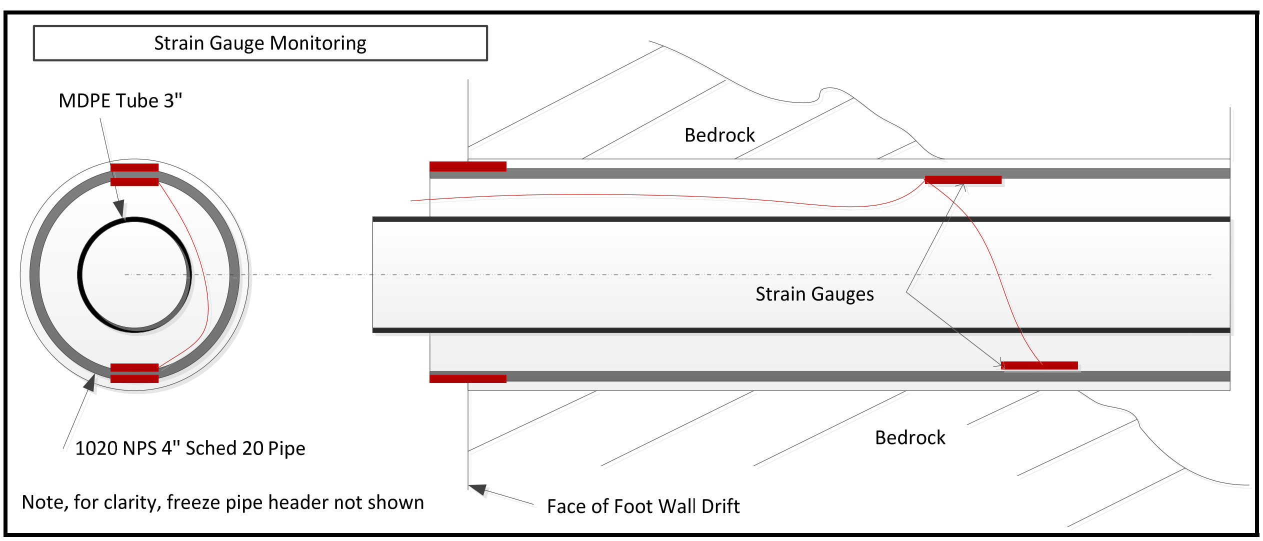
The refrigerant system temperatures and pressures will be monitored continuously throughout the freeze plant, the surge tanks, and at the freeze pipe headers. The entire system will be remotely monitored with near real-time (<2s) exception reporting and daily summary reports. These will be reviewed daily for both safety and system efficiency purposes. The growth of the frozen zone will be observed by installing thermistors at 0.3m spacing (see Figure 20) in horizontal monitoring holes drilled below the freeze pipe holes at ~5m centers (see Figure 16).

Figure 19 - Temperature Monitoring Holes



Strain gauges will also be installed on a freeze pipe nearest the centerline of the cross cut drifts at the collar and where the pipes pass through the bedrock/frozen ground boundary (see Figure 20) to monitor vertical shear deflections resulting from the potential for creep which could cause the pipes to deform plastically and/or rupture.

Figure 20 - Strain Monitoring



During the exploration and sampling of at least the first two blocks, frequent monitoring of the geomechanical instrumentation will be used to evaluate the roof deflection with time in the production crosscuts and the stopes. Although it is not anticipated that there will be any surface subsidence, a survey line will also be established and monitored daily using a series of permanent targets. The monitored zone will extend from 100m NE of the mine workings to 100m past the edge of the 1st phase mine block; subsequently it will progress with the mining in a SW direction.

Any observed flow of water in or around the frozen ground, or any thermal anomalies will be immediately reported to the Mine Engineer and management team, as well as the freeze plant operator. The freeze plant operator will be responsible for maintaining all records relating to the progress of the freezing operation, and making these available to mine manager in the form of daily reports of the freeze wall status. The data presentation will be fully automated.

Deflection monitoring of the back of the crosscut drift will be hampered by ice build-up and any screening or insulation installed. Therefore immediately after establishing the cross-cut drift and before the build-up of any significant frost, the newly exposed surface will be studied and mapped by the mine geologist for any visible defects before determining roof deflection measurement locations (and if necessary screening and support locations). Once measurement locations have been selected, short (<300mm) GRP dowels will be installed into the roof at intervals of no less than 3m. These dowels will support survey targets which will be used to monitor the vertical displacement of the back. Any necessary insulation subsequently applied to the back will not mask off these targets so that they remain clearly visible.

Initially, deflection measurements will be taken prior to each shift, plotted for reference, and provided to the Mine Manager and Mine Engineer. As creep rates of the frozen structure are better understood, the mine engineer may recommend reducing the frequency of deflection measurements. Deflections greater than 4mm or an accelerating deflection rate will be further investigated as a possible safety hazard by the Mine Engineer. Any additional ground support, as recommended by the mine engineer would be installed.

In addition to the foregoing detailed description of freeze monitoring, a routine program examining the mine workings in general will be instituted looking for indications of ground failure or inadequate ground support. Approximately quarterly, an independent ground control specialist will be brought to site to assess the ground control program.

10.9 Bulk Sample Production

After approximately 30 day freezing time, and only after data from the freeze monitoring system (see Section “Ground Freezing”) confirms that the freeze structure is sufficiently competent, and at the correct temperature to meet design requirements, will the Mine Engineer approve entering the mining panels.

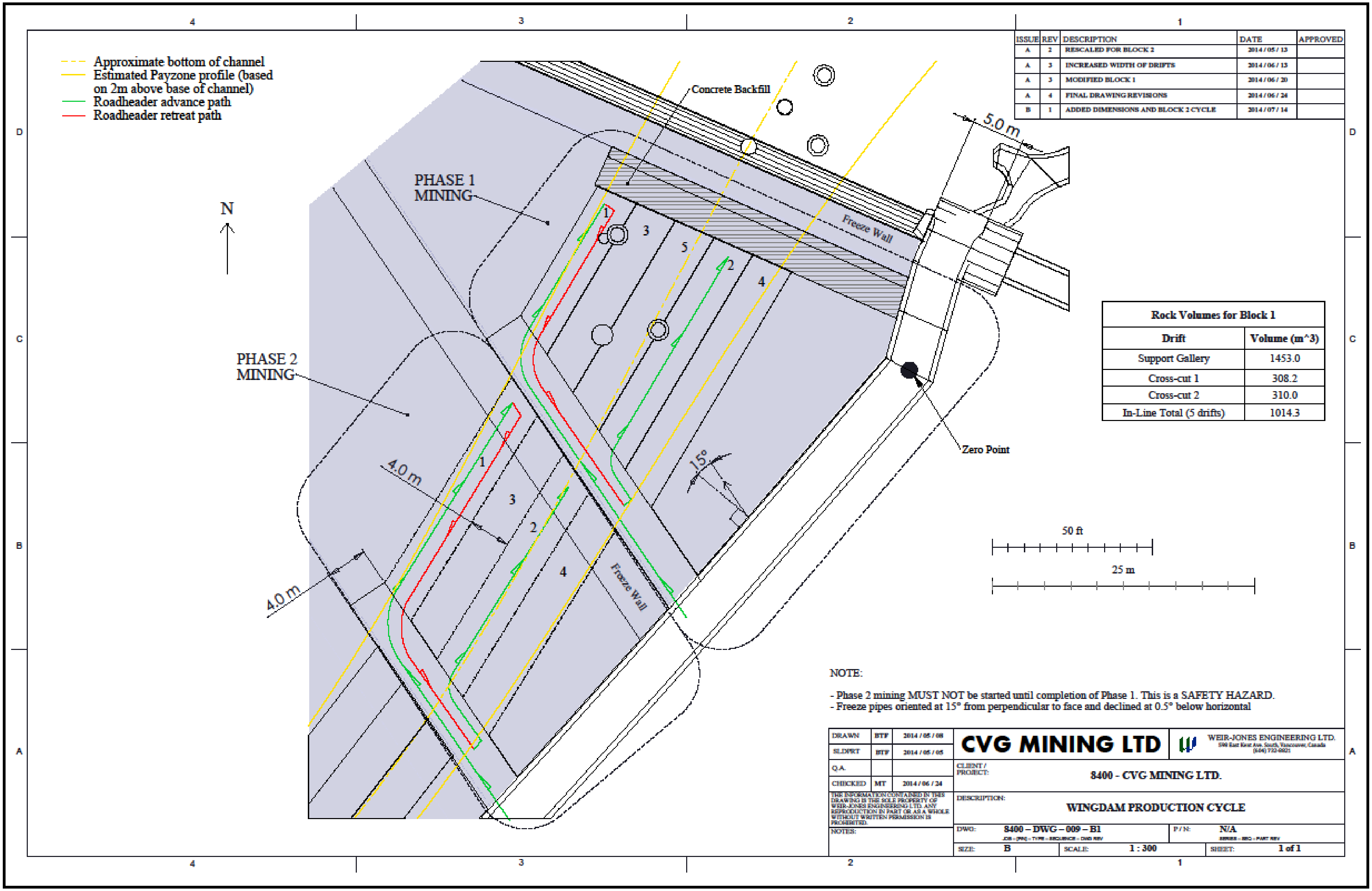
Initially the mining panels will be depressurized and dewatered through drainage holes drilled from the footwall drift and then entered via a production cross-cut driven from the footwall drift perpendicular to the channel’s downstream axis. The cross-cut drifts will be 4m wide x 2.5m high. The first mining block will be entered via two cross-cuts to accurately locate the channel boundaries and determine the gold grade distribution across the channel. In subsequent mining blocks only the downstream cross-cut will be necessary. The cross-cuts will be initially through bedrock and into the channel intersecting the deepest part of the Paleo-channel about 1-1.5 m above the floor of the cross-cut.

These cross-cuts will remain open for approximately 30 days and will be constantly monitored for deflection. Ground support will be added where needed to protect mine workers from localized spalling or where dewatered unconsolidated gravels are encountered in the sidewalls of the cross cut (see section “Ground Control”)

The placer channel will then be mined via a series of longitudinal drifts (stopes) typically 4m-wide x 2.5m-high x 25m-long driven parallel to the channel using mechanized mining equipment to extract the pay zone delineated by the grade distribution of the cross-cut drift. These stopes will be entered from the downstream cross-cut and driven upstream to allow for natural drainage of water from the face by gravity. The first stope will be driven along the edge of the pay zone farthest from the footwall drift. Alternating stopes will be mined progressively back towards the footwall drift leaving a ~ 4m wide pillar between them. As one stope is mined a previously mined stope will be backfilled with a cemented waste material. Subsequently, once the cemented backfill has gained sufficient strength, these pillars will be mined starting with the one farthest from the footwall drift and effectively retreat-mined back to the last pillar nearest to the footwall drift. The intended stope mining sequence for Blocks 1 and 2 is shown in Figure 22. This mining sequence reduces the potential for pillar deformation due to creep, and therefore reduces the flexural stresses on the freeze pipes at the bedrock to slum/gravel contact as the frozen roof beam deflects under the overburden load.

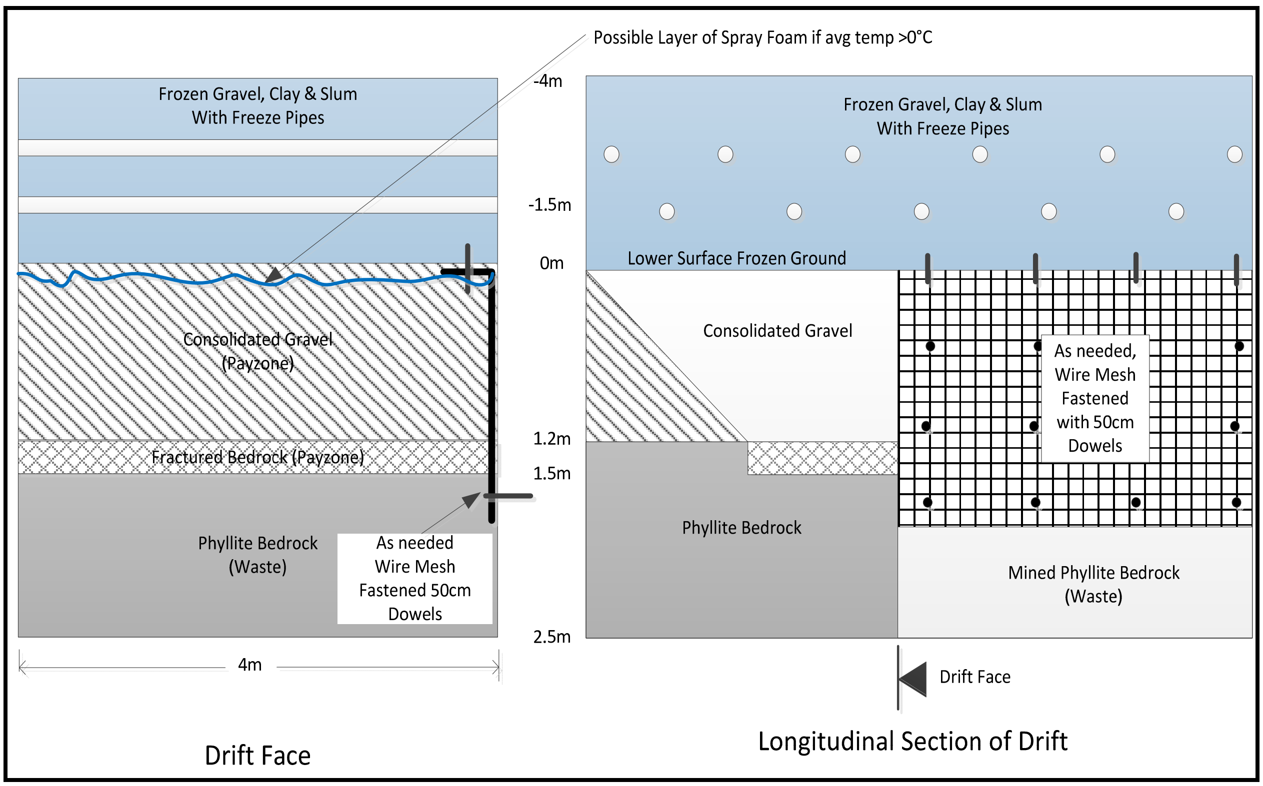
Note that significant variations in pillar height are not desirable. For this reason, as the geophysical interpretation of 2015 is confirmed, changes to the mining geometry to minimize the height of the pillars will be necessary (See Recommendations 1, 4, 5 & 6)

Figure 21 - Typical Mining Sequence Prior to 2015 Seismic Interpretation



All production drifting (cross cuts and stopes) will be carried out to maintain the bottom of the pay zone at an elevation ~1m above the floor of the drift. The face of the production stopes will be advanced in three steps: first the ~1.5m Pay Zone at the top of the face will be extracted; second, the bench formed by removing the Pay Zone and floor will be cleaned of any gold particles which may have fallen during removal of the Pay Zone; and third, the remaining bedrock block will be removed in larger pieces (see Figure 23). Once mined, the floor of the stope will be cleaned of fallen gold particles and immediately backfilled with cemented fill (see "Ground Control").

Figure 22 - Mine Horizon Section



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As the mining in the panel retreats, the last drift to be backfilled will be the downstream cross cut forming the boundary with the next mining block. Backfill placed in this drift will have a higher cement content to block ground water flow and to provide competent support for the upstream boundary of the next panel being frozen downstream. This will ensure safety and avoid sterilizing the Pay Zone under the frozen support pillars. Once backfilled, the cross cut drifts into the panels will be sealed by installing an engineered bulkhead pinned into the bedrock at the entrance to the cross cut. This barricade will be designed to be able to withstand a 50-60 meter hydrostatic head (~540Kpa) that could be generated by slum and water (see "Ground Control").

The method of freeze block mining described above will be repeated in a sequence downstream of the first mining block to sample the proposed 300m of paleo-channel. The sequence of panel mining will be managed such that as first panel is being mined, the second one in sequence is being frozen, the third panel in sequence is being drilled and installed with freeze pipes, and the footwall drift for the fourth panel in sequence is being driven.

Testing of bedrock materials and experience gained from mining the pay zone gravels during CVG's initial bulk sample has shown that the ground appears to be suitable for continuous mining by the appropriate mining machines (see "Mechanized Mining Equipment") with support from a small fleet of LHD scoops. Only as a last resort will conventional drilling and blasting techniques be used in the event that larger erratic boulders or steeply dipping, harder, quartzitic layers are encountered. The use of mechanized mining equipment will minimize any negative impact that blasting could have on the freeze pipe and freeze structure installation, and will improve the structural safely of the mine.

Generally, all material excavated from the footwall drift will be trammed to surface and placed on a permitted rock dump, this will be ~12,000 m^3 of broken rock. Based on the experience gained by CVG processing gravels in the Exploration Drift, all material mined from production drifting will be passed over a 4 inch grizzly and 1.5 inch vibrating screen deck installed at the bottom of the decline in an existing room. Reject from the grizzly and > 1.5 inch reject from the screen will report to an underground stock pile and be used in backfill. The < 1.5 inch fraction passing the screen will be combined with the material from cleaning the face during mining and the floor of the production drift prior to backfilling to form a concentrate which will be brought to surface for gold recovery by gravity separation (see "Gold Processing and Production"). The waste fines from gold recovery will be stockpiled and brought back into the mine for backfilling.

In total the planned 12 blocks of mining will result in a mined volume of approximately 28,900 BCM. At a 2.7 SG, this will generate approximately 78,030 tonnes of material. Based on experience, a swell factor of about 30% can be anticipated and therefore approximately 22,230 BCM (60,021 tonnes) will be used for backfill. The balance of approximately 6,670 BCM (18,009 tonnes) will be spoiled on surface. Together with the waste rock mined while advancing the footwall drift a total of approximately 16,850 BCM (45,495 tonnes) is anticipated to be spoiled on surface in the permitted waste rock dump.

CVG expects that it will require 8 months to mobilize, dewater and initially develop the mine to a point where gold production can begin and estimates that it will take a further 12 - 14 months to mine the planned 12 mining blocks.

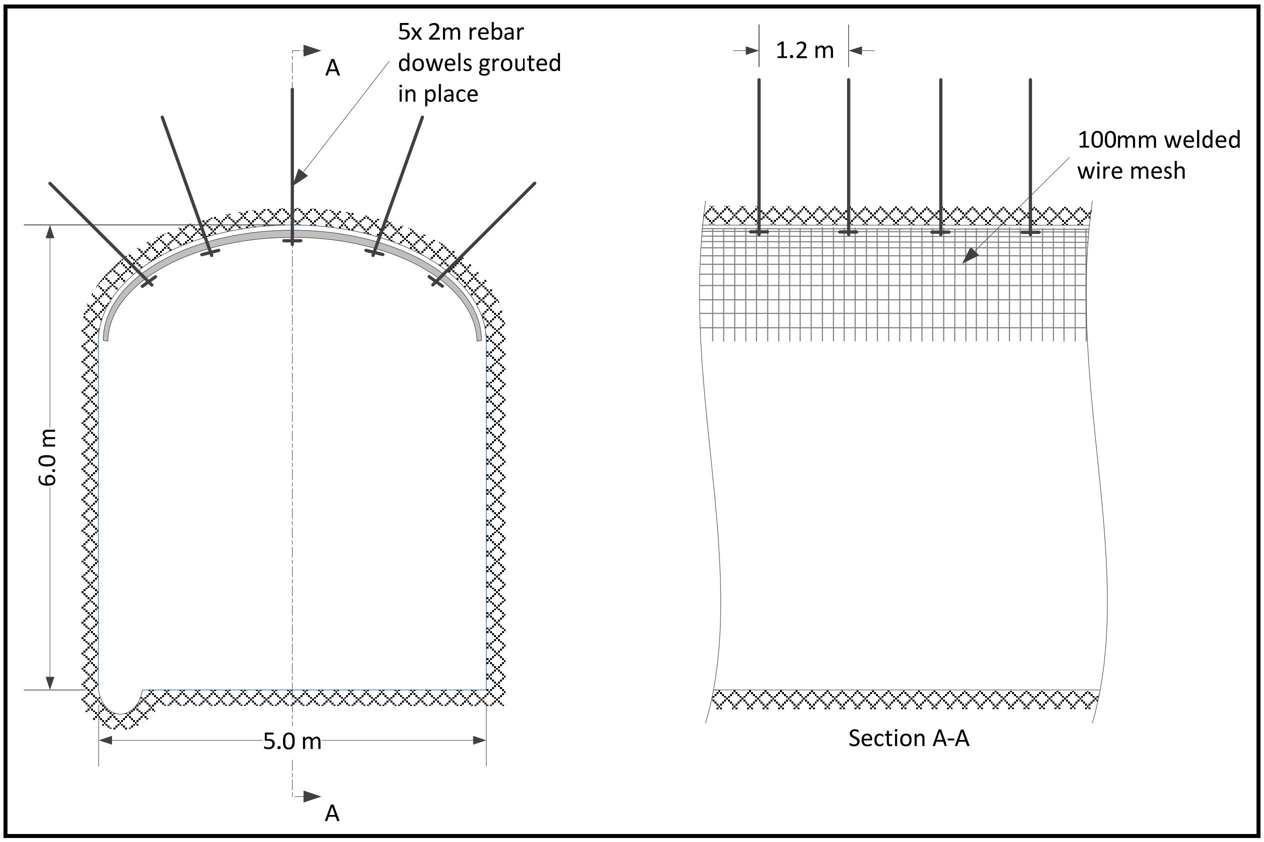
10.10 ARD Monitoring

Samples of broken waste rock/muck will be collected on a regular basis and analyzed to ensure that the chemical composition of the waste rock being spoiled on surface does not have a detrimental impact on the receiving environment. Based on previous drainage monitoring and analytical results it is expected that waste rock brought to surface will be non-acid generating and will not cause deleterious impacts to the environment. The routinely collected samples will each be analyzed at a certified laboratory for a typical static test package including Sobek ABA with siderite correction, Total C, Inorganic C, Sulphur Speciation, and Solid Phase Metals.

10.11 Ground Control

The footwall drift driven parallel to the paleo-channel is a permanent opening in bedrock used to drill freeze holes, access the mining blocks, and to bring in mine services and supplies. When CVG Mining completed the Exploration Drift in 2011 the footwall drift developed was safely supported by bolting and screened using 100mm welded wire mesh with five 2m grouted rebar bolts per ring and 1.2m ring spacing. This method of ground support will continue to be installed as the footwall drift is advanced. At intersections or where work places are excavated in bedrock, these larger spans will be supported with 3m grouted rebar dowels and 100mm mesh. If necessary, the ground support will be supplemented with shotcrete in areas where adverse structure or faults are encountered. In accordance with current practice the back support will be monitored and assessed, with enhanced support provided on an "as needed" basis. A typical layout of footwall drift support is shown in Figure 23.

Figure 23 - Typical Footwall Drift Support



With the anticipated production rates of the mechanized mining equipment, production drifts under the frozen structure are not expected to be open for more than a few days prior to backfilling. Results of WJEL modeling indicate that a 6m span of frozen structure will be self-supporting during this period of time and thus the ~4m span of the production drift is not anticipated to require roof screening and bolting except if needed to control localized spalling. Given the irregularity of the bedrock contact shown by the 2015 geophysical interpretation, sloughing of the pillars requires special attention and the mining plan of 2015 must therefore, be updated to reflect variations in the elevation of the bedrock contact across and along the channel (See Recommendation 4).

Any localized spalling at the back will be supported with short (0.5m to prevent unintentional puncturing of the freeze piping) grouted dowels or split sets and light mesh as needed. The dowels and screen are not intended to create a laminated beam, but will be installed to protect personnel from the localized spalling of loose material.

Sloughing of unfrozen unconsolidated gravels from the pillars between the bedrock/channel contact and the bottom of the frozen structure should be anticipated. As a safety precaution, light mesh and short dowels will be used as needed to control any potential sloughing of the drift wall as shown in Figure 23.

In the event that the air temperature in the stopes, or during the pillar recovery, rises above 0ºC, CVG Mining will have the ability to insulate the exposed frozen roof beam to minimize thermal transfer and keep the workings dry. Insulated tarpaulin blankets held in place by metal spring battens spanning the wall surfaces. The tarpaulin construction will be two outer layers of rip stop polyethylene with an inner core of closed cell polyethylene. The tarpaulins will be removed during backfilling of the stopes and reused.

To minimize the time that frozen spans remain unsupported and to enable the pillars to be subsequently mined, the stopes will backfilled (Figure 25) with three layers of waste material mixed with a binder as follows:

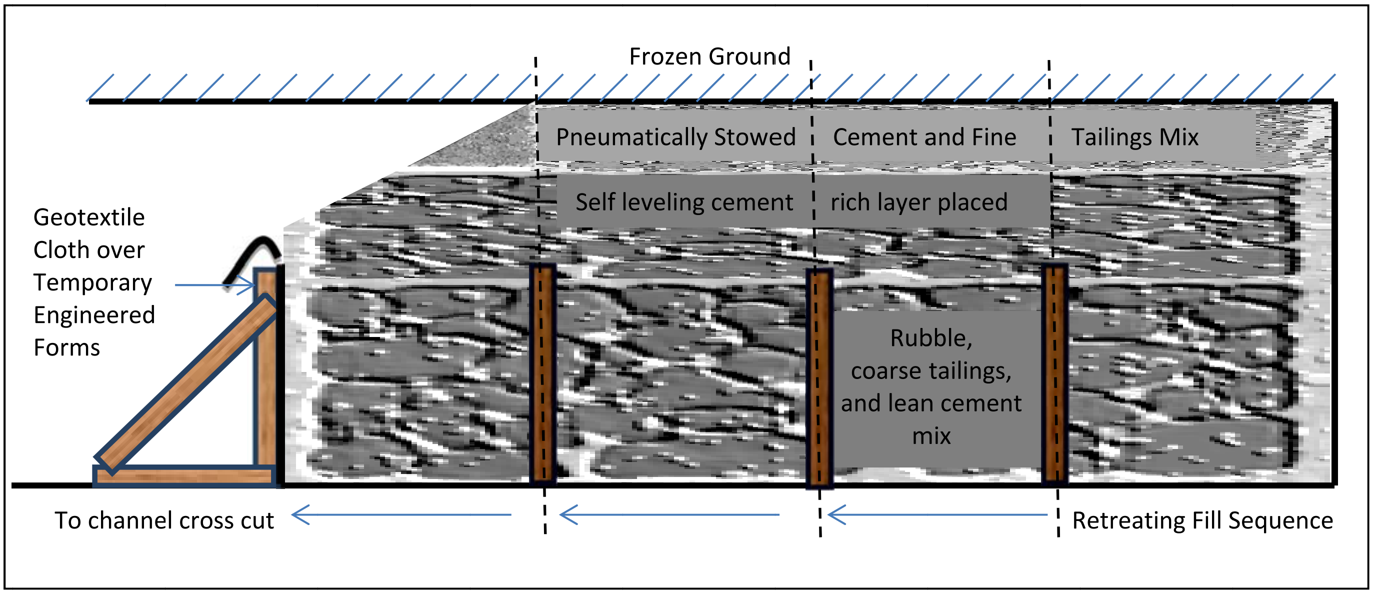
Primary Layer:
Waste material from the grizzly output underground is expected to be -12” +4” in size (see Section 4.9 “Gold Processing and Production”). This material will be transported with a 4 yd LHD Scoop and will be mixed with a lean cement mixture prior to being backfilled. The material would be mixed with the lean cement slurry by vibratory action as the LHD moves to the application point and then dumped in place behind formworks to a height of between 1.5-2 meters.

2nd Layer:
Waste material from the vibrating screen reject is expected to be -4 +1.5” in size (see Section 4.9 “Gold Processing and Production”). This wet material will be mixed with cement and supplied to the application point from the mixer by LHD and deposited on top of the primary layer to a height of 1.8-2.2 meters. Expected slump of this mix would be in the range of 150mm to 200mm.

3rd Layer:
Minus 1.5” waste material from the wash plant on surface (see Section 4.9 “Gold Processing and Production”) will be brought underground in bulk and stored. When needed, it is to be drum mixed with a cement binder (15-33% by weight and 3-6% moisture) for > 1 minute and fed into a dry shotcrete nozzle for pneumatic stowing/filling the remaining void. This process will allow for the anticipated frequent stop/starts the filling process will require as effective backfill working lengths are determined.

To ensure that the subsidence/deflection of the frozen roof beam remains within design limits CVG intends to verify that this plan and cement/tailings ratios will achieve a UCS of 4MPa and an effective Young’s Modulus of at least 700 MPa within 72 hours. Accordingly, the final backfill plan and type of cementitious binder utilized may be subject to modification prior to initiation of mining. Further study of this method and process are needed. Trials of potential mix ratios should be started as soon as possible (See Recommendation 12)

Figure 24 - Typical Back Fill Transverse View



When abandoning a mining block, the backfilling of the downstream production crosscut will incorporate a mix with a suitable particle size distribution of Portland cement, waste rock and fly ash or other suitable materials to achieve a permeability of < 10^{-7} m/s and equivalent strength properties to the rest of the backfill and the frozen soil. This pillar will support the upstream edge of the next (downstream) mining block, and act as an upstream barrier to ground water ingress as the mined out block is allowed to thaw.

After each panel has been mined and abandoned an engineered bulkheadwill be installed in the longitudinal footwall pillar between the access drift and the mined panel. These bulkheads will be designed to withstand the full hydrostatic head of 50-60 meters (~540kPa) that is anticipated to develop as the upstream freeze structure thaws. It is anticipated that these bulkheads will be similar in design to those installed by CVG Mining upon completion of the Exploration Drift and that the bulk permeability of the Portland cement enhanced backfill behind these bulkheads will be low, thus the potential for water flow at these bulkheads will also be low. Despite this, pressure relief lines will be installed in the bulkheads, and the pressure monitored on a long term basis.

The time for the frozen support beam to thaw will depend on a number of factors; these include the ambient temperature of the adjacent unfrozen ground which has been assumed to be ~10°C, the moisture content of the adjacent material and, most importantly, the velocity of the groundwater flowing across the surface of the frozen block. Typically, the temperature of the frozen material will commence to rise as soon as the refrigerant circulation ceases, and it will be several weeks or even months before the ground is completely thawed. However, the creep characteristics and the strength of the frozen material may be reduced considerably in the first week after the active freezing is terminated. For this reason it is important that the production sequence outlined above is followed closely. It is also important that specified backfilling materials and the placement procedures are strictly followed to support the proposed frozen spans. It is also important to eliminate any possible subsidence of overlying material into the mined out panels during thawing.

10.12 Gold Processing and Sampling

Material mined from the pay zone as described above will be taken by LHD and passed over a grizzly with a 4 inch flat bar deck. The oversize material is not expected to be gold bearing, and will be transported back to an active backfill stope on the backhaul with the LHD. The undersize material will then be washed over a 1.5 inch vibrating screen deck equipped with high pressure spray bars. The oversize material from the screen deck is also not expected to be gold bearing and will also be backhauled by LHD to an active backfill stope. The undersize material or “concentrate” will be collected together with any gold bearing floor cleanings from the production stopes and securely transported and stored at surface for further processing. Based on experience, the mine is expected to make water through groundwater seepage at a rate of 550 – 850 m^3/day (100-150 US gpm) and CVG plans to use this water to process material over the grizzly and screen deck and collect it in an underground settling sump so that any solids can be settled out and then either reused or pumped to surface for further treatment in the permitted settlement ponds. Additional make up water, if required to operate the screening equipment, will be sourced from surface.

CVG intends to use the same equipment used during the CVG Exploration Drift project consisting of a trommel, boil boxes, sluice box and tables located in a secure facility at surface. The trommel will consist of a hopper and a 3.5-foot diameter by 8-foot long rotating steel drum containing a 4-foot long scrubbing area and a 4-foot long screening area. A high-pressure spray bar will run through the entire length of the drum interior with another located in the hopper. The screened slurry made up of minus 0.5-inch mesh aggregate and water will enter a 2-foot wide by 8-foot long steel flume containing 3 boil boxes. The slurry will pass through an attached 2-foot wide by 8-foot long sluice box lined with nomad carpet and sheets of raised expanded metal. Final clean-up of the concentrate will be completed using Deister tables in an assay lab. Water to operate this processing equipment will be sourced from the downstream end of the permitted surface settlement ponds and discharged into the upstream end of the ponds.

A major benefit of this type of processing compared to a conventional gold mill, is that no cyanide and minimal chemical usage is required for gold recovery, while compared to a typical placer wash plant, the majority of suspended solids are trapped and settled underground, with surface ponds cleaning up the remainder. This process will have minimal environmental impact.

The pay zone is generally expected to be 1.2m in height encompassing 0.9m of the gravel above the bedrock interface and 0.3m of weathered underlying bedrock below the bedrock interface to capture material that has fallen into weathered bedrock cracks. The pay zone is small compared to the overall development, and as such, the overall development volumes will be used for cost determination, but for revenue projections, grade in grams/m2 is a more useful number. As such, the volumes from the Exploration Drift were converted into areas with grade assigned.

The location of the first phase (300m) of mining was chosen based on results of the CVG Exploration Drift and the two historical churn drill fences, “C line” and “D line”, in close proximity to the east and west boundaries of the first phase of mining.

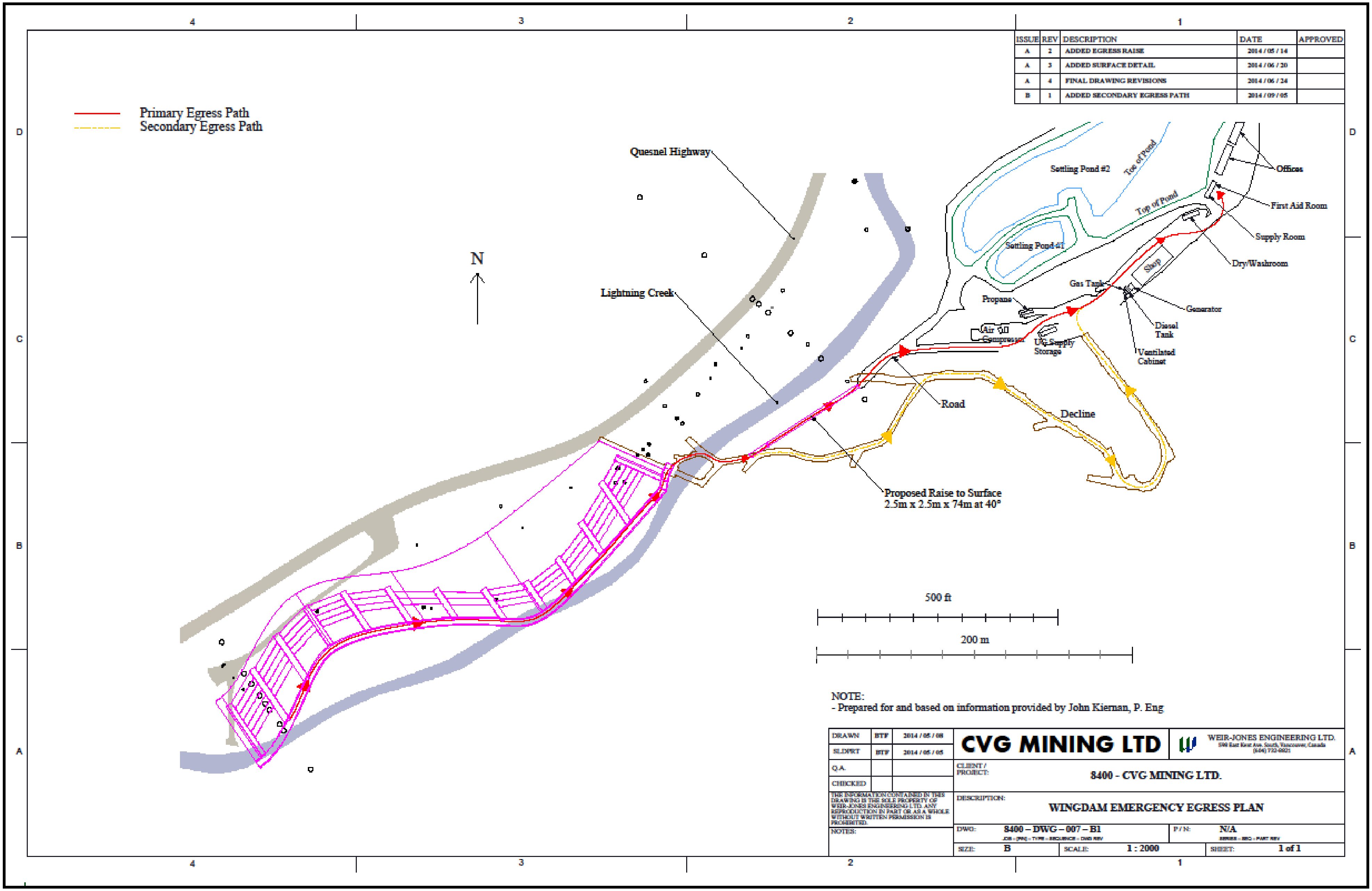
The 2.44m x 2.44m x 23.5 m long Exploration Drift showed three distinct areas of grade distribution: the first 3.8 m on the south rim which yielded 19.860 TrOz of placer gold (2.142 TrOz/m^2), the middle 14.8 m which yielded 146.436 TrOz of placer gold (4.055 TrOz/m^2), and the remaining 4.9 m on the north rim which yielded 7.2 TrOz of placer gold (0.602 TrOz/m^2). The overall recovery from the exploration drift was 173.496 TrOz of placer gold equal to a grade of 3.026 TrOz/m^2.

The overall grade shown in the bulk sample compares favourably with the average grade indicated by 7.5” diameter historical churn drill holes C1, C7, C8, C9, D1 and D2 drilled to the channel bottom near the east and west bounds of the first phase of mining. In the aggregate these holes reported recovery of 253 grains of placer gold from a 1.84 ft^2 of drilled area resulting in a grade of 3.082 TrOz/m^2.

10.13 Mine Safety and Second Egress

Prior to the proposed Phase 1 test sampling, safety procedures will be established and recorded in a mine safety manual for each job underground to meet the requirements of the Mines Act and to provide a safe working environment as a first priority. Planned emergency egress paths are shown in Figure 28. Note that if the seismic interpretation of 2016 is confirmed, this egress plan may need to be evaluated with each undulation of the footwall drift.

Figure 25 - Emergency Egress Plan



Pre-shift safety briefings will be held between crew and foremen every day, and regularly scheduled weekly safety meetings will be held to encourage awareness of the various hazards that can be found around a mine site. Use of PPE will be monitored and enforced at all times.

Prior to development of future vent raises and points of egress as the mine advances beyond the initial ~300m of channel, portable refuge station(s) will be located proximal to the furthest development faces on the footwall drift as required.

A mine rescue agreement will be established with another company that operates an underground mine in BC. Potential candidates include: QR, Myra Falls, and New Afton. In addition, workers at the Wingdam Mine will be trained in mine rescue, and a number of BG174's or equivalent will be maintained on the property for a first response.

Site security will be strictly enforced. Monitoring cameras will be in place in the underground and surface processing areas and in the assay lab. Any gold recovered from the assay lab will be taken to a secured location prior to transportation to market.

10.14 Mechanized Mining Equipment

A survey of current vendors of Mechanized Mining equipment has been compiled by WJEL and is presented in the WJEL Mine Plan Report of 2014. The rock conditions expected to be experienced at Wingdam will require a flexible machine capable of mining and mucking wet silt, gravels, cobbles, sand and broken bedrock. Specifically, a suitable machine must be capable of meeting the following constraints at a minimum:

Table 10 - Machine Requirements

Requirement	Dimension
Rock Material Footwall	90% Phyllite 50 MPa with occasional quartz veins
Rock Material	60% Consolidated Gravel with occasional cobbles 40% Phyllite 50 MPa Bedrock
Machine Access	2.5 x 2.5 m Maximum Machine cross section dimensions. Although possible, reassembly discouraged.
Machine turn radius	<6m Ideally within 4m space.
Machine Cutting workspace	Minimum 2.5m vertical above, 0.5 below, 4m side to side and 1m advance reach
Mucking and conveying system	To move material loose from working face to haulage space behind only. Must not clog when moving wet clay. Must be easy to clean and capable of handling up to 0.5m particle sizes.

There are two distinct applications for mechanized mining machines: cutting or breaking the bedrock in the footwall access drifts and bottom of the production drifts; and mining the pay zone of gravels and fractured bedrock in the production drifts. Experience has shown that the gold particles are coarse and that inevitably some of these particles can be expected to drop out of the pay zone gravels onto the underlying barren bedrock bench and floor as the production drift is advanced. Ideally the lower ~1 m of barren bedrock in the production face would be cut in pieces large enough to easily separate them from the auriferous material by simply scalping the run of mined material at the grizzly. Furthermore, all pay zone material should be transported in LHD scoops or trucks, and not by scraper chain or flexible belt conveyors as the later would likely result in the loss of gold particles by gravity or by smearing along the haulage route and at the transfer point.

Due to the narrow width of the stopes necessitated by the need to support the frozen roof, any machine used will need to muck out mined material from the face to the rear of the machine using a short conveyor system that is easy to clean and capable of working with saturated fine sands and silt. All mechanized mining equipment should be reviewed with this detail in mind, and therefore machines with small gathering arms are likely to be unsuccessful.

The paleo channel contours predicted by the 2015 geophysical interpretation this machine must also be capable of traveling and mining at gradients of > ± 15º. This may require risk mitigation plans for material handling, and / or the abandonment of some of sections of the channel in favour of the higher producing locations (See Recommendations 4 and 8 for Mine plan update and Machine Requirements).

Based on the foregoing requirements, CVG has narrowed its selection of machine to either the Alpine / Terex / Schaeff / ITC 120 family of machines, or the Dosco / Webster 2000 series machines. CVG is working with the distributors and manufacturers of these machine to finalize its selection but both families of machines are certified for underground mine use and may be capable of meeting the requirements set out above.

10.15 Mine Energy Considerations

The Wingdam mine does not have ready access to 3-phase grid power, a single phase line runs past the mine site from Quesnel to Barkerville. This line is operating at a capacity with brownouts in Barkerville being a common occurrence. The current mine plan calls for 3 MW of 3-phase service and although BC Hydro is capable of providing this capacity via extension of a 3-phase overhead line from Quesnel to the mine site, the Company has been informed by BC Hydro that the capital cost for extending the line would be approximately $7 million, excluding the costs of an engineering study, environmental mitigation, land acquisition, right of way, public consultation, temporary construction services and any civil cost and installation of civil works on private property.

BC Hydro is willing to contribute $520,000 towards extension of the overhead line. However, management estimates that that the overall costs of the project would fall between $9 -10 million which, when amortized over the life of mine would result in a fully loaded cost of grid power of approximately $0.165 per kWh. This results in an estimated annual power costs of approximately $3.5 million. This cost compares favourably to the costs of alternative power sources (Table 13), however, with an estimated lead time of 2 years, grid power is not a viable alternative for near term completion of the 300m test sample.

The Company has investigated several different power supply solutions, based on available fuel sources and related equipment, to determine an alternative to grid power with an acceptable lead time. These solutions ranged from conventional large scale generation of electrical power using diesel, LNG, CNG or LPG (propane), combined with conventional screw compressor ammonia chiller technology, to conventional electrical generation using fossil fuels combined with absorption ammonia chiller technology and fossil fuel or wood pellet fuelled thermal fluid heaters as a driving heat source for the chiller. The possibility of waste heat recovery from the fossil fuel generator was also investigated. CNG and LPG were dismissed as an alternative power sources due to higher costs and challenging logistics of delivering the required quantities by truck. The basic economics of the remaining alternatives are summarized in Table 13. The scenarios were run assuming a delivered cost of \$0.90/liter for diesel, \$14.00/GJ for LNG and \$140/tonne for white fiber wood pellets. The annual power costs were calculated including fuel, repairs, maintenance, and amortization of capital cost over a 6 year life of mine assuming a salvage value of the equipment.

The results of this investigation indicate that the most cost effective solution, next to grid power, is the use of LNG to power electrical generation and as a driving heat source combined with waste heat recovery from electrical generation to power ammonia absorption chillers for ground freezing. Accordingly, management has assumed in its financial modeling that this solution will be utilized. However, it is noteworthy that wood pellets, which are readily available in the area are in fact the least expensive source of fuel on a delivered cost per energy content basis. It is the amortization of high capital costs of an industrial sized wood pellet thermal fluid heater over the relatively short mine life of Wingdam which results in a fully loaded cost greater than an LNG solution. The availability of a used wood pellet fuelled thermal fluid heater may warrant further investigation.

Table 11 - Summary of Power Alternatives

Type of Chiller	Screw Compressor	Screw Compressor	Absorption Chiller	Absorption Chiller	Absorption Chiller
Electricity Generation	Grid	Diesel Genset	Diesel Genset	Diesel Genset	NG Genset
Driving Heat Genertation	N/A	N/A	Wood Pellets	NG Boiler	NG Boiler
Without Heat Recovery					
Capital Costs of Electrical Power	$9,500,000	$2,200,000	$1,100,000	$1,100,000	$1,500,000
Capital Costs of Chiller	$4,000,000	$4,000,000	$1,500,000	$1,500,000	$1,500,000
Capital Costs of Boiler	$0	$0	$7,700,000	$1,200,000	$1,200,000
Total Capital Costs	$13,500,000	$6,200,000	$10,300,000	$3,800,000	$4,200,000
Refrigeration Costs ($/kW)	$0.173	$0.333	$0.256	$0.231	$0.229
Electrical Generation Costs ($/kW)	$0.165	$0.328	$0.328	$0.328	$0.285
Total Estimated Annual Power Cost	$3,411,692	$6,664,094	$5,786,133	$5,500,361	$5,105,547
With Heat Recovery	N/A	N/A			
Capital Costs of Genset			$1,500,000	$1,500,000	$1,900,000
Capital Costs of Chiller			$1,500,000	$1,500,000	$1,500,000
Capital Costs of Boiler			$7,700,000	$1,200,000	$1,200,000
Total Capital Costs			$10,700,000	$4,200,000	$4,600,000
Refrigeration Costs ($/kW)			$0.187	$0.155	$0.139
Electrical Generation Costs ($/kW)			$0.349	$0.349	$0.309
Total Estimated Annual Power Cost			$5,186,265	$4,815,938	$4,289,982

11.0 Existing Infrastructure

The Wingdam Mine project has extensive existing infrastructure including a gated mine entrance, lay down area, workshop, rock dump, portal, decline and settlement ponds.

12.0 Data Verification

Given that none of the mineralization outcrops on surface and that underground access is not available, no representative or data verification samples were taken. Historical project infrastructure **on surface** and numerous project drill sites were observed.

13.0 Environmental Studies, Permitting and Social or Community Impact

13.1 Surface Wetlands

During CVG's operation of the Exploration Drift in 2012, the Ministry of the Environment and the Department of Fisheries and Oceans raised concern over the "drying up" of surface wetlands in the vicinity of the mining activity and pointed to the dewatering of the mine as a reason for this drying up.

These wetlands are recharged from two sources: by the natural seasonal surface drainage in the area, and by groundwater flowing from old, uncapped monitoring wells drilled into the confined lower aquifer. The artesian pressure in the confined lower aquifer is sufficient to cause these old wells to overflow onto the ground, enhancing the surface flow of water through the wetlands.

Historically, including the CVG Exploration Drift, mining activities have involved dewatering the gravels in the confined lower aquifer by pumping from one or more of the Melvin Shaft, Decline and/or dewatering wells. The dewatering of the lower aquifer temporarily eliminated the artesian flow from the old wells and, therefore, temporarily eliminated this supplemental source of recharge water into the surface wetlands. Once mining operations were completed and dewatering activities ceased, the artesian pressure returned to the confined lower aquifer and the old wells once again flowed. As demonstrated by analysis of the CVG Exploration Drift well monitoring data contained in the Lorax Report, the confined lower aquifer is not connected to the upper unconfined surface aquifer and, therefore, dewatering or depressurizing the lower aquifer has no effect on the natural surface drainage through the wetlands.

It is important to note that CVG's current plan is not to dewater the confined lower aquifer but rather depressurize it by pumping and installing a well upstream of the mine area and pumping this well at a rate sufficient to reduce the hydraulic gradient over the mine area to near zero. This is being done for a variety of reasons as highlighted in Section "Groundwater Management" but primarily to enhance mine safety by improving the efficacy of the CVG's planned ground freezing program.

This strategy of reducing the hydraulic gradient over the mine area will reduce the artesian pressure in the confined lower aquifer sufficiently to also temporarily reduce or eliminate the ground water recharge of the surface wet lands from the old wells. CVG management has discussed this situation with the Department of Fisheries and Oceans. As a result, the parties have tentatively agreed that the issue would be mitigated by directing a portion of the groundwater being pumped to depressurize the confined lower aquifer to the surface wetlands as "make up" water. This groundwater is being pumped upstream of the mine and accordingly will not be influenced by mining activities. Therefore, the water quality and physical characteristics will be identical to the current artesian flow of groundwater from the old wells. The amount of groundwater discharged into the wetlands will be sufficient to maintain the water level in the wetlands at a seasonally adjusted elevation as approve by DFO. Any groundwater not utilized to replace the artesian flow will be discharged into the permitted surface settlement ponds. Once mining has been completed the depressurization pumping will stop and the artesian pressure will, once again, cause ground water to flow to the surface through the old wells.

13.2 Surface Disturbance

All contemplated surface activities related to mining will be conducted within the boundaries of, and in accordance with CVG's permit #P-11-612. Therefore, there is no planned surface disturbance outside this permitted area.

In 2011, while CVG was dewatering the confined lower aquifer in preparation for conducting its Exploration Drift program, a sink hole developed and the mine was temporarily flooded until CVG was able to mitigate the problem. This sink hole developed as a result of remobilization of a previous sink hole. The redevelopment of the sink hole and subsequent remediation is discussed in detail in a June 14, 2011 report prepared by Clifton and Associates Ltd.

The original sink hole occurred when Gold Ridge Resources was trying to first enter the gravels from the foot wall drift at the base of the decline. Although they had dewatered, they encountered saturated slums which flowed into the mine and created a sink hole to surface. The sinkhole had no connection to the creek so no mine flooding occurred and once the slums finished flowing, GRR was able to recover the mine. GRR remediated the sinkhole from surface by filling it with material on hand and installed a wooden bulkhead underground to restrict any further flow of material into the mine.

GRR then moved a few meters downstream and again tried to enter the gravels with a similar result except that while saturated slum flowed into the mine, no sink hole formed to surface. This second attempt was also abandoned and a wooden bulkhead was installed to restrict any further flow of material into the mine through this entry.

In contrast, the remobilization of this sinkhole in 2011 occurred under quite different circumstances. When the confined lower aquifer was being dewatered. Lightning Creek had swollen its banks due to the unusual size of the freshet, and this had saturated the soils including the material use by GRR to loosely fill the old sink hole. Ultimately this, and the surrounding material, collapsed again, reforming the sink hole and allowing surface water to flow into the mine around the wooden bulkhead installed by GRR. This connection was recharged by Lightning Creek which by that time had eroded its original bank in the vicinity of the sinkhole and flowed freely into the mine. The mine flooded to the level of the ventilation raise drift where CVG was able to maintain its level by pumping until it could remediate the situation with a grout plug. To install the grout plug, CVG drilled holes from the vent raise drift and pumped polyurethane grout behind the wooden bulkhead. As this grout expanded, it successfully sealed off the inflow from the sink hole and allowed the mine to be pumped out. This grout plug was considered temporary, and subsequently, CVG sealed both of the GRR entries into the gravels from the footwall drift by installing engineered shotcrete bulkheads capable of withstanding the hydraulic pressures.

The circumstances leading to the development 2011 sink hole were unique and are not expected to reoccur. There is no known (recorded) previous mining along the 300m channel proposed to be mined and, therefore, CVG does not anticipate any possible remobilization of historical sink holes. Furthermore, CVG does not plan to dewater the confined lower aquifer which eliminates any possible flow from the upper aquifer to the lower aquifer through old (unknown) connections that may exist and minimizes any risk of a sink hole developing or reforming.

In the unlikely event that a “window” (see “Groundwater Management”) develops in the frozen structure isolating the mining block, saturated material from the high pressure area external to the frozen structure could flow into the low pressure (mine works) area internal to the frozen structure. This might lead to a new sink hole forming to surface. As outlined in Section “Ground Freezing” and elsewhere in this report, CVG intends to minimize this risk through routine monitoring of the frozen structure, and the groundwater pressures (hydraulic gradient) in the confined lower aquifer.

13.3 Geotechnical Stability of Highway 26

During CVG’s operation of the Exploration Drift, the Ministry of Transportation (“MOT”) raised some concerns that the geotechnical stability of Provincial Highway 26 might be compromised by dewatering the mine area. These concerns are valid, and are discussed below.

The geotechnical stability of the Highway is controlled by the surface aquifer, the soil moisture content of the road bed and the volume of heavy traffic. As mentioned in the previous section the confined lower aquifer is not connected to the upper unconfined surface aquifer and, therefore, dewatering or depressurizing the lower aquifer has minimal effect on the surface aquifer. Notwithstanding this, CVG's groundwater management plan does not involve dewatering the confined lower aquifer. CVG's plan to depressurize the confined lower aquifer is not expected to have any significant effect on the surface soil stability.

During the Exploration Drift program, CVG in consultation with MOT conducted a monitoring program of the driving surface on the highway while mining was being conducted. CVG will continue to work with the MOT to establish control points on the Highway Right of Way to monitor and identify any potential impacts that CVG's mining activities may have and work with MOT, as needed, to mitigate any potential damage to the Highway.

Along the highway edge there is a clear line of sight from a point above the initial cross cut drift downstream to seismic line SL14-01. This segment of the highway is offset from but closest to the proposed mining area. During mining operations, CVG proposes to monitor the elevation of this segment of highway including an additional distance of 100m east and west of the segment. Stationary targets will be installed at intervals along the edge of the road shoulder and periodically surveyed from two convenient observation locations in road pullouts off of the edge of the highway so that there are no interruptions to traffic, and so that the work can be performed safely. This monitoring will ensure that there are no negative effects to the highway surface due to the freezing, or any settlement issues which might occur post backfilling.

13.4 Wingdam Permit

On January 21, 2015 CVG Mining received an amendment to the existing BC Mines Act Placer Permit authorizing mining at the Wingdam project. The amendment authorizes the company's plan to sample approximately 300m of ancient paleo-channel of Lighting Creek. The permit contemplates the production and processing of 20,000 cubic meters of gold bearing pay gravels per year.

In conjunction with the BC Mines Act Permit, CVG holds an active Discharge Permit under the provisions of the BC Environmental Management Act. Unlike most mines, the only effluent discharge from mining activity at the Wingdam is groundwater sourced from drawdown wells and retained water from the initial pumping and drawdown from the flooded decline. This water will be pumped and held in temporary retention in the established settling ponds after which the water may be discharged directly into Lightning Creek under the provisions of the Discharge Permit.

14.0 Interpretations and Conclusions

Significant gold mineralization has been identified during the many phases of exploration and development on the property dating back to the late 1800's. The depth to the enriched gold zone coupled with the difficulties of mining through unstable sediments that are unconsolidated, fine-grained and water-saturated have been factors that have hindered development of this project in the past.

Recent work by CVG Mining Ltd. and Omineca has been important in confirming the viability of developing the gold found in the Deep Lead Channel on the Wingdam Property.

CVG's 2012 drift sampling program demonstrated that freeze ring technology may be effectively applied to maintain ground support during conventional mining operations, as well as maintaining a barrier for water inflow. The 2012 program was also successful in extracting and processing gold from the mineralized zone. Results from this program were in line with historical gold grades from previous programs on the Property. The gold-enriched zone in CVG's drift was up to 1.20 meters thick and extended 23.5 meters across the width of the channel. The average gold grade across this width amounts to 34.55 m^3 when diluted through the entire thickness of a 2.44-meter drift height. The average grade amounts to 41.84 g/m^3 across an 18.6-meter distance when the low grade (6.88 g/m^3) section across the north rim is excluded.

CVG completed a compilation of project data in 2014 and prepared a preliminary mine plan. In 2015 a subsequent geotechnical drilling and test program recovered representative core samples which were analyzed to determine frozen material characteristics. These results confirmed the preliminary assumptions of the ground-freezing plan. The drill programme also included downhole seismic refraction geophysics which augmented five lines of seismic done on surface. The interpretation of this data is included in the current report, but has not yet been integrated into an updated mine plan.

The geotechnical and seismic work generated a more accurate profile of the Deep Lead Channel at the gravel-bedrock interface, which lead to a new interpretation of the channel geometry. This work also helped to generate a 3D model of the channel, showing that the walls of the channel were steeper than originally assumed. All of this leads to a possible increase of about 6% in the volume of the channel, when compared to earlier estimates based upon assumptions about the historical channel profiles. The new model is also indicates the presence of a series of depressions along the Channel which are thought to have been potential natural traps for the placer gold during deposition of the gravels.

The results of this seismic interpretation became available in late 2015/early 2016, and they are summarized in Figures #5 and #6 of this report. It is apparent from examining this interpretation that the actual paleochannel is both more irregular in plan view, and also in vertical elevations, than the previously idealized shape of the payzone.

The profile of the current inferred channel is dominated by five erosional features which extend below the previously inferred elevation of the bedrock contact by up to 5.5m. These variations in the bed of the paleochannel mean that the mining horizon will not follow a uniform slope over the proposed 300m length of the initial twelve mining panels.

The most recent profiles indicate a significantly more complex geometry in both the longitudinal and transverse directions. For this reason it will be necessary to review the proposed mining geometry, the relative position of the footwall access and freeze hole drilling drift, and the impact of these modifications on the development, ground freezing and mining costs. This work will have to be done as a preparation of a more detailed mine plan and freezing layout. Specifically the following matters will have to be addressed:

- The variation in elevation of the bedrock contact below the paleochannel means that the elevation of the mining horizon in the 25m long production panels will vary both along the length of the production stope ~ 21m, and also across the width of the panel. Detailed longitudinal and transverse sections of each panel and stope need to be prepared and presented as 3-D images so that the mining geometry can be evaluated. Not only will the relative positions of the stopes and pillars vary within each panel, but there will be significant variations in the mean mining elevations from panel to panel.

- The annotated Figure #6 shows the mean invert elevation of the stopes, and it is immediately apparent that, in some instances, the payzone elevation may vary by as much as 6m along the length of the stope. This means that the pillar height will vary by the same amount if the 4m frozen roof beam is unchanged from the current planar profile.

  To use Panel #4 as a specific example the erosion feature in the paleochannel means that the frozen roof beam would be approximately 8m above the payzone at the bedrock contact at the centre of the channel. There is also a change in elevation across the 30m wide channel of more than 4m, > 15º. Obviously this means that the pillar between the two longitudinal slopes would have a W/H ratio of ~ 0.5, rather than ≈ 2.0. Given the anticipated ground conditions pillars having these proportions would not be stable, and thus the need to change the panel boundaries and beam angles.

- A related design consideration associated with the irregularity of the elevation of the paleochannel will be the collaring of the freeze holes required to stabilize the 4m thick roof slab. As can be seen from the annotations on Figure #6 in the longitudinal direction the freeze hole collars will either have to be located in a linear pattern which will give rise to excessively high pillars and the need for a very high foot wall drift. Alternatively they will have to follow an undulating profile which is parallel to the payzone contact on the south side of the paleochannel. Irrespective of the collaring pattern there will also be a transverse variation in pillar height across the channel.

- There is also the issue of the variation in the invert elevations of the stopes in adjacent mining panels. One solution to this would be to step the roof beams from panel to panel. This is most pronounced between panels #2, #3 and #4, and between #9, #10 and #11.

All these issues will require additional engineering design work however, they are all solvable by modifying the geometry of the mining layout, and adjusting the shapes and position of the footwall drift or, as a last resort, sterilizing some sections and pillars.

In 2015 CVG Mining also received an amendment to the existing BC Mines Act Placer Permit authorizing mining at the Wingdam project. The amendment authorizes the company's plan to sample approximately 300m of ancient paleo-channel of Lighting Creek.

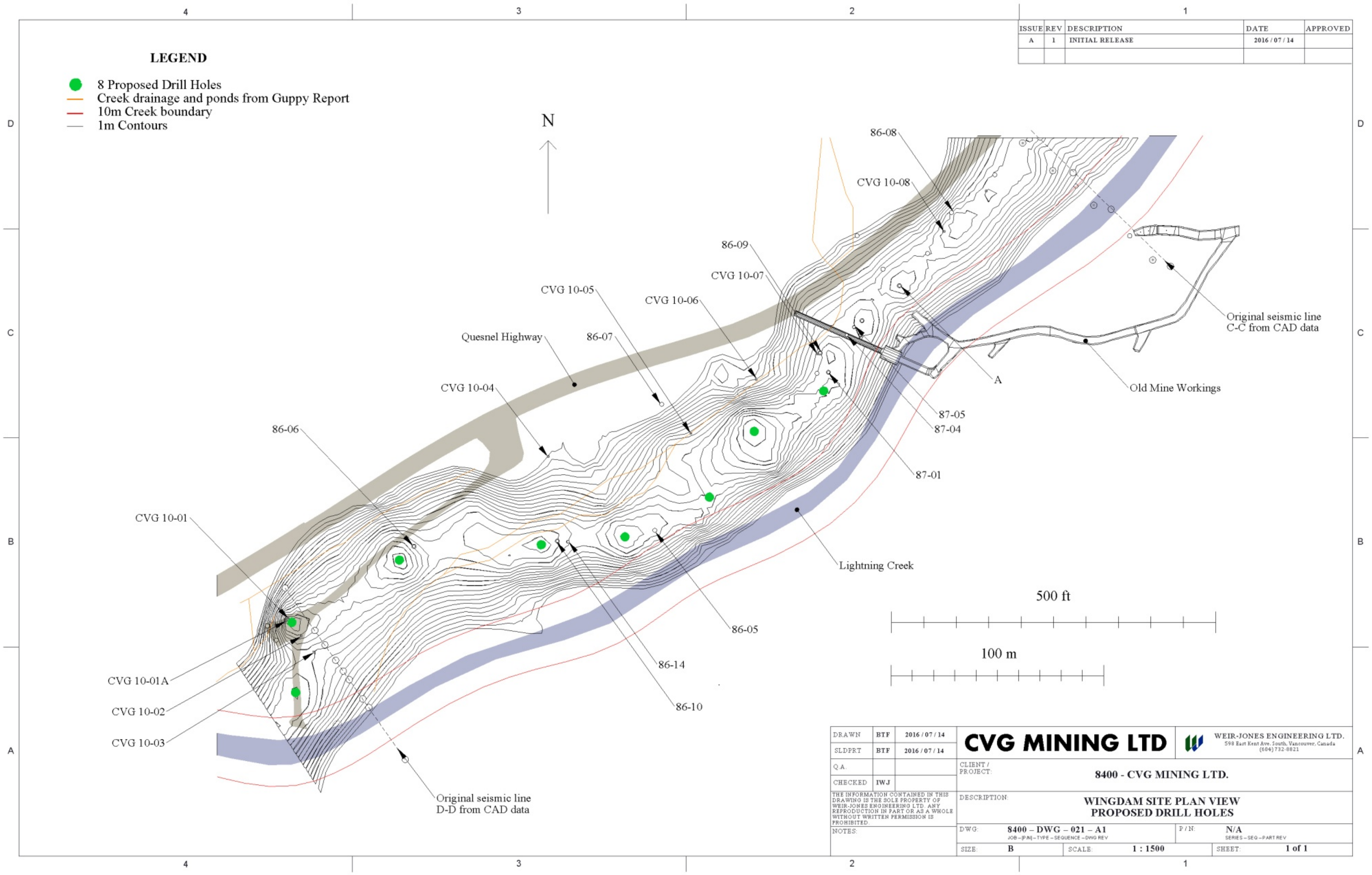
15.0 Recommendations

The following work is recommended for the Wingdam property. Further geotechnical analyses and engineering to finalize the parameters for underground freeze mining should be completed. The estimated cost for this work is $465,000 (Table 15

Specific recommendations for the mining operations and risk mitigation include:

1. The seismic interpretations from the 2015 drilling program portray an undulating profile of pools and benches along the thalweg of the bedrock contact. It is a significant departure from the previously assumed 2° slope, and will require the mine plan to be adjusted to include provision for an undulating profile along the footwall drift. The panel boundaries, pillar dimensions and mining zone will also need to be adjusted. It is recommended to validate the Frontier Geoscience bedrock contact model by drilling eight holes along the thalweg at the pool and bench locations to confirm bedrock elevation, and to obtain additional samples from bedrock contact -1.5m, to the roof elevation as before. The recovered material will be used for subsequent material property studies as necessary. The proposed drill locations are shown in the Figure 26 below:

Figure 26 – Recommended Drill Locations



2. Develop cross sections and longitudinal sections based on Frontier's current bedrock profiles for both the mining panels and the footwall drift. Prepare to revise these cross sections after #1 is completed, and if necessary. An accurate 3D surface of the bedrock contact, together with the advance drilling from the footwall drift, will assist in adjusting the mining horizon to ensure that the panel elevations and pillars heights follow the bedrock contact accurately, and thus reduce the amount of mined material.

3. The creep properties are of an increased importance given the potential change to the stope heights created by the discovery of the depressions in the bedrock contact predicted by the latest seismic interpretation. The UCS, UTS and creep properties of the frozen soil, as well as bedrock point loads should therefore be measured from the material recovered in Recommendation #1. A chart of the target recovery from the holes is shown below in Figure 30.

4. Re-Develop options for block size, block limits, freeze hole layout and alternate mining layouts based upon the confirmed bedrock contours from #1, and the cross sections and 3D geometry from #2. This will reduce the potential for unanticipated changes to the mine plan as mining of the footwall drift progresses.

5. Investigate the drained behavior of the material between the frozen roof and bedrock contact. Specifically, investigate the ability of this material to be dewatered, the effect of permeability on dewatering, and the amount of consolidation associated with dewatering of unfrozen pillar material at different pressures. As an example, what is the zone of influence of a dewatering hole, or is the slum impermeable and thixotropic? These investigations should lead to a model for the behaviour of dewatering on settlement of roof beam.

6. Investigate the effect from the studies in #5 on the freeze pipes due to consolidation and/or pillar compression. It will be necessary to investigate the probable time dependent deformation of the frozen roof beam at temperatures just below freezing to establish if the roof can be allowed to creep down (settle) as dewatering proceeds in the parts of the panel where the pillars would be excessively high. The question is whether or not it is safe to allow the roof to settle while the slum consolidates due to dewatering, without fracturing or damaging the freeze pipes.

7. Investigate possible end point slip modes of beam failure under various relaxation conditions at the bedrock to frozen slum contacts due to a potential for a concave back profile in some sections.

8. For all potential mining and drilling layouts re-evaluate the operational capabilities of the mining and drilling equipment in both the mining panels and in the FW drift. Some of the panels may have dip angles in the neighborhood of ± 15° to the horizontal. Whether the conveyors on the equipment can operate effectively at this slope will have an effect on the mining rates and potentially the elevation of the footwall drift. The freeze drilling pipes may also require less than ideal dip angles to accommodate material recovery.

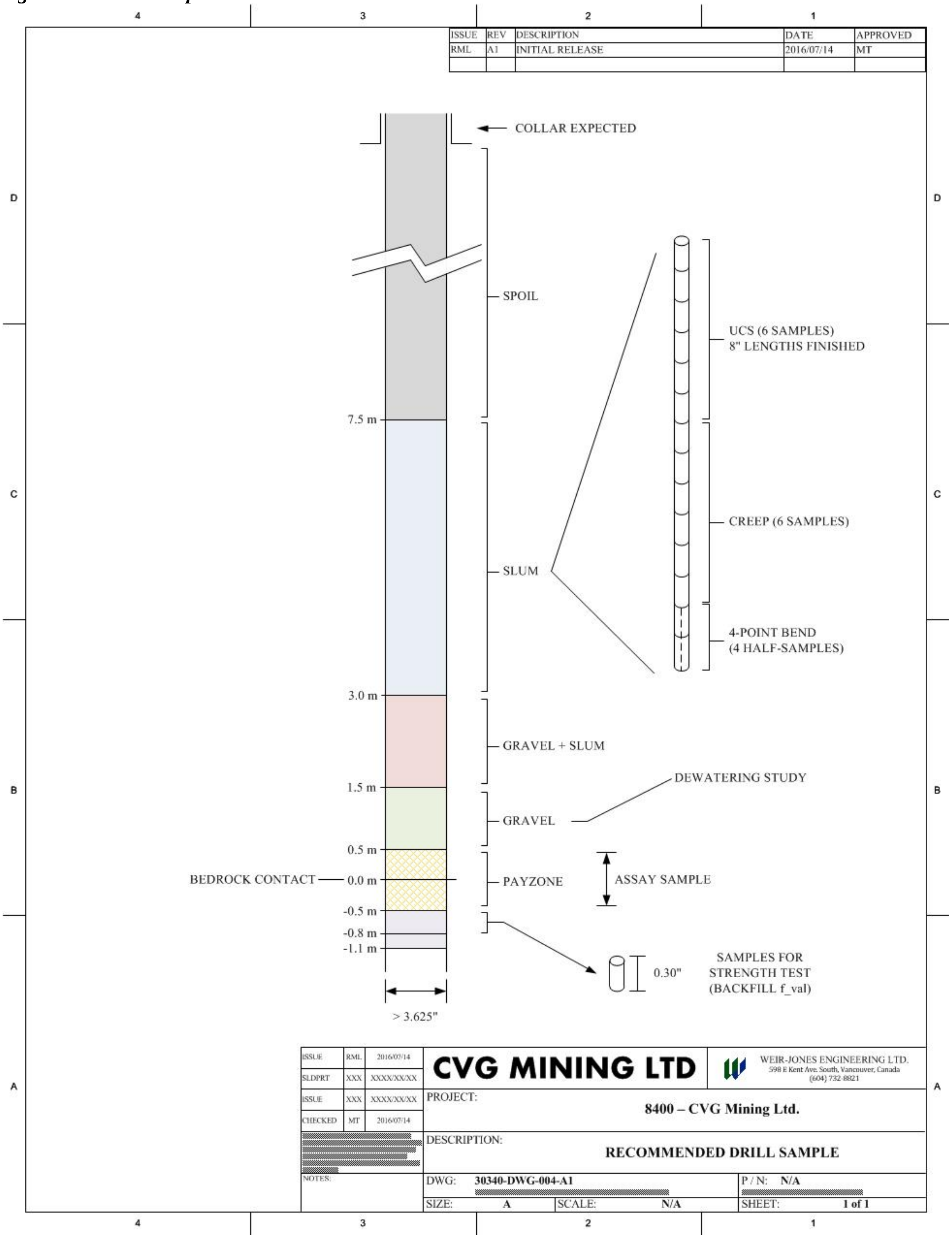
9. Prepare new summaries of both mining and drifting volumes for various mining scenarios. The new seismic interpretations have increased the available pay zone, but also have increased the volume of material to be mined.

10. Detailed design work remains to define the freeze plant piping connections underground, as well as the instrumentation layout and monitoring communications link(s). Also, the drainage of undulating thalweg/mining profile and egress concerns in both stopes and FW drift need to be studied. The original assumption that any spilled coolant or water flows will gather at the face of the footwall drift will no longer apply. As each panel progresses, the egress plan and drainage plan must be updated to reflect the new geometry.

11. The integrity of the bulkheads on the panel access drifts must be reconsidered. They need to be located in competent ground so that they can withstand the maximum anticipated head after the panel roof beam has thawed, and the assumptions for this must be reconsidered given the potential for taller pillar designs from Recommendation #4.

12. To confirm the current working assumptions for back fill material, hydration times, compressive strength and modulus, it is recommended that some of the material collected from #1 also be diverted to backfill studies. The new seismic data, if and when confirmed, will very likely increase the amount of slum removed as a percentage of the total mined material, and therefore there will be a need to return more of this material back into the stopes, to facilitate removal of the pillars.

Figure 27 – Drill Sample Plan



Further testing is also highly recommended as the initial sample size was very small. In order to properly assess the uniaxial compressive strength of a frozen soil, ASTM D7300 standard recommends a minimum of 15 samples for a given temperature. Because of the two different grain sizes observed, a minimum of 30 samples should be tested to satisfy the requirements. 15 of these should be fine-grained while the other half should be coarse-grained. Therefore, an additional 18 samples should be tested at each temperature, for a total of 54 samples. However, because the nature of this testing is to design a freeze wall and not for scientific purposes, a lower number of samples may be considered by increasing the safety factor in the freeze wall design. A minimum of 8 samples at each temperature and each soil type may be sufficient, and the results separated by soil type.

Likewise, with the compressive creep testing and the flexural creep testing, a minimum of 15 samples should be tested to determine more accurately the creep properties. Again 8 may be sufficient if the samples and results are separated by soil type. With regards to the flexural creep test, a similar number of samples should be tested to failure to determine the flexural strength with greater reliability. At this point more samples can be tested under creep using a percentage of this new value as the applied force.

As a final confirmation check, creep testing of representative samples from the horizontal drilling of the pillar freeze holes should be carried out at -20°C, and at the design loads before the gallery entry.

Table 12 - Proposed Phase 1 Budget - Geotechnical and Engineering

Item	Cost
Geotechnical Drilling Program 8 holes to define bedrock contacts and collect samples for geotechnical work	$200,000
Sample Testing geotechnical testing of slum / gravel / bedrock samples	$45,000
Freeze Instrumentation and Design temperature and flow modelling, hardware design	$60,000
Backfill Study	$85,000
Engineering Support revised mine plan based on channel profile confirmation / geotechnical results	$75,000
TOTAL:	**$465,000**

16.0 References

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Appendix I:
Statement of Qualifications

CERTIFICATE OF AUTHOR

I, Stephen Kenwood, P.Geo., hereby certify that:

I am an independent Consulting Geologist and Professional Geoscientist residing at 13629 Marine Drive, White Rock, B.C. V4B 1A3 (Telephone: 604-619-7504)

I graduated from University of British Columbia, Vancouver B.C., in 1987 with a Bachelor's Degree in Science (B.Sc.), in the field of Geology.

I have practiced my profession as a Geologist for the past 29 years since graduation, and I have been involved in exploration for precious and base metals in western North America, Panama, Peru, Chile, Slovakia, Mexico, and China. Although I have limited experience with placer gold projects, my past experience with a wide range of projects at varying stages of development from grassroots to production provides me with the ability to assess the Wingdam Property to be a prospective early stage gold exploration project. More specifically, in British Columbia I worked for Cominco at the Snip Gold Project from 1987-1989 where my experience included planning and overseeing an extensive underground exploration program, including mapping and sampling in underground cross-cuts and raises and overseeing an underground drill program of over 700 drill holes. Also in British Columbia, I was Project Geologist at Eskay Creek from 1989-1991, where my experience included planning and overseeing the inaugural underground exploration on that property for Corona Corp. and Prime Resource Group; duties included overseeing underground development, and mapping and sampling throughout.

I am a registered as a Professional Geoscientist (P. Geo.) in the Province of British Columbia (No 20447) and I am entitled to use the Seal, which has been affixed to this report.

I have prepared this report titled Technical Report, Wingdam Technical Report for Omineca Mining and Metals Ltd. ("Omineca") dated July 27, 2016, based on a visit to the subject property on April 13, 2016, a review of all available data concerning the subject property supplied by the present property owners.

For the purposes of this Technical Report I am a Qualified Person as defined in National Instrument 43-101. I am responsible for all of the items in this technical report, with the exception of Items 10.1 - 10.15, 14.0 and 15.0. I have read the Instrument (NI 43-101) and this report is prepared in compliance with its provisions.

I am not an employee, insider, director or partner of Omineca or any related party to Omineca and have no direct or indirect interest in the property which is the subject of this report. I do not hold, directly or indirectly, any securities in Omineca or any related company to Omineca, nor do I intend to acquire any such securities in Omineca or any related company, in full compliance with all provisions of Section 1.5 of National Instrument 43-101.

At the effective date of the technical report, to the best of the qualified person's knowledge, information, and belief, the technical report, or part that the qualified person is responsible for, contains

contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at White Rock B.C. this July 27th, 2016 (Effective date)

Respectfully submitted,

Stephen Kenwood, P.Geo Qualified Person

CERTIFICATE OF AUTHOR

I, Iain Weir-Jones, P.Eng., Ph.D., FGS, hereby certify that:

I am a Mining Engineer residing at 3126 West 48th Avenue, Vancouver, B.C., V6N 3P6. My place of business is at 598 East Kent Avenue South, Vancouver, B.C., V5X 4V6. My company, Weir-Jones Engineering Consultants Ltd., was established in 1971.

I graduated from University of Newcastle Upon Tyne, UK in 1965 with a Bachelor of Applied Science degree in Mining Engineering. In 1968 I was awarded a Ph.D. in Geomechanics from the same university. I am also a Fellow of the Geological Society of London, FGS.

I have practiced my profession as a Mining Engineer since graduation, and I have been involved in mining and ground control projects in North America, and in more than forty other countries in South America, Africa, Europe, Asia, and Australasia.

I am registered as a Professional Engineer, P.Eng, in the Provinces of British Columbia (No. 8228), and Alberta (No. 108877) and I am entitled to use the Seals. My BC Seal has been affixed to this report.

I am responsible for sections 9.4.1, 10.0, 11.0, 13.0, 14.0, 15.0, 16.0. of the report entitled "Technical Report on the Wingdam Property" dated July 27th, 2016, based upon work carried out under my direct supervision by staff at Weir-Jones Engineering Consultants Ltd.

My most recent visit to the Wingdam property was in 1974.

With regard to Technical Report, I am a Qualified Person as defined in National Instrument 43-101. I have read the Instrument (NI 43-101) and this report is prepared in compliance with its provisions.

I have no direct or indirect interest in the Wingdam property of this report, and I am completely independent of Omineca Mining and Metals Ltd. On this basis, I believe that I am in full compliance with all provisions of Section 1.5 of National Instrument 43-101.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the technical report, or the part for which I am responsible, contains all scientific and technical information that is required to be disclosed to ensure that the technical report is not misleading.

Dated at Vancouver, B.C., this July 27th, 2016 (Effective date).

Yours sincerely,

Iain Weir-Jones, P.Eng., Ph.D., FGS

Appendix II:
Deep Lead Channel
Drift Sample Metalurgical Certificates

ALS Minerals

ALS Canada Ltd.
2103 Dollarton Hwy
North Vancouver BC V7H 0A7
Phone: 604 984 0221 Fax: 604 984 0218 www.alsglobal.com

To: CVG MINING LTD
BOX 4580
QUESNEL BC V2J 3J8

Page: 1
Finalized Date: 7-JUN-2012
Account: CVGMIN

QC CERTIFICATE VA12114777

Project: Wingdam Project
P.O. No.:
This report is for 1 Pulp sample submitted to our lab in Vancouver, BC, Canada on 22-MAY-2012.
The following have access to data associated with this certificate:

STEPHEN KOCSIS	LEN SINCLAIR	

SAMPLE PREPARATION

ALS CODE	DESCRIPTION
WEI-21	Received Sample Weight
LOG-24	Pulp Login - Rcd w/o Barcode

ANALYTICAL PROCEDURES

ALS CODE	DESCRIPTION	INSTRUMENT
ME-GRA24	Au Ag in Bullion FA-GRAV	WST-SIM

To: CVG MINING LTD
ATTN: LEN SINCLAIR
BOX 4580
QUESNEL BC V2J 3J8

This is the Final Report and supersedes any preliminary report with this certificate number. Results apply to samples as submitted. All pages of this report have been checked and approved for release.

Signature:
Colin Ramshaw, Vancouver Laboratory Manager

ALS Minerals

ALS Canada Ltd.
2103 Dollarton Hwy
North Vancouver BC V7H 0A7
Phone: 604 984 0221 Fax: 604 984 0218 www.alsglobal.com

To: CVG MINING LTD
BOX 4580
QUESNEL BC V2J 3J8

Page: 2 - A
Total # Pages: 2 (A)
Finalized Date: 7-JUN-2012
Account: CVGMIN

Project: Wingdam Project

QC CERTIFICATE OF ANALYSIS VA12114777

Sample Description	Method Analyte Units LOR	ME-GRA24 Au fineness 1	ME-GRA24 Ag fineness 1
DUPLICATES			
CC1-PG-1		909	82
DUP		909	82
Target Range - Lower Bound		885	79
Upper Bound		933	85

FAXED

TECHNIC CANADA

UNIT #1 – 3551 VIKING WAY, RICHMOND, B.C. V6V 1W1
PHONE (604)270-8778 FAX (604)270-6529 TOLL FREE 1-800-555-1350

REC.REPORT # 25719
REF.LOT # 11173
FIELD FORM #

C.V.G. MINING
BOX 4580
QUESNEL, B.C. V2J-3M4

DATE RECEIVED: MAY 30/12
DUE FOR SETTLEMENT:

SUPPLEMENTAL INFORMATION SHEET ONLY, THIS IS NOT AN INVOICE

MATERIAL DESCRIPTION: FRASER CANYON PLACER // WINGDAM PLACER

ACTUAL RECEIVED WT: A) 12.720 TR.OZ NET WT. AFTER MELT: A 11.840 TR.OZ
B) 164.500 TR.OZ B) 162.700 TR.OZ

ASSAY OF MATERIAL CONTENT AND PAYOUTS: (Based on Net Weight After Melt)

A)	GOLD:	90.180%	PAYOUT:	99.00%	=	10.570 TR. OZ
	SILVER:	8.550%	PAYOUT:	96.00%	=	0.971 TR. OZ
B)	GOLD:	90.560%	PAYOUT:	99.00%	=	145.867 TR. OZ
	SILVER:	8.020%	PAYOUT:	96.00%	=	12.526 TR. OZ

REFINING CHARGES:

Refining: $ 265.83
Freight:
Other:
HST $ 31.90
Royalties:

TOTAL CHARGES: $ 297.73

TECHNIC CANADA LTD., PER Keith Law

DETAILS / COMMENTS: