RESOURCE CAPITAL GOLD CORP

Preliminary Economic Assessment

Dufferin Gold Deposit, Nova Scotia

RESOURCE CAPITAL GOLD CORP

Revised Preliminary Economic Assessment

of the

Dufferin Gold Deposit

located in

Nova Scotia, Canada

45° 00′ North, 62° 24′ West

NTS 11D/16C

April 3, 2017 (Revised)

Original Dated December 30, 2016

Effective Date December 30, 2016

By

Patrick Hannon, M.A.Sc., P.Eng., MineTech International Limited

Wm. Douglas Roy, M.A.Sc., P.Eng., MineTech International Limited

Greg Mosher, M.Sc., P.Geo., Global Mineral Resource Services

MINETECH INTERNATIONAL LIMITED

HALIFAX, CANADA

Table of Contents

Appendices

Error! Not a valid heading level range.

Index of Figures

Index of Tables

1 Summary

Introduction

This report is a Preliminary Economic Analysis (PEA) of the Dufferin Property (the Property) with the presently defined Mineral Resources. The mining plan is based on using underground hard rock mining, trackless equipment and conventional gravity-flotation processing. A gold doré brick and a gold flotation concentrate will be the saleable items from the mine.

The Property consists of two portions, “East Dufferin” and “West Dufferin” that are fault-separated. At East Dufferin, Resource Capital Gold Corp., through its subsidiary Maritime Dufferin Gold Corp., currently holds three separate but contiguous exploration licences consisting of 54, 40-acre claims for a total of 2,160 acres (874 ha). Exploration License 50561 covers the known mineral resources at East Dufferin. This licence was previously held as Mining Lease 94-2. The company also holds an Environmental Approval, and Industrial Approval for the East Dufferin portion of the Property, both of which remain in place and valid.

At West Dufferin, through a separate subsidiary, Resource Capital Gold Corp. owns two exploration licenses contiguous with those at East Dufferin, comprising 50, 40-acre claims for a total of 2,000 acres (809 ha). These are Exploration License 06219, which covers the known mineral resources at West Dufferin, and Exploration License 06271, which joins 06219 to the west. The total project thus comprises 104, 40-acre claims totalling 4,160 acres (1,683) under the control of Resource Capital Gold Corp.

The Property is accessible by 8 kilometres of all-weather gravel road from Port Dufferin. Port Dufferin is on the Eastern Shore of Nova Scotia, approximately 140 kilometres east-north-east of the city of Halifax. Surface rights for the facilities are owned by Resource Capital Gold Corp. Parts of the tailings management facility are on private land. The company has an agreement with this landowner. The other landowner is the Province (Crown Land), and the company has a lease with the Province for the use of the land in connection with the operation of the Property..

Gold was discovered at West Dufferin in 1880, and underground work began at the East Dufferin site in 1994. Gold mineralization is found in quartz veins, the veins being concentrated on the crest and limbs of an anticlinal fold. Several companies have held the properties, completing surface drilling and extending the underground workings. Since 2008, surface and underground diamond drilling, metallurgical studies, and a limited amount of underground production has been carried out at East Dufferin.

Caution to the Reader

This preliminary economic assessment (PEA) is preliminary in nature, in that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. A PEA is a conceptual study of the potential viability of mineral resources. The economic viability of the inferred mineral resources has not been demonstrated.

Geology

The Dufferin deposit is a turbidite-hosted gold deposit. This type of gold deposit is hosted in sediments that were deposited in deep water, the result of a turbidity current of sediment flowing down the continental slope.

Exploration and mining of gold in the Meguma Group of Nova Scotia has taken place since the 1860s. The gold fields of Nova Scotia occur within Meguma Structural Terrane composed predominantly of Cambrian to Lower Devonian sediments and turbiditic metasediments. These sediments have been subdivided into two groups, the Cambro-Ordovician Meguma Group, which consists of the basal Goldenville Formation quartz-wackes, the overlying Halifax Formation slates, and the Silurian to Lower Devonian clastic sediments and volcanics.

The host rocks for the Dufferin deposit were subject to compression forces after deposition. These forces folded the rocks into a series of anticlines and synclines.

Late during the Acadian Orogeny the closure of the proto-Atlantic created a zone of intense compression. These forces folded the rocks into a series of anticlines and synclines. Granitoid plutons intruded the Meguma Group, through the re-melting of the lower part of the package. This mass of melted material further deformed and metamorphosed the turbidite sequence. The east-west folding created a penetrative slaty cleavage perpendicular to the fold axis in the argillite units. This cleavage played an important role in the development of the gold deposits. Hydrothermal, structurally controlled, concordant, auriferous quartz veins were squirted into zones of lesser pressure, such as the crest.

At the Dufferin deposit, this folding created a series of stacked saddle-veins draped over a chevron-style anticlinal fold called the Crown Reserve Anticline at East Dufferin. Previous mining on this anticline, at the Crown Reserve and Maple Leaf mines, consisted of minor development on the south limb of the fold. At West Dufferin, this same anticline is called the Dufferin Mines Anticline, and was explored and mined historically beginning in 1868. Mineralization on the Dufferin project comprises one deposit, cut roughly in half and offset along the Harrigan Cove fault; the eastern half is referred to as East Dufferin, and the western half as West Dufferin.

Previous Work

Gold was first discovered in the project area in 1868, at West Dufferin. Various enterprises conducted early production at West Dufferin and the first modern exploration was conducted by Cominco in 1935 – 1942 with the first drilling on the property, and sinking of a new shaft. Various companies owned West Dufferin and drilled selected holes between then and 1986, when Jascan Resources began its three campaigns of drilling. This totalled 60 drill holes, which constitutes the bulk of drilling on the project. In 2010, Nycon Resources Inc., the project vendor, acquired the project and has since drilled 40 holes on the project, validating the old drilling to NI 43-101 standards.

At East Dufferin, exploration in the early 1980s led to the discovery of the gold mineralization in saddle-veins. This work culminated in an "in-house" feasibility study on the property by Corner Bay Minerals Inc. in 1989.

There are no prior NI 43-101-compliant estimates of Mineral Resources for the Dufferin Property. Several operators completed historical mineral resource estimates for the deposit.

These estimates are historical in nature, were not prepared in compliance with the CIM definitions required by National Instrument 43-101 and should not be relied upon.

Newfoundland Goldbar Resources resumed exploration on the property in 1999. They carried out diamond drilling and a limited amount of production.

In 2003, Azure Resources Corp. acquired the Dufferin Property. Azure dewatered the mine, sampled portions of the first and second saddles (the uppermost saddles), surveyed the surface and tailings pile, carried out metallurgical studies and refurbished the camp and mill. A limited amount of production was carried out.

In 2006, Jemma Resources added a gold flotation circuit to the existing mill and reprocessed the existing tailings. A total of 1,602 ounces (49,828 grams) was recovered from 31,745 tonnes of tailings.

A 2012 Technical Report on the property recommended a three phase program including underground sampling, diamond drilling, and feasibility work. Ressources Appalaches completed part of the recommended work while attempting to put the mine back into production. They carried out drilling, refurbished the mill, and carried out a limited amount of production.

The head grade of the material processed through the mill was approximately 2 grams per tonne. Approximately 3,700 tonnes out of the 18,000 were above the 3 grams/tonne cut-off, averaging about 6 grams per tonne. Included with the 3,700 tonnes were 570 tonnes with a head grade of about 16 grams/tonne.

In September 2014 the mine and plant were put on care and maintenance.

Resource Capital Gold Corp purchased the East Dufferin portion of the project in August 2016, and the West Dufferin portion in November, 2016.

Current Mineral Resource Estimate

An updated mineral resource estimate was prepared for this report.

Exploratory data analysis indicates that 200 g/tonne is an appropriate top-cut grade for limiting the effect of high-grade outliers.

Ordinary block kriging was used for block grade estimation. A 2 g/tonne block-cut was used for identifying mineral resources. For the entire deposit, including both West Dufferin and East Dufferin, at a 2 g/tonne block cut-off grade, there are 151,000 non-diluted tonnes at a grade of 11.9 g/tonne, for 58,000 *in situ* ounces in the indicated resource category, and 703,000 non-diluted tonnes at a grade of 6.6 g/tonne giving 150,000 *in* situ ounces in the inferred category. Saddle 6 is the deepest saddle that contains a significant quantity of +2 g/tonne material.

The extent of the main block of resources, at a 2 g/tonne block cut-off, is 2170-2850 metres East and 800-1045 metres elevation (using the site grid), or 680x245 metres in longitudinal area. A smaller area of +2 g/tonne blocks is centred on Section 3260 metres East.

Mineral Resources were subdivided into crest and leg domains because they would likely be mined using different mining methods.

No Mineral Reserves were identified.

	Volume (m^3)	Tonnes	Ounces	Average Grade (g/tonne)
East Dufferin				
Indicated	57,200	151,500	58,000	11.9
Inferred	163,800	434,100	96,800	6.9
West Dufferin				
Inferred	101,800	269,800	53,200	6.1
Combined				
Indicated	57,200	151,500	58,000	11.9
Inferred	265,600	703,900	150,000	6.6

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65;
[3] Gold price = $US 1250 per ounce.
[4] West Dufferin top-cut: 100 g/tonne; East Dufferin top-cut: 200 g/tonne.

Infrastructure

Infrastructure at the East Dufferin site includes facilities required for a 300 tonne/day mining operation. Current infrastructure includes:

- Underground workings, in good condition, including 2,253 m of underground development over 5 levels reaching Saddles 1 through 4, with a vertical depth of 100 m and an east-west extent of 470 m (see Table 14-3). A very small section is currently flooded, as a sump. Some areas require rehabilitation.
- A 300 tpd gravity/flotation mineral processing facility, including a crusher and a grinding mill, currently under care and maintenance;
- Three-phase, grid-connected power;
- A workshop; and,
- Several office and camp trailers.

Tailings Facility

The current tailings impoundment is in good condition, though it is nearly full. It would need to be expanded in the near term.

Mining Plan

Inferred Resources are used in this PEA to indicate the direction for the eventual development of the deposit. Indicated Resources are also used.

A conceptual mining plan for the deposit was developed in order to provide an idea of the project's capital and operating costs. Inferred Resources are used in this PEA to indicate the direction for the eventual development of the deposit. Indicated Resources are also used.

Further development is required before the operation can have enough working faces to provide 9,000 to 10,000 tonnes per month to the processing plant.

Management may decide that Dufferin will be mined by contractor; if so, this PEA can always be used as a basis for contract mining arrangements. Contract mining has several advantages when there is a short mine life, especially if the contractor is working on target prices with bonuses and penalties. Some operations start with a contractor who gets the operation going, trains the crews, sets up an efficient mine and leaves after a period of time.

Narrow vein mining is required for the fold limbs and drift and slash mining is planned for the crest of the veins. Approximately 64% of the gold is in the limbs and 36% is in crest areas, so the dominant mining method has to consider thin dipping structures which must be defined prior to mining.

After definition drilling, the limb portions of the saddles will be accessed by crosscutting from a haulageway to the vein limb. A sill wide enough for a mechanical excavation would then follow vein by drifting along the strike and down the plunge of the saddle vein where the vein begins to pinch out down dip. A raise would connect the sill to the crest portion of the saddle. Stoping depends upon the width and continuity of the limb. Some dextral faulting can be expected to offset the deposit to the north as mining is extended to the east and this will have an effect on the mining method chosen for a particular saddle vein. A longhole method has been assumed in this PEA for cost purposes.

The thermal fragmentation method should be tested at Dufferin. There are several North American mines that have tested the method and the company with the patented technology is interested in lease arrangements. Dilution is kept very low and veins as thin as 30cm can be mined.

Other narrow vein mining methods such as modified shrinkage, cut and fill, resuing and stull stoping can be considered but these methods are labour intensive.

An overall gold processing recovery of 95% has been used in this study.

Capital and Operating Costs

All capital items would be purchased during the pre-production period and the first three months of production. It is assumed that the majority of equipment and facilities will be acquired through lease agreements.

The capital costs and operating costs are summarized below.

Major Items and startup inventory	total Year 1	Year 2
Pre-Production Mine Exploration & Development Year	$3,265,000	
Upgrade Tailings Facility	$250,000	
Mine Capital	$587,375	$136,000
Mill Capital, miscellaneous	$1,049,100	$203,000
Diamond Drilling	$100,000	$320,000
Construction supervision	$40,000	
Miscellaneous equipment and Contingency, 20%	$1,060,000	$140,000
Total	**$6,360,000**	**$840,000**

Economic Analysis

As part of this PEA, a cash flow was performed for the Dufferin Mine Project using the capital and operating costs generated for the mine plan. This economic analysis includes both East Dufferin and West Dufferin.

The cash flow does not include considerations for outstanding debts from previous operations, nor does it take in to account any possible corporate tax credits that a potential buyer may hold from other operations. A base case gold price of US$1,250/oz. and a base case exchange rate of 0.75 CDN/US was used for economic modelling.

Sensitivities were performed; the project was found to be most sensitive to, in order, the Canadian dollar exchange rate and grade, the price of gold, operating cost and capital cost.

Economic results in the PEA show the production of 216,050 ounces of gold during a 10-year mine life, with a pre-tax IRR of 158% and a capital payback period of 1.3 years with a pre-tax net present value of CAD$121,100,000 at a 5% discount rate. These economic numbers do not include any of the planned production from the stockpiled materials described in the November 9 press release, which would be in addition to such amounts. An after tax cash flow of $89 million was estimated for the project. The pre-tax and post-tax cash flow estimates are summarized in the table below.

The PEA includes a 6-month pre-production period followed by 6 months of initial mining ramping up to full 300-tpd production at the end of Year 1. The pre-production period has already started, with Company staff onsite for the last two months in preparation for beginning mining activities.

The PEA anticipates at total of CAD$9.85M in capital expense, which includes $5.29M in plant, equipment, and pre-production development cost; $1.20M of reclamation bonding; $2.29M of working capital; and $1.06M in contingency.

The following economic results do not include any of the planned production from the stockpiled materials described in the November 9 press release, which would be in addition to such amounts.

Economic Analysis Assumptions and Results

Gold price	USD$1,250
Exchange rate USD-CAD	0.75
Total tonnes processed (includes dilution and development material)	1,231,000 tonnes
Average processing rate	290 tonnes per day
Diluted head grade East Dufferin	6.83 g/tonne Au
Diluted head grade West Dufferin	5.46 g/tonne Au
Gold recovery rate	95%
Average annual gold production	21,604 ounces Au
Capital, working capital, pre-production expense	CAD$9.85M
Pre-tax NPV_5	CAD$121.1M
Pre-tax IRR	158%
Post-tax NPV_5	CAD$89.2M
Post-tax IRR	121%
Payback	1.3 years
Average cash cost per ounce	CAD$617
Mine life	10 years
Total gold ounces recovered	216,050
Cumulative pre-tax cash flow	CAD$170.4M
Cumulative post-tax cash flow	CAD$126.3M

Conclusions

This underground saddle vein gold deposit has good potential to produce for several years if there are good accounting, good grade control and sensible mining engineering practices.

The deposit is hosted by hard rocks of the metamorphosed and folded Meguma Group Goldenville Formation, a turbidite series of rocks originally deposited along the coast of an ocean existing prior to the present Atlantic Ocean. The folded rocks were injected with quartz veins during a time of compression of the rocks as the ocean before the Atlantic closed and the Appalachian mountains were thrust up like the Himalayans are today.

Several companies have invested in this gold project – a 300 tonne per day processing plant has been built and a considerable amount of underground development has been done. There are reasonable surface facilities, good access and a well trained work force available in the immediate area.

Resue mining can be done using long period and millisecond delays when blasting. Leg mining will have to use a narrow vein mining method and there are several methods, in particular thermal fragmentation and longhole drilling.

Recommendations

Phase 1 recommendations include further diamond drilling, metallurgical work, test stoping, and preliminary tailings impoundment design work. This phase is expected to cost $590,000.

Phase 2 consists of a prefeasibility study to (a) determine whether, updated and more detailed information, the project remains viable, and (b) establish Mineral Reserves. This phase is expected to cost $130,000.

Mineral Resource Report and Preliminary Economic Assessment for the Dufferin Gold Property

2 Introduction

This report is a Preliminary Economic Analysis of the Dufferin Gold Deposit, located on the Eastern Shore of Nova Scotia. This report was originally commissioned in November 2014 by Ressources Appalaches, which was since put into receivership. Resource Capital Gold Corp ("Resource Capital Gold") is a Canadian mineral exploration company that currently holds the Dufferin gold deposit. This report was updated to reflect the property's current ownership.

This report is in compliance with National Instrument 43-101 (Standards of Disclosure for Mineral Projects) and Form 43-101F1 (Technical Report and Related Consequential Amendments).

MineTech International Limited ("MineTech") is an independent geological and mining engineering consulting firm based in Halifax, Nova Scotia. Global Mineral Resource Services (GMRS) is an independent geological and resource estimation consulting company based in Vancouver, British Columbia.

On June 11th, 2009, MineTech released an NI43-101 compliant Technical Report on the property (MineTech, 2009). An updated report was completed in May of 2012. Since the 2012 report, the mill was refurbished, the tailings pond was enlarged, the underground mine was pumped out, surface and underground diamond drilling was carried out, and there was some underground development and production of the mine.

The Property consists of two portions, "East Dufferin" and "West Dufferin." At East Dufferin, Resource Capital Gold Corp., through its subsidiary Maritime Dufferin Gold Corp., currently holds three separate but contiguous exploration licences consisting of 54, 40-acre claims for a total of 2,160 acres (874 ha). Exploration License 50561 covers the known mineral resources at East Dufferin. This licence was previously held as Mining Lease 94-2. The company also holds an Environmental Approval, and Industrial Approval for the East Dufferin portion of the Property, both of which remain in place and valid.

At West Dufferin, through a separate subsidiary, Resource Capital Gold Corp. owns two exploration licenses contiguous with those at East Dufferin, comprising 50, 40-acre claims for a total of 2,000 acres (809 ha). These are Exploration License 06219, which covers the known mineral resources at West Dufferin, and Exploration License 06271, which joins 06219 to the west. The total project thus comprises 104, 40-acre claims totalling 4,160 acres (1,683) under the control of Resource Capital Gold Corp.

2.1 Purpose of Report

This report was prepared to provide a Preliminary Economic Assessment (PEA), as defined in National Instrument 43-101, of the Property. It uses information gathered during drilling and underground work. National Instrument 43-101 (NI 43-101) defines a PEA as "a study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources."

2.2 Caution to the Reader

The reader is cautioned that this PEA uses Inferred Mineral Resources. NI 43-101 Part 2, Section 2.3(1)(b) and Companion Policy 43-101CP, Part 2, Section 2.3(1) Restricted Disclosure, prohibits the disclosure of the results of an economic analysis that includes or is based on inferred mineral resources, an historical estimate, or an exploration target.

The "CIM Definition Standards," adopted on May 10, 2014 define an Inferred Mineral Resource as:

> *"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*
>
> *"An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
>
> *"An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.*
>
> *"There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource."*

Companion Policy 43-101CP, Part 2, Section 2.3(1), Restricted Disclosure states that:

"CIM considers the confidence in inferred mineral resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. [National Instrument 43-101] extends this prohibition to exploration targets because such targets are conceptual and have even less confidence than inferred mineral resources. The Instrument also extends the prohibition to historical estimates because they have not been demonstrated or verified to the standards required for mineral resources or mineral reserves and, therefore, cannot be used in an economic analysis suitable for public disclosure."

However, under NI 43-101, Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 Section 2.3(3), a Preliminary Economic Assessment is allowed to use inferred mineral resources and to carry out an economic assessment in order to inform investors of the potential of the property. Investors must be informed that the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and that there is no certainty that the production and/or economics foreseen by the preliminary economic assessment would be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

2.3 Sources of Information

Much of the data that was used to prepare this report was provided by Ressources Appalaches and Nycon Resources, the former mineral rights holders for the Dufferin Mine property and the Dufferin East property.

MineTech and GMRS have reviewed the data provided by Ressources Appalaches and Nycon Resources and/or by their agents. The authors have also reviewed other information sources, such as databases hosting technical reports and mineral rights information for the province of Nova Scotia.

This report contains references to geological information obtained from government geologists and mineral resource estimates prepared by other Professional Geoscientists and previous operators of the Dufferin Property. This report constitutes part of Resource Capital Gold Corp's disclosure record and will be available on SEDAR.

MineTech and GMRS have conducted a review and appraisal of the information used in the preparation of this report, and believes that the information used in the preparation of the report, as well as for its conclusions and recommendations, is valid and appropriate considering the status of the project and the purpose for which the report is prepared.

2.4 Site Visits

The authors, Mr. Patrick Hannon, M.A.Sc., P.Eng., and Mr. Douglas Roy, M.A.Sc., P.Eng. visited the Dufferin Mine property numerous times. Mr. Roy and Mr. Hannon last visited the property on December 14, 2016.

Mr. Hannon first visited the Dufferin Mine Property in 2000, while working as Director of Mines for the Department of Natural Resources. He first visited the East Dufferin portion of the property in 1987 while working for ACA Howe Int. Ltd.

Mr. Roy first visited the area in 2003, when he completed a survey of the surface facilities using a transit and EDM. Mr. Hannon and Mr. Roy also visited the property several times between 2012 and December 2014. These visits were generally between a month and 5 weeks apart. Mr. Hannon and Mr. Roy also provided Ressources Appalaches with engineering support on an as-needed basis during 2013 and 2014.

Mr. Hannon and Mr. Roy are very familiar with the property. They have examined the surface geology, have examined the underground geology and underground mine numerous times, and have examined drill core and taken independent samples.

Mr. Roy and Mr. Hannon completed various jobs on site including surface surveying, geophysical surveys (VLF), tailings dam inspection and reporting, and dewatering studies.

At West Dufferin, old foundations and old mine shafts were observed. During the most recent site visit on December 14, 2016, the ground was snow-covered which prevented the examination of historical drilling sites. The area has been clear cut several times so travel over the slash is a challenge. There is very little outcrop.

Photos were always taken during the field visits.

2.5 Authorship

Mr. Roy completed the sections on Mineral Resource Estimates for Dufferin Mine and East Dufferin sites, and contributed to all other chapters of the report. Mr. Dickie contributed the chapters on Mining Methods, Recovery Methods, Project Infrastructure Capital and Operating Costs and the Economic Analysis of the project in the 2014 PEA report. These sections were updated and revised by Mr. Hannon for this 2016 report.

Mr. Hannon and Mr. Roy, as Qualified Persons, take responsibility for all sections of the report except for those authored by Mr. Mosher.

Mr. Mosher of Global Mineral Resource Services completed the sections on Mineral Resource Estimates for West Dufferin and is responsible for those portions of the report.

Ms. Leah Page, P.Geo., completed the wireframes in support of resource modeling at West Dufferin. Mr. Greg Mosher critically reviewed Ms. Page's work and takes full responsibility for it.

Table 2-1 - Authorship Table

Chapter	Title	Author(s)		
1	Summary	Patrick Hannon	Doug Roy	
2	Introduction	Patrick Hannon	Doug Roy	
3	Reliance on Other Experts	Patrick Hannon	Doug Roy	
4	Property Description and Location	Patrick Hannon		
5	Accessibility, Physiography, Climate, Local Resources, and Infrastructure	Patrick Hannon		
6, 6.1	History – East Dufferin	Patrick Hannon		
6.2	History – West Dufferin			Greg Mosher
7	Geological Setting and Mineralization	Patrick Hannon,		
8	Deposit Types	Patrick Hannon		
9, 9.1	Exploration – East Dufferin	Patrick Hannon		
9.2	Exploration – West Dufferin			Greg Mosher
10	Drilling	Patrick Hannon		
11	Sample Preparation, Analysis and Security		Doug Roy	
12	Data Verification		Doug Roy	
13	Mineral Processing and Metallurgical Testing	Patrick Hannon		
14	Mineral Resource Estimates		Doug Roy	Greg Mosher
15	Mineral Reserve Estimates		Doug Roy	
16	Mining Methods	Patrick Hannon		
17	Recovery Methods	Patrick Hannon	Doug Roy	
18	Project Infrastructure	Patrick Hannon		
19	Market Studies and Contracts	Patrick Hannon		
20	Environmental Studies, Permitting, and Social or Community Impacts		Doug Roy	
21	Capital and Operating Costs	Patrick Hannon		
22	Economic Analysis	Patrick Hannon		
23	Adjacent Properties	Patrick Hannon	Doug Roy	

24	Other Relevant Data and Information	Patrick Hannon	Doug Roy	
25	Interpretations and Conclusions	Patrick Hannon	Doug Roy	Greg Mosher
26	Recommendations	Patrick Hannon	Doug Roy	Greg Mosher
27	References	Patrick Hannon	Doug Roy	

3 Reliance on Other Experts

MineTech and GMRS have not relied on any other experts in the commission of this report.

4 Property Description and Location

4.1 Location and Access

The Dufferin Property is located in Halifax County, Nova Scotia, on the Eastern Shore of Nova Scotia, approximately 140 kilometres by paved highway east-northeast of Halifax and 8 kilometres north from the settlement of Port Dufferin. Access from Provincial Highway 7 connecting at Port Dufferin is by the gravelled Dufferin Mines Road. Logging roads provide good access to other parts of the property (see Figure 4-1 and Figure 4-2).

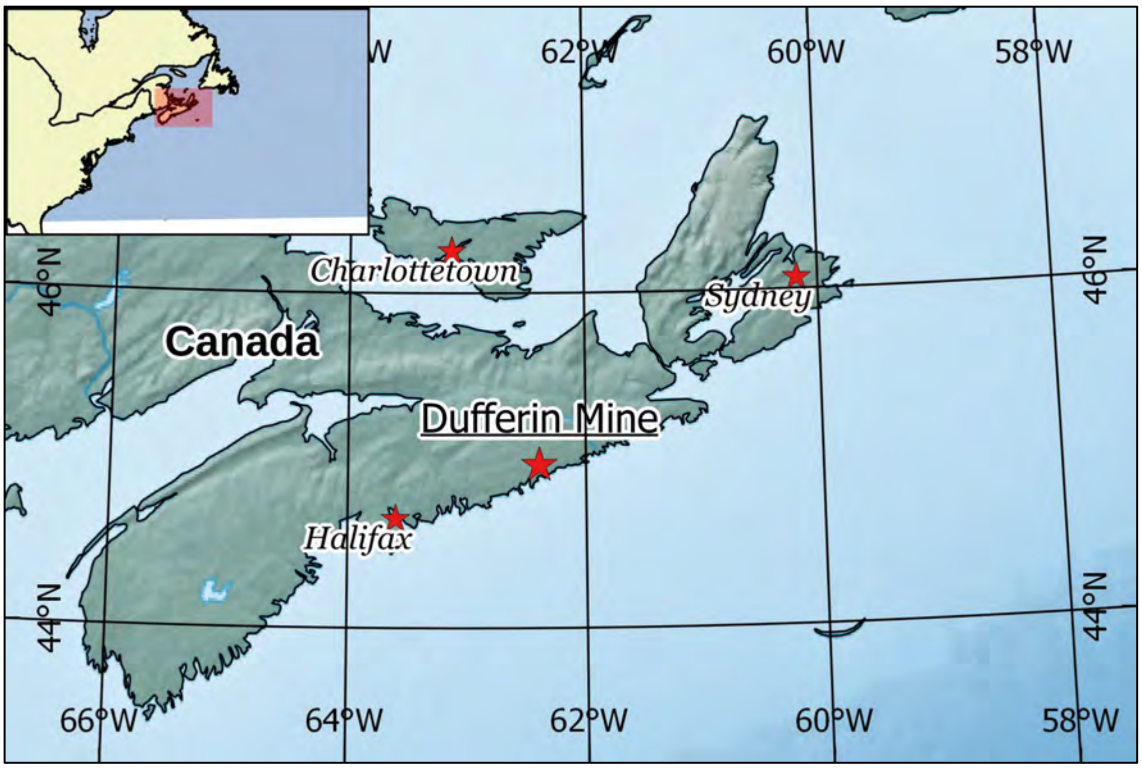


Figure 4-1 - Location Map

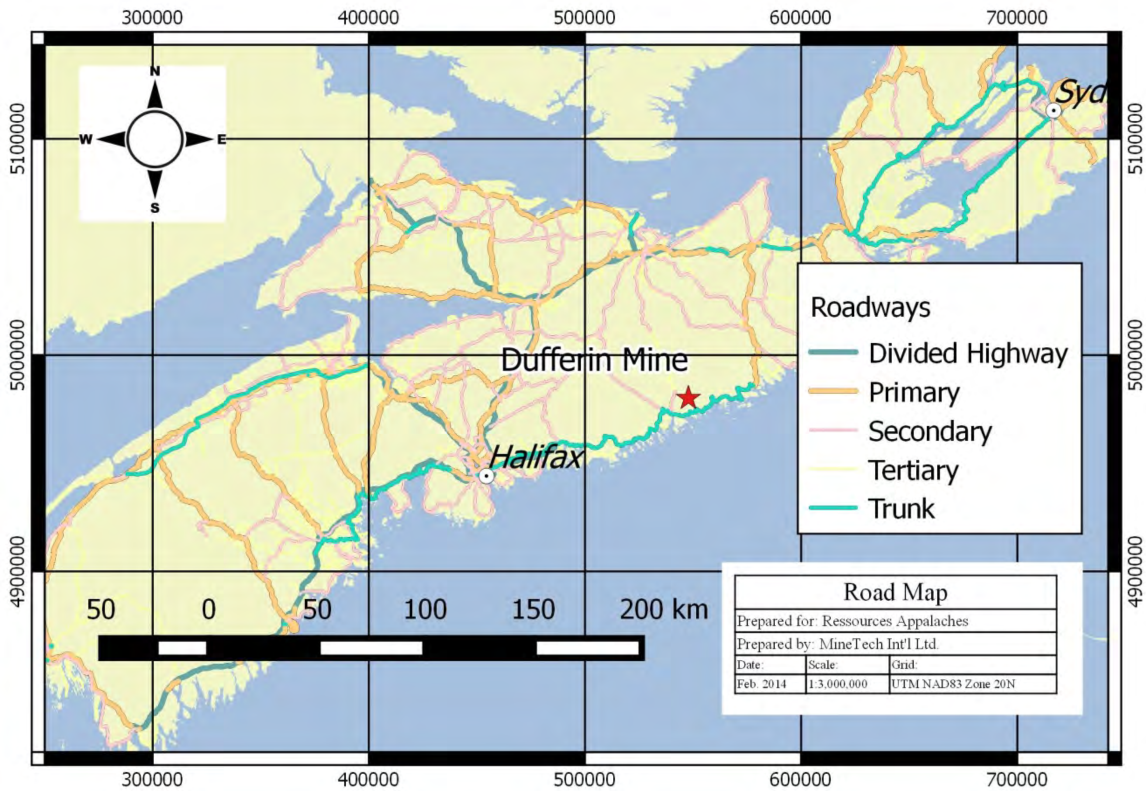


Figure 4-2 - Road Map.

The approximate centre of the property is at 44° 58’ 33” North Latitude and 62° 23’ 08” West Longitude on NTS map-sheet 11D/16C. The UTM coordinates of the approximate centre of the Property are 4,980,000N and 548,000E using UTM NAD 83 Coordinate Zone 20N.

4.2 Mining Rights and Permits

Section 6.1 gives the history of the property title as well as Resource Capital Gold Corp’s purchase of the property. The Property consists of five separate but contiguous exploration licences consisting of 104, 40-acre claims for a total of 4,161 acres or 1,684 ha (see Figure 4-3 and Table 4-1). The Licenses are valid for all minerals with the exception of uranium, salt, potash, and coal, minerals in abandoned tailings and waste rock, and geothermal resources.

The Property consists of two portions, “East Dufferin” and “West Dufferin.” At East Dufferin, Resource Capital Gold Corp., through its subsidiary Maritime Dufferin Gold Corp., currently holds three separate but contiguous exploration licences consisting of 54, 40-acre claims for a total of 2,160 acres (874 ha). Exploration License 50561 covers the known mineral resources at East Dufferin. This licence was previously held as Mining Lease 94-2. The company also holds an Environmental Approval, and Industrial Approval for the East Dufferin portion of the Property, both of which remain in place and valid.

At West Dufferin, through a separate subsidiary, Resource Capital Gold Corp. owns two exploration licenses contiguous with those at East Dufferin, comprising 50, 40-acre claims for a total of 2,000 acres (809 ha). These are Exploration License 06219, which covers the known mineral resources at West Dufferin, and Exploration License 06271, which joins 06219 to the west.

The MRA Regulations, section 62 (1) states that "A licensee shall obtain a letter of authorization, in the prescribed manner, before commencing bulk sampling for the purpose of extracting 100 tonnes or more of mineral-bearing material."

Table 4-1 A Tabulation of the Exploration Licences controlled by Resource Capital Gold Corp.

Right Number	NovaROC ID	Holder Name	Right Type	Location	Issue Date	Expiry Date	Age	No. of Claims	Status
07351	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 77 CLAIMS N MAP 11D16C TRACTS 78 CLAIMS N,O,P,Q MAP 11D16C TRACTS 90 CLAIMS J,K,P,Q MAP 11D16C TRACTS 91 CLAIMS A,B,C,F,G,H,J,K,L,M,N,O,P,Q MAP 11D16C TRACTS 92 CLAIMS D,E,M,N	2007-05-17	2017-05-17	10	27	Active
08619	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 80 CLAIMS L,O,P MAP 11D16C TRACTS 89 CLAIMS B,C,F,G,H,J MAP 11D16C TRACTS 90 CLAIMS E,F,L,M	2009-05-20	2017-05-20	8	13	Active
50561	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 79 CLAIMS N,O,P,Q MAP 11D16C TRACTS 80 CLAIMS Q MAP 11D16C TRACTS 89 CLAIMS A MAP 11D16C TRACTS 90 CLAIMS A,B,C,D,G,H MAP 11D16C TRACTS 91 CLAIMS D,E	1986-09-02	2017-09-02	30	14	Active
51383	564782	Maritime Dufferin Gold Corp.	Mineral Lease	MAP 11D16C TRACTS 79 CLAIMS N,O,P,Q MAP 11D16C TRACTS 80 CLAIMS Q MAP 11D16C TRACTS 89 CLAIMS A MAP 11D16C TRACTS 90 CLAIMS A,B,C,D,G,H MAP 11D16C TRACTS 91 CLAIMS D,E			0	14	Active
The West Dufferin properties, consisting of exploration licenses 06219 and 06274, were purchased by Resource Capital Gold Corp., through a separate subsidiary company, 100% from Nycon Resources Inc.									
6219	201519	NYCON RESOURCES, INC.	Mineral Exploration Licence	MAP 11D16C TRACTS 65 CLAIMS J,K,L,M,N,O,P,Q MAP 11D16C TRACTS 66 CLAIMS M,N,O,P,Q MAP 11D16C TRACTS 79 CLAIMS A,B,C,D,E,F,G,H,J,K,L,M MAP 11D16C TRACTS 80 CLAIMS A,B,C,D,E,F,G,H,J,K	26/07/1979	26/07/2017	38	35	Active
6274	201519	NYCON RESOURCES, INC.	Mineral Exploration Licence	MAP 11D16C TRACTS 63 CLAIMS E,F,J,K,L,M,P,Q MAP 11D16C TRACTS 64 CLAIMS M,N,O,P,Q MAP 11D16C TRACTS 81 CLAIMS A,B	28/09/2001	28/09/2017	16	15	Active

		Exploration Licence	Area (ha)	Area (acres)	Claims
East	Dufferin	50561	227	561	14
		08619	210	519	13
		07351	437	1080	27
West	Dufferin	06219	567	1401	35
		06274	243	600	15
		Totals	1,684	4,161	104

Figure 4-3 Surface Area of the Licences Held by RCG.

The exploration licences must be renewed each year. To renew all of the claims for another year will cost $18,860 plus tax and at least $59,000 worth of assessment work will have to be submitted, for a total of $77,860 plus tax (see Table 4-1).

Annual renewal fees and work assessment requirements depend upon the age of the mineral right. Section 35 of the Regulations states that "The minimum value of acceptable assessment work that must be submitted for the renewal of an exploration licence is:

Year of Licence	Dollars per Year per Claim
1st to 10th	$200
11th to 15th	$400
16th and after	$800

And, section 70 outlines the Fees payable under the Act.

For the issuance of exploration licence it is $10.00/claim. The annual renewal of the exploration licence depends upon the age of the licence:

for years

2 to 10...$20.00/claim

11 to 15...$40.00/claim

16 to 25...$160.00/claim

26 and after...$320.00/claim

The rental for a lease is $120.90/claim/year.

An exploration licence holder can make a payment in lieu of assessment work once in a five year period. For years

1 to 10...$242.00/claim

11 to 15...$484.05/claim

16 and after...$968.05/claim

A Mineral Lease is required to reopen the mine. The Property previously held this permit. Government officials have stated in meetings that, if there are no significant changes to the plans, the previously-submitted application can be re-submitted and 'should' be able to be approved quickly.

A report to describe the expanded Dufferin East and Dufferin West would be required with Form 14, the Application for Lease.

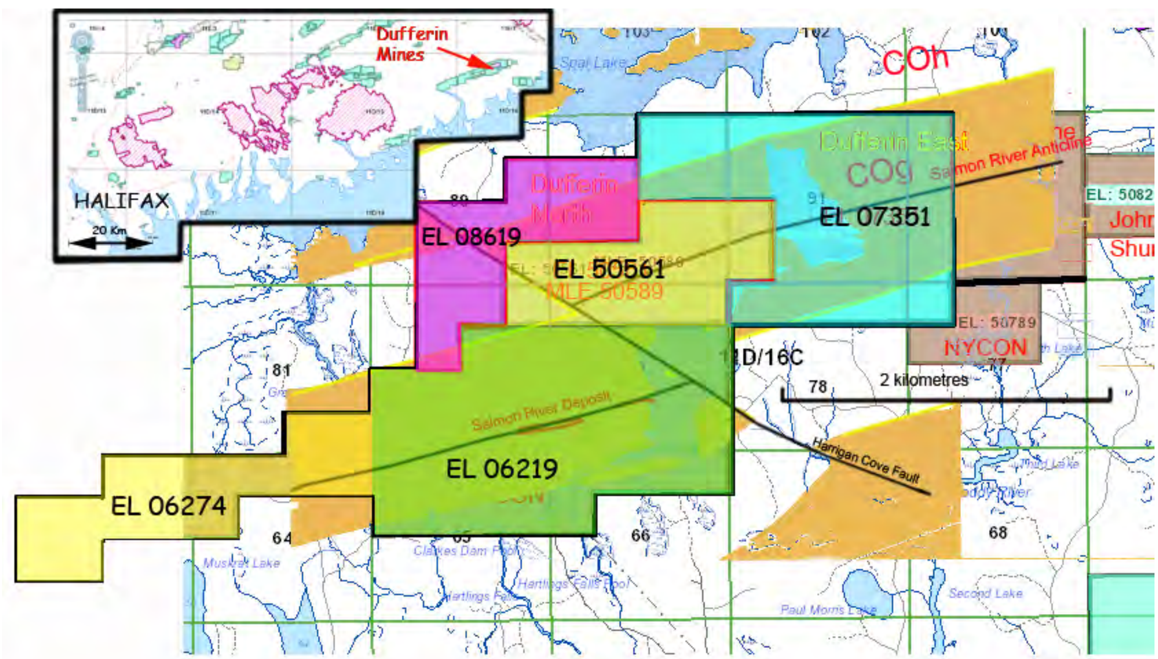


Figure 4-4 - Claim Map illustrating the Exploration Licences controlled by / optioned by RCGC.

4.3 Surface Rights

4.3.1 East Dufferin Surface Rights

Surface rights for the portion of the mine property affected by mining operations are owned by Resource Capital Gold Corp, John McLellan and the Crown (see Figure 4-4). The portal for the decline lies on Crown land that is leased by Resource Capital Gold Corp. Parts of the tailings management facility and other works impinge on the property of John McLellan. Ernst & Young confirmed in March, 2015 that that the leases covering these lands are in good standing.[1]

Landholders controlling surface rights within the mining lease include the Crown, Northern Timber Nova Scotia Corp., Resource Capital Gold Corp, Sandra I. Murphy, John McLellan, Susan I. Gammon, Ralph D. Balcom, and Hugh R. Schofield.

There are no known aboriginal rights issues in the area. Any future development mineral claims where the Crown is the landowner requires the mineral rights holder to consult with First Nations prior to developing that part of the mineral right.

[1] Personal communication with George Kinsman, March, 2015

4.3.2 West Dufferin Surface Rights

West Dufferin surface rights are illustrated in Figure 4-5 with details in Table 4-2.

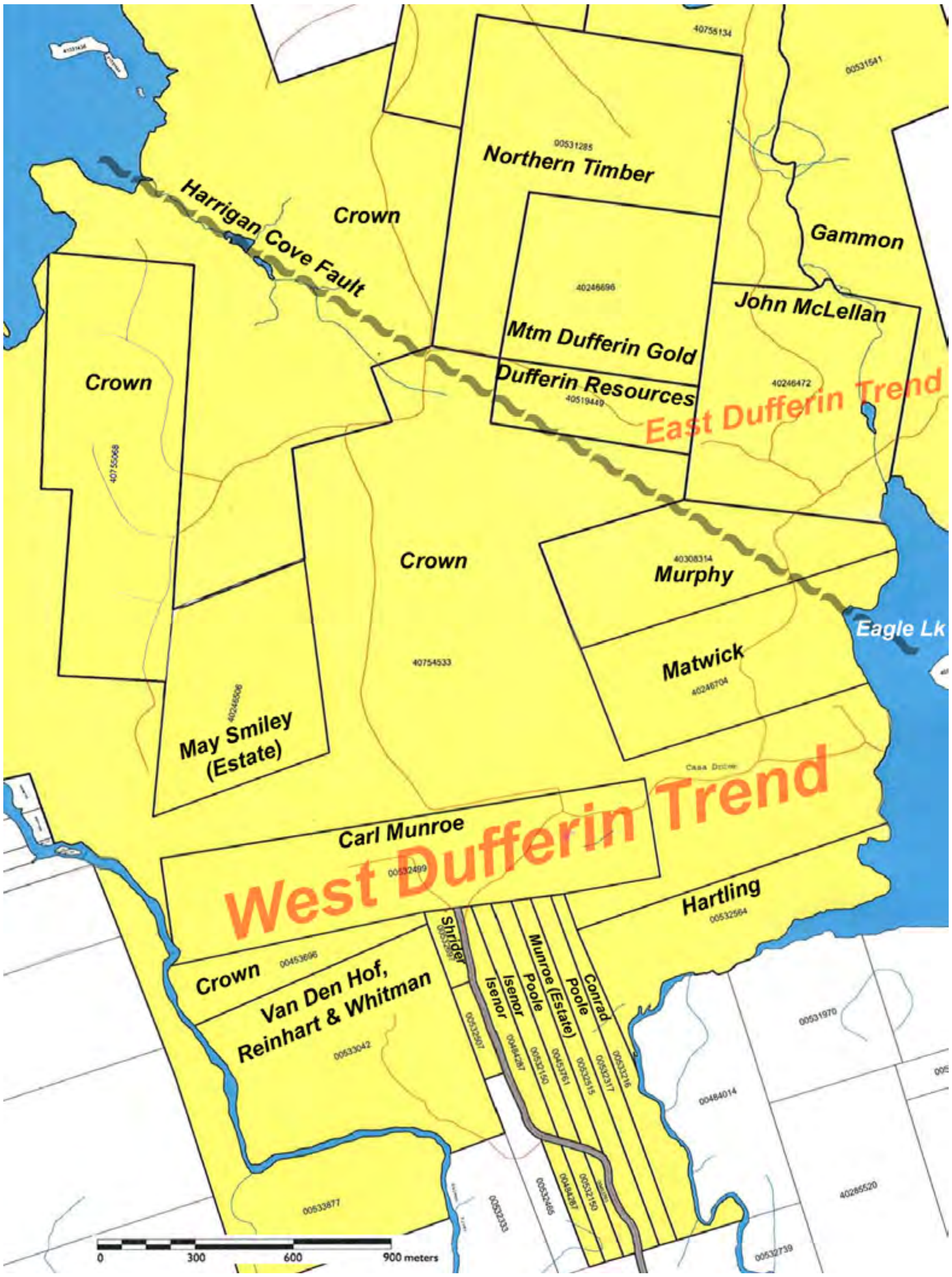


Figure 4-5: Real property (surface) rights.

Table 4-2: Real property (surface) rights.

PID	Size (Hectares)	Owner(s)	Size (Acres)
West Dufferin:			
40246506	123.5	May Smiley (Estate)	50
40754533	387.8	Dufferin Resource Incorporated (Party to Agreement)	157
		Maritime Dufferin Gold Corp (Assignee)	
		Lascaux Resource Capital Partners LLC (Mortgagee)	
		NS Department of Natural Resources (Owner)	
40308314	123.5	Sandra Isabel Murphy & Donald Michael Murphy	50
40246704	181.2	Jack J Plouffe & Linda L Matwick	73.35
		East Coast Credit Union (Mortgagee)	
532499	279.1	Carl E Munroe	113
453696	59.3	NS Department of Natural Resources	24
533042	247.0	Patricia A Van Den Hof, Sandra M Reinhart, & Audrey J Whitman	100
532697	12.4	Henry E Shrider	5
532507	7.4	Carl E Munroe	3
484287	39.5	Violet E E Isenor	15.98
532150	40.3	Violet E E Isenor	16.33
453761	37.1	Cathy Poole & Clarence C Poole	15
532515	42.0	Thomas W Munroe (Estate)	17
532317	37.1	Cathy Poole & Clarence C Poole	15
533216	51.5	Gayle Elaine Conrad	20.87
532564	130.9	Karen Louise Hartling & Douglas Victor Hartling	53
East Dufferin:			
40755068	247.0	NS Department of Natural Resources	100
40754590	965.8	NS Department of Natural Resources	391
40755126	247.0	NS Department of Natural Resources	100
40755134	190.2	NS Department of Natural Resources	77
531541	247.0	Susan I Gammon & Phillip R Gammon	100
531285	308.8	Northern Timber NS Corp	125
40246696	179.4	Maritime Dufferin Gold Corp (Fee Simple)	72.62
		Lascaux Resource Capital Partners LLC (Mortgagee)	
40519449	76.5	Maritime Dufferin Gold Corp (Fee Simple)	30.98
		Lascaux Resource Capital Partners LLC (Mortgagee)	
40246472	247.0	John McLellan (Fee Simple)	100
		Dufferin Resources Incorporated (Lessee)	

4.4 Property Boundaries

The surface rights boundaries for East Dufferin were legally surveyed during 2008.

The mineral lease and mineral claims boundaries have not been surveyed. Nova Scotia uses a map staking system whereby the province is divided into a latitude-longitude-defined, regular grid of claims of approximately 40 acres each (16.2 ha). Unless a dispute with a mineral rights neighbour arises, it is not normally required to physically survey or mark the claim boundaries.

4.5 Locations of Mineralization and Infrastructure

The locations of mineralization and infrastructure are illustrated in Figure 4-5 to Figure 4-7. The gold mineralization follows the anticlinal axes.

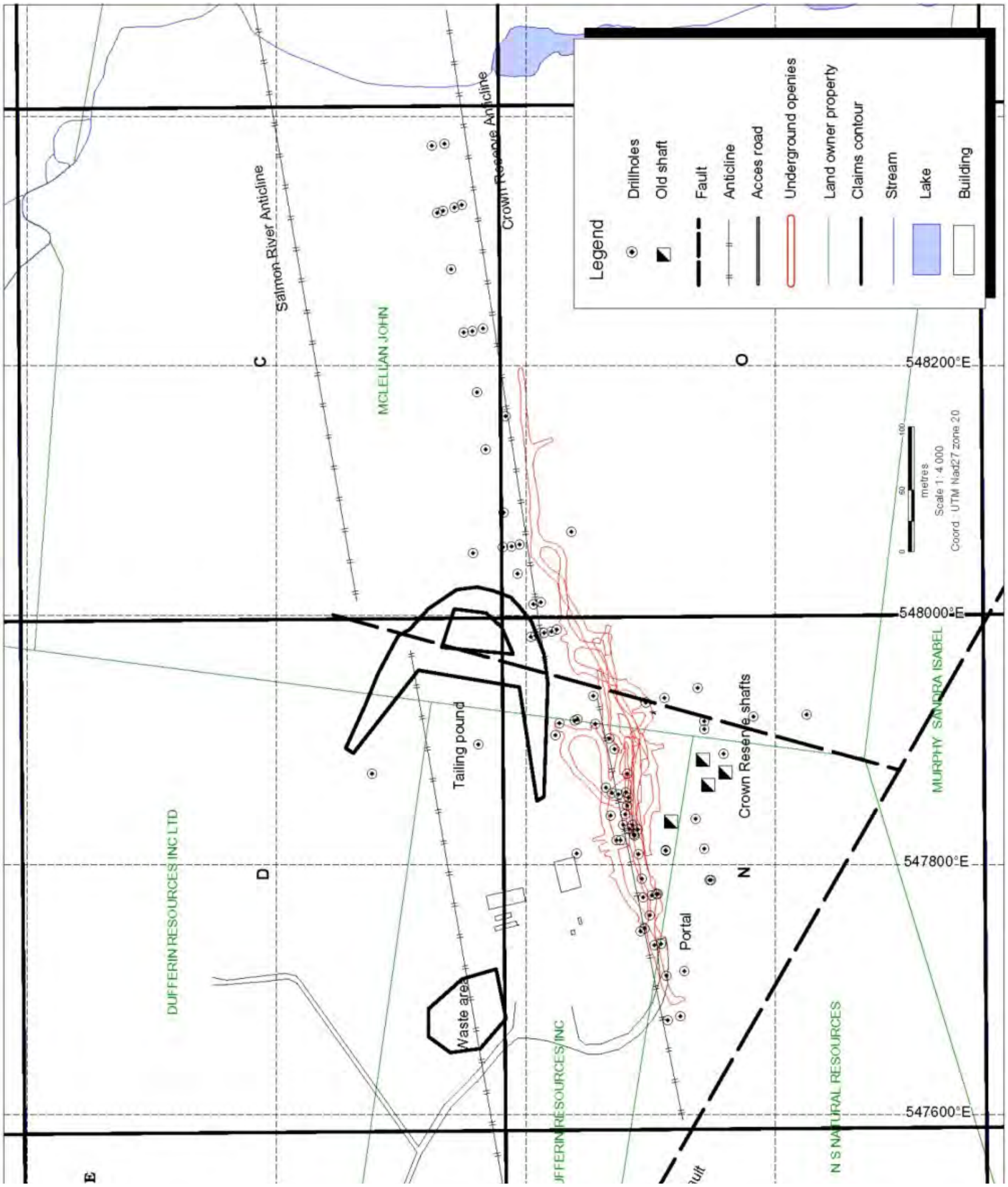


Figure 4-6 - Locations of underground workings and surface infrastructure at East Dufferin.

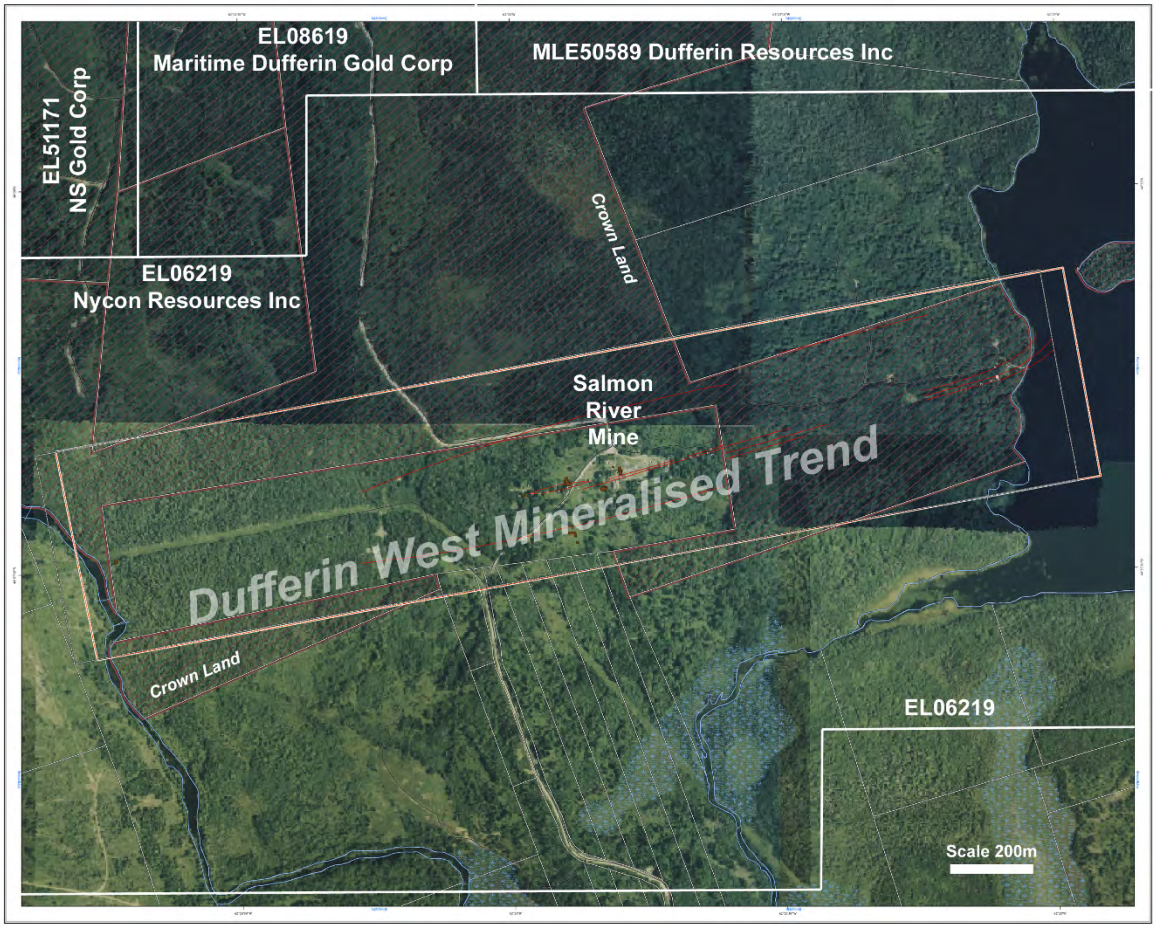


Figure 4-7: West Dufferin infrastructure and mineralised trend.

4.6 Environmental Considerations

Environmental considerations for the project are described in detail in Section 20.

Prior to obtaining a mining permit on the East Dufferin portion of the project, Dufferin Resources commissioned an Environmental Assessment of the project. In 1994, Jacques, Whitford and Associates Limited of Fredericton, New Brunswick completed the Environmental Assessment (Pheeny, 1994). The Industrial Approval which was granted in 2013 was based in part on the 1994 EA.

The 1994 EA included a baseline water quality study, land usage considerations, evaluation of plant, wildlife, raptors, fish and waterfowl resources, and the socio-economic impact of the mining project.

The report stated that:

> “..... the attractiveness of the proposed project was further enhanced by the documentation of environmental considerations such as:
>
> • mineralised rock and waste rock is non-acid generating and contains low levels of heavy metals such as copper and mercury.

• A small project area (under 10 ha) in which no sensitive environmental issues were identified.

• A project site which has been disturbed by previous activities and the current work will mitigate this existing ground exposure.

• Complete isolation of the project area from the hydrologic basin due to topography, creating a single point discharge sub-basin which can be easily engineered and monitored.

• A mill/tailings water balance circuit which limits the total site discharge to the minimum level. i.e. Once start-up is complete, process makeup water will be by reclaim from tailings, thus creating a closed mill circuit.

• Power and road access current are available on site so any impacts have been defined.

• No direct impact on the existing water courses in the area and only a minimum effect on existing natural lands during construction and operation ".

4.7 Nature of Resource Capital Gold's Interest

4.7.1 East Dufferin

The East Dufferin portion of the project, comprising exploration licenses 50561, 08619, and 07351, was acquired from Ernst & Young (the court-appointed receiver of the property following bankruptcy of previous operator Ressources Appalaches) on September 30, 2016. The overall purchase agreement grants the senior secured creditor of the property, Lascaux Resource Capital, the following terms:

1. USD$1,500,000 upon closing of the transaction, Friday October 7, 2016.
   This sum has been paid.
2. USD$1,600,000 annually on the first through fifth anniversaries of closing.
3. Net Smelter Return royalty of 1% beginning on the fifth anniversary of closing.

The exploration licenses were purchased by Maritime Dufferin Gold Corporation, which is 100% owned by Maritime Gold Corporation, which is in turn owned 90% by Resource Capital Gold Corp. The remaining 10% of Maritime Gold Corp is owned by ACT2 Pty Ltd (ACT2), as settlement of C$1,187,500 of previous debt owing from RCG to ACT2

To the best of the Authors' knowledge, the East Dufferin properties are not subject to any other royalties, back-in rights, payments, or other agreements and encumbrances.[2]

[2] Ressources Appalaches Press Release, 2012-09-27, "Ressources Appalaches Receives Code of Practice Approval for Dewatering Operations of its Dufferin Mine" ('[the] Dufferin Mine … is 100% owned and free of all royalties'); Personal Communication with NSDNR listing all agreements attached to Lease 94-2, sent 2015-02-18.

4.7.2 West Dufferin

The West Dufferin properties, consisting of exploration licenses 06219 and 06274, were purchased by Resource Capital Gold Corp., through a separate subsidiary company, 100% from Nycon Resources Inc. for the following consideration:

1. USD$50,000 upon signing of a binding letter agreement. This sum has been paid.
2. USD$250,000 upon closing the transaction, which *was* planned to occur on or before December 27, 2016. The deadline was extended to January 30, 2017.
3. USD$350,000 on the first and second anniversaries of closing.
4. Net Smelter Return (NSR) royalty of 2%, with the option to purchase 1% for USD$750,000 and the option to purchase the remaining 1% for USD$1,000,000.

Resource Capital Gold acquired these assets through its subsidiary, Maritime Gold Corporation.

To the best of the Authors' knowledge, the West Dufferin properties are not subject to any other royalties, back-in rights, payments, or other agreements and encumbrances.

4.8 Environmental Liabilities

The only significant environmental liability is the existing tailings facility at East Dufferin. The rock is not acid-generating, although mineralised rock and waste rock have high arsenic values. The facility will have to be reclaimed upon closure. Environmental liabilities are discussed in detail in Section 20. Major reclamation items include:

- Removing of buildings and other surface structures.
- Removing the milling equipment from the site to be reused on another site or sold.
- Decommissioning and sealing of the portal vent raise and other access points.
- Chemicals, fuels and explosives will be removed from site and disposed in accordance with all provincial and federal regulations.
- Tailings will be capped and vegetated.
- Surface contouring of the mine and mill site, including the area of the tailings impoundment.
- Monitoring for 3 years after reclamation is complete.
- The site will be fully reclaimed 2 years after production ends.

Mercury was used during historical milling at West Dufferin. As was common practice at the time, tailings were simply "released" to the environment – spilled over the land and into streams. This is a "potential" rather than a "known" existing environmental liability because sufficient investigation has not yet been carried out.

5 Accessibility, Physiography, Climate, Local Resources, and Infrastructure

The topography of the area slopes to the gently southeast to sea level from an elevation of about 70 m at the Dufferin site. Glacial drumlins elongated in a south-east – north-west direction and measuring about 1 km by 350m, rise 30 m or so above the surrounding ground.

5.1 Accessibility

The Property is in Halifax County, Nova Scotia. It is located on the Eastern Shore of Nova Scotia, approximately 140 kilometres east-northeast of Halifax. Access by road from Halifax is by paved highway 107 to Musquodoboit Harbour along paved highway 7 to Port Dufferin, a hamlet approximately 14 kilometres east of the village of Sheet Harbor. At Port Dufferin there is a highway sign directing travelers to Dufferin Mines, approximately eight kilometres north from that settlement, along an all-weather gravel road. There are also Logging roads in the area which provide good access to all parts of the property.

5.2 Physiography

The property is situated in an area of moderate relief in undulating terrain of linear swampy intervals and low rolling hills with maximum elevation of 100 metres above sea level. Drainage is controlled by the southwest branches of the Salmon River on creeks draining the numerous small lakes. Drainage direction may be controlled by south to south-westerly trending cross faults. Eagle Lake lies east-southeast of the mine site, Spar Lake lies to the north and Nowlin Lake lies to the east. Glacial till of between two and ten metres in thickness covers most of the property. Glacial direction in the area is 170°+/-5°, that is to say, material has been transported in a southerly direction by the glaciation. The vegetation consists of mixed pine and birch forests on the higher ground with white spruce swamps and peat bogs in the low-lying areas. Alders and willows are common throughout. The trees are mostly second growth and are of small merchantable size.

5.3 Climate

The closest weather monitoring station is at Malay Falls, Nova Scotia, approximately 7km east-north-east of Property. The Malay Falls station is no longer active, but records exist for most years between 1987 and 2000. The coldest month is January, with a mean temperature between 1988 and 1999 of -5.75 °C; the hottest month is August, with a mean temperature between 1987 and 2000 of 17.6 °C. Mean annual precipitation between 1987 and 2000 was 1643 mm, which includes a mean annual snowfall of 124.5 cm (using a conversion of 1 cm of snow to 1 mm of rain).

As the Malay Falls site record is only up to year 2000, the record for Collegeville NS was also examined. This station has collected data since at the 1930s. The data shows that average daily temperature has been warming slightly with a 0.4°C rise over the 50 year period, January 1961 –

December 2010. (Source: http://climate.weather.gc.ca/historical_data/search_historic_data_e.html). Both the temperature and total precipitation are illustrated below.

This site is drier that the Malay Falls site. The total precipitation for the area was 1380 mm per year for 1961-1980; 1384 mm/yr for the period 1971-2000 and 1315 mm/year for the period 1981-2020.

A review of these records did not identify any climate impediments in the area of the West and East Dufferin claims.

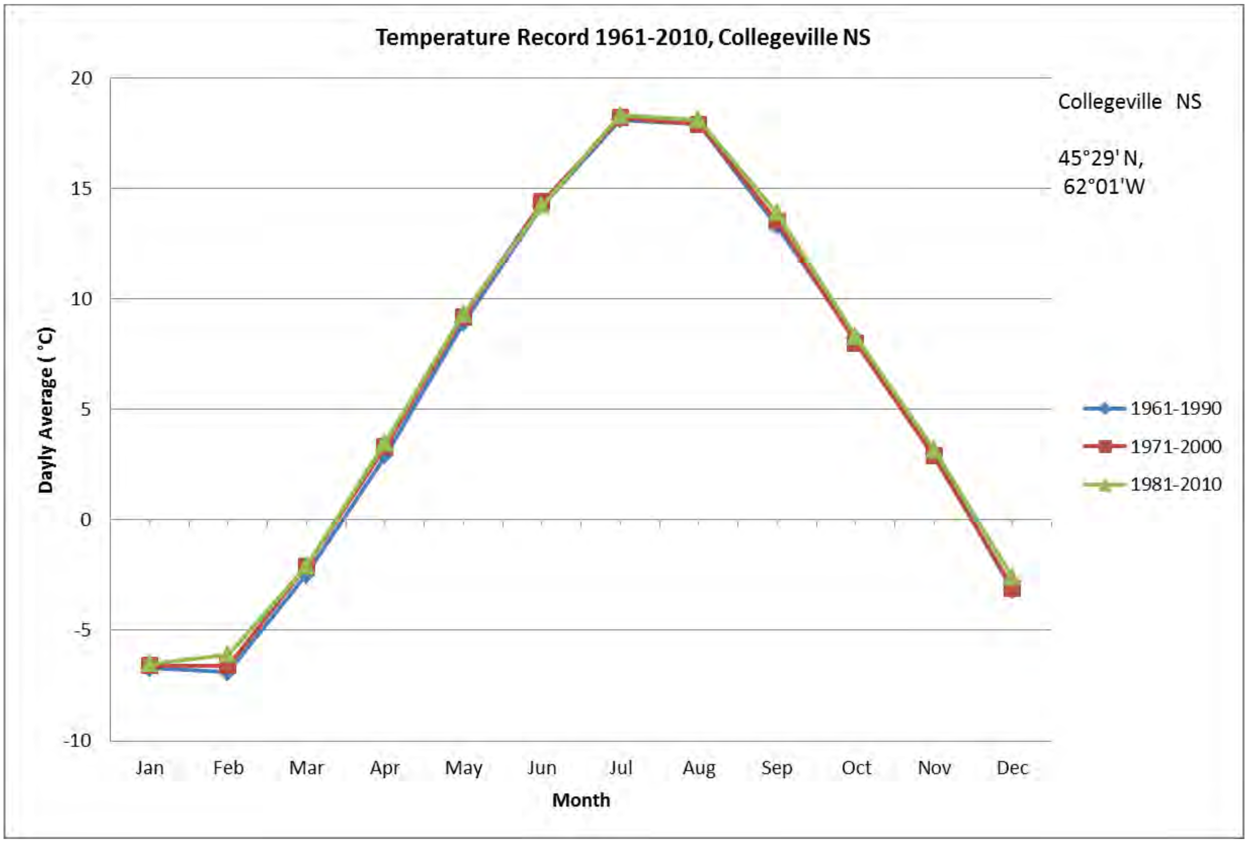


Figure 5-1 Temperature data from Collegeville NS Station, 1961-2010.

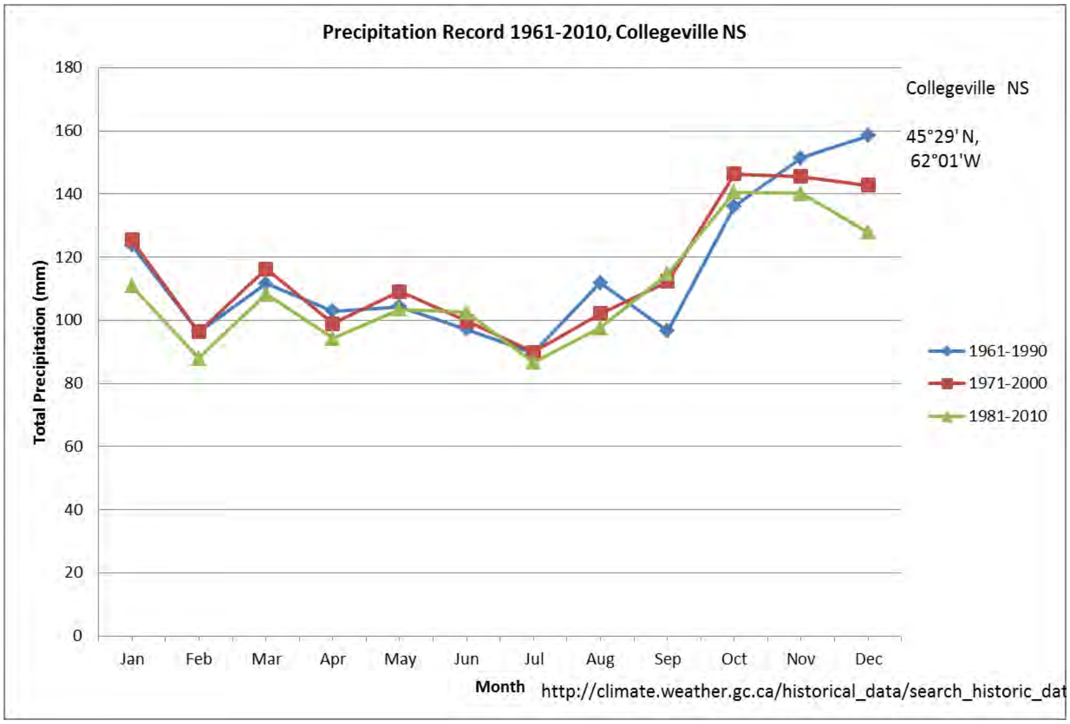


Figure 5-2 Total Precipitation, Collegeville NS, 1961-2010.

Extreme precipitation events for the site records indicate more extreme precipitation during the late 1930s and early 1940s.

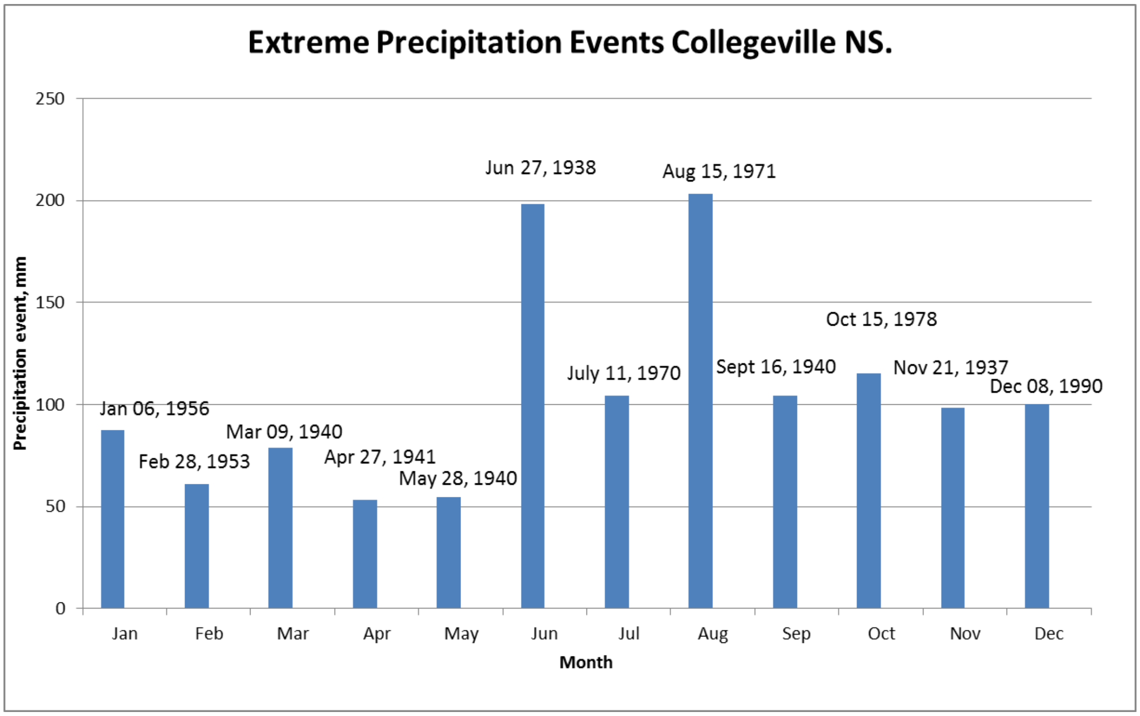


Figure 5-3: Climate Normals, approx. 1988 to 1997, Malay Falls

5.4 Local Resources

Modern sawmills are located within 50 kilometres to provide necessary timber for mine and camp purposes.

The property is well located with respect to utilities. It is connected to three-phase power, telephone lines and data lines. A communications tower on-site provides cellular phone connectivity.

Modern towns and cities exist within reasonable commuting distances for supply of provisions, mine supplies and as a source of skilled and semi-skilled labour.

Within Sheet Harbour and area there are a number of businesses including: accommodations & dining, arts, painting & photography, contractors, crafts, flowers & gifts, financial services, forestry contractors, fuel & automobile repair, funeral homes & monuments, groceries & produce, heating & plumbing services, insurance services, laundromats & laundry services, legal services, port of sheet harbour service companies, real estate, recreational services, refuse, recycling & septic, retail services, seafood & aquaculture, taxation services, trucking services, internet access services, sheet harbour public library, churches, community development, community halls / service clubs, Royal Canadian Legion, educational services, volunteer fire departments, health services, eastern shore memorial hospital services, libraries, museums & historical societies, postal outlets, justice of the peace, Sheet Harbour RCMP, government services federal and provincial services.

5.5 Infrastructure

The East Dufferin infrastructure is described in Section 18. Figure 5 2 shows an aerial view of the East Dufferin portion of the property.

Infrastructure at West Dufferin is limited (refer to Figure 4 7). It consists of:

- an access road,
- very close proximity to the infrastructure at East Dufferin;
- very close proximity to 3-phase power; and,
- very close proximity to streams and lakes.

There is sufficient land available at Dufferin West and Dufferin East for mining operations. There is 3-phase power on site, sufficient water for mining, trained mining personnel in the area and sufficient land for tailings storage area. A 300 tonne per day gravity-flotation processing plant is on site.

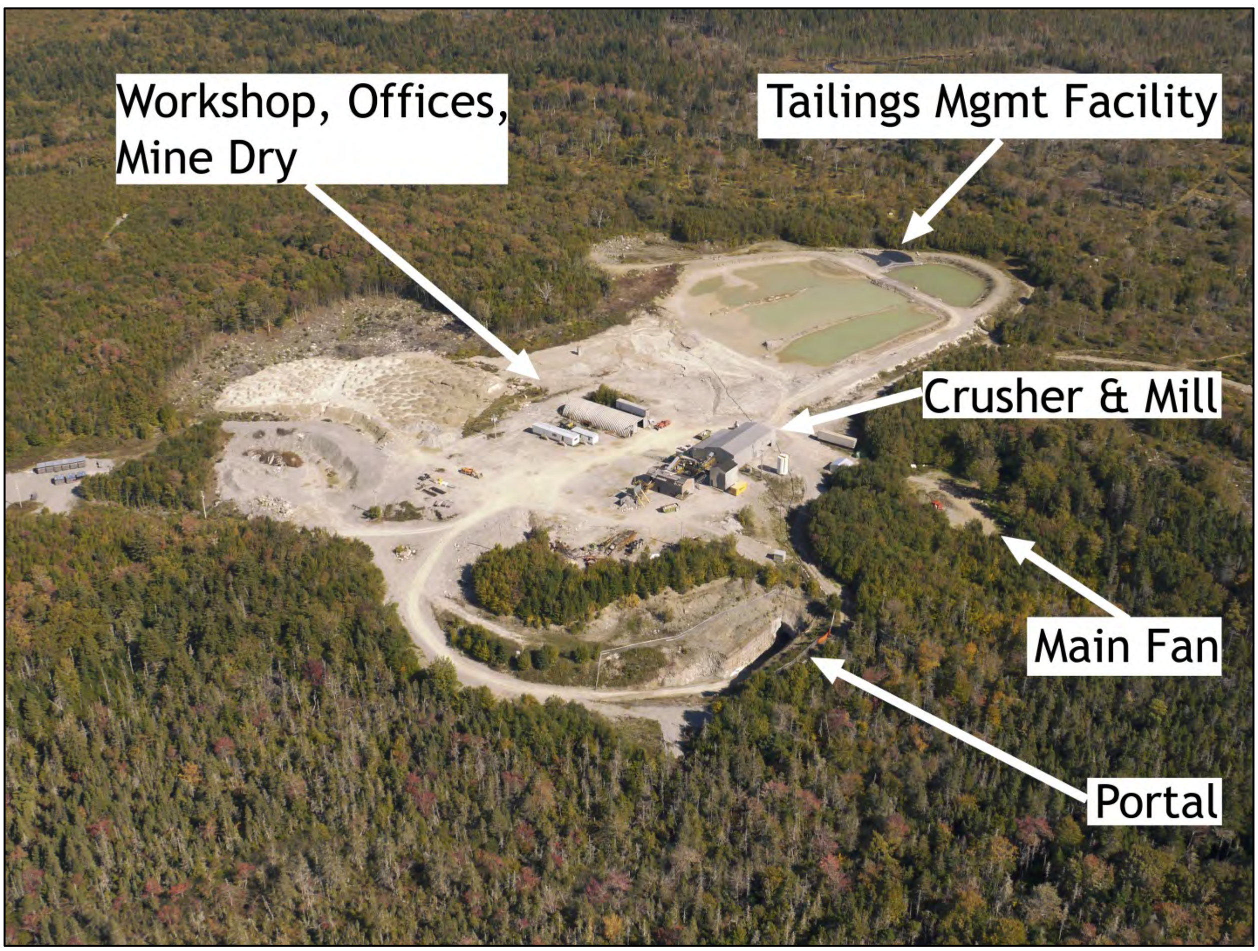


Figure 5-4 - Air photo of the East Dufferin Property in 2013 (taken October 4th, 2013; photo courtesy of the Nova Scotia Department of Natural Resources. Markup added by MineTech.)

6 History

6.1 East Dufferin History

6.1.1 Prior Ownership and Ownership Changes

Exploration License 50561 started life as Exploration Licence 11818 in 1986, owned by Seabright Resources Inc. Those claims were transferred to Seabright Explorations Inc. in 1987.

In 1991, Seabright Explorations changed its name to Corner Bay Minerals Inc. Corner Bay transferred the title to Dufferin Resources in June 1994. Dufferin Resources has held the title to the property since then.

In September 1994, Mining Lease 94-2 was issued, consisting of the claims that were listed in Table 4-1. A change in the legislation in 2004 reclassified Mining Leases as Mineral Leases.

In 1998, Newfoundland Goldbar Resources acquired Dufferin Resources. Newfoundland Goldbar later sold Dufferin Resources to Azure Resources in 2003. In 2005, Azure Resources sold Dufferin Resources to Jemma Resources.

Ressources Appalaches signed an agreement in November 2008 to acquire 100% of Dufferin Resources from Jemma Resources. The agreement included the acquisition all of Dufferin Resources' assets, including the ramp-access Dufferin Mine, together with its processing plant (mill), mining lease, environmental permits, and all other infrastructure required for production.

Table 6-1 - Ownership History

From	To	Owner	Controlling Subsidiary	Property	Notes
1986	1987	SR	Same as owner	Exploration License 11818	Initial Creation
1987	1991	SE	Same as owner	Exploration License 11818	Change in company name
1991	1994	CBM	Same as owner	Exploration License 11818	Change in company name
1994	1994	DR	Same as owner	Exploration License 11818	Purchase of Exploration License by DR
1994	1998	DR	Same as owner	Mining Lease 94-2	Conversion of Exploration License to Mining Lease
1998	2003	NGR	DR	Mining Lease 94-2	Purchase of DR by NGR
2003	2004	AR	DR	Mining Lease 94-2	Purchase of DR by AR
2004	2005	AR	DR	Mineral Lease 94-2	Change in legislation reclassified of all Nova Scotia Mining Leases as Mineral Leases
2005	2008	JR	DR	Mineral Lease 94-2	Purchase of DR by JR
2008	2015	RA	DR	Mineral Lease 94-2	Agreement by RA to acquire DR from JR. Final payment made in 2011.
2015	Aug 2016	RA	DR	Exploration License 50561	Mineral Lease converted to Exploration License
Aug 2016	Present	RCGC	RCGC	Exploration License 50561	

Table 6-2 - Company Name Acronyms

Acronym	Full Name
AR	Azure Resources Corp.
CBM	Corner Bay Minerals Inc.
DR	Dufferin Resources Inc.
JR	Jemma Resources Corp.
NGR	Newfoundland Goldbar Resources Inc.
RA	Ressources Appalaches Inc.
RCGC	Resource Capital Gold Corp.
SE	Seabright Explorations Inc.
SR	Seabright Resources Inc.

With the final payment made to Jemma Resources in October, 2011, Ressources Appalaches gained 100% interest in the Dufferin Mine property. The cost of the transaction was $4,000,000, paid in four instalments spread over three years.

Ressources Appalaches later received financing from Lascaux Resource Capital (LRC). In January, 2015, Ressources Appalaches failed to meet the requirements under a forbearance agreement with LRC and fell into receivership. Ernst & Young Inc. are acting as the Receiver.

In February 2015, Mineral Lease 94-2 was converted into Exploration License 50561.

In September 2016, Resource Capital Gold Corp. purchased Exploration Licenses 50561, 08619, and 07351 from the Receiver (see Section 4.7, above).

6.1.2 East Dufferin Exploration and Development Work by Previous Owners & Operators

The first discovery of gold bearing quartz boulders in the Dufferin area was reported in the area of Port Dufferin in 1868. Development started in 1880 at West Dufferin, often referred to as the Old Dufferin Mine. Mapping by E. R. Faribault between 1860 and 1905 aided in identifying the regional structures and forms of the gold mineralized deposits.

The east-west trending Salmon River Anticline consists of two flexures but only the southern fold, 75 metres to the south and called the Dufferin Mines Anticline at West Dufferin, carries quartz and economic gold mineralization. The Salmon River Anticline has been faulted off east of the Lake Eagle Mine by the Harrigan Cove Fault. This moved the anticline fold extension to the northwest by approximately 1.5 kilometres of sinistral strike-slip separation. A significant dip-slip displacement is suggested by the variance in separation of the trace of the Salmon River and Dufferin Mines anticline west of the fault and the Salmon River and Crown Reserve anticlines east of the fault (refer to Figure 7-2), (Horne and Jodrey, 2002).

The Old Dufferin Mine, and the Lake Eagle Mine, which is 700 meters to the east of the Old Dufferin Mine, were the first producers in the area, both located on the West Dufferin portion of the project. They are situated approximately 1.5 kilometres south of the mineralized zones on the East Dufferin portion of the project. Production from mines in this district between 1883 and

1925 was reported in a contemporary document to be 35,301 ounces of gold from 110,576.5 tons of mineralised rock crushed (Malcolm, 1929). A more recent published figure was reported as 41,805.4 ounces between 1881 and 1935 (Bates, 1987).

In 1923, approximately 900 metres east of the actual Dufferin Mine, the Maple Leaf Gold Mining Co. sank a shaft 11.3 metres deep and drifted west along the vein approximately 50 metres. A second shaft was excavated 30.5 metres east of the first shaft.

In 1934 Crown Reserve Mines sunk a shaft on a 35.6-cm wide vein to a depth of 25.9 metres within a southward dipping slate belt, around 50 metres south of the present Dufferin Mine on the East Dufferin portion of the project. A second 10-cm wide quartz vein was uncovered south of the original vein and two additional shafts were sunk to a depth of 40 metres with limited drifting and stoping.

6.1.2.1 Summary of Historical Work

A total of 58 diamond drill holes with a combined length of 5,260 meters were drilled before Ressources Appalaches gained control of the Property. Table 6-3 summarizes the historical drilling:

Table 6-3 – Historical Diamond Drill Hole Summary (pre-Ressources Appalaches)

Company	Year	Holes Nos.	No. of holes	Total Length (m)
Corner Bay Minerals/Seabright Expl.	1987-88	PD87-01,02, PD88-01-33	35	3,237
Dufferin Resources	1993	PD93-01-11, 13-14	13	659
Newfoundland Goldbar	1999	PD99-15-24	10	1,364
Total			*58*	*5,260*

A summary of underground mining on the Property can be found in Table 14-3.

6.1.2.2 Summary of Modern Work (1986 to present)

Between 1986 and 1993, Seabright Explorations Inc. (later named 'Corner Bay Minerals') explored the site using geochemistry, geophysics and diamond drilling. They also built infrastructure, including a beginning for the decline, an access road, installation of 3-phase power, and some permitting.

Between 1993 and 1998, Dufferin Resources carried out more diamond drilling, as well as some metallurgical test work. They also prepared two resource estimates and one feasibility study. In terms of infrastructure, they advanced the decline to the face of the first saddle, and built both the tailings management facility and the mill. Dufferin Resources operated the mine for a few months in 1995, and milled at least 1,400 tonnes from the First Saddle.

Between 1998 and 2003, Newfoundland Goldbar Resources owned Dufferin Resources. Newfoundland Goldbar's main activity during this time was diamond drilling. They drilled a number of holes, extending the known strike length and depth of the deposit.

Between 2000 and 2003, EnviroGold Technologies was in a mining contract with Newfoundland Goldbar. After dewatering the mine, they drifted along the First Saddle for 150 metres and drove the decline to the Second Saddle and 150 metres to the face of the Third Saddle. Over 55,000 tonnes were milled producing 7,397 ounces.

Between 2003 and 2005, Azure Resources Corp. held an option to acquire the property. During this period, they partially dewatered the underground, refurbished the mill, drove the ramp to the Fourth Saddle, and mined mineralised rock on the First through Third saddles.

Between 2005 and 2008, Jemma Resources held an option to and eventually acquired the Property. During this time they refurbished the mill and processed tailings, gaining approximately 1,600 ounces Au from approximately 31,745 tonnes of tailings.

Between 2008 and 2015, Ressources Appalaches operated the Property.

From 2015 to present, the Receiver, Ernst and Young, has maintained the property under care and maintenance and the underground mine has been kept dry.

6.1.2.3 1986-1993: Seabright / Corner Bay Minerals

From 1986 to 1989 Seabright Explorations Inc., later renamed Corner Bay Minerals, conducted geological mapping, soil geochemistry, geophysical surveys (magnetometer, IP, and VLF-EM), trenching and 3,237 metres of diamond drilling in 35 holes. The company also completed some site preparation including the construction of the access road, installation of a three phase power transmission line and permitting. A decline was planned, but the project was suspended before work started on the decline.

6.1.2.4 1993-1998: Dufferin Resources

In 1993, an option on the Property was acquired by Dufferin Resources. In 1993, the company drilled 13 holes totalling 659 metres. The drill program was designed to confirm and define the size, configuration and existence of higher grades at the fold crest of the first two saddle veins.

All drill hole collars were surveyed. Core was logged in detail, photographed and compiled in a consistent format into a database to correlate major units and veins. While most quartz veins were assayed, several of the quartz veins were left intact and were kept in storage or on site as "witness samples". Core from several of the holes was stored at the Nova Scotia Department of Natural Resources core storage facility in Stellarton. The remaining unmineralized drill core was destroyed (Duncan and Graves, 1993).

Two resource estimates were prepared in 1993, first by Duncan and Graves (Duncan and Graves, 1993), and later by Atkinson (Atkinson, 1993). These historical estimates are described in Section 6.1.2.10.2.

Metallurgical and assay test work on core samples carried out at M-Tech Incorporated of Elmsdale, Nova Scotia indicated that the high grade mineralization (>10 g/tonne Au) is carried by coarse, particulate free gold which is liberated from the gangue when ground to 100% passing -10 mesh (1,700 microns). Recovery rates of 90% to 95% were achieved using the KMS technique (Duncan and Graves, 1993).

Underground development was started in 1994. The decline was collared and advanced 50 metres to the face of Saddle 1.

An environmental study (Jacques, Whitford and Associates Limited) as well as gravity metallurgical test work and construction of the tailings pond were also completed.

The gravity mill was constructed in 1994. The mine operated briefly in early 1995 but closed on April 28th, 1995 for economic reasons. A reported 3,418 tonnes from Saddle 1 was milled altogether. After closing, the underground was allowed to flood and the mill was placed on care and maintenance.

6.1.2.5 1998-2003: Newfoundland Goldbar Resources / EnviroGold Technologies

In 1998, Newfoundland Goldbar Resources Inc. acquired a 100% interest in Dufferin Resources. Exploration work on the Dufferin Property resumed in 1999. A further 10 diamond drill holes totalling 1,364 metres tested a further 300 metres of strike length along the eastern extension of the Crown Reserve Anticline at 50 metre intervals. This drilling confirmed the depth potential of the Upper and Middle gold bearing quartz saddle-veins.

During this period one hole (PD-99-24) was drilled to a depth of 396 metres on Mine Grid Section 2310E. This hole intersected 10 additional argillite units below saddle 1 with quartz veins and significant mineralization (Woodman, 1999).

This work established approximately 700 metres of strike extension for the two upper saddle-reef veins and potentially up to 13 stacked saddle-reef zones.

In late 2000 EnviroGold Technologies Inc. entered into a Mining Contract on the Property and started underground development. The contract between Dufferin Resources and EnviroGold Technologies was intended to test new milling and gold recovery technology proposed by EnviroGold. After dewatering, mining continued along the first saddle for a distance of 150 metres in a 3m x 3m drift. The main decline was driven to the second saddle and developed over 350 metres to the face of the third saddle. Development of the second saddle consisted of drifting at 4m x 5m along a lateral extent of 300 metres. In excess of 55,000 tonnes was milled during the period and 7,397 ounces of gold were recovered until the operations stopped in January 2002 for economic reasons. The workings were again allowed to flood (Sanguinetti and Ainsworth, 2003).

6.1.2.6 2003-2005: Azure Resources Corp.

In June 2003 Azure Resources Corp negotiated an option to acquire a 51% interest in the Property with Newfoundland Goldbar Resources Inc.

Work carried out on the Property from June 2003 to February 2004 consisted of partial dewatering of the underground, sampling of portions of the saddles and legs of saddle Veins 1 and 2, surface surveying, surveying and sampling of the tailings pile, metallurgical studies, reconstruction of the camp, and redesign and reconstruction of the mill (Sanguinetti, 2004).

From February to November 2004, Azure continued dewatering the mine, refurbishing the mill as well as preparing general surface work. Mining of mineralised rock took place mainly in the first, second and third saddles, and a ramp was driven to the fourth saddle.

6.1.2.7 2005-2008: Jemma Resources

In 2005, Jemma Resources signed an option to acquire the property. In 2006, tailings were reprocessed in the grinding and flotation sections of the mill. A total of 1,601.9 ounces was recovered from the 31,745 tonnes of tailings processed. Refer to Section 13 for discussion of Mineral Processing and Metallurgical Testing.

6.1.2.8 2008 to 2014: Ressources Appalaches

On November 10, 2008 Ressources Appalaches announced that they had come to agreement terms with Jemma to purchase the property for $4 million, spread over three years, subject to a five month due diligence period.

Exploration work by Ressources Appalaches is described in Section 9.

In early 2009, MineTech was commissioned to prepare a Technical Report on the property (MineTech, 2009). The report was authored by Douglas Roy and Patrick Hannon, and is available on SEDAR.

The 2009 Technical Report was updated in 2012 to report on diamond drilling campaigns on the property (MineTech, 2012). The report was authored by Douglas Roy and Patrick Hannon, and is available on SEDAR.

In September of 2012, Ressources Appalaches received Code of Practice Approval to dewater the mine from the Nova Scotia Department of Labour and Advanced Education, Occupational Health and Safety Division. Also in 2012, a manager was appointed for the Dufferin property, and the company began hiring personnel to dewater the mine and refurbish the mill.

By May of 2013, Ressources Appalaches completed a $10-million loan agreement from New York financing firm Lascaux Resource Capital Partners, LLC. Funds were dedicated to dewatering the mine, refurbishing the mill, enlarging the tailings pond and purchasing some equipment for mining. By June of 2013 the portal and the outside portion of the ramp was rehabilitated with steel screen, rebar, shotcrete and cable bolts.

In August of 2013 a plan was submitted and accepted by the Provincial Regulators allowing Ressources Appalaches to proceed with rehabilitating the underground mine. In November of 2013, they received the Industrial Approval – the final Permit required to allow them to proceed with production.

In April of 2014, an underground diamond drilling program started.

In July of 2014, Ressources Appalaches announced that the first gold bar was poured at the Dufferin Mine and delivered to Johnson Matthey refinery in Ontario for final refining.

By August 2014, Ressources Appalaches announced that the plant had run the mill up to 245 tonnes per day.

6.1.2.9 East Dufferin Historical Production

Historical production from the East Dufferin portion of the property is summarized in Section 13.1.

Relevant tables include:

- Table 14-3 - Previous mining.
- Table 13-1 – Reported East Dufferin Mine Production previous to 2014.

6.1.2.10 East Dufferin Historical Mineral Resource Estimates

There are no previous NI43-101-compliant Mineral Resource estimates for the Property (West or East Dufferin).

All of the estimates described below are historical in nature, were not prepared in compliance with the CIM definitions required by National Instrument 43-101 and should not be relied upon.

A current mineral resource estimate has been prepared, and is described in Section 14.

6.1.2.10.1 Corner Bay Minerals

Corner Bay Minerals interpreted early drillhole data from the Dufferin deposit in 1988. The Corner Bay Minerals estimate is mentioned in the Duncan & Graves report (Duncan and Graves, 1993).

Interpretation of early drill hole data by Corner Bay Minerals outlined three stacked saddle vein systems over a strike length of 215 metres and an in situ reserve of quartz vein material of 23,000 tonnes averaging 16.2 g/tonne Au in the Upper (1) saddle-vein and 61,000 tonnes averaging 14.2 g/tonne Au in the Middle (2) saddle-vein.

The Corner Bay Minerals estimate is relevant as it is the first known estimate made since the current boundaries of the Property were established. It is not reliable, because detailed information on the methods and data used are unavailable, and it is based on a more limited set of data and experience than is currently available.

The key assumptions, parameters, and methods used to prepare the estimate are not known.

The Corner Bay Minerals estimate did not use the resource/reserve categories described in sections 1.2 & 1.3 of NI 43-101. There are no plans to upgrade this historical estimate to a current mineral resource or mineral reserve.

The Qualified Person has not done sufficient work to classify this historical estimate was current mineral resources or mineral reserves. The issuer is not treating this historical estimate as current mineral resources or mineral reserves.

6.1.2.10.2 Duncan and Graves, 1993

A historical resource estimate was prepared by D.R. Duncan & Associates Ltd. based on the exploration work of Dufferin Resources in 1993 and earlier diamond drilling programs (Duncan and Graves, 1993).

The Duncan and Graves estimate is historically relevant because it provided the basis for much of the work on the Property completed before RA acquired the property. It also gave an estimate of the character of the mineralized saddles. However, it is not reliable. Some of the material they describe may have been mined, and it is based on a more limited set of data and experience than is currently available.

Duncan and Graves calculated their estimate using a polygonal cross sectional method. Holes were plotted on cross sections at intervals of 20 or 30 metres along the strike of the deposit, using surveyed collar coordinates for every hole and corrections for downhole surveys.

The specific gravity value used was 2.65 for both quartz vein material and waste (greywacke). The various intersections were weight averaged to produce an average grade over a specific width. Anomalous grades of less than 1 g/tonne were used to assist the geological interpretation. No cut-off grade is noted in the calculations disclosed in the available reports.

In their 1993 report, Duncan and Graves noted that: ''Estimating the gold grade for a given saddle vein is complicated due to several factors including:

- the presence of coarse, particulate free gold
- the crest of the saddle is enriched relative to the legs of the saddle[3]
- drill core size and sample representativeness
- drill core recoveries
- sample preparation and assay methods'' (Duncan and Graves, 1993, Section 5.6.1)

Tonnages reported in the estimate were not rounded. The "in situ geological mineral inventory" of the various quartz saddle veins indicated a total of 147,736 tonnes over a strike length of 450 metres between sections 2162E and 2612E. The Upper (1) "A" saddle vein was estimated to contain 20,611 tonnes at an average uncut grade of 17.5 g/tonne gold. The Middle (2) "A" saddle vein was estimated to contain 80,740 tonnes at an average uncut grade of 12.0 g/tonne gold. Grade estimates for the material in the other saddles (46,385 tonnes in the Upper "B", "Upper "C", Middle ''B" and lower "A" saddles) were not calculated due to lack of adequate sampling in those saddles.

The Duncan and Graves estimate did not use the resource/reserve categories described in sections 1.2 & 1.3 of NI 43-101. Specifically, Duncan and Graves use the now-deprecated "Proven", "Probable" and "Possible" categories. The Issuer does not intend to upgrade or verify the Duncan and Graves estimate.

[3] The current resource estimate (see Section 14) prepared for this report does not agree with Duncan and Graves' statement that 'the crest of the saddle is enriched relative to the legs of the saddle'. See section 14.14 for a discussion of this finding.

The Qualified Person has not done sufficient work to classify the Duncan and Graves estimate as current mineral resources or mineral reserves. The Issuer is not treating the Duncan and Graves estimate as current mineral resources or mineral reserves.

6.1.2.10.3 Internal Ressources Appalaches Resource Estimates

Two internal estimates were prepared by RA as part of their conceptual mining plan.

The internal estimates are relevant as they were used by RA as a basis for their mine plan. However, they are not reliable. The outlines used for resource delineation appear to have been constructed for mine planning purposes rather than mineral resource estimation. Grades were simple arithmetic averages of samples without consideration of the samples' volumes of influence or relative locations. More accurate methods for resource estimation are available.

The first internal conceptual mining plan was made by RA in May of 2013 for saddles 1 through 6, between the Easting 2225 E and 2875 E (650m).

A total of about 310,000 tonnes grading 6 g/tonne was the initial "Mineable Reserve Estimate". This work indicated about 480 tonnes per metre of strike length for the deposit. The estimate was for internal use and it is historical.

Table 6-4 - "Mineable reserve estimate" from Dufferin Mine Conceptual Mine Plan – May 2013 (not rounded).

Saddle	Total		
	Tonnes	g/tonne	(tonnes*g/t)
1	34,755	7.61	264,486
2	78,145	5.76	450,115
3	20,846	4.49	93,599
4	58,410	7.55	440,996
5	38,512	4.27	164,446
6	83,595	5.54	463,116
Totals	314,263	5.97	1,876,757

Another historical estimate of the mineable resources was made by RA in May 2014[4]. This estimate outlined mining blocks for Saddles 1 through 6. Approximately 260,000 tonnes were considered mineable between 2140E and 3220E, a distance of 1080m, indicating approximately 240 tonnes mineralised rock per metre of strike length for saddles 1 through 6. The assay information was available for 160,000 tonnes and this indicated a diluted grade of 4.3 grams per tonne.

[4] *May 23, 2014, "Dufferin-mining-blocks-ver2.xlsx", André Rancourt*

The internal RA estimates did not use the resource/reserve categories described in sections 1.2 & 1.3 of NI 43-101. The internal RA estimates used the term "Mineable Reserve Estimate", but the exact definition of that term is not known. The Issuer does not intend to upgrade or verify the RA estimates.

A Qualified Person has not done sufficient work to classify the internal RA estimates as current mineral resources or mineral reserves. The Issuer is not treating the internal RA estimates as current mineral resources or mineral reserves.

6.2 West Dufferin Exploration History

The following project history was modified from Mitchell (1989), Woodman (2004a, b), Kennedy and Yule (2010), and Yule (2012, 2013, 2014, 2015).

6.2.1 West Dufferin Exploration and Development Work by Previous Owners & Operators

6.2.1.1 1880: Mr. George Stewart and Mr. Alex Kent

Following discovery of gold-mineralised rock and mapping by E.R. Faribault in 1868, Mr. George Stewart and Mr. Alex Kent sunk a shaft into a bedding-parallel auriferous quartz vein at what was then called the Salmon River property at West Dufferin. When mined at depth, this vein was discovered to be a saddle reef-type vein along a fold hinge.

6.2.1.2 1881-1925: Various

Sporadic mining occurred in the Salmon River/West Dufferin area for the next 44 years. Production from mines in this district between 1883 and 1925 was reported in a contemporary document to be 35,301 ounces of gold from 110,576.5 tons of mineralised rock crushed (Malcolm, 1929). A more recent published figure was reported as 41,805.4 ounces between 1881 and 1935 (Bates, 1987). The 6 m-thick No.2 South Vein was the focus of most of the mining and was stoped across the crest of the host anticline. Mismanagement during this time period resulted in disputed mineral rights and repeated transfers of interest.

6.2.1.3 1897: Montreal Gold and Silver Development Company

The company acquired the Salmon River/West Dufferin property and built a 60-stamp mill. They milled 24,339 tons of mineralised rock sporadically over several years that contained 2,502 ounces of gold. They enlarged and deepened the main shaft to 122 m, with minor drifting.

6.2.1.4 1909-1910: Eagle Lake Mining Syndicate

The company sunk a shaft on the western shore of Eagle Lake. The Harrigan Cove Fault was discovered during this time and exploration for the faulted eastern extension of the Salmon River Anticline began.

6.2.1.5 1934: Crown Reserve Mines

The company sunk a shaft on a 35 cm-thick quartz vein to a depth of 26 m. This shaft was ca. 700 m west of the Maple Leaf shaft and on the southern limb of the Salmon River Anticline. A second 10 cm-thick quartz vein was discovered south of the previous vein and was mined through two additional shafts that were sunk to 40 m, with limited stoping and drifting.

6.2.1.6 1935-1941: Consolidated Mining and Smelting Company of Canada Limited (Cominco)

Cominco acquired the Salmon River Mine on the West Dufferin portion of the project and surrounding area in 1935. They completed a diamond drill program totaling 2,554 m, with four of the holes intercepting significant gold mineralized zones (~1-2 m intervals containing 3 - 101 g/tonne Au). Based on these results, they sank a shaft to 161 m and drifted on the 120 m and 150 m levels with mediocre results. Cominco then focused on the more promising Caribou property. In 1941, Cominco completed 488 m of drifting on the 61 m level of the Salmon River Mine/West Dufferin.

6.2.1.7 1972: Lons Mining Corp.

The Salmon River Mine at West Dufferin remained idle until 1972 when Lons Mining Corp. trenched and drilled 93 m of diamond drill holes on the property. Lons focused on the eastern extension of the Salmon River Anticline and near the Eagle Lake shaft, where one core drill hole intercepted 0.5 m containing 34.2 g/tonne, and another core drill hole intercepted a 2.1 m-thick quartz vein that was not assayed. Assays of samples collected from trenches reported low gold values, even though some samples contained visible gold. Mapping of the trenched areas defined the fold axis.

6.2.1.8 1974: Tri-Bridge Consolidated Gold Mines Ltd.

Tri-Bridge Consolidated Gold Mines Ltd. acquired the West Dufferin/Salmon River property and cut grid lines for magnetic/VLF and soil geochemical surveys that outlined several anomalous zones.

6.2.1.9 1979: St. Joseph Exploration Ltd.

St. Joe re-cut Tri-Bridge's lines and conducted soil geochemical and IP surveys near the hinge of the Salmon River Anticline. An IP anomaly was interpreted west of the Dufferin mines and continued to Eagle Lake. This IP anomaly was tested with two core drill holes that intersected 4-5 m of sericite-ankerite mineralized schist that contained 1-2% pyrrhotite and pyrite. The best intercept in this schist contained 2.1 g/tonne Au over 3 m.

6.2.1.10 1980-1981: Pan East Resources

Pan East completed an airborne magnetic and VLF survey of the Salmon River Gold District in 1980, and followed up with a stream-sediment and humus sampling program south of the West Dufferin/Salmon River gold mine in 1981.

6.2.1.11 1984: US Borax and Chemical Corp.

The company completed three diamond drill holes to test St. Joe's IP anomaly on the southern limb of the Salmon River Anticline. Visible gold was reported in these core drill holes, but assays reported a maximum of 0.5 g/tonne over a 0.6 m interval.

6.2.1.12 1986-1988: Jascan Resources Inc.

The company completed a three-phase, 60 hole drill program totaling 8,740 m at West Dufferin. These core drill holes tested the southern limb of the Salmon River Anticline, in between the Salmon River gold mine and Eagle Lake. The drill program defined 100 m-long gold mineralized zones, with assays as high as 170 g/tonne.

6.2.1.13 1988: Kidd Creek Mines

Kidd Creek Mines completed regional and detailed soil geochemistry surveys along the Salmon River Anticline. The data outlined several gold anomalies near the Quoddy River property that were never followed-up.

6.2.1.14 1994-1995: Bruce Mitchell

The project was idle until 1994, when Mr. Mitchell, who held the licenses covering the western extension of the Salmon River anticline, conducted small ground geophysical surveys (magnetics and VLF), soil geochemical surveys, trenching and prospecting. He found several mineralized quartz boulders in the trenches.

6.2.1.15 2001-2016: Nycon Resources Inc.

Nycon Resources acquired license area 06274 in 2001 on the West Dufferin portion of the project, and in 2002 conducted a soil geochemical survey on the Muskrat Lake property (western extension of the Salmon River Anticline) that outlined broad arsenic and gold anomalies coincident with the anticline axis. They drilled two holes to test the eastern and western extensions of the Salmon River Anticline for gold-bearing quartz veins with poor results.

In 2004, three holes were drilled to define the Salmon River Anticline axis and test the western extension of the Salmon River gold mineralized zones at West Dufferin. The anticline axis was successfully defined and thin (20 cm max.) gold mineralized, bedding parallel quartz veins were intercepted. Further west, another DDH was drilled for the same purpose, but did not intercept the fold axis, or any quartz veins.

In 2007, Nycon contracted Brian McPherson to conduct a soil sampling survey north and south of the Salmon River Anticline axis. The samples contained 23-72 ppb Au (average 38 ppb Au) over a 500 m-long anomalous zone in the southern part of the survey area.

In 2008, Nycon contracted Brian McPherson to conduct a Mobile Metal Ion (MMI) geochemical sampling program to identify gold bearing veins at depth and within the Salmon River Anticline. The samples defined anomalous trends in Au values north of the Salmon River Anticline axis. These data showed that lead, zinc, and copper anomalies broadly coincided with elevated gold values.

In 2009, Nycon drilled 686 meters in 8 core holes at West Dufferin. Although not assayed, core photos show that numerous quartz veins were intersected.

In 2010, Nycon contracted Mercator Geological Services (Mercator) to compile and review past geological and geochemical data to guide future exploration in License area 06274. Mercator also completed reconnaissance mapping, prospecting and conducted a B-horizon soil survey that comprised 170 samples taken from two grids and at 50 m intervals. In addition, Nycon drilled 2,006 meters in 17 core holes at West Dufferin. Assays returned many promising intervals, and corroborated the pre-43-101 drilling (see Drilling, below).

In 2012, Nycon contracted Mercator to review the previous mapping and sampling results to guide a new soil geochemistry program. This new survey comprised 105 samples taken at 50-m intervals near North West Lake, and did not identify highly anomalous gold contents in soils. Limited reconnaissance mapping north of the soil grids did not identify any new outcrops.

In August of 2013, Mercator was contracted to collect B-horizon soil samples in License area 06274 to extend the 2012 sampling grid and infill samples taken in the 2010 and 2012 soil sampling programs. The sampling program comprised 47 B-horizon samples taken at ca. 50m intervals. The highest gold value was 128 ppb Au, in a sample taken from the northwest corner of the western soil grid. 48 ppb Au and 22 ppm As were measured in a sample taken on the south side of Muskrat Lakes road and approximately 150 m south of the projected Salmon River Anticline axis.

In 2014, Nycon contracted Mercator to complete outcrop mapping and a B-horizon soil sampling program in License area 06274. The 2014 soil sampling program extended and infilled the western sampling grid to better understand the distribution of anomalous Au values in soil samples from the previous programs. The sampling program comprised 91 B-horizon samples taken at ca. 50 m spacing. Overall, only mildly anomalous gold and arsenic values were measured in soil samples taken during the 2014 program. New outcrops of greywacke and altered greywacke were recorded along a ridge at thirty three stations. The geological data gathered at these stations indicated that the Salmon River Anticline axis was ca. 200 m south of its previously inferred location.

In 2015, Nycon contracted Mercator to complete a B-horizon soil sampling program in license area 06274. The program comprised 60 soil samples taken to extend and infill previous soil sampling surveys. Significant gold values in soil samples were not measured, but coincident positive gold and arsenic values were interpreted to represent the western extension of the Salmon River Anticline.

6.2.1.16 West Dufferin Historical Production

As described above, production from mines at West Dufferin between 1883 and 1925 was reported in a contemporary document to be 35,301 ounces of gold from 110,576.5 tons of mineralised rock crushed (Malcolm, 1929). A more recent published figure was reported as 41,805.4 ounces between 1881 and 1935 (Bates, 1987).

6.2.1.17 West Dufferin Historical Mineral Resource Estimates

There are no previous NI43-101-compliant Mineral Resource estimates for the West Dufferin Property.

A current mineral resource estimate has been prepared, and is described in Section 14.2.

7 Geological Setting and Mineralization

7.1 Regional Geology

The gold fields of Nova Scotia occur within the Meguma structural terrane, a tectonic subdivision of the northern Appalachians.

The Meguma terrane is predominantly composed of Cambrian to Lower Devonian sediments and turbiditic metasediments (Figure 7-1), and is found only in mainland Nova Scotia. Rock units of the terrane were folded around north-easterly trending, sub-horizontal axes during the Early Devonian Acadian Orogeny when the terrane docked with the eastern side of the Avalon terrane.

These sediments have been subdivided into two groups:

a) The Cambro-Ordovician Meguma Group, which consists of the basal Goldenville Formation quartz wackes and overlying Halifax Formation slates; and

b) The Silurian to Lower Devonian-aged infolded keels of clastic sediments and volcanics of the White Rock, Kentville, New Caan and Torbrook formations, which occur at the terrane's west and north-easterly limits (Zalnieriunas, 1997).

The Meguma Group was intruded by granitoid plutons of Middle Devonian to Early Carboniferous age. These consisted of granite, granodiorite, granodiorite porphyry, granite and lesser quantities of tonalite and trondhjemite. Intrusives range in size from a few square kilometres to that of composite batholiths. The intrusions deformed and metamorphosed the turbidite sequence during the Acadian Orogeny. The main feature of the deformational history of the Meguma Terrane is the formation of a series of major east-west trending upright symmetric to slightly reclined asymmetric folds. A penetrative slaty cleavage was developed in the argillaceous units during this episode as well as a pervasive pressure solution cleavage in the greywackes. This folding and cleavage have an important role in the development of the gold deposits. Regional metamorphism is of greenschist to upper amphibolite facies. Contact metamorphism occurs adjacent to granitoid intrusions.

A pervasive alteration which may extend for several kilometres along strike on either side of individual gold deposit consists of silicification, carbonitization, sericitization and sulphidization. A less pervasive and more restricted alteration within the gold districts is characterized by carbonate and chlorite (+/- arsenopyrite) within gold districts (Smith and Kontak, 1990).

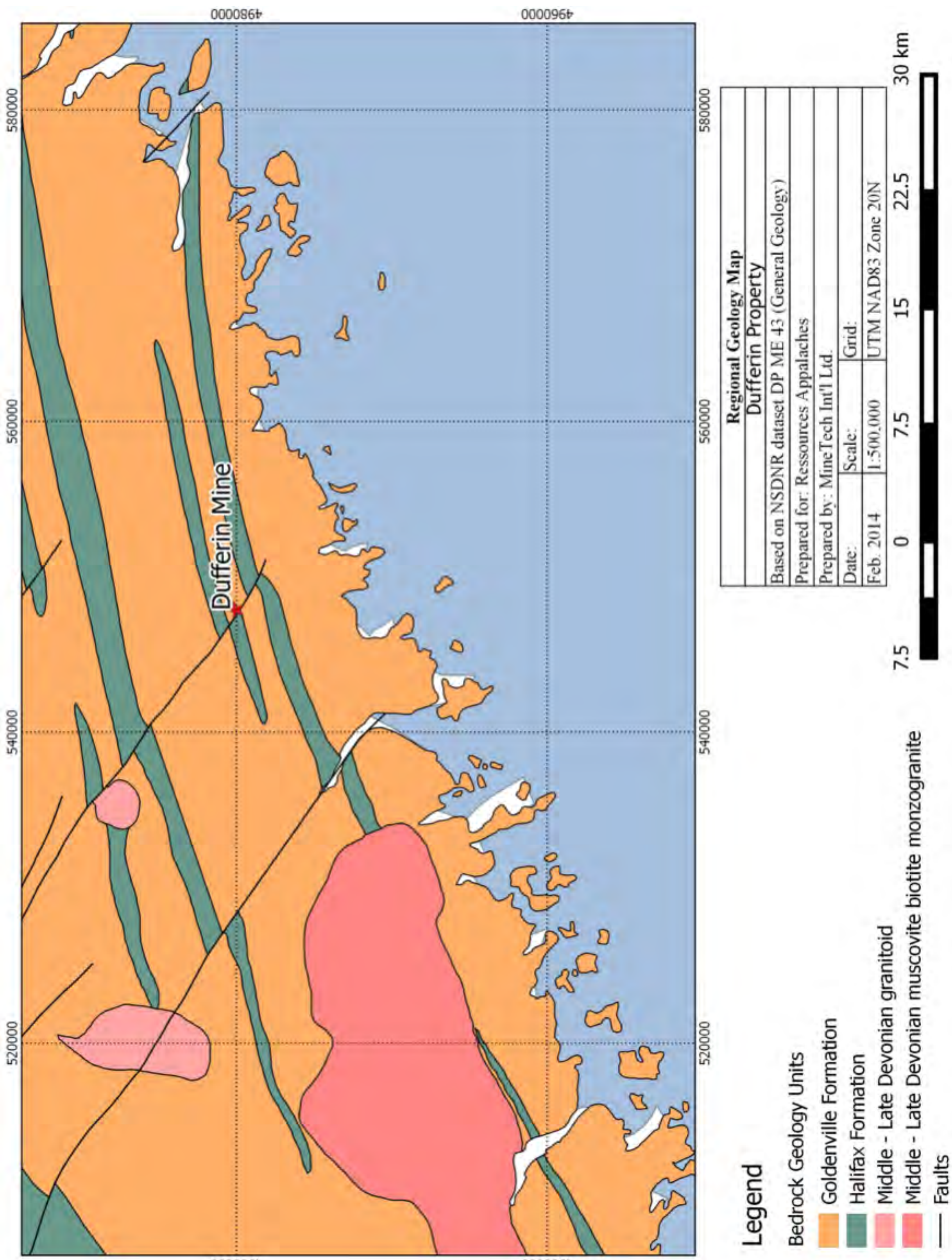
Mineral occurrences within the Meguma occur as three styles as follows:

a) Concordant, syn-depositional or diagenetically related deposits such as the Eastville base metal occurrences associated with the Goldenville Halifax Formation Transition Zone (GHT) and Clinton-type iron formations.

b) Hydrothermal, structurally controlled deposits such as the 370 Ma-aged auriferous concordant Goldenville Formation quartz veins.

c) Mineralization associated with Acadian plutonism such as the East Kemptville or Millet Brook uranium deposits (after Zalnieriunas, 1997).

Figure 7-1 - Regional Geology



7.2 Property Geology

The Dufferin Property is underlain by folded metasediments of the Goldenville Formation, chiefly as greywacke with minor interbedded argillite, and the Halifax Formation, composed of black, graphitic slate. While confined to the northern end of the property, rocks of the Halifax Formation serve as an excellent marker horizon. These rocks are folded into a series of gently east-plunging, upright anticlines and synclines.

The Salmon River and Crown Reserve anticlines are separated by the Ruth Falls Syncline, trending east-northeast across the Property. The western ends of these structures are terminated by the northwest striking (sinistral) Harrigan Cove Fault. This major fault has displaced the Salmon River and Crown Reserve anticlines along the northwest trace of the fault by approximately 1.5 kilometres of sinistral strike-slip separation. West of the Harrigan Cove Fault, both anticlines have been identified at West Dufferin, where the Salmon River anticline retains the same name, but the Crown Reserve anticline is called the Dufferin Mines anticline. A significant dip-slip displacement on the Harrigan Cove Fault is suggested by the variance in separation of the trace of the Salmon River and Dufferin Mines anticlines west of the fault and the Salmon River and Crown Reserve anticlines east of the fault (Figure 7-2) (Horne and Jodrey, 2002).

The Crown Reserve Anticline at East Dufferin is a tight chevron-style fold, steeply inclined to the south, the hinge zone of which is a rounded arc-shaped structure 5 to 10 metres across. The limbs (leg-reef veins) are uniform and straight. The south limb has a dip of approximately 65°, and the north limb has a dip of approximately 78°.

There are three significant faults on the East Dufferin portion of the Property that offset the Crown Reserve Anticline and vein array (see Figure 7-2). These faults are referred to as Faults 1, 2, and 3 (numbered from west to east). Faults 1 and 3 trend northwest, parallel to the Harrigan Cove Fault whereas fault 2 trends north-south. Each of these faults displays oblique, sinistral, east-side-down displacement with the dip-slip displacement less than the strike-slip displacement (Home and Jodrey, 2002).

Mining activity at East Dufferin during 2001 by EnviroGold Technologies concentrated on saddle-veins along the Crown Reserve Anticline. Gold bearing quartz saddle-veins occur within dilation zones within argillite units along the hinge of the fold axis as well as within quartz leg-reef veins primarily on the steeper dipping north limbs. While no marker units were identified, the stratigraphic section was subdivided into a series of units which could be correlated from limb to limb and along strike.

The units were classified (Woodman, 1999) as:

- Massive greywacke with little or no argillite
- Interbedded silty-greywacke and argillite
- Argillite and quartz veins.

Alteration of the massive units includes silicification, and carbonate alteration of arsenopyrite to siderite. Other alteration of the host rock includes chloritization, sericitization which altered the original feldspar minerals, and silicification and carbonatization, caused by the introduction of a silica-CO_2 rich fluid.

The quartz saddle-veins are of milky white to grey coarsely crystalline quartz containing thin layerings of argillite and/or chlorite. The veins are generally thicker at the fold apex with sharp contacts between quartz and argillite on both the hanging wall and the footwall. Common gangue minerals in the quartz include ankerite, siderite, calcite, kaolinite and chlorite. Sulphide minerals in order of decreasing abundance include arsenopyrite, pyrite, galena, sphalerite, chalcopyrite, pyrrhotite and stibnite. Gold is often observed with galena and arsenopyrite, with galena being considered the best indicator sulphide for gold (Woodman, 1999). Arsenopyrite, up to a few percent, occurs within the veins and the wall rocks.

Mining of the saddle-reef veins and leg-reef veins has shown the complexity of structure in each vein. Massive, coarsely laminated, bedding-parallel quartz veins may occur with a zone of *en echelon* veins. Strong boudinage features are common in some *en echelon* veins reflecting high shear strain. The saddle-reef veins at the Dufferin Property have a pronounced asymmetry with saddle-reef development mainly in the hinge and the north limb of the fold. Similar asymmetry has been noted in the Bendigo-Castlemaine goldfields of Australia, as described in Section 8.2 (Horne and Jodrey, 2002).

Figure 7-2 - Structural geology of the site.

7.3 Mineralization

The gold-bearing anticline structures at Dufferin have been shown to have a strike length of at least 3 km and a depth of at least 400 metres. Eighteen or more stacked, saddle-reef veins have been indicated on the East Dufferin portion of the project through diamond drilling, mapping and underground mining. The veins are numbered Saddles M1, M2, 0 – 13, 1A, 1B etc. The saddles have a crest and associated leg-reef veins. The veins are sub horizontal, juxtaposed one above the other with 20 to 40 meter spacing. Saddle 1, which begins at the East Dufferin portal, has been followed for a strike length of 1,200 metres eastwards; new veins have been located above Saddle 1 in the eastern part of the Property (Ressources Appalaches press release, Aug 26, 2014). Mineralized veins have been traced over a total strike length of 1.6 km on East Dufferin.

The West Dufferin portion of the project hosts eighteen or more saddle-reef structures,—many of which appear to correlate with those at East Dufferin—as indicated by diamond drilling, mapping, and underground mining. These mineralized structures have been drilled over a strike length of 1.9 km west from the Harrigan Cove fault, which separates the two halves of the deposit.

Most of the largest gold deposits in Meguma Terrane are located in the Goldenville Formation, within and adjacent to quartz veins. These quartz veins are better developed in slate and their equivalent schist than in the more competent greywacke and quartzite strata. Gold bearing quartz saddle-veins occur within dilation zones. These zones are within argillite horizons along the hinge of the fold axes as well as within quartz leg-reef veins primarily on the steeper dipping north limbs.

The most common forms of gold mineralization are as free gold in fine films near crack-seal laminae, along vein-wall contacts, as coarse-grained aggregates in the white quartz, within sericitic fractures, and as smears within quartz veins. Gold also occurs along selvages within the argillite and as inclusions, fracture fillings or attached to sulphides such as galena and arsenopyrite. Most of the gold associated with quartz veining contains approximately 5% silver (Ryan and Smith, 1998).

Gold has been detected within all vein types, including saddle-reef, laminated leg-reef, en echelon and discordant veins; this is consistent with the interpretation of synchronous formation of all veins. The most abundant visible gold was noted in the north leg of saddle-reef 2a at East Dufferin, where it commonly occurs near slate septa within the vein. The correlation of visible gold and galena has been observed during the mining of the deposit (Coles, 2002). Gold usually occurs within a few centimetres of galena and may actually occur attached to the sulphide crystals. Because of this relationship, galena is used as an indicator of gold mineralization in the exploration of the Dufferin veins.

8 Deposit Types

The Dufferin deposit is orogenic, turbidite hosted quartz–carbonate vein deposit. This type of deposit is also found in the Bendigo area of Australia (see Section 8.2), in the Stawell-Juneau region of Alaska-and in the Meguma Group rocks of Nova Scotia.

8.1 Meguma Gold Deposit

To date, over 60 deposits have been exploited in the Meguma Terrane of Nova Scotia, yielding over 1.2 million ounces of gold.

Most of the gold is from narrow, high-grade veins in relatively shallow (<300 metre) underground mines. The more significant gold deposits in the Meguma Terrane are located in the Goldenville Formation, in and adjacent to quartz veins. These quartz veins are better developed in slate and their equivalent schist than in the more competent greywacke and quartzite strata.

One difficulty in planning underground development on a narrow vein gold deposit in the Meguma is that gold distribution tends to be 'nuggety'; that is, it is not distributed evenly within the quartz veins. For this reason, determining the location, size, shape, rake and grade of mineable shoots is best done with a combination of diamond drilling and mine development.

R.V. Zalnieriunas developed a simplified classification of auriferous veins and vein systems within the Meguma. The system is based on the vein mineral content and its relationships to structure and stratigraphy. It is summarized as follows:

"Historically, the types of quartz veining which has been recognized within the Meguma consists of:

Interbedded Veins:

> The majority of the "ore-bearing" veins are of this bedding parallel type. They vary in thickness from a few millimetres up to a metre. They are stratiform, with good lateral and vertical continuity. They are generally developed within the slates and schists and are composed of quartz with carbonate, minor amounts of sulphides and variable amounts of tourmaline, scheelite, stibnite and cassiterite. Texturally the veins are laminated to brecciated, hosting variable amounts of wall rock inclusions. Gold appears as fine to coarse-grained free particles in quartz, carbonate, sulphides and slate.

Fissure Veins:

> Fissure veins are mineralogically similar to interbedded veins but lack the laminated textures. They cross-cut stratigraphy and may be persistent for considerable lateral distances. These veins may host gold mineralization and are also referred to as "cross veins".

Angular Veins:

> Angular veins are discordant veins which pass from or through laminated veins into the adjacent sedimentary wall rock. They may pinch out or grade into stockwork zones. The veins may be barren to weakly mineralized. They may also locally host high grade gold and sulphides close to the laminated bedding parallel veins.

Bull Veins:

> Bull veins consist of massive, white quartz. The veins may be up to several metres thick and can be oriented either conformable or disconformable to the sedimentary strata ".

In addition, other characteristics of gold deposits noted within the Meguma are described by earlier workers such as Faribault and Malcolm (Malcolm, 1929) as part of the general classification of these deposits (after Covey, 1987).

Belts:

Areas where a number of laminated veins are located in one or a series of closely spaced slate or schist units are termed "belts ".

Rolls:

During progressive deformation, auriferous veins may be further localised in hinge zones by the mechanisms of folding and hinge zone thickening associated with incipient similar folding.

Crumples:

An important gold concentration mechanism occurs during later stages of deformation by buckling or kinking of the slate units, and coincident fracturing of more competent rocks. These features are seen as bands orientated at an angle to the axial plane of the regional folding, This phenomenon causes fore-shortening of the kinked or buckled area.” (Zalnieriunas, 1997)

The majority of former Nova Scotia gold producers worked individual leads or belts which were typically characterized by narrow mining widths and relatively erratic but sometimes very high grades. In many Meguma gold deposits “ore” shoots are small target areas containing 5,000 t to 20,000 t in one or more thin leads when diluted to a 1 m to 1.2 m mining width. Many of these operations, standing alone, would not be commercially viable by today's standards because of difficulties in maintaining sustained production. Larger deposits, however, while they exhibit geological characteristics similar to the smaller ones, have additional structural or lithological features as a means of concentrating the gold. Such features may include shears, hinge zone thickenings, wider argillite beds or cross-faulting.

8.2 Bendigo-Castlemaine Goldfields

Similarities between the saddle-vein deposits in Nova Scotia to those of the Bendigo - Castlemaine goldfields of the State of Victoria, Australia have been noted by numerous authors (including Chace, 1949; Fowler and Winsor, 1996; and Horne, 2002). The Bendigo-Castlemaine goldfields lie within a package of chevron-folded Lower Ordovician turbidites composed primarily of regularly alternating slates, argillites, greywacke and quartzites which were later intruded by local Devonian granites.

8.3 Orogenic Gold Deposits

Turbidite-hosted Meguma gold deposits are a sub-type of orogenic gold deposits. Orogenic gold deposits form near or soon after peak metamorphism in collisional metamorphic terranes of all ages. Displaying strong structural control in 2^{nd}- and 3^{rd}-order brittle faults and ductile shear zones as quartz-dominated stockworks, breccias, sheeted veins, vein arrays, replacements, and disseminations, most deposits formed at greenschist facies (250-350°C, 1-3 kbar, 2-20 km deep) in compressional-transpressional settings at convergent plate margins near 1^{st}-order deep crustal fault zones with complex structural histories, especially where these faults change direction (Goldfarb et al, 2005; Groves et al, 1998). Orogenic gold systems can be huge—with the largest up to 2-10 km long, 1 km wide, and 2-3 km deep—and contain some of the planet’s largest concentrations of gold, such as deposits in the Kalgoorlie (39M ounces) and Bendigo

(26M ounces) districts in Australia, and the Timmins (64M oz) and Kirkland Lake (24M oz) districts in the Canadian Shield.

Ore occurs in quartz veins and altered wall rock, with generally high gold:silver ratios and high fineness, accompanied by 2-5% sulfides. Historically, high-grade veins were exploited (5-30 g/tonne), but many deposits comprise large volumes of lower-grade, bulk-mineable ore. Alteration consistently adds CO_2, S, K, H_2O, SiO_2 to wall rocks in the form of carbonates (ankerite, calcite, dolomite), sulfides (pyrite, arsenopyrite, pyrrhotite), and silicates (muscovite, biotite, K-feldspar, albite, and chlorite); scheelite and tourmaline are common, and at higher metamorphic grades amphibole, diopside, and other skarn-like replacement minerals occur. The typical geochemical signature is elevated As, B, Bi, Hg, Sb, Te, and W, with generally low Cu, Pb, and Zn. Gold was transported as sulfide complexes in reduced, near-neutral metamorphic fluids of high CO_2 and low salinity and deposited by pressure decreases during episodic seismic events (leading to the characteristic banded quartz veins) or by desulfidation reactions with wall rocks.

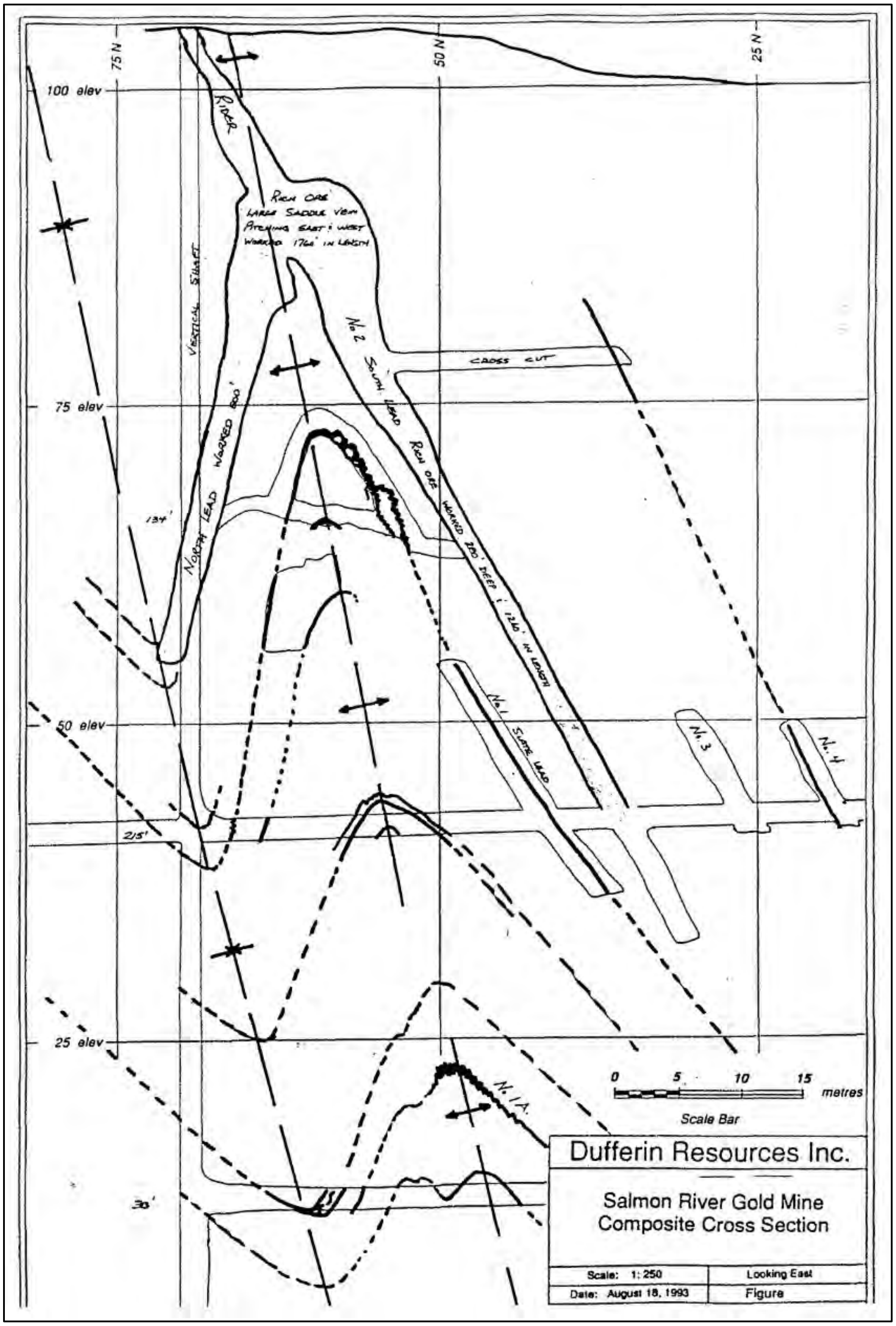


Figure 8-1 - Cross Section through the Salmon River Mine at West Dufferin (Duncan & Graves, 1993)

9 Exploration

9.1 East Dufferin Exploration

Recent exploration work at the East Dufferin site includes diamond drilling, a LiDAR survey, a ground magnetometer survey, metallurgical testing, and underground sampling, as well as prospecting and mapping.

9.1.1 LiDAR Survey

An airborne LiDAR survey was carried out by LiDAR Services Ltd. of Calgary and all exploration licences were covered. The data "permitted the company to identify surficial deposits, geological structures and folding on several of the properties and will be used to help focus follow-up ground geological surveys, prospecting and sampling."[5] A detail of the survey covering the Dufferin property is shown in Figure 9-1.

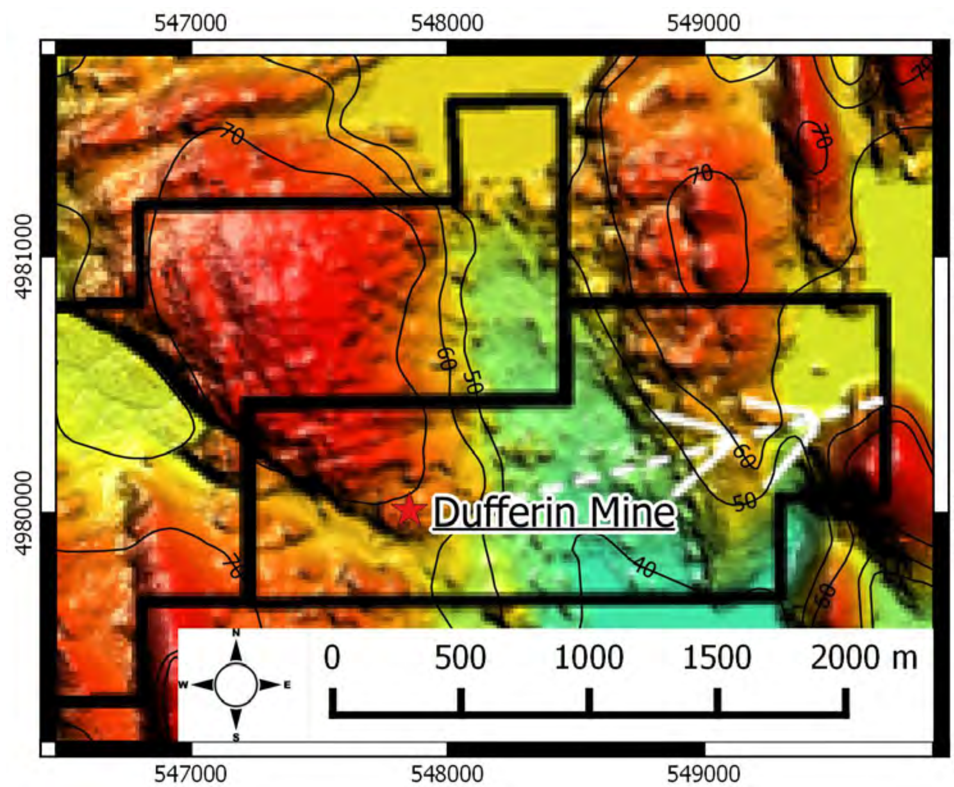


Figure 9-1 - Detail of LiDAR Survey (UTM NAD 83, Zone 20N coordinates)

[5] Ressources Appalaches Press Release, October 25, 2010, "Ressources Appalaches: Drilling Doubles Dufferin Mine Gold Zone from 700m to 1.4Kms along Strike, Second Drill Program Underway"

9.1.2 Ground Magnetometer Survey

The ground magnetometer survey was carried out at East Dufferin between June 24th and July 1st, 2010, by Geosig Inc. of Québec City, Québec. It covered 25 grid lines at the property (from 25+25E to 37+75E, with a 50 metre spacing); each line has a length of approximately 1 km. Total line distance covered was 28.5 km. The western half of survey was completed in walking mode. The remaining lines were done in mobile mode with readings every 5 metres. Lines were not cut; the survey was done using GPS across the bush.

Geosig Inc. identified two major magnetic anomalies as well as one smaller magnetic anomaly crossing the grid, with a roughly ENE-WSW trend. At least 6 faults with a NW trend were identified. Displacement of the southern magnetic anomaly was minor, except near the creek where displacement of more than 50 metres was identified. The interpreted magnetic map is shown in Figure 9-2.

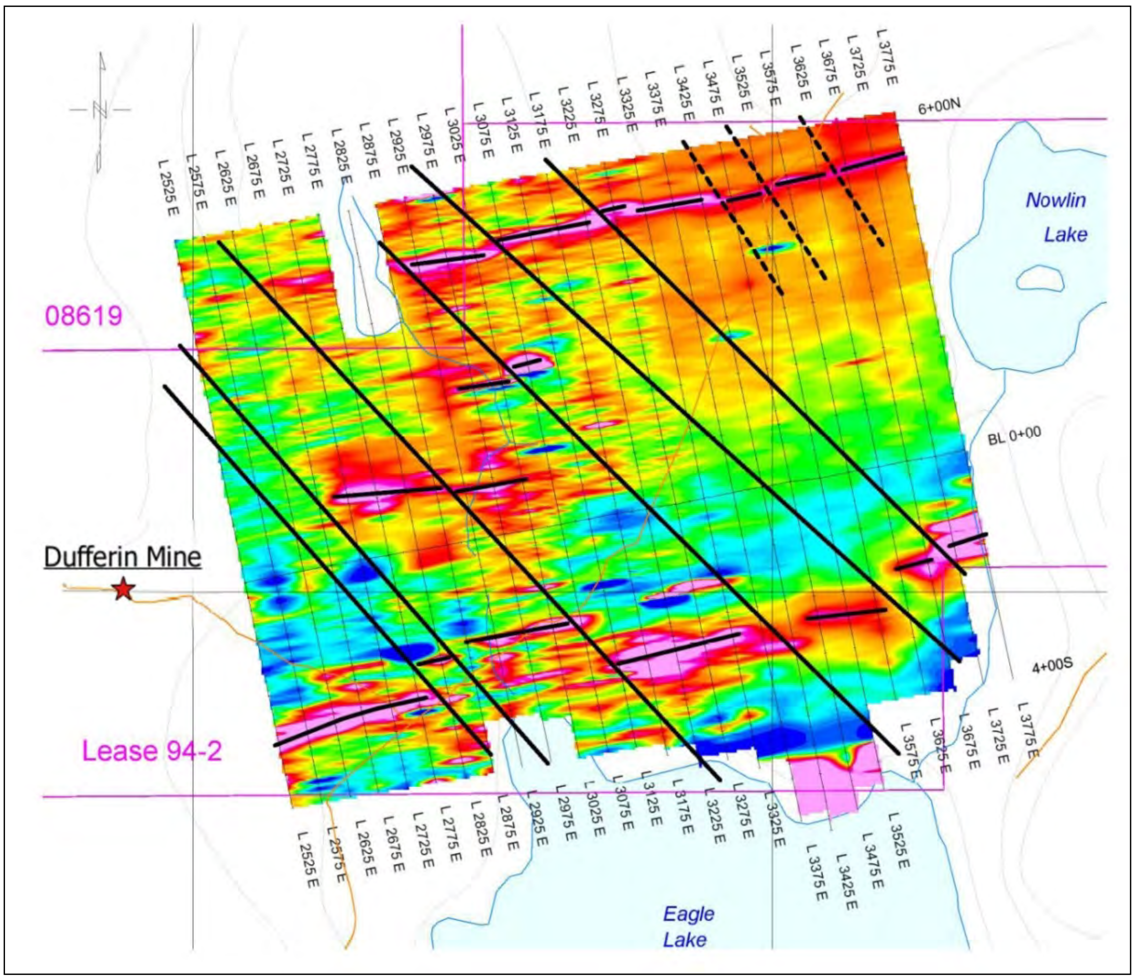


Figure 9-2 - Dufferin Magnetic Survey Map Interpretation

9.1.3 Metallurgical Testing

Metallurgical testing was commissioned by Ressources Appalaches. It is discussed in Sections 13.10, 13.11, and 13.12.

9.1.4 Underground Sampling

Underground samples were collected in 2014. Unfortunately, no coordinates were recorded at the time of sampling. However, coordinates were later assigned "forensically" for the purpose of using them for mineral resource estimation.

In carrying out their underground sampling, Ressources Appalaches staff used an Ingersoll Rand handheld pneumatic chipping hammer along with chisel and maul. Typically a tarp was laid out beneath the face to catch all coarse and fine material. Typically they took 3-5 samples weighing 1 to 2 kg each across the vein incorporating the argillic buffer at the contacts.[6]

Average sample grades were calculated for each saddle (Table 9-1). No top-cut grade was used. Channel samples graded much higher than muck samples, particularly for Saddle 1. The average muck sample for Saddle 1, the source of the vast majority of mill feed, was 1.5 g/tonne, which compared quite well with the overall estimated mill feed grade of 1.7-2.0 g/tonne.

The average channel sample widths for all saddles were fairly comparable, in the 0.6-0.8 metre range.

Table 9-1 - Underground sampling.

	Average Sample Grade (g/tonne)			Number of Samples			
Saddle	Channel Sample	Muck Sample	'Representative' Sample	Channel Samples	Muck Samples	'Representative' Samples	Average Channel Sample Width (m)
Saddle 1	6.6	1.5	5.5	271	110	69	0.64
Saddle 2	34.8	8.5	3.9	48	35	24	0.79
Saddle 3	1.8	0.3	0.5	46	4	2	0.76
Saddle 4	29.1	9.1	36.9	43	23	1	0.62

* No top-cut was applied.

Underground channel sampling, muck sampling from production, and the estimated block model grades were compared (see Table 9-1). For Saddle 1, which was the source of most of 2014's production, the average estimated block grades for the mined portion was in-between the channel and muck sample grades.

[6] Pers. Comm, Leah Page, Geologist with Ressources Appalaches, Jan 2015

Table 9-2 - Comparison of underground sampling, muck sampling, and estimated block grades.

Saddle	2014 Mining Location	Average Channel Sample Grade (g/tonne)	Average Muck Sample Grade (g/tonne)	Average Crest Domain Block Model Grade (g/tonne)
Saddle 1	2238-2308 m East and 2436-2626 m East, Mainly on 1A	6.6	1.5	3.9
Saddle 2	2451-2469 m East	34.8	8.5	0.8
Saddle 3	2427-2446 m East	1.8	0.3	2.6
Saddle 4	2253-2283 m East	29.1	9.1	2.6

Compared against muck sampling, estimated block grades for Saddle 2 were lower, higher for Saddle 3, and lower for Saddle 4. However, very little material was mined from these saddles.

Saddle 1 saw the most production. The average estimated block grade for the mined areas was 3.9 g/tonne, which is less than the average channel sample grade. Those samples were taken more selectively than how the zones were modelled, as the model takes the minimum mining width into account. In other words, we would expect the block model to be more diluted than the channel samples, which may explain the lower average grade of the blocks.

The average muck sample grade was lower than the estimated block model grade. The minimum mining width of 1.5 metres for the block model, compared with the actual mining height of approximately four metres in Saddle 1, means that the muck was much more diluted than the block model, which may explain the lower average muck sample grade.

In portions of quartz vein Saddles 1, 2, and 4, underground grade-control samples were taken by Ressources Appalaches during mining activities. These samples cut across the mineralized quartz vein structures in vertical channel samples taken at approximately three-meter intervals along strike. In order to validate the assay results of these samples, which were originally analyzed at Ressources Appalaches' onsite mine lab, RCG undertook a program of re-assaying these samples. Coarse rejects from previous grade-control samples were sent to an independent laboratory (ALS Minerals) for analysis by screen fire metallics, the preferred testing method for mineralization with relatively coarse gold particles, as at Dufferin. These new results validated the previous grade-control sample assays, in some cases returning higher values.

The underground grade-control samples from these portions of Saddles 1, 2, and 4 have a weighted average grade of 34.3 g/tonne Au. More widely spaced drill-hole assays that cut these same portions of the same saddles have a weighted average grade of 6.1 g/tonne Au. Thus, at least in these locations, drill assays considerably underestimated the grade of quartz vein mineralization at East Dufferin. This may have implications for actual mined grade in other portions of the deposit. Actual mined grade from these blocks is not available, due to mill feed blending during Ressources Appalaches operations.

The wide discrepancy between the drilled grade and underground sample grade in these portions of Saddles 1, 2, and 4 may be explained by the difference in volume and density of the drill sampling vs. underground sampling in a deposit with coarse gold displaying a nugget effect. In similar deposits, research and testing has repeatedly demonstrated that smaller assay charges, smaller sample volume, and lower sample density (wider-spaced samples) will tend to understate the actual grade in narrow-vein coarse gold deposits (Dominy et al, 2001, 2008;

Johansen, 1987). Dominy et al (2001) state that "the greatest understatement of grade comes from surface diamond drill holes and the least from bulk sampling." The authors presents data from two narrow-vein coarse-grained gold deposits; in one deposit, diamond drilling understated the calculated mill head grade by 85 – 94%; in another, grade-control sampling underestimated calculated mill head grade by 65%.

Thus, the drill sampling in Saddles 1, 2, and 4 was lower grade than the grade-control sampling because the drill sampling constituted smaller, more widely spaced samples. This implies that the calculated resource grade of the Dufferin deposit, based largely on drill results, may underestimate the grade that will be attained during actual mining.

9.2 West Dufferin Exploration

The following summary of exploration activities in the West Dufferin project area is modified from Mitchell (1989), Woodman (2002, 2004a, b), Kennedy and Yule (2010), and Yule (2012, 2013, 2014, 2015).

9.2.1 Lons Mining Corp.

After production into the 1930s, the Salmon River Mine remained idle until 1972 when Lons Mining Corp. completed trenches on the property. Lons focused on the eastern extension of the Salmon River Anticline and near the Eagle Lake shaft. Assays of samples collected from trenches reported low gold values, even though some samples apparently contained visible gold. Mapping of the trenched areas defined the fold axis.

9.2.2 Tri-Bridge Consolidated Gold Mines Ltd.

Tri-Bridge Consolidated Gold Mines Ltd. acquired the Salmon River property and cut grid lines for magnetic/VLF and soil geochemical surveys that outlined several anomalous zones.

9.2.3 St. Joseph Exploration Ltd.

St. Joe re-cut Tri-Bridge lines and conducted soil geochemical and IP surveys near the hinge of the Salmon River Anticline. An IP anomaly was interpreted west of the Dufferin mines and continued to Eagle Lake.

9.2.4 Pan East Resources

Pan East completed an airborne magnetic and VLF survey of the Salmon River Gold District in 1980, and followed up with a stream sediment and humus sampling program south of the Salmon River Gold Mine in 1981.

9.2.5 Seabright Explorations Inc.

The company completed a detailed exploration program to define the western extensions of the Salmon River Anticline. This program comprised ground geophysics (VLF, magnetics, and IP), and soil and till geochemical surveys (Mitchell, 1989). Gold values up to 44 ppb in till and 18 ppb in soil were reported north of Muskrat Lake, and a quartz boulder found near East Lake assayed 50.75 g/tonne (Mitchell 1989).

9.2.6 Bruce Mitchell

Mr. Mitchell held the licenses covering the western extension of the Salmon River anticline and conducted small ground geophysical surveys (magnetics and VLF), soil geochemical surveys, trenching, and prospecting. Excavation of a 225 m-long trench on a soil anomaly north of Halfway Brooke Lake rarely exposed bedrock. Few structural data were gathered from rocks exposed during the trenching. Quartz boulders uncovered during trenching contained 0.01 to 2.61 g/tonne Au, with two unassayed boulders containing visible gold.

9.2.7 Nycon Resources

Nycon acquired license area 06274 in 2001, and in 2002 conducted a soil geochemical survey on the western extension of the Salmon River Anticline that outlined broad arsenic and gold anomalies south of the projected anticline axis. 131 B-horizon soil samples were taken at 100 m intervals on 100 m line spacing, and were prepared and analyzed for Au and As at Dalhousie University using aqua regia digestion and an atomic absorption finish. The highest measured Au value in a soil sample was 30 ppb. Many of the historical geochemical anomalies were interpreted to be spurious as a result of contamination by mine material (Woodman, 2002).

In 2007, Nycon contracted Brian McPherson to conduct a soil sampling survey north and south of the Salmon River Anticline axis. 54 B-horizon soil samples and 4 stream sediment samples were collected. Soil samples were concentrated and sent to Dalhousie University to be analyzed for gold and arsenic. Soil samples contained 23-72 ppb Au (average 38 ppb Au) over a 500 m-long anomalous zone in the southern part of the survey area. Stream sediment samples apparently did not contain detectable Au.

In 2008, Nycon contracted Brian McPherson to conduct a mobile metal ion (MMI) geochemical sampling program to identify gold bearing veins at depth and within the Salmon River Anticline. The samples defined anomalous trends in Au values north of the Salmon River Anticline axis. These data showed that lead, zinc, and copper anomalies broadly coincided with elevated gold values.

In 2010, Nycon contracted Mercator Geological Services (Mercator) to compile and review past geological and geochemical data to guide future exploration in License area 06274. Mercator also completed reconnaissance mapping, prospecting, a LIDAR topography survey, and a B-horizon soil survey that comprised 170 samples taken from two grids and at ca. 50 m intervals. None of the soil samples contained highly anomalous concentrations of gold.

In 2012, Nycon contracted Mercator to review the previous mapping and sampling results to guide a new soil geochemistry program. This new survey comprised 105 samples taken at approximately 50 m intervals near North West Lake, and did not identify highly anomalous gold contents in soils. Limited reconnaissance mapping north of the soil grids did not identify any new outcrops.

In August of 2013, Mercator was contracted to collect B-horizon soil samples in License area 06274 to extend the 2012 sampling grid and infill samples taken in the 2010 and 2012 soil sampling programs. The sampling program comprised 47 B-horizon samples taken at ~50m intervals. The highest gold value was 128 ppb Au, in a sample taken from the northwest corner of the western soil grid. 48 ppb Au and 22 ppm As were measured in a sample taken on the south side of Muskrat Lakes road and approximately 150 m south of the projected Salmon River Anticline axis.

In 2014, Nycon contracted Mercator to complete outcrop mapping and a B-horizon soil sampling program in License area 06274. The 2014 soil sampling program extended and filled in the western sampling grid to better understand the distribution of anomalous Au values in soil samples from the previous programs. The sampling program comprised 91 B-horizon samples taken at ~50 m spacing. Overall, only mildly anomalous gold and arsenic values were measured in soil samples taken during the 2014 program. New outcrops of greywacke and altered greywacke were recorded along a ridge at 33 stations. The geological data gathered at these stations indicated that the Salmon River Anticline axis was about 200 m south of its previously inferred location.

In 2015, Nycon contracted Mercator to complete a B-horizon soil sampling program in license area 06274. The program comprised 60 soil samples taken to extend and fill in previous soil sampling surveys. Significant gold values in soil samples were not measured, but coincident positive gold and arsenic values were interpreted to represent the western extension of the Salmon River Anticline.

10 Drilling

10.1 East Dufferin Drilling

10.1.1 Drilling Summary

One hundred (100) NQ diamond drill holes with a combined length of 15,325 metres were drilled on the East Dufferin portion of the property by Ressources Appalaches between 2008 and 2014 (see Table 10-1).

Table 10-1 - Summary of Drilling by Year

Year	Drill Holes	Metres Drilled
2008	23	2,995
2009 (New Holes)	19	3,029
2009 (Extensions)	4	312
2010	31	5,290
2014 (Underground)	9	600
2014 (Surface)	14	3,099
Total	*100*	*15,325*

10.1.2 Review of Previous Programs

The objective of the 2008 drill program was to follow the anticlinal axis to the East and to locate additional saddle reef veins at depth. Drilling took place in November and December of 2008. Holes were surveyed in UTM NAD 27 zone 20 with a Trimble GeoXH GPS. The diamond drill holes were NQ-sized. Tropari tests were used to check the orientation and dip. All casing was left in place and the core was stored on site. All the cores were logged by Caroline St-Onge, geologist of Géominex. Dominique Gagné, also a geologist of Geominex, supervised the field work and creation of the cross sections. The entire program was supervised by Alain Hupé, senior geologist for Géominex (later the President of Ressources Appalaches).

The objective of the 2009 drilling program was resource definition. The first six Saddle Reef type veins were systematically drilled at intervals of 75 metres to 150 metres over a length of 600 metres and down to a maximum depth of 210 metres. A few holes were also drilled to test veins located within 50 metres of the surface, and to confirm the continuation of these veins over a length of 650 metres.

The objective of the first part of the 2010 drill program was to extend the primary axis of Dufferin gold mineralization, to investigate fold structures and to test the host turbiditic greywackes for different styles of mineralization. Ressources Appalaches reported that the mineralized Crown Reserve anticlinal axis has been extended for an additional strike length of 700 metres, bringing the total strike length of mineralization to at least 1,400 metres.

The objective of the second part of the 2010 drill program was to confirm the depth of the mineralization to below 300 metres. Tropari downhole surveys were used to check the orientation and dip. Many holes were not surveyed due to equipment malfunctions. As a result, downhole surveys for some holes were unreliable; to date these holes have not been resurveyed.

During all programs, issues arose with downhole azimuth readings being approximately 30 degrees off. No cause has been found; magnetics in the area are normal. Leah Page, geologist for Ressources Appalaches, corrected the results using the declination and a Brunton compass in the field. For holes with no downhole surveys, the surface azimuth was used as the azimuth for the entire hole.[7] Dips have been reliable.

10.1.3 2014 Drilling Program

The primary objective of the 2014 drill program was to explore Veins 3 through 6, between depths of 75m and 200m from surface. The secondary objective was to explore veins towards the east of the project.[8]

A total of 23 NQ-sized diamond drill holes were drilled, with a combined length of 3,699 meters. Of those, 9 holes with a length of 600 meters were drilled underground, and 14 holes with a length of 3,099 meters were drilled on surface. Drill core from the program is stored on site. The collar locations for the 2014 holes can be found in Appendix I.

Drilling was contracted to two companies: Forages de l'Est and Forages La Virole, both from Rimouski, Quebec. Drill holes were spotted and logged by Ressources Appalaches staff and outside contractors. The drill program was overseen by Alain Hupé of Ressources Appalaches.

10.1.4 Drilling, Sampling, or Recovery Factors

There are no known drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the East Dufferin results.

10.2 West Dufferin Drilling

10.2.1 Drilling Summary

Drilling on the West Dufferin portion of the project consists of a total of 163 surface and underground core drill holes totalling 16,453 m drilled by six companies between 1935 and 2010 (Table 10-3). Of these holes, 127 (15,361 m) were drilled from surface and 36 (1,091 m) were drilled underground from the Cominco mine workings. Assays are available for 97 of the drill holes, and these comprise the drill assay database on the West Dufferin portion of the project.

---

[7] Personal Communication, Leah Page, January 2015

[8] Ressources Appalaches Press Release, Aug. 26, 2014, *"287 g/tonne Au drilled over 1 meter at Dufferin Mine"*

Table 10-3 - Summary of Drilling by Year

Year	Drill Holes	Metres Drilled		Company
1935	9	2445.5	surface	Cominco Ltd.
1937 - 1942	36	1091.28	underground	Cominco Ltd.
1972	13	92.99	surface	Lons Mining Corp.
1981	2	75.43	surface	St. Joseph Exploration Ltd.
1984	3	436.47	surface	US Borax and Chemical Corp.
1987 - 1988	60	8717.72	surface	Jascan Resources Inc.
2002	12	501.2	surface	Nycon Resources Inc.
2004	3	400	surface	Nycon Resources Inc.
2009	8	686	surface	Nycon Resources Inc.
2010	17	2006	surface	Nycon Resources Inc.
Total	*163*	*16,452.59*		

10.2.2 Review of Previous Drill Programs

The first drilling on the West Dufferin portion of the project was conducted by Cominco in 1935-1936, with 9 surface core holes totalling 2,446 m on the western end of the known mineralized trend. Promising results led to Cominco sinking the Cominco Shaft to an eventual depth of 161 m and completing 99 m of crosscuts and 147 m of drifts (Wightman, 1936, 1937). During the following six years, 1937-1942, Cominco drilled 36 underground core holes totalling 1,091 m.

Lons Mining Corp. drilled 13 small-diameter “Winkie” holes totalling 93 m in 1972 on the eastern end of the Salmon River anticline, with the best result being 0.46 m grading 34.3 g/tonne Au.

In 1981, two core holes were drilled by St. Joseph Exploration on the far western edge of the project. This was followed by three core holes drilled by US Borax in 1984.

The bulk of the drilling on the project was performed by Jascan Resources Inc. during 1987 – 1988, when 60 holes totalling 8,718 m were drilled. Although narrow, some results were extremely high in grade, including 3,167.9 g/tonne Au over 0.04 m (hole 87SR-61, 14.94 – 14.98 m) and 1,148.2 g/tonne Au over 0.06 m (87SR-32, 8.38 – 8.44 m). Results from this drilling outlined the currently known deposit at West Dufferin.

10.2.3 Nycon Resources Drilling

Nycon Resources drilled 40 core holes totalling 3,593 m in four programs between 2002 and 2010.

For unknown reasons, the 2002, 2004, 2009 Nycon drill programs were not assayed. These holes tested shallow eastern and central portions of the West Dufferin deposit (2002, 2009) and adjacent to the historical Cominco drilling on the western end of the deposit (2004). Resource Capital Gold is attempting to locate the drill core from these programs, and if found will evaluate it for possible assaying in order to add to the drill database for the project.

Drilling was contracted to Logan Drilling Group International, of Stewiacke, Nova Scotia. Drill holes were spotted and logged by D.R. Duncan & Associates geologists Stewart Yule and Jim

Barr. The drill program was overseen by David Duncan of D.R. Duncan & Associates (Duncan, 2011).

The 2010 drilling was distributed throughout the central portion of the deposit. These holes intersected many of the vein structures cut in previous drilling. Several holes—10SR-121, 113, and 125—extended below the depth of previous drilling and intersected new previously undrilled saddle-reef vein structures. Hole 10SR-121 in particular intersected 25 new saddle reef structures that extend mineralization to a depth of 305 m below surface. Highlights of the 2010 drilling included 158.6 g/tonne Au over 1 m (10SR-121, 158.2 – 159.2) and 9.9 g/tonne Au over 0.3 m (10SR-121, 59.4 – 59.7 m).

10.2.4 Drilling, Sampling, or Recovery Factors

There are no known drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the West Dufferin drill results.

11 Sample Preparation, Analyses and Security

11.1 East Dufferin

11.1.1 Previous Programs

11.1.1.1 On Site Sampling Preparation and Quality Control

No written methods or procedures were available for the pre-2008 work.

The sampling method and approach for the 2008 - 2010 drilling was reviewed by the authors, who have determined that drilling and sampling were carried out in a systematic and professional manner.

Figure 11-1 - Core storage

Figure 11-2 - Sampling

Though ALS Chemex, the assay lab used during the 2008 to 2010 drill programs, has its own internal QMS procedures, Ressources Appalaches did not submit any of its own standards, duplicates or blanks to ALS Chemex or any other laboratory for the 2008-2009 drilling.

11.1.1.2 Laboratory Sampling Method and Approach

Previous drill programs by Ressources Appalaches in 2008, 2009 and 2010 used the same laboratories as the current program (the ALS labs in Sudbury and Val D'Or). The sample preparation methods used in previous programs were very similar to the methods used for the current program. The analytical methods used on some samples differed: the current program used screened metallics for all samples, while previous programs used fire assay with atomic absorption finish for most samples, and used screened metallics and ICP-AES on less than 10% of samples.

11.1.2 2014 Drilling Program

11.1.2.1 On-Site Sample Preparation and Quality Control

11.1.2.1.1 Sample Preparation

Drill core sections were chosen for sampling and assaying based on the presence of visible sulphide mineralization, quartz veins, and argillite. In addition, both walls of the selected zones of interest were sampled. The minimum length of the core sample was 15 cm and the maximum length was 1.9 m with the exception of 2 samples over 200 cm long. The average length of the samples was 71 cm. A total of 774 samples were collected over the entire 23 hole drilling program giving an average of 34 samples per drill hole. The total length of sampled core was 551 metres which represents almost 15% of all the core drilled.

After being examined and logged, core was sampled in the following manner:

1. the core of the section to be sampled was cut in half with a hydraulic core splitter;
2. one half was put aside to be sent to the laboratory;
3. the second half of the core was returned to its place in the core box and a tag bearing the sample number was placed at the beginning of the split core forming the sampled length; and,
4. the two metallic collection bowls (one on either side of the splitter), the core splitter and the working table were then thoroughly brushed clean before proceeding to the next sample.

Samples were taken by a geologist or technician primarily to identify the presence of gold mineralization.

11.1.2.2 Quality Control

11.1.2.2.1 Blanks

A total of 61 blanks were sent with the 2014 samples. Of these, 59 assayed at 0.05 ppm or below, including 34 assaying below the detection limit of 0.01 ppm. One assayed at 0.45 ppm.

One blank, sample number J722224, assayed at 7.08 g/tonne. However, this may be the result of a standard being mislabelled as a blank. See Section 11.1.2.2.4 for more information.

11.1.2.2.2 Standards

A total of 24 CDN-GS-2M standards, 19 CDN-GS-7F standards, and one HiSilP1 sample were submitted to the lab. Both the CDN-GS-2M and CDN-GS-7F standards, as assayed, had averages that were close to the expected values, but the range of values was wider than expected. In both cases, values were generally lower than would have been expected. One CDN-GS-2M standard and one CDN-GS-7F standard assayed at a near-nil grade; these are considered to be outliers and are discussed in Section 11.1.2.2.4.

The HiSilP1 sample came within its expected range.

Table 11-1, Table 11-2, Figure 11-1, and Figure 11-2, below, summarize the results of the standards submitted to the lab. The summary statistics do not include the results from the outliers.

Table 11-1 - Reference Material Specifications

Standard	Mean (average) (g/tonne)	95% Confidence Interval (~2 standard deviations) (g/tonne)	-2 Standard Deviations	+2 Standard Deviations
CDN-GS-2M	2.210	0.244	1.966	2.454
CDN-GS-7F	6.90	0.41	6.49	7.31
HiSilP1	12.05	0.13	11.92	12.18

Table 11-2 - Reference Material Results (not including the two outliers)

Standard	Number Submitted (not including outliers)	Number Submitted (including outliers)	Mean (average) (g/tonne)*	Min (g/tonne)*	Max (g/tonne)*	2 standard deviations (g/tonne)*
CDN-GS-2M*	23	24	2.168	1.30	2.55	0.633
CDN-GS-7F*	19	20	6.85	6.37	7.31	0.57
HiSilP1	1	1	12.15	12.15	12.15	n/a

**Calculations do not include the outliers; see Section 11.1.2.2.4*

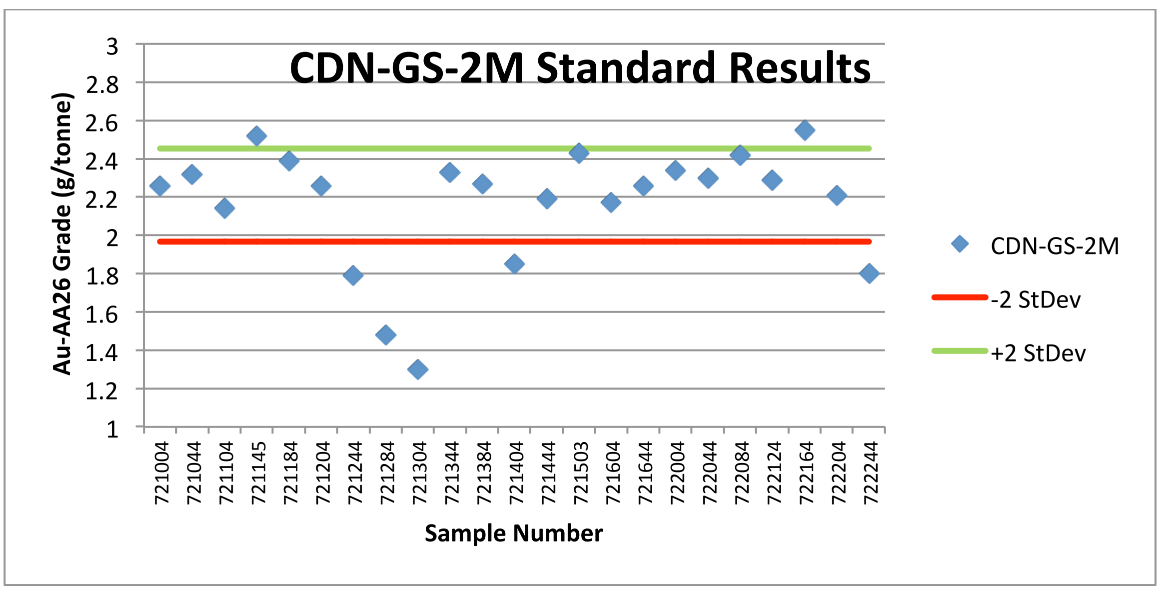


Figure 11-1 - CDN-GS-2M Standard Results (NB: Does not include 1 outlier that assayed at 0 g/tonne)

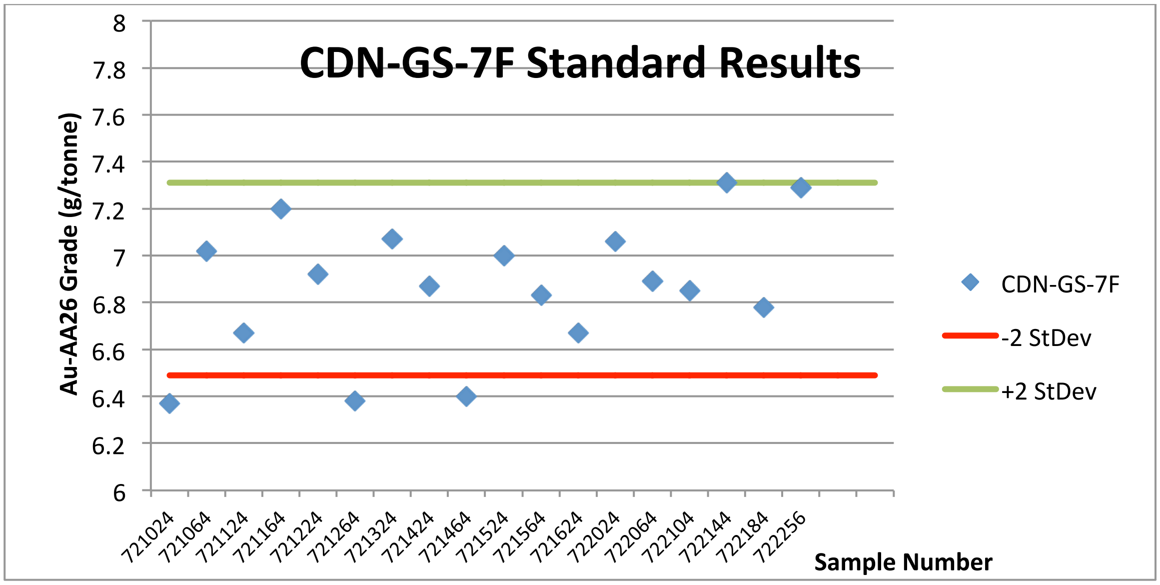


Figure 11-2 - CDN-GS-7F Standard Results (NB: Does not include 1 outlier that assayed at 0 g/tonne)

11.1.2.2.3 Duplicates

A total of 47 duplicate samples were submitted. One duplicate sample, number J722029, did not appear in the assay certificates, leaving 46 pairs of duplicate samples.

A wide spread between original and duplicate values was found for individual samples, which is expected, given the nuggety style of gold mineralization in at the Property. However, a strong correlation was found overall, with a coefficient of correlation *r* of 0.94. Figure 11-3 - 2014 Drill Program Duplicate Assays, below, illustrates the results.

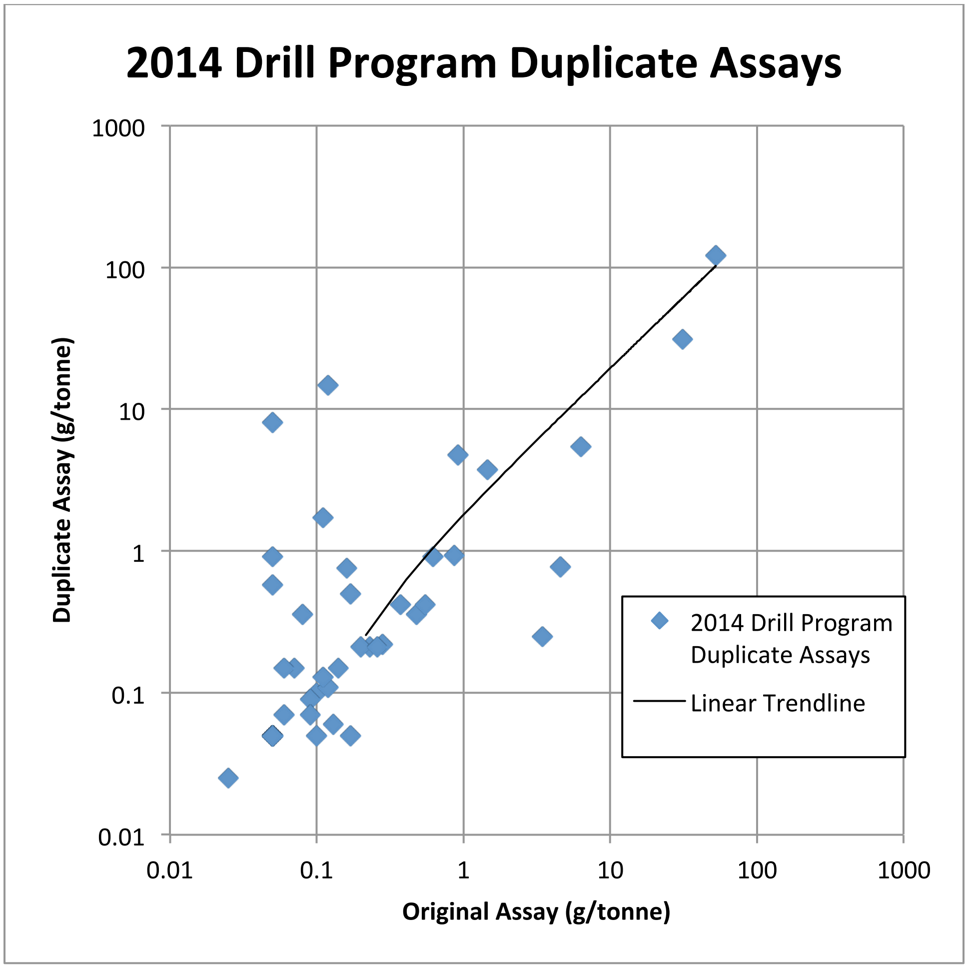


Figure 11-3 - 2014 Drill Program Duplicate Assays (log scale)

11.1.2.2.4 QA/QC Outliers

One sample, J722224, was listed as a blank in the Ressources Appalaches database but assayed at 7.08 ppm. This may have been the result of a standard being mislabelled as a blank. Sample J722225, assayed at below detection limit, was listed as a CDN-GS-7F standard in the Ressources Appalaches database. CDN-GS-7F standards have a grade in the range of 6.90 ± 0.41 g/tonne. It seems likely that these two samples, J722224 and J722225, were either mis-tagged or were switched in the lab. If this is true, it may indicate a problem with sample handling.[9] Because this problem was apparently missed, it may also indicate a problem with RA's QA/QC procedures.

[9] J722224 and J722225 are in assay certificate SD14117403.csv; J721584 is in SD14110174.csv . They are also in the Ressources Appalaches database, file GD_Local_Comp_RA_2014.accdb, in the table Drillholes_QC_ASSAYS

Another sample, J721584, is shown with a value of 0.04 g/tonne but is listed in the database as being a CDN-GS-2M standard, which has an expected average of 2.210 g/tonne and a 2-standard-deviation range of 0.244 g/tonne.

11.1.2.2.5 Security and Tracking

Each sample was assigned a numbered identification ticket, which was put in an individual bag together with the rock sample. A witness sample, identified by a ticket bearing the same number, was left as evidence in the core box. Each bag, after being closed and tied up, was then put in a larger bag. This larger bag, containing about twenty (20) samples, was then delivered by the Ressources Appalaches staff to a courier (either Day & Ross or Purolator) in Sheet Harbour. The courier then shipped the samples to the ALS Chemex's assay laboratory in Val-d'Or, Quebec.

11.1.2.3 Laboratory Sampling Method and Approach

Sample preparation was done at the ALS Minerals laboratory in Sudbury, Ontario. Assaying was done at the ALS Minerals laboratory in Val d'Or, Quebec. Both labs are owned by ALS Minerals, 2103 Dollarton Hwy, North Vancouver, BC, V7H 0A7, (604) 984 0221 (www.alsglobal.com).

The ALS labs are independent of the issuer. Both labs are accredited to the ISO 17025 standard by the Standards Council of Canada for mineral assaying, including Au-AA and Au-GRA.

11.1.2.3.1 Sample Preparation

All samples from the 2014 drill program were assayed by screened metallics. The method used was as follows. Before analysis, samples were logged and weighed. Then, samples are prepared using the following steps:

- Drying;
- Crushing to 70% passing <2mm;
- Splitting using a riffle splitter; and then
- Pulverizing to 95% < 75 microns by grinding in a ring mill pulveriser using a carbon steel ring set.

11.1.2.3.2 Assaying

Samples were assayed using ALS's Au-SCR24 method. Analysis begins by passing the material through a 100 micron (150 mesh) screen. The plus fraction (material larger than 100 microns) is analysed in its entirety by fire assay with gravimetric finish and reported as the Au (+) fraction result. The minus fraction (the portion that passed through the screen, that is, the portion that was smaller than 100 microns) is homogenized, and two 50 gram subsamples are analysed by fire assay with AAS finish, with the average of the results from those two subsamples used as the Au (-) fraction result. All three values are used in calculating the overall average result for the sample.

Table 11-3 - Gravimetric Finish (Au-SRC24) Limits

Determination Reported	Description	Units	Lower Limit	Upper Limit
Au Total (+)(-) Combined	Total gold content of sample as determined by metallics calculation above.	ppm	0.05	1000
Au (+) Fraction	Gold content of plus fraction determined by Au-GRA22.	ppm	0.05	100,000
Au (-) Fraction	Gold content of minus fraction. Reported as average of two sub-samples.	ppm	0.01	1000
Au-AA26	Gold content of first minus fraction sub-sample.	ppm	0.01	1000
Au-AA26D	Gold content of second minus fraction sub-sample.	ppm	0.01	1000
Au (+) mg	Weight of gold in plus fraction.	mg	0.001	1000
WT. (+) Fraction Entire	Weight of plus fraction.	g	0.01	1000
WT. (-) Fraction Entire	Weight of minus fraction.	g	0.1	100,000

11.1.2.3.3 Laboratory Quality Control

Samples were assayed at ALS Chemex assay laboratory, Val-d'Or, Quebec. ALS Chemex has a Quality Management System (QMS) to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

The Val-d'Or facility operates under ALS Laboratory Group's global Quality Management System and is in compliance with ISO 9001:2000 for the provision of assay and geochemical services according to QMI Management Systems Registration.

The Quality Assurance program at ALS Chemex is a multi-level program that involves quality control procedures for sample preparation and analysis, plus a quality assessment stage that includes data review and statistical analysis.

The routine quality control testing includes the analysis of blanks, reference materials and duplicates. All results from quality control samples become part of a separate database that is used for Quality Assessment. All reference materials are either primary, certified references or in-house reference materials that have undergone a rigorous validation process.

Clients can retrieve their data electronically in a secure fashion over the internet. The internal security system maintains a record of any activity in a client work order file, including the act of viewing a file, and records the name of the user and the time, date and nature of the activity. In this way they can check whether any unauthorised activities have taken place in a client file.

11.2 West Dufferin

11.2.1 Previous Programs

11.2.1.1 On Site Sampling Preparation and Quality Control

No written methods or procedures were available for on-site sampling preparation and quality control for the Cominco drilling between 1935 and 1942.

Similarly, no records exist of onsite sampling methods for the Jascan drill programs during 1987 – 1988, although the most comprehensive report on the work, ACA Howe (1988) contains complete assay certificates and drill logs. Jascan was a reputable company, and it is reasonable to expect that its exploration was done under industry standard exploration practices at the time, (which would have included sawing or splitting the drill core and sending sawn/split core to the laboratory for all sample preparation), particularly because ACA Howe, a very respectable mineral engineering firm, produced the assessment reports for the drilling.

11.2.1.2 Laboratory Sampling Method and Approach

The laboratory for the Cominco drilling is unknown.

The 1987 – 1988 Jascan drill programs used Chemlab Inc. of St. Johns, New Brunswick for all assays. No certifications for Chemlab are known, although it was independent of Jascan. Almost all of the Jascan drill assays were run using screen-fire methods. This is a more costly but more accurate method for fire assaying gold deposits with relatively coarse gold particles, as it tests the gold in various size fractions of the sample and then reports an overall average for the sample. This increases confidence in the reliability of the Jascan drill results, which is borne out by the nearby 2010 Nycon drilling (see below).

11.2.2 2010 Nycon Drilling Program

11.2.2.1 On-Site Sample Preparation and Quality Control

Duncan (2011) reported that NQ-sized core drilled during the 2010 drill program was sawed onsite using a diamond-blade saw, with one-half core sent for analysis and one-half remaining in the core box. Core is currently stored at Eastern Shore Cartage in Watt Section, Nova Scotia, near Sheet Harbor.

Data gained by analyzing samples of drill sludge at the drill rig were disregarded. Only analyses of sawn drill core are included in the West Dufferin drill database.

11.2.2.2 Quality Control

11.2.2.2.1 Blanks, Standards, and Duplicates

Included in the 54 samples of sawn drill core submitted from the 2010 drilling were two blind blanks, one blind standard, and no duplicates. The first blank (sample 22173) returned a value of 5 ppb Au. The second blank (sample 22208) returned a value of 13 ppb Au. Although above detection limit, this does not raise any particular concerns, as the blank used was anhydrite from a different drill program and thus a natural material that may have had some small amount of gold within it, rather than a purchased certified blank material.

The single standard included (sample 22197) was CDN-GS-2C, a certified standard reference material purchased from CDN Resource Laboratories, of Vancouver, British Columbia, a well-known and reputable provider of standard reference materials to the mining and minerals industry. Standard CDN-GS-2C is a gold-ore reference standard with a recommended gold concentration of 2.06 g/tonne Au and "between laboratory" standard deviation of ±0.075 g/tonne Au. The assay returned for this sample was 1,956 ppb Au, well within three standard deviations of the standard's accepted value. No duplicates were submitted with the 2010 sawn drill core samples.

11.2.2.2.2 Security and Tracking

Duncan (2011) reported that drill "samples were placed in sealed plastic bags inside 20-liter plastic pails and shipped to Eastern Analytical…via Day and Ross." As D.R. Duncan & Associates was an independent contractor to the property owner Nycon Resources, we have no particular concerns with assay security and tracking.

11.2.2.3 Laboratory Sampling Method and Approach

All samples from the 2010 drilling program were sent to Eastern Analytical Inc., in Springdale, Newfoundland. Eastern Analytical is an independent laboratory certified to ISO 17025 (competence of testing and calibration laboratories), and accredited by the Canadian Association for Laboratory Accreditation (CALA), which reports that "Accreditation is the formal recognition by CALA of the competence of an environmental analytical laboratory to carry out specified tests, conforming with the requirements of ISO/IEC 17025. This formal recognition is based on an on-going evaluation of laboratory capability and performance," (CALA, 2016).

11.2.2.3.1 Sample Preparation

Laboratory sample preparation at Eastern Analytical for the Total Pulp Metallic Sieve Procedure included crushing the entire samples to 80% -10 mesh, pulverizing the entire sample to 95% -150 mesh, and separating the entire sample into -150 mesh and +150 mesh fractions (Duncan, 2011).

11.2.2.3.2 Assaying

Assays for the 2010 drill program were done by Eastern Analytical's Total Pulp Metallic Sieve Procedure. This entails 1) assaying the entire +150 mesh fraction as one sample and the weight recorded; 2) weighing the -150 mesh fraction and performing a one-assay-ton (30-gram) fire assay on this fraction; 3) calculating a weighted average of the two analyses. Fire assays were performed by standard methods, with Au determination by atomic absorption (Duncan, 2011).

11.2.2.3.3 Laboratory Quality Control

No internal laboratory quality control data have been provided by Eastern Analytical related to the sawn drill-core samples. However, the lab is ISO 17025 certified, and as such is expected to have a rigorous internal quality control regimen. The lab did report results of two blanks, one for each batch of drill-sludge samples submitted, which they reported a both as 5 ppb Au, and results of two standards, although the standards are not identified. We are satisfied that Eastern Analytical's internal quality control were appropriate and sufficient.

12 Data Verification

12.1 East Dufferin

Doug Roy, M.A.Sc., P.Eng. (one of the report's authors) first visited the property in 2003 when Azure Resources was operating. Mr. Roy first visited the property under Ressources Appalaches' tenure on April 8th, 2009. This visit was made to observe the results of the work that Ressources Appalaches carried out during 2008, examine the core and take quarter-core verification samples.

During the 2009 site visit, Mr. Roy examined the core and selected intervals to re-sample. The re-sampling was carried out by a Géominex geologist under Mr. Roy's continuous supervision. Mr. Roy took custody of the samples and was in continuous possession of them until delivery to the Minerals Engineering Centre at Dalhousie University, Halifax, Nova Scotia. There, the samples were subjected to fire assay (30 gram sample) with an atomic absorption spectroscopy finish.

The purpose verification sampling was to confirm the presence of gold on the property in a similar concentration to that reported. Because of the coarse nature of the gold, it was not expected that the assays from verification sampling ("verification assays") would equal the original sample assays with any degree of precision.

Results are reported in Table 12-1. The author expected that, even with the small number of samples (eight in total) and the small sample size (quarter-core), the precision (compared with the original sample assay) would be poor but that the number of verification samples that were higher than the originals would be roughly the same as the number that were lower. The result was that six verification assays were lower and two were higher than the originals.

The authors feel that the results from this verification sampling were acceptable. In other words, gold is indeed present on the property in potentially economic concentrations. However, the results also illustrate the erratic effects that coarse gold can have on sampling and assaying. Coarse gold requires large sample sizes for representative sampling. Samples from drill core, even larger diameter core are too small to be representative of grade at a particular point. Having a high sample value at a particular point is a good indication that there is gold with an economic concentration. However, the indication is by no means absolute.

Table 12-1 – Data verification sample results, 2009.

Hole	From (m)	To (m)	Interval (m)	Original Sample Number	Verification Sample Number	Original Assay (g/tonne)	Verification Sample Assay (g/tonne)	Percent Difference
08-07	163.5	164.4	0.9	904,714	900351	7.68	0.40	-95%
08-07	125.0	126.0	1.0	904,679	900352	125.50	33.71	-73%
08-16	28.2	29.4	1.2	903,090	900353	10.50	20.75	98%
08-11	26.0	27.0	1.0	904,755	900354	70.30	0.28	-100%
08-11	85.9	86.4	0.5	904,810	900355	18.90	5.84	-69%
08-17	211.8	212.2	0.4	903,219	900356	22.10	12.23	-45%
08-17	212.2	212.6	0.4	903,220	900357	0.13	0.21	64%
08-17	212.6	213.8	1.2	903,221	900358	7.71	1.50	-81%

On December 8, 2014, a MineTech mining engineer, Lucas Dickie E.I.T., under the direction of Mr. Hannon and Mr. Roy, collected 8 check samples from the veins exposed underground, 4 samples from the stockpiles and 8 samples from the 2014 diamond drill core stored at the site. Results from the MineTech check samples are presented in Table 12-2. The verification samples show a variance from the originals similar to the 2009 check sampling, verifying again the presence of gold on the property in potentially economic concentrations. The authors feel that the results from this verification sampling were acceptable.

Table 12-2 – Data verification sample results, 2014.

Samp #	Area	Location Description	Vein Width (cm)	MineTech SampleID	RA Sample ID	RA Grade	RA Certif ID	MineTech Assay Results	MineTech Reassay Results
1	Level 1035W	NL Side of Face (Low)	30	J725110				3.96	
2	Level 1035W	S Side of Face (Mid)	70	J725111				0.23	
3	Level 1028 NL	Ring 416(LH)	30	J725112				0.11	
4	Level 1028 NL	Ring 410(LH)	30	J725113				1.55	
5	Level 955	NL Side of Face (High)	100	J725114				0.80	
6	Level 955	SL Side of Face (High)	60	J725115				1.46	
7	Level 944	NL Side of Face (Low)	40	J725116				8.03	
8	Level 944	NL Side of Face (Mid)	60	J725117				2.91	
9	Crushed Reject Pile	West Side of Pile		J725118				1.11	
10	Crushed Reject Pile	North Side of Pile		J725119				0.06	0.05
11	Crushed Reject Pile	East Side of Pile		J725120				0.20	
12	Crushed Reject Pile	South Side of Pile		J725121				0.10	
13	F14-04 (67.7 to 68.50)			J725122	J721068	4.76	SD14077133	1.53	
14	F14-10 (76.8 gto 77.20)			J725123	J721203	4.37	SD14087925	0.62	
15	F14-14 (158.33 to 158.79)			J725124	J721463	6.55	SD14104020	3.10	
16	F14-15 (157.40 to 157.90)			J725125	J721420	3.49	SD14098611	1.80	
17	F14-17 (55.95 to 56.30)			J725126	J721542	5.61	SD14110174	19.52	
18	F14-18 (215.40 to 216.00)			J725127	J721628	8.05	SD14110174	0.08	
19	F14-18 (213.80 to 214.60)			J725128	J721625	2.85	SD14110174	2.24	
20	F14-22 (199.70 to 200.45)			J725129	J722205	6.45	SD14117403	13.16	7.50

Since the visit in 2009, Mr. Roy has visited the property on numerous occasions, the most recent being December 14, 2016. He has toured the mineral processing facility, examined the portal and underground workings, the tailings facility, and he has examined the diamond drill hole locations in the field.

Patrick Hannon, an author of this report, visited the property several times over the last two years, the most recent visit being the December 14, 2016. He has inspected the underground workings and diamond drill core, tailings facility, and processing plant.

Minor errors were found in the 2009 and 2010 drill hole collar database. These errors were reported to Ressources Appalaches and were corrected.

Downhole surveys for several holes (F09-5, 11, 14, 15, and 22 as well as F10-1, 2, 3, 6) show a greater than expected deviation. Those holes should be re-surveyed at the next opportunity.

Mr. Hannon checked a random sample of the 2009, 2010, 2011 and 2014 assay database values against the assay certificates provided by the assay laboratory, and found that the database values were an acceptable match for the values reported by the assay laboratory.

A check of the assayed value of blanks, standard, and duplicates was carried out. The results are discussed in Section 11.1.2.2.

Data verification results are considered adequate for the purposes of this technical report.

12.2 West Dufferin

Because of the similarity in mineralization, proximity to well-verified East Dufferin, obvious continuity between East and West Dufferin across the Harrigan Cove Fault, and long history of gold production and exploration by previous companies, no data verification samples were taken at West Dufferin.

Resource Capital Gold supplied the drilling database to Mr. Greg Mosher, who reviewed the database. He corrected some minor errors and omissions and performed a final quality check to ensure the integrity of the database.

A check of the assayed value of blanks and standard was carried out, as discussed in Section 11.2.2.2.

Data verification results are considered adequate for the purposes of this technical report.

13 Mineral Processing and Metallurgical Testing

Mineral processing and metallurgical testing for the East Dufferin portion of the project is discussed below. There has been no modern mineral processing or metallurgical testing work done on the West Dufferin portion of the project; historical production for West Dufferin is discussed above in Section 4, History.

13.1 Previous Mine Production

Pre-Ressources Appalaches Mine production at the East Dufferin Property is summarized in Table 13-1.

Table 13-1 – Reported East Dufferin Mine Production previous to 2014.

Year	Company	Tonnes (mineralised rock)	Gold Recovered (troy oz.)	Notes
1994				Site Preparation & Excavation for Portal
1995	Dufferin Resources	3,418	117	Operated until April 28, 1995
2000	Newfoundland Goldbar	4,000		mineralised rock stockpiled, not milled.
2001	Newfoundland Goldbar	51,172	7,287.7	55,172 tonnes were reported milled, likely including the 2000 mine production.
2004	Azure Resources	23,144	1,648.5	Operated until November, 2004
2006	Jemma Resources		1,601.9	Tailings re-processing: 31,745 tonnes
Total		**81,734**	**10,656**	

13.2 Initial Testwork for Mill Design (1993)

During December of 1993 a ramp portal was drilled, blasted and partially excavated to expose the upper saddle vein (Duncan and Graves, 1993). Approximately ten tonnes of quartz vein material was excavated for a bulk sample in the vicinity of drill holes PD-88-18 and PD-93-01. Three thousand to four thousand kilograms of this material was delivered to the Minerals Engineering Centre of the Technical University of Nova Scotia[10], for metallurgical test work.

M-Tech Incorporated of Elmsdale, Nova Scotia received approximately 300 kilograms of crushed material (-1/4 inch material, jaw and cone crushed) from the Technical University of Nova Scotia. The material was further reduced to –60 mesh by use of jaw, cone and roll crushers. A recovery

[10] The Technical University of Nova Scotia has since merged with Dalhousie University

rate of 92% was reported on a split of this material by use of the KMS System and the head grade was determined to be 15.14 grams per tonne gold (Raymond, 1994).

The gravity mill was constructed in 1994 with a crusher, jig and table unit. The mining permit and mill permit were issued in January of 1995. The mine operated for a few months in 1995, shutting down on August 28, 1995 after producing 3,640 grams (117 troy ounces) from 3,418 tonnes of rock, for an average recovered grade of about 1 gram per tonne. Mill recovery was less than 50%. The Dalhousie Mineral Engineering Centre completed some additional test work on gold samples from Dufferin, using gravity and cyanide leaching methods, but results were not available.

13.3 2012 Testwork

Ressources Appalaches carried out cyanidation studies and gravity and flotation mineral processing studies. Both studies were carried out using core samples.

The cyanidation work (St-Jean, 1999), carried out at Laboratoire LTM Inc., achieved overall recoveries in the 98.6% to 99.6% range from a 19 kilogram sample.

Gravity/flotation work was carried out at Met-Solve Laboratories Inc. (Lum, 2012) using 52 kilograms of drill core. Using gravity concentration followed by flotation of the gravity tailings, they were able to recover approximately 90% of the gold in the gravity section for an overall recovery of approximately 99%. The grind size was approximately 120 microns and the Bond ball mill work index was 14.3 kWhr/tonne (i.e.: moderate hardness).

One of Lun's recommendations was that the work be repeated using a fresher, larger sample because as the drill core ages, the surfaces of the sulphides weather, which can affect the flotation recovery. Also, Lun postulated that the nugget effect was quite high in the sample and that a larger sample would be more representative.

The final flowsheet was titled "30453_F00_01-Rev01(PFD_Mass balance)_RevA-2 (07Aug_2012).pdf". A comparison was made between Met-Solve's results and the final flowsheet. The major differences were:

Table 13-2 - Comparison of Met-Solve Results & Final Flowsheet

Item	Testwork	Final Flowsheet	Comment
Number of Falcon Stages	2	1	Actual gravity-recovered proportion of gold recovery would be lower.
Process Water	Presumably Fresh	Mostly Reclaimed	In the past, using entirely fresh water has resulted in higher recoveries.
Gravity Section Recovery	+92%	72%*	Final flowsheet value is more reasonable, based on past production at Dufferin and other NS gold deposits.
Flotation Section Recovery		24%*	
Overall Recovery	99%	96.3%*	Final flowsheet value is more reasonable, based on past production at Dufferin and other NS gold deposits.

* From the flowsheet:

- Feed is 100 g/hr
- Table Produces 72 g/hr, 72% recovered by gravity
- Flotation Produces 24.3 g/hr, 24 % recovered by flotation
- Overall recovery 96.3% (calculated)

There was a small discrepancy in the overall gold recovery values that were listed on final flowsheet. In the bottom right corner of the sheet, the overall recovery is listed as 95%. However, adding the gravity-recovered and flotation-recovered gold yields an overall recovery of 96.3%. For this report, an overall recovery of 95% was used.

Generally, the gravity-recovered proportion of gold, as well as the overall gold recovery, were lower in the final flowsheet. The author (Mr. Roy) agrees with this because actual mill recovery is typically lower than bench-scale, laboratory recovery.

13.4 Mill Modifications (2000-2001)

In September of 2000 the underground was dewatered and the workings were rehabilitated. Mine development began in October 2000.

The milling plant was changed from a jig - table unit to a high centrifugal apparatus and a bullion furnace was added. A “shatterbreak” impact crusher was added and two Falcon concentrators, with a maximum centrifugal field of 300G, were added (only one was used). Screens, slurry pumps and an air compressor were also installed.

In 2001, a bulk sample was tested in three test lots. Test # 1 was an initial mill test of 10,000 tonnes with gravity separation and resulted in “poor recovery”. Test # 2, also of 10,000 tonnes used a form of impact grinding[11], but also had poor recovery. Test # 3 added a ball mill for grinding, improving recovery to 77.6%.

Production results for the year 2001 show a total of 55,172 tonnes (60,691 tons) milled and 7,397 ounces of gold poured, with an "estimated" 200 ounces in the circuits and 2,191 ounces in the tailings (based on a tailings grade of 1.7 g/tonne). This indicates a calculated head grade of 5.5 g/tonne. Estimated recovery was 77.6% (EnviroGold Memo, 2002).

13.5 Grinding, Flotation and Settling Work for Reprocessing Tailings (2003)

During 2003, test work into the reprocessing of tailings was completed (Hannon, 2003). Tailings samples had an average *in situ* grade of approximately 4 g/tonne gold with a sample range from 1.5 g/tonne gold to 33 g/tonne gold.

The work was completed by Ed Thornton P.Eng. of MineTech International Limited at Dalhousie University's Mineral Engineering Centre.

Twenty two samples were taken from the tailings pond at various depths during August of 2003. Each sample was split into two equal portions with a sample splitter. One portion was forwarded to Knelson Gravity Solutions in Langley B.C. via Day and Ross Transportation.

From the other portion of the 22 samples, a head portion of each sample was taken and assayed for gold. The remaining portion of the head sample was split into sixteen, 2,000 gram samples. They were weighed for subsequent flotation tests. A head sample was taken. All of this work was completed on the tailings material remaining at the site from the previous operation.

13.5.1 Grinding

This mineralised rock was found to be very hard. A ten minute grind produced a product of 40 % -200 mesh another 30 minutes of grinding only increased the fineness of grind to 55 % -200 mesh. Therefore a work index that represents 100% quartz was recommended to be used in calculating the required horsepower.

A grind of 50% to 55% -200 mesh is required to prevent sanding out in the flotation cells. Cyclones in the grinding circuit will cause the gold and arsenopyrite to be preferentially ground finer.

The finer grinds improved the metallurgy but they also created much more surface area. This means more collector is required to recover the gold. In one test, 2 ½ times the amount of collector that was used in the rougher float was required for cleaner flotation after a regrinding of 15 minutes.

[11] The crushed "ore" is fed to a chamber that has a screen to remove fines – the crushed material grinds itself by impact as new material is constantly added. For this method to work, the material must be free flowing.

Finer grinds prior to the rougher flotation and cleaner floatation do little to overcome the arsenic problem.

13.5.2 Flotation Work

The recommended flotation reagents were Alloid Colloids CA838A and methyl isobutyl carbinol (MIBC) (Hannon, 2003). The dosage values were 10 and 5 grams per tonne of feed, respectively. Based on a water throughput of 15 m^3/hr, 6 and 3 grams of reagent per cubic metre of water (6 and 3 ppm), respectively, are required. Copper sulphate was examined during flotation test work; however, it was determined that is was not required.

A settling test on the Dufferin tailings was carried out on a 300 gram sample after a bottle roll cyanidation test (Hannon, 2003). The test was carried out in a 2 litre cylinder at 15% solids. The cylinder had a diameter of 76.2 mm and a height of 441.5 mm.

The sample was flocculated with Ciba (Allied Colloids) Percol E10. A 0.1%, solution was used at a rate of 15 grams per metric tonne.

The settling rate was measured from the top of the water (clear zone), and took about 17 minutes.

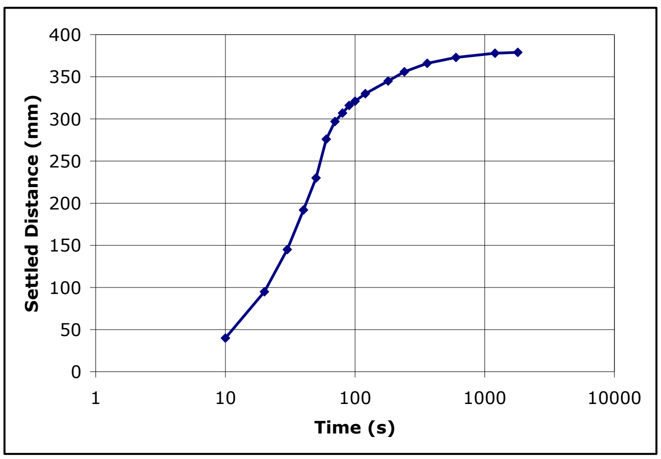


Figure 13-1 - Settling Rate

13.5.3 Arsenic

From the start of the work, it was difficult to comprehend why arsenides responded to the floatation process. Normally the addition of copper sulphate and a strong collector such as a xanthate are required for arsenopyrite to float. The collectors that were used in these tests are very selective and should not have caused the arsenic to float to this extent. The flotation of the arsenopyrite may have been caused by the close mineralogical association of gold and arsenopyrite. This must be confirmed by microscopic studies on polished sections of flotation concentrate.

The assay of sulphur in the feed shows that almost all of the sulphur occurs as arsenopyrite. It seemed that Z-200 [sodium ethyl thionocarbamate] was more effective for arsenic flotation than R208, and because of this, its use should not be ruled out. During the test work it was thought that some galena was floating and for this reason Z200 was tried, as it is known that Z200 is not a good collector for lead; however, barely any galena existed in the mineralised rock.

13.5.4 Forecasted Metallurgy for Tailings Reprocessing

With the proper equipment, it was predicted that the following metallurgy could be obtained:

- Gold Rougher flotation only 94 % recovery of gold, with a grade of 300 g/tonne.
- Gold cleaner flotation 91. 5 % recovery with a grade of 700 g/tonne.

13.6 Gravity Work, Knelson Concentrators – Tailings (2003)

The test work on the tailings material produced predictable results, with finer grinding liberating more gold for both gravity and flotation recovery. The gravity recoverable gold ("GRG") of the tailings composite was surprisingly high considering the material had already been processed in a gravity circuit prior to disposal in the tailings impoundment. Recovery in excess of 60% was predicted on this material at grind of 80% passing 75 microns (P80).

Twenty-two tailings samples were used in gravity concentration test work at Knelson's testing centre in Langley.

The primary objective of this test work was to determine if the gold contained within the samples is readily recoverable through gravity concentration. A description and a flow chart of the procedure used for the test are shown below.

The samples were individually mixed, and split to form a 3.7 kg composite for processing. The remaining samples were re-bagged and stored.

1. The test sample was processed through a 3" Laboratory Knelson Concentrator (KC) at an RPM set to produce the equivalent of 60 G-force.
2. During the test, sub-samples of the tailings stream were collected for assays.
3. At the end of the concentration stage, the concentrate was washed from the inner cone of the KC.
4. The KC concentrate was panned to produce a panned concentrate and panned tails (middlings) sample.
5. The concentrate and tailings samples were labelled, dried, weighed and sent to an independent local lab for assaying.

13.6.1 Results

Table 13-3 – Gravity Work, Knelson Concentrators.

	Mass		**Assay**	**Units**	**Dist'n**
Product	**(g)**	**(%)**	**Au (g/tonne)**	**Au**	**(%)**
Pan Concentrate	15.0	0.40	330	132.0	**50.7**
Pan Tails	67.7	1.80	8.36	15.1	**5.8**
Sample Tails	3668	97.8	1.16	113.4	**43.5**
Totals (Head)	3751	100.0	**2.61**	260.5	**100.0**
Knelson Conc.	82.7	2.20	66.7		**56.5**

The recovery for the single pass test was 56.5% in a mass pull to concentrate of 2.2%. The calculated feed grade of the sample was 2.61 g/tonne gold, and is considered to be high for a tailings sample. The concentrate was readily upgradeable indicating the recovered gold was relatively liberated. Visual observation of the panned concentrate indicated an abundance of sulfides and the possibility of a gold flake. No microscopic analysis of the concentrate was performed.

13.7 Gravity Work, Knelson Concentrators – Coarse Rock (2003)

A 100 kg coarse mineralised rock sample was submitted for gravity concentration test work. The GRG testing scheme was based on the fact that progressive size reduction allowed for the determination of the precious metal recovery as liberated without over-grinding and smearing coarse precious metal particles. The GRG value is used as a basis for estimating actual gold recovery via mathematical modelling.

The primary objective of this test work was to determine the gravity recoverable gold (GRG) content and the distribution of the GRG by particle size distribution.

The general procedure for the test program is provided below.

i. An initial 10 kg sample was used to determine grind times required to produce size distributions for the test procedure.
ii. The 3" Laboratory Knelson Concentrator (KC) operating parameters were set as follows; fluidization water flow rate of ~3.5 lpm, cone RPM corresponding to a force of 60 g's.
iii. An ~20 kg of composite sample was slowly processed through the KC.
iv. During the test, sub-samples of the tailings stream were collected for assaying and screen analysis.
v. After processing, the KC-MD3 cone concentrate was removed.
vi. The concentrate and tailings samples were dried, screened, weighed and stored for subsequent assaying.
vii. The bulk tailings were ground in a rod mill and steps ii) to vi) were repeated.
viii. The tailings were then further re-ground and steps ii) to vi) were repeated.

ix. All samples were sent for assay and the final bulk tails were stored.

13.7.1 Results

An overall recovery value and recovery by size were presented in Tables 15-2 and 15-3, respectively for each stage.

Table 13-4 - KRTS 20046 Overall GRG Results.

Grind Size	Product	Mass		Assay	Dist'n
(microns)		(g)	(%)	Au (g/tonne)	(%)
P80 = 722	Stage 1 – Conc.	109.4	0.53	1099	**35.5**
	Sampled Tails	464.0	2.24	14.65	**2.0**
P80 = 187	Stage 2 – Conc.	128.6	0.62	466	**17.7**
53.5% -75mm	Sampled Tails	316.2	1.53	11.18	**1.0**
P80 = 75	Stage 3 – Conc.	119.5	0.58	356	**12.6**
	Final Tails	19581.2	94.51	5.40	**31.2**
	Totals (Head)	20719	100.0	**16.35**	**100.0**
	Knelson Conc.	357.5	1.73	623.0	**65.8**

The overall GRG value was found to be 65.8% at a final grind of 75 microns (P80) with a head grade of 16.4 g/tonne Au.

First stage recovery at 35.5% indicated that gold is liberated in the crushing stage. Further grinding and recovery in stages 2 and 3 provide additional recoveries of 17.7 and 12.6% of the total gold. No microscopic examination was performed on the concentrates. Gold was visibly observed in the second panned concentrate with flakes as large as 1mm being noted.

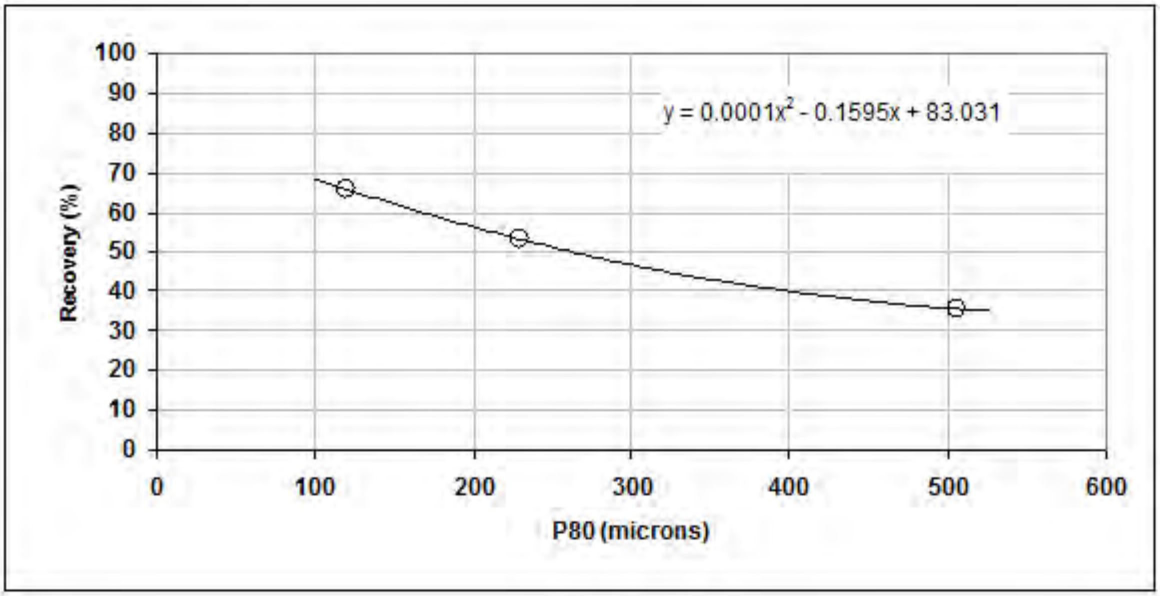


Figure 13-2 - Cumulative recovery as a function of grind size (p80) in microns.

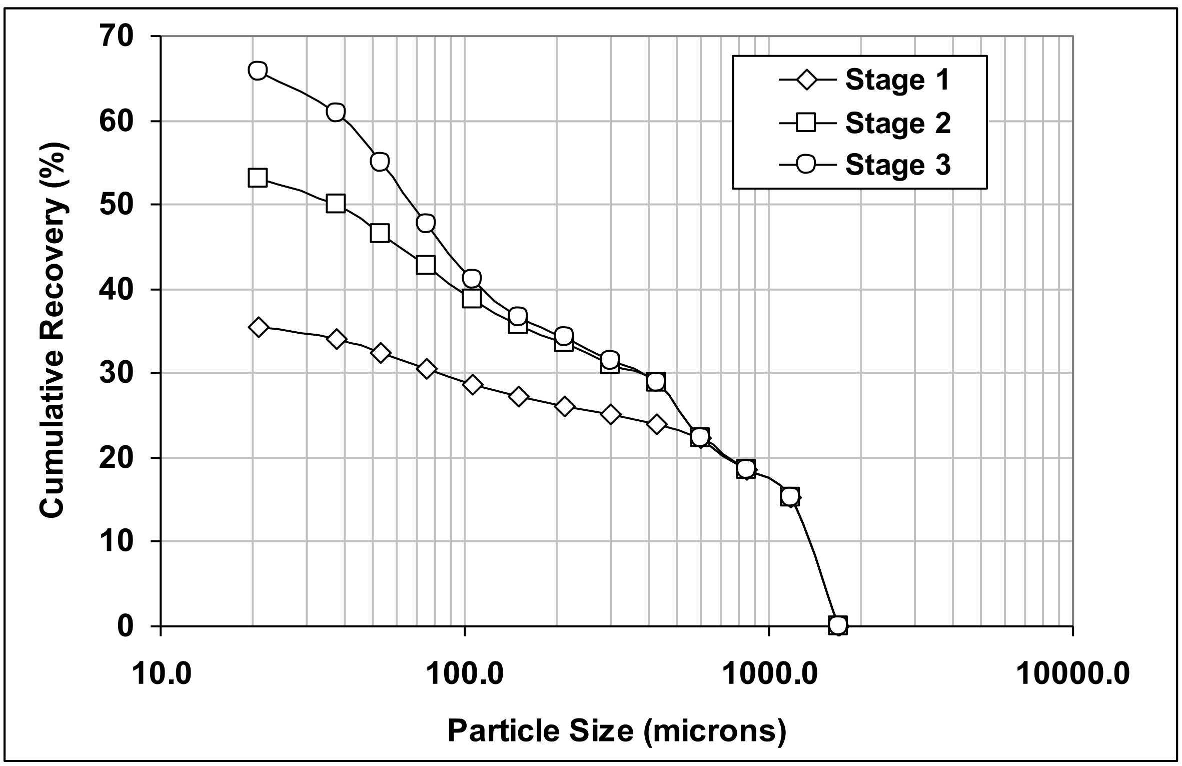


Figure 13-3 - Cumulative Recovery (GRG distribution) as a function particle size in microns.

Table 13-3 - KRTS 20046 Overall Fractional Results.

Size		**Recovery of Au (%)/ Fraction**			**Cumulative Recovery (GRG)**		
US Mesh	**µm**	**Stage 1**	**Stage 2**	**Stage 3**	**Stage 1**	**Stage 2**	**Stage 3**
12	1700	0.0	0	0	**0.0**	**0.0**	**0.0**
16	1180	15.2	0.0	0.00	**15.2**	**15.2**	**15.2**
20	850	3.3	0.0	0.00	**18.5**	**18.5**	**18.5**
30	600	3.9	0.0	0.00	**22.3**	**22.3**	**22.3**
40	425	1.7	4.9	0.09	**24.0**	**28.9**	**28.9**
50	300	1.1	1.0	0.29	**25.1**	**31.0**	**31.4**
70	212	1.1	1.4	0.36	**26.2**	**33.5**	**34.2**
100	150	1.0	1.1	0.42	**27.2**	**35.6**	**36.8**
140	106	1.4	1.7	1.17	**28.6**	**38.7**	**41.1**
200	75	1.9	2.1	2.65	**30.5**	**42.7**	**47.7**
270	53	1.8	2.0	3.36	**32.3**	**46.6**	**54.9**
400	38	1.8	1.5	2.59	**34.2**	**50.0**	**60.9**
-400	-38	1.3	1.9	1.65	**35.5**	**53.2**	**65.8**
Total		**35.5**	**17.7**	**12.56**			

Results from the standard three stages of the GRG test provided an overall GRG value of 65.8**%** at a final grind size of P80 = 75 microns. The recovery in the first stage, on the as-crushed material, was relatively high at 35.5% indicating that a significant portion of the GRG is liberated early in the crushing and grinding process.

The calculated feed grade for the sample tested was 16.4 g/tonne gold.

13.8 Mill Alterations (2006)

Several alterations to the mill were made by Jemma Resources in order to re-process the tailings material from previous operators.

Eight, 300 cubic foot Outokumpu flotation cells were installed for gold rougher flotation recovery in 2006. Twelve Number 15 Denver Sub A flotation cells arranged for three stages of cleaning in a 5 - 4 - 3 arrangement. The balls in the 6 foot diameter by 14 foot long ball mill were changed to 1 inch diameter to better accommodate the incoming fine feed.

A 4 foot diameter disc filter with ancillary vacuum pump and filtrate pump was installed. A backup 4 foot diameter by 6 foot long ball mill was available for regrinding of the rougher concentrates, along with a 1 inch diam. ceramic cyclone. But, this assembly was not used.

The outside and inside fine “ore” bins were modified with steeper walls and vibrators were added to improve the feeding process of the moist tailings. Tailings were excavated and removed, followed by screening to remove rock and debris. A screen and stacker were brought on site for this purpose.

The flotation cells were placed on staging so that all the concentrates could run by gravity to the individual pumps located on the bottom floor. Around all the flotation cells there was a working platform. The final concentrates were placed in large metal containers for drying. The gold concentrates in these containers were then manually shovelled into fibreglass two tonne tote bags for shipment to the Belledune smelter in New Brunswick.

The plant started operating on June 6th, 2006 and ceased operations on December 21st, 2006 for the duration of the winter.

Reagent 208 and sometimes reagent X 523, both supplied by Cytec were used at the rate of 0.3 lb/ton as the gold flotation collector and methyl isobutyl carbinol is used as the frother at the rate of 0.03 lb/ton.

13.9 Screened Metallics (2008)

A selection of 96 samples were re-assayed using screened metallics with gravimetric finish to determine the coarseness of the gold. With coarse gold there is the danger a nugget will bias the sample. Several authors, including Rawthshorne (2004), have noted the coarse nature of Dufferin's gold. Good recovery values using gravity methods (better than 50 %) supports that observation.

Knelson Concentrators carried out test work on fresh rock samples in 2004 as described in Section 13.7. They observed that approximately half the gold was recoverable using gravity methods using a grind size (80% passing size) of 100 micron. In other words, half the gold was gravity-recoverable when the sample was crushed to 100 micron.

For the majority of the samples that were tested in 2008 using screen metallics, more than 99% (by mass) of the gold passed through the Tyler 150 Mesh screen. In other words, more than 99% (by mass) of the gold particles were finer than 100 micron.

13.10 Cyanide Work (2009)

In January 2009, Laboratoire LTM of the Val D'Or area, Quebec ran a series of tests on Ressources Appalaches diamond drill core reject obtained from ALS Chemex Laboratory (Vancouver). The samples were a composite of 19 diamond drill samples that had been previously analysed for gold. The total sample weight was about 7 kilograms.

The seven samples were ground to various amounts passing 200 mesh (75 µ), then leached with a cyanide solution to liberate the gold. Recovery was very good, averaging approximately 99% and with a range of 98.6 to 99.6 grams. The tails generally averaged less than 0.1 g/tonne.

Table 13-4 - Grinding - Recovery Test Work Using Cyanide Leach.

Test #	% passing 200 mesh	Head Grade g/tonne	Tails g/tonne	Recovery %
AD-1	> 100	6.77	0.08	98.8
AD-2	95	6.04	0.05	99.2
AD-3	90	18.74	0.07	99.6
AD-4	85	13.71	0.11	99.2
AD-5	80	12.79	0.14	98.9
AD-6	75	5.6	0.07	98.7
AD-7	70	7.36	0.10	98.6

The conclusions reached by Laboratoire LTM included the observation that the mineralised rock seems to have characteristics which are suitable for a process of direct cyanidation such as the Merrill-Crowe process. The gold tested appeared to be coarse. The consumption of cyanide might be reduced by gravity concentration of the gold after crushing.

13.11 Met-Solve Laboratories Work (2012)

Met-Solve Laboratories of Langley, British Columbia was contracted to perform metallurgical analyses, including gravity concentration tests, bond work index tests, froth flotation tests, and work on the leaching characteristics of the concentrates.

The bond work index tests allowed the grinding capacity of existing on-site equipment to be estimated. Gravity concentration tests established the recoveries to be expected using a centrifugal concentrator. Froth flotation work helps determine optimum conditions (grind, reagents, flotation kinetics) for concentrating the mineralised rock by froth flotation. For future consideration, leaching characteristics concentrates were also established.

The test demonstrated that most of the gold can be recovered by gravity concentration, and produce doré bullion by smelting on site. Tests were ongoing to treat the gravity tailings to check the additional recovery that could be obtained by flotation. The anticipated high gold recovery by gravity and smelting on site meant that only a modest part of the production, from the flotation stage, would be subject to external smelter charges.

Sampling:

- A composite 50-kg sample was submitted to Met-Solve Laboratories in British Columbia.
- The sample was made up of 92 NQ core samples from 13 boreholes, which had been split a second time (quarter core).
- The drill intersections were taken from the central portion of the mineralized zone, which includes the first eight Saddle Reef-type veins, distributed over a length of 700 m and down to a depth of 275 m.
- A portion of the 50-kg sample was analysed by fire assay with gravimetric finish to determine the basic grade at 19.1 g/tonne Au.

- The other quarter cores were retained; all drill cores are stored at the mine site.

Gravity Test Work Description:

- The sample was crushed to about 1.5 mm, passed through a 20 mesh screen and the screen undersize was passed through a Falcon L40 laboratory concentrator. The concentrate from the L40 was then panned to produce a concentrate and tailings, both of which were assayed.
- The L40 tailings and the +20 mesh material were then ground and passed through the L40 once again, with the concentrate being panned and the two products assayed.
- The L40 tailings were then ground a second time to a P80 of 130 microns and passed through the L40 once again, the concentrate being panned and products assayed.
- The average grade of the three concentrates was 491 g/tonne and the recovery to this concentrate was 94.1%.

13.12 Recent Work since the Previous Report

During 2014, Ressources Appalaches' processing plant treated approximately 17,000 tonnes of underground development rock.

14 Mineral Resources

14.1 East Dufferin Mineral Resources

East Dufferin mineral resources were estimated by Doug Roy, M.A.Sc., P.Eng.. Patrick Hannon, M.A.Sc., P.Eng. assisted with geological interpretation.

14.1.1 Software

Micromine 2014 software, Version 15.0.8, was used to facilitate the resource modelling for East Dufferin.

14.1.2 Mine Grid

Previous project owner Ressources Appalaches ("RA") supplied the East Dufferin mine grid transformation parameters. The western end of the mine grid starts at Section 2160E, which is defined in Table 14-1.

Table 14-1 - Section 2160E coordinates.

Point	East (UTM)	North (UTM)
1	574,733.676	4,979,964.235
2	547,745.767	4,979,915.719

Based on those points, the mine grid has a rotation of 13.994° counter-clockwise. The mine grid's zero northing passes through Point 1 in Table 14-1.

For ease of use, the supplied drilling database and wireframes were transformed to site grid coordinates. In other words, the site grid was used for geological interpretation and block modelling.

A datum of "1,000 metres equals Mean Sea Level" was used so that there would be no negative elevations. In other words, the data was shifted up by 1,000 metres.

14.1.3 Exploratory Data Analysis

The most common sample length was 0.5 metres (Figure 14-1). Sample lengths were regularised to 0.5 metres for grade estimation.

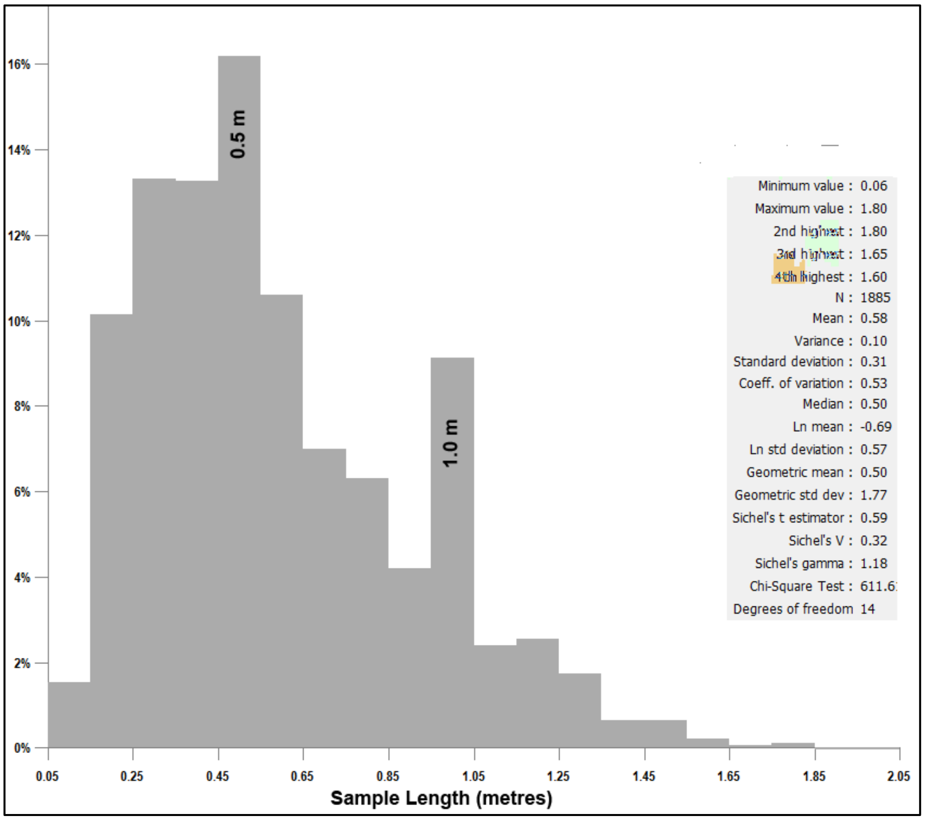
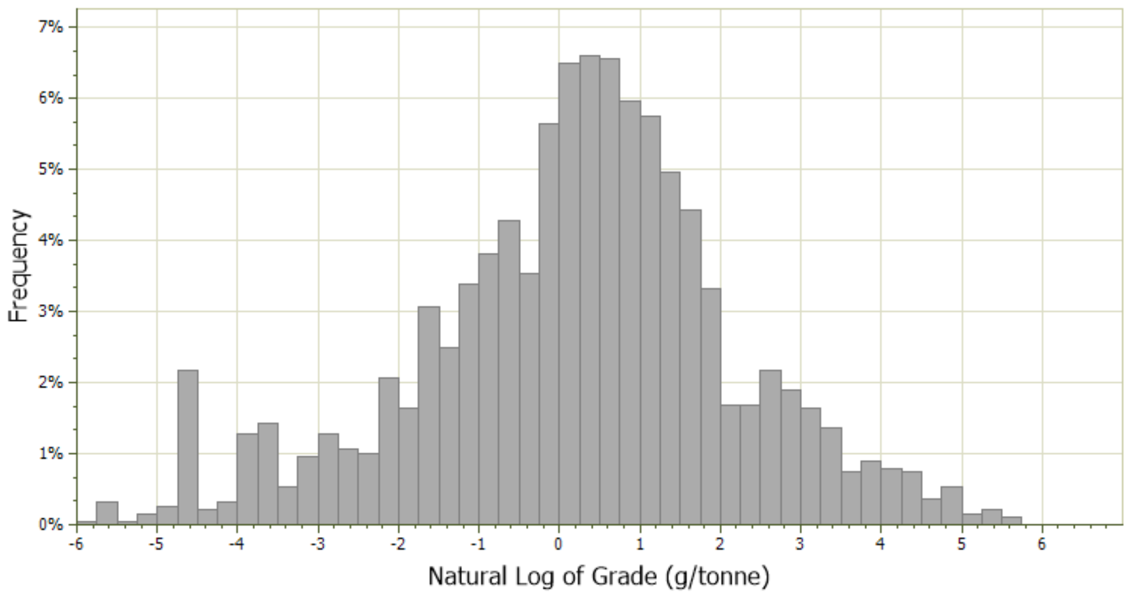


Figure 14-1 - Histogram of sample lengths within the interpreted mineralized zones.

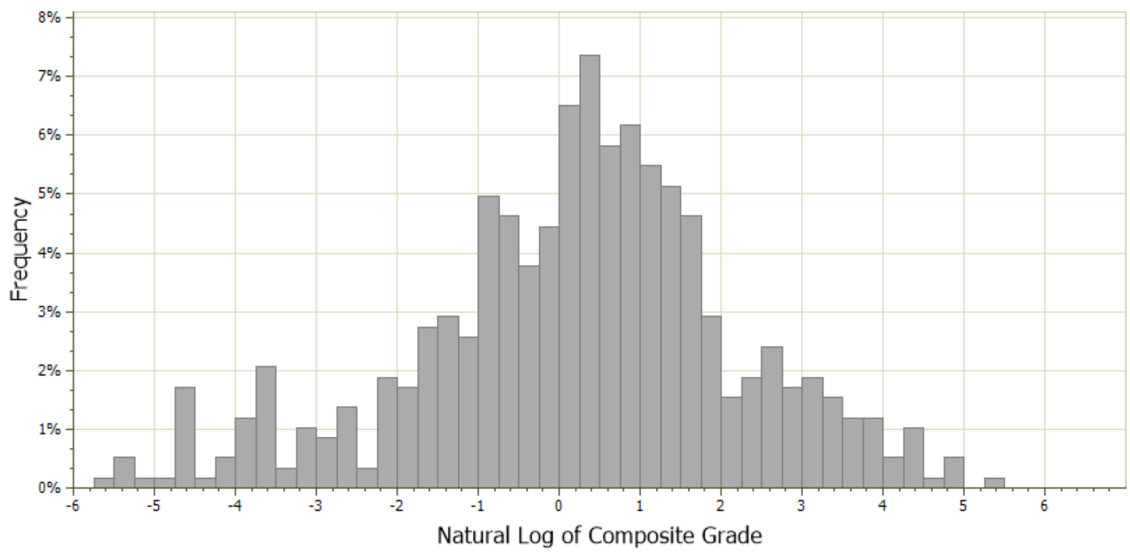
The mean length-regularised sample grade within the outlined mineralized zones was 6.95 g/tonne with a standard deviation of 21.04 g/tonne (refer to Figure 14-2).

The coefficient of variation (CV), or the ratio of the standard deviation to the mean, is a relative measure of dispersion and an indication of the level of difficulty in performing local estimation due to the presence of extreme values (outliers). Dufferin's CV is 3.0 -- CVs greater than two generally indicate that extreme values may cause a problem for local grade estimation. A description of the top-cutting method is presented in Section 14.1.6.



Minimum value	0.00	Variance	442.83	Geometric std dev	7.41
Maximum value	287.00	Standard deviation	21.04	Sichel's estimator	8.99
2nd highest	287.00	Coeff. of variation	3.03	Sichel's V	4.01
3rd highest	220.75	Median	1.40	Sichel's gamma	7.41
4th highest	215.53	Ln mean	0.19	Chi-Square Test	370.19
N	1894	Ln std deviation	2.00	Degrees of freedom	43
Mean	6.95	Geometric mean	1.21		

Figure 14-2 - Histogram of length-regularised gold samples within the interpreted mineralized zones (non-top-cut).



Minimum value	0.00	Variance	315.89	Geometric std dev	7.41
Maximum value	239.17	Standard deviation	17.77	Sichel's estimator	9.13
2nd highest	128.05	Coeff. of variation	2.70	Sichel's V	4.01
3rd highest	125.50	Median	1.45	Sichel's gamma	7.38
4th highest	122.00	Ln mean	0.21	Chi-Square Test	83.12
N	584	Ln std deviation	2.00	Degrees of freedom	36
Mean	6.58	Geometric mean	1.24		

Figure 14-3 - Histogram of composited intercepts (non-top-cut).

14.1.4 Trend Investigation

Trend investigation was carried out for log-transformed, length-regularised samples within the mineralized zones. Through a visual comparison of trend models and sample grade, a linear trend seems a best fit for the data and correlates with what block grade estimation shows: a decrease in grade as one moves east along strike. This may simply be an artefact of decreasing drill density as one moves east and a corresponding decrease in the density of saddle crest intercepts.

The linear trend plot is shown in Figure 14-4. The equation for the linear trend is:

VALUE = 3.628 + -0.00177 X + -0.00123 Y + 0.000272 Z

The loss in grade as one moves eastward is approximately 0.18 g/tonne per one hundred metres. Because the trend is weak, no trend functions were used during the block grade estimation process.

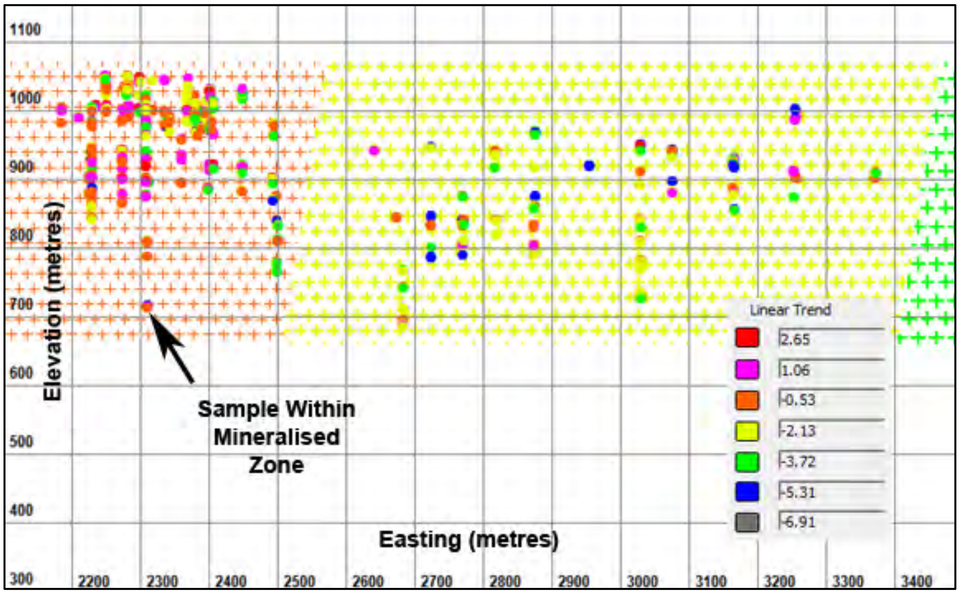


Figure 14-4 - Linear trend plot.

14.1.5 Adequacy and Representativeness of Data

The author believes that the available information and sample density allow a reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the Mineral Resource categories in the CIM Standards.

14.1.5.1 Surveying

A combination of UTM grids and site grids have been used historically. While the author believes that survey control is *adequate* for the purpose of resource estimation, there is definitely room for improvement.

Much of the historical surveying was simply reported in metres, with no indication of map projection or datum. The magnitude of the values suggests that the coordinates are UTM. But, a Modified Transverse Mercator ("NS-MTM") grid has also commonly been used in Nova Scotia. For the same point, UTM and NS-MTM coordinates are close to each other, but not identical.

Proper survey monuments were installed for the first time at the Dufferin site in 2014. The author suggests that these monuments should be used for site survey control in future surveys and that map projections and datums be clearly identified. A selection of important features, such as drill collars and mine openings, should be re-surveyed, using those monuments for control, to confirm the current survey.

14.1.6 Top-Cut

A top-cut analysis was carried out. Based on a cumulative probability analysis of the length-regularised samples within the mineralized zones, an appropriate top-cut grade was estimated to be 200 g/tonne (refer to Figure 14-5).

The highest 11 samples out of a sample database of 2,306 samples[12], or 0.5%, were limited to 200 g/tonne. Top-cutting those samples causes a 3-4% decrease in the average block grade.

This top-cut value may seem high. However, past mining has encountered areas that averaged many ounces of gold per tonne of rock.

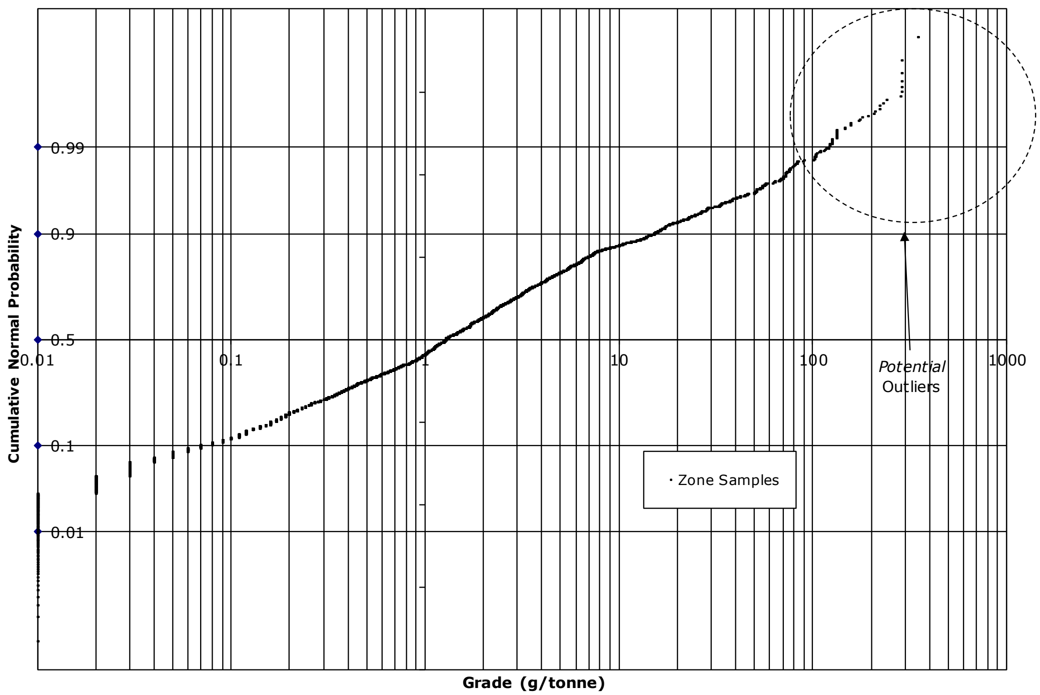


Figure 14-5 - Cumulative probability curve for top-cut analysis.

14.1.7 Specific Gravity (Bulk Density)

A small number of specific gravity (SG) testing has been carried out. The average SG value was 2.65. Therefore an average SG of 2.65 was used for this study.

Because the SG work was limited in extent, the author recommends that further SG testing be carried out. This work should be carried out during the next round of exploration work, and should take approximately 3-5 days to complete.

[12] 0.5-metre, length-regularised samples within interpreted mineralised zones.

14.1.8 Variography

Downhole, omni-directional and directional semi-variograms were constructed using natural-log-transformed, length-regularised samples from within the interpreted mineralized zones.

A variogram "map," created from 0.5-metre, length-regularised samples within the mineralised zones show slightly greater grade continuity in the vertical direction than the along-strike direction. Directional semi-variograms seem to agree. However, when one compares the number of sample pairs, there are overwhelmingly more sample pairs in the vertical direction compared to the along strike direction. Therefore, the author feels that the seemingly greater continuity in the vertical direction is simply an artefact of the greater number of sample pairs in that direction.

Examination of the sample intercepts[13] produced similar results.

Therefore, the author elected to use the omni-directional semi-variogram model for block grade estimation.

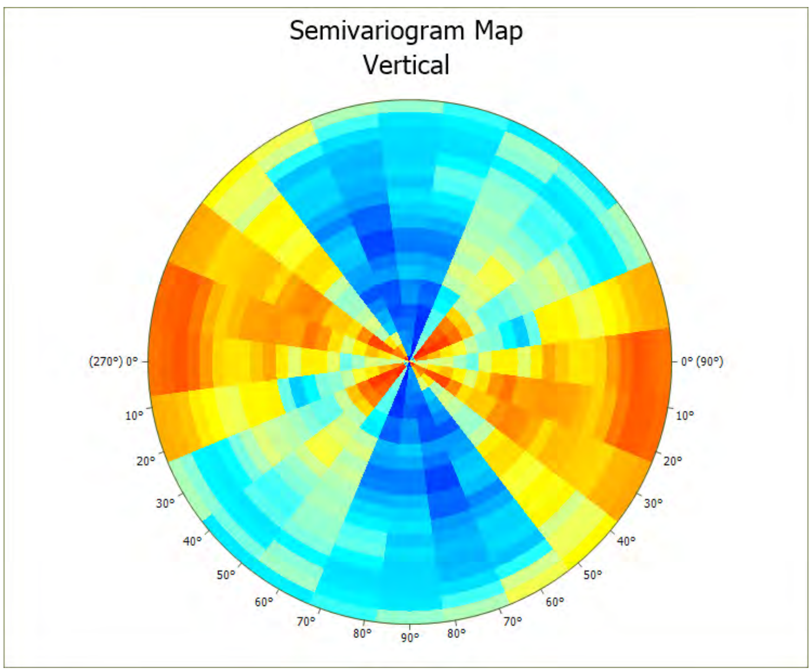


Figure 14-6: Semi-variogram map, vertical section, facing north. Blue represents lower variance; amber: higher.

[13] The entire drill intercept as opposed to length-regularised samples.

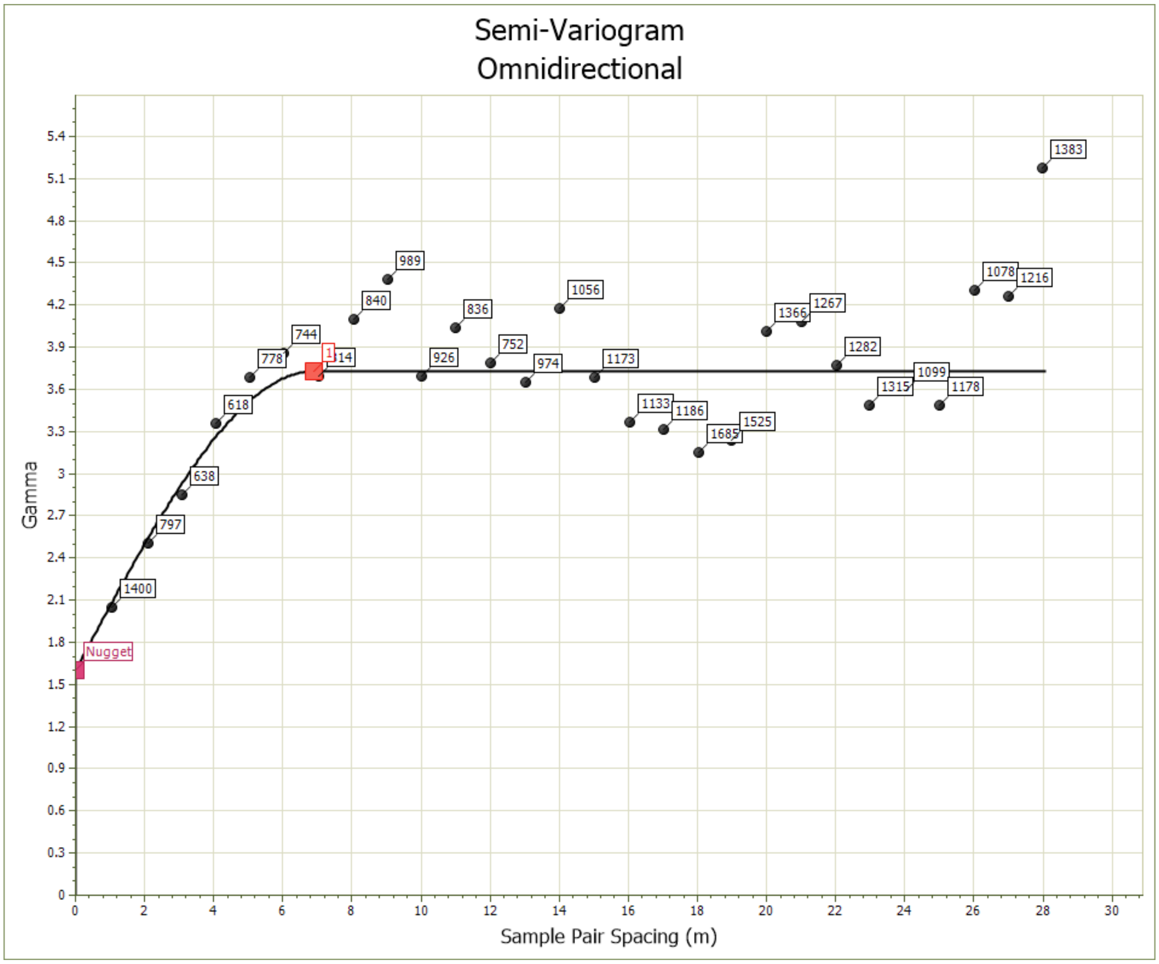


Figure 14-7 - Omni-directional semi-variogram for length-regularised samples within mineralized zones.

Table 14-2 - Semi-variogram parameters.

Parameter	Value
Direction	Omnidirectional
Model Type	Spherical
Nugget	1.6
Range	7 metres
Partial Sill	2.1

14.1.9 Interpretations of Mineralized Zones

Mineralised zones were interpreted on each cross-section using the following guidelines.

The north limb was extended down-dip by 10-15 metres past the last intercept. The south limb, which is thought to pinch out more rapidly than the north limb, was extended 5-10 metres down-dip, past the last intercept.

Zones were extended halfway along strike to the adjacent under-mineralized cross-section or 25 metres, whichever was less.

Intercepts were included in the model if they were “gold positive,” were bedding parallel (where core angle information was available), and had the correct lithology (logged as either vein or argillite).

RA supplied wireframes for their geological interpretation of the mineralized zones. The zones were invariably interpreted to “pinch out” down-dip. The “tips” of the zones did not quite meet minimum mining width criteria. To rectify this, squared-off tips were added at the minimum mining width of 0.5 metres.

The minimum mining thickness was 0.5 metres. This presumes very selective, “low-productivity” mining. For the crest of the fold, this presumes a “resuing” type method whereby the waste is mined separately from the mineralised rock. The term “low-productivity” simply means that the “tonnes-per-person-shift” will be low compared with other, more mechanised mining methods. However, the objective would be to send ‘ounces’ to the mill rather than tonnes. In other words, the objective would be to minimise dilution and maximise mined grade.

For the steeper “limbs” of the deposit, this presumes a very selective, open-stope mining approach such as narrow vein longhole or thermal fragmentation.

There have been several different, and not very well documented survey grids that have been used over the years. Reportedly, Ressources Appalaches installed the first permanent surface survey stations in 2014. Also, downhole surveys have not been well documented and, in some cases, suspect. Many holes were not surveyed down-the-hole. As depth increases, surface and down-hole survey errors multiply, sometimes causing confusion during mineralised zone interpretation. Where reasonable, mineralised zone outlines followed the hanging-wall and footwall contacts of the interpreted intercepts. Where the author was confident in a zone’s continuity but the interpreted intercepts did not quite “line up” as well as they could have, a “best fit” path for the mineralised zone through the composited intercepts was chosen.

All saddles were modelled, regardless of whether portions had been mined-out, so that their assays would be included during the grade estimation process. Mined-out blocks were removed post-grade-estimation (refer to Section 14.1.9.5).

14.1.9.1 Faulting

A number of faults are present that, in places, offset the veins horizontally and vertically, sometimes by as much as 15-25 metres. Three main faults have been identified within the existing underground workings. These were considered during modelling. Additional faulting probably exists to the east of the current workings; however, the current drill spacing and fence orientation precludes detailed fault modelling at this stage.

14.1.9.2 Wireframing

Typically, wireframes are constructed by joining cross-section outlines section-to-section to create a three-dimensional 'solid.' Because of the numerous fault offsets that were observed while mining Saddle 1 in 2014, wireframe solids were constructed by extruding the outlines east-west to the cross-section extents, at a plunge of seven degrees east – the average plunge of the underground drifts that follow Saddles 1 and 2.

The author believes that this accurately represents the volumes of the mineralized zones.

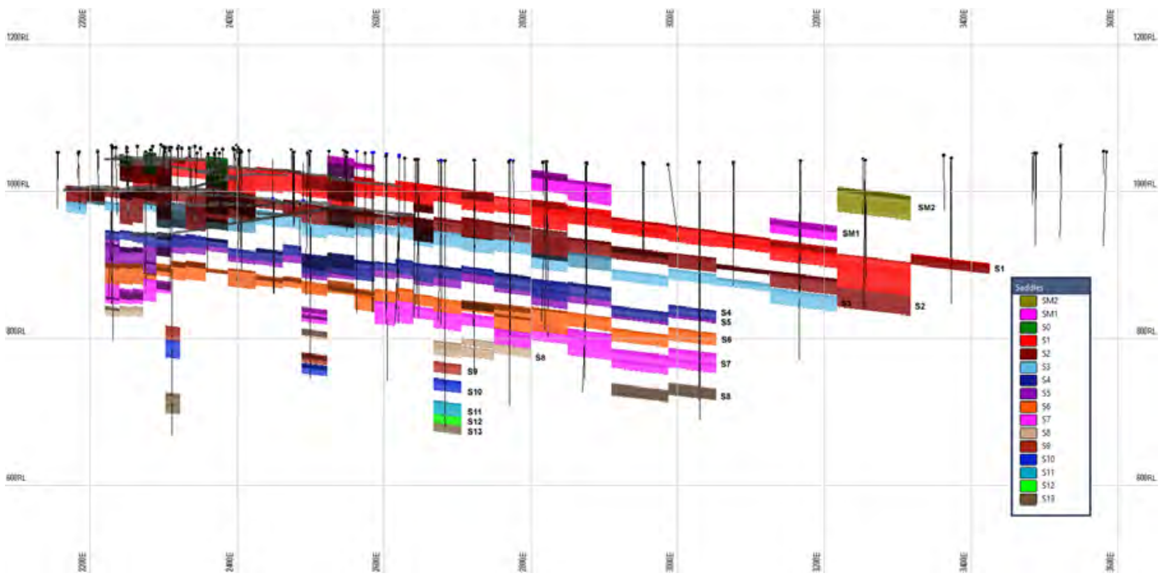


Figure 14-8 - Vertical longitudinal section showing mineralized zones (saddles), facing grid north.

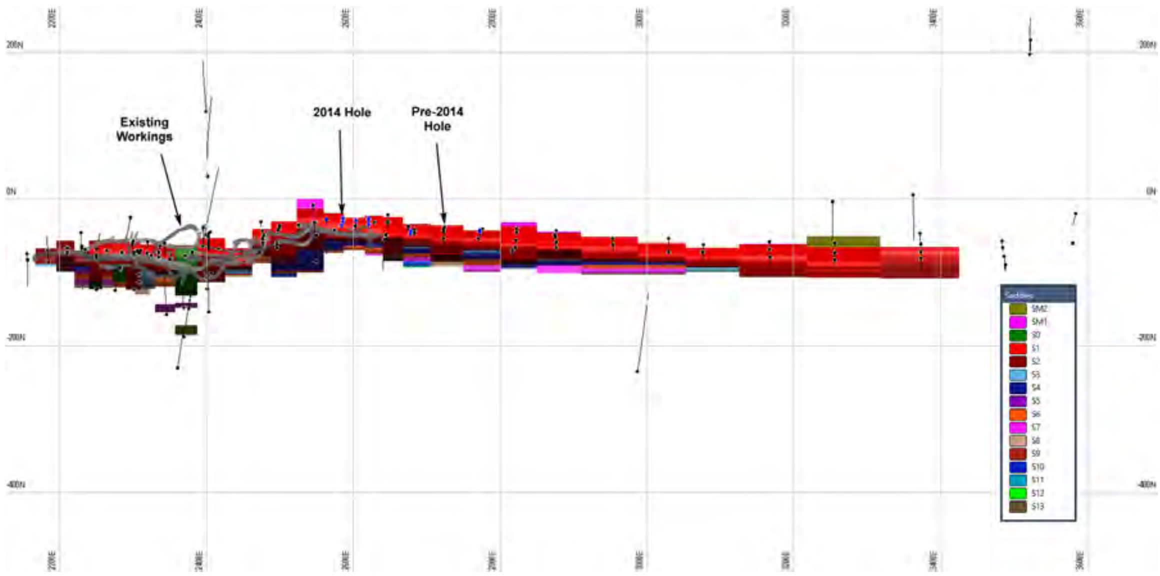


Figure 14-9 - Plan view of mineralized zones.

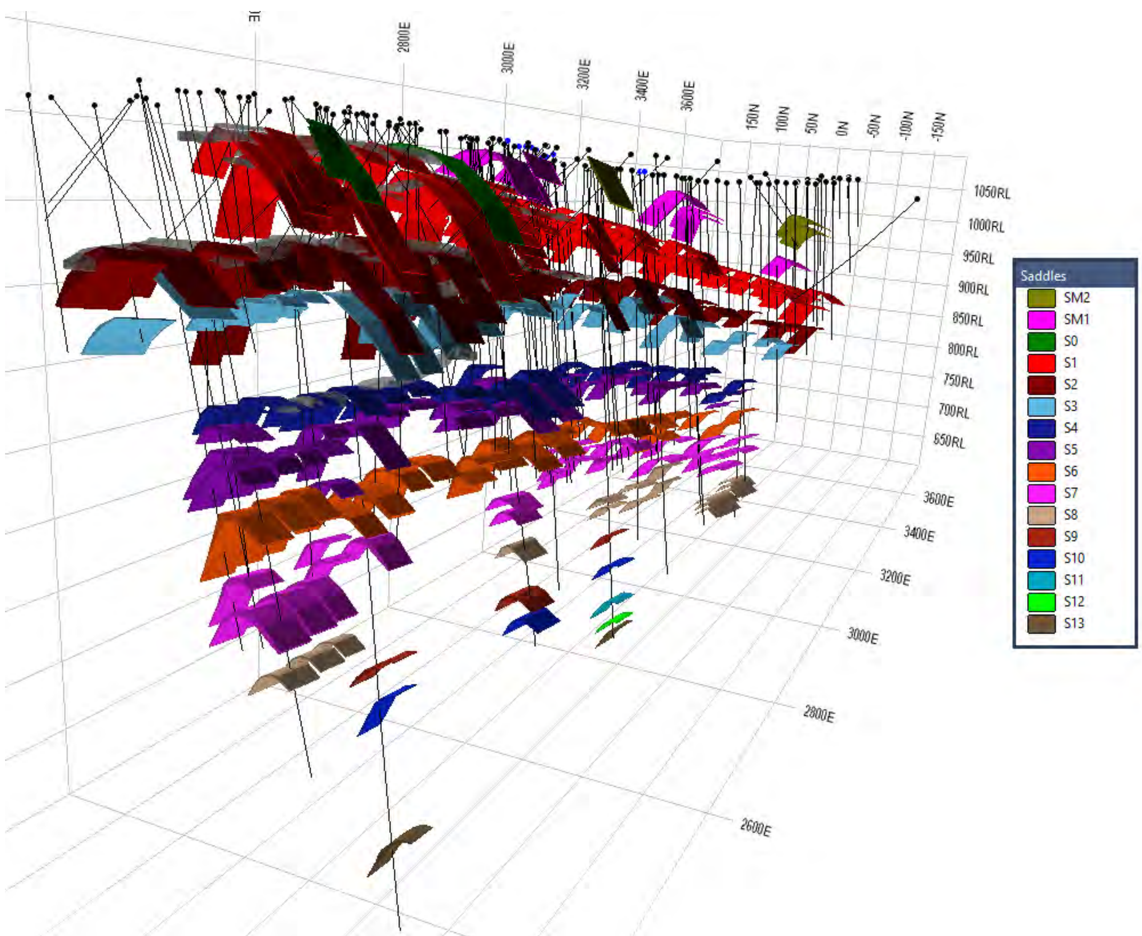


Figure 14-10 - Three-dimensional view of mineralized zones (saddles), facing grid northeast.

14.1.9.3 Zone Compositing

Samples within the mineralized zone wireframes were tagged. Each interpreted sample intercept was graphically reviewed section-by-section and manually adjusted where needed.

14.1.9.4 Core Angles

Core angles were used to aid geological interpretation. The core angles were apparently measured relative to the core axis. But, the dip direction was not recorded in the drill logs. In some cases, the author used judgment, based on RA's previous geological interpretations, to reverse the dip directions by 180°.

14.1.9.5 Modelling of Previous Mining

The vast majority of previous mining consisted of drifting along the crests of the saddles. In some areas, the walls were slashed and/or limited benching was done. In a few places, the northern down-dip or "leg" portion was mined.

Table 14-3 - Previous mining.

Area	Description
Saddle 1	Drifting from 2215 m - 2630 m East. Limited leg mining of north leg, 2455 m – 2465 m East between elevations of 1022 m and 1028 m.
Saddle 2	Drifting from 2265 m - 2685 m East. Leg mining in "Roger's Drift," 2300 m – 2400 m East (approximate). Not surveyed. 5-10 metres high (reportedly).
Saddle 3	Drifting from 2305 m - 2470 m East.
Saddle 4	Drifting from 2250 m - 2310 m East.

There were several places where it was reported that the underground workings followed the saddle crest, but the survey model and the drilling-interpreted saddle models did not quite overlap. To mitigate this problem, workings polygons were digitised over the saddle crests, where appropriate. The affected blocks were assigned a "void" attribute so that the mined material would be accounted for.

14.1.9.6 Crest and Leg Domains

For the purpose of separating blocks into the crest and leg mining domains, horizontal surfaces were digitised, section by section, vein by vein, at the floor of the crest.

The proportion of each block that was within the crest domain was stored in the "Proportion Crest Domain" field in the block model.

14.1.10 Block Modelling

A blank block model was created that was constrained by the mineralized zone wireframes. The model uses sub-blocking rather than block factoring.

A parent block size of 5.0x2.5x2.5 metres (East x North x Elevation) was chosen. That size is appropriate for the drill spacing in the more densely drilled areas of the deposit, with fences spaced 20-25 metres. Using the smaller-scale mining methods that are being considered for this deposit, that size block can be mined selectively. A larger block size, such as 10x10x10 metres, *may* be too coarse to adequately model the local grade variation that is observed both in core samples and underground, particularly in areas of relatively denser sampling.

Table 14-4 - Block model description.

Dimension	Range	Parent Block Size	Number of Blocks	Number of Sub-Blocks	Modelling Resolution
East	2100-3500 m	5 m	281	5	1 m
North	-250-100 m	2.5 m	141	10	0.25 m
Elevation	600-1100 m	2.5 m	201	10	0.25 m

14.1.11 Grade Estimation

Only “parent blocks” were estimated. Ordinary kriging was carried out in several runs (see Table 14-5).

Each saddle was considered as a separate domain for the purpose of grade estimation.

Semi-variogram model parameters are shown in Table 14-2.

Sample grades were top-cut (refer to Section 14.1.6).

Table 14-5 - Kriging plan.

Parameter	Run 1	Run 2	Run 3
Search Sphere Radius	24 m	48 m	100 m
Minimum Holes	2	2	1
Minimum/Maximum Samples	3/36	3/36	3/36
Discretisation	2x2x2	2x2x2	2x2x2

14.1.12 Block Cut-off Grade

Various block cut-off grades were considered. A prime consideration was the continuity of the blocks at various cut-offs. At block cut-off grades in the range of 3-4 g/tonne, the resources are *reasonably* continuous (refer to Figure 14-16 and Figure 14-17).

This deposit would most likely be mined using labour-intensive (low tonnes per person-shift) mining methods such as drift-and fill, shrinkage stoping, or narrow longhole methods. The unit mining cost of these methods are higher than high-productivity methods such as blasthole stoping.

For the purpose of selecting a reasonable block cut-off grade for resource identification, the following assumptions were made:

1. A unit mining cost in the range of $50-80 per tonne is appropriate.
2. In the past, processing recovery has been as high as 95%.
3. Processing costs are likely to be in the range of $20-30 per tonne.
4. Unplanned mining losses of 10-15% and unplanned dilution values of 10-20% are reasonable.
5. A gold price in the range of $US 1100-1300 per ounce is reasonable.
6. An exchange rate in the $0.70-1.00 is reasonable.

Using those assumptions, a sensitivity analysis was carried out which resulted in cut-off grades within the range of 2-5 grams of gold per tonne (in situ).

A 2 g/tonne block cut-off was selected for mineral resource identification. It shows reasonable prospects for economic extraction, reasonable block continuity, and a reasonable marriage between average grade, tonnes, and metal content (refer to Figure 14-17).

14.1.13 Mineral Resource Categorisation

The CIM Standing Committee on Reserve Definitions define Inferred, Indicated, and Measured Mineral Resources in “CIM Definition Standards for Mineral Resources and Mineral Reserves” (2014):

> *Inferred Mineral Resource*
>
> *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.*
>
> *An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
>
> *Indicated Mineral Resource*
>
> *An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*
>
> *Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.*
>
> *Measured Mineral Resource*
>
> *A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

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

This deposit has strong geological continuity but a strong nugget effect is also present. Therefore, a sample may indicate that an area is "gold-positive." But, the sample's grade is not a reliable measurement of the "local" grade.

For this deposit, "Run 3" of the kriging plan (refer to Table 14-5) was the least constrained and allowed block grade estimation using a minimum of three samples up to one hundred metres away (Table 14-5). In reality, because of the geological constraints that were applied to the block model, only 10% of the blocks were estimated in Run 3 (Figure 14-22). The +2 g/tonne, Run-3-blocks were estimated using an average of 15 samples, with an average distance between block and samples of 49 metres.

For those reasons, the authors believe that all +2 g/tonne blocks in the block model may reasonably be classified as Inferred.

For identifying Indicated Mineral Resources, the author selected a geometrical-geostatistical approach. The variogram range is 7 metres (Table 14-2). The area with the greatest drilling density lies around the existing underground workings (Figure 14-11). This area also contains the highest concentration of blocks that were estimated using samples from two holes with an average distance of less than 7 metres.

The author believes that the geological continuity and the drill sample spacing are adequate for classifying Indicated mineral resources in Saddles 1-4 in that area.

The author believes that there are isolated pockets of mineralization around the existing underground workings for which the geological continuity and the drill sample spacing may be adequate for identifying a small amount of Measured mineral resources. However, because those pockets would be so isolated and the volume would be so relatively small, no Measured mineral resources are identified in this report.

Inferred Category:

- Fair confidence in geological continuity.
- At least one intercept within 100 metres.

Indicated Category:

- Good confidence in geological continuity.
- Drill fence spacing 25 metres or less.

Table 14-6 - Location of Indicated mineral resources.

Saddle	Easting
Saddle 1	2,220-2,655 metres
Saddle 2	2,220-2,655 metres
Saddle 3	2,575-2,655 metres
Saddle 4	2,250-2,290 metres

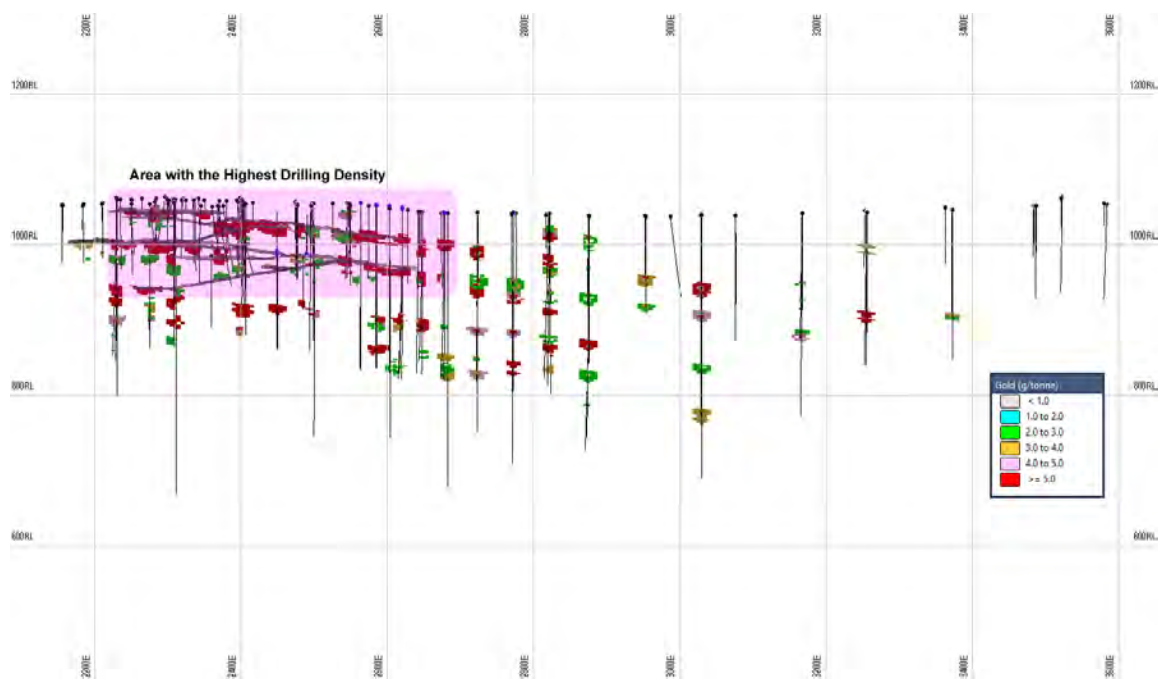


Figure 14-11 - Longitudinal view showing blocks that are within seven metres of at least two intercepts.

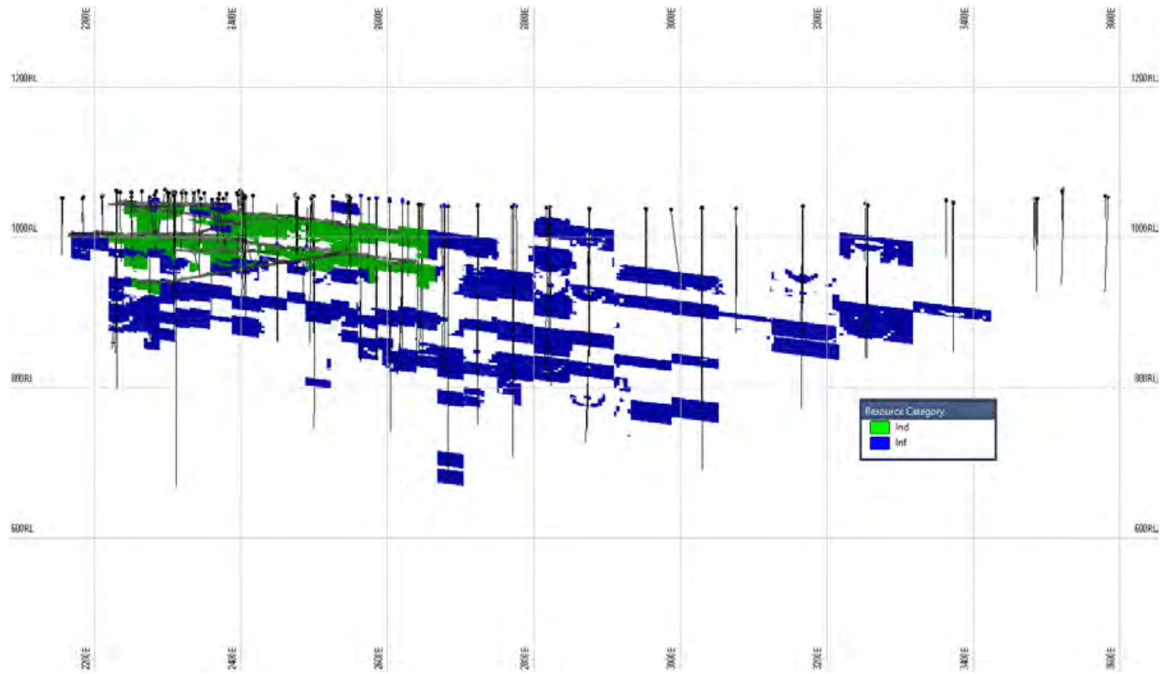


Figure 14-12 - Indicated ("Ind") mineral resources, Longitudinal View (Facing North).

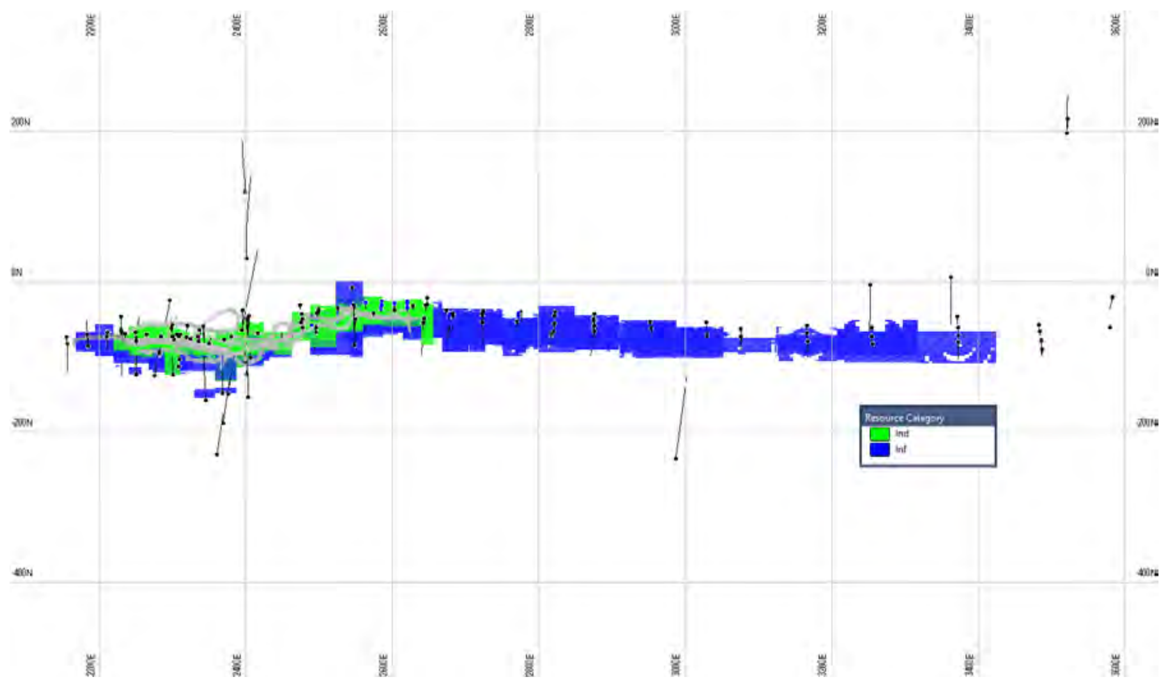


Figure 14-13 - Indicated ("Ind") mineral resources, Plan View.

14.1.14 Results and Discussion

The results of grade estimation at various block-cut-off values are shown in Table 14-8. Figures illustrating the results are shown in Figure 14-17, Figure 14-18, Figure 14-19, and Figure 14-20.

Initially, it was thought that the crests of the saddles held the majority of the metal content in the deposit. Detailed modelling of the drill hole and underground information available to date shows that the limbs and crest have similar grades, but approximately 2/3 of the tonnes and 2/3 of the metal are contained in the limbs (see Table 14-11).

If one were to use a 3 g/tonne block-cut-off grade for mine planning and production scheduling purposes, the +3 g/tonne material is not especially continuous along strike (see Figure 14-17). Periodic breaks in the +3 g/tonne blocks mean that a good deal of development work in sub-grade material should be expected.

At a 2 g/tonne block cut-off grade, when one views the resources, the gaps between mineralization are likely attributed more to lack of sampling than to “gold-negative” samples.

The continuity of the +2 g/tonne blocks is greater than the +3 g/tonne blocks (Figure 14-17). And, a 2 g/tonne block cut-off grade is approximately the breakeven cut-off grade for mining and milling an “exposed block.” For those reasons, a 2 g/tonne was selected as the block cut-off grade for the reporting of mineral resources.

Saddle 6 is the deepest saddle with a significant amount of +2 g/tonne material.

From Figure 14-17, the extent of the main block of resources at a 2 g/tonne block cut-off is 2200-3200 metres East and 800-1045 metres in Elevation, or 1000x245 metres in longitudinal area.

Table 14-7 - Summary of mineral resources by resource category.

	Volume (m3)	Tonnes	Ounces	Average Grade (g/tonne)
Ind	57.2k	151.5k	58.0k	11.9
Inf	163.8k	434.1k	96.8k	6.9

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

Table 14-8 - Summary of non-categorised resources for grade-tonnage curve.

Cut-off Grade (g/tonne)	Volume (m^3)	Tonnes	SG	Average Grade (g/tonne)	Ounces
4.0	*125k*	*332k*	*2.65*	*12.3*	*132k*
3.5	*141k*	*374k*	*2.65*	*11.4*	*137k*
3.0	*159k*	*420k*	*2.65*	*10.5*	*142k*
2.5	*191k*	*505k*	*2.65*	*9.2*	*149k*
2.0	**221k**	**586k**	**2.65**	**8.2**	**155k**
1.5	*270k*	*716k*	*2.65*	*7.0*	*162k*
1.0	*311k*	*825k*	*2.65*	*6.3*	*166k*
0.5	*339k*	*900k*	*2.65*	*5.8*	*168k*
0.0	*355k*	*941k*	*2.65*	*5.6*	*168k*

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

Table 14-9 - Mineral resources by resource category and saddle.

	Volume (m3)	Tonnes	Ounces	Average Grade (g/tonne)
Ind	57.2k	151.5k	58.0k	11.9
S1	31.0k	82.1k	30.1k	11.4
S2	24.7k	65.5k	25.0k	11.9
S3	0.3k	0.9k	0.1k	2.2
S4	1.1k	3.0k	2.8k	29.1
Inf	163.8k	434.1k	96.8k	6.9
SM2	7.6k	20.1k	1.9k	2.9
SM1	5.2k	13.8k	1.4k	3.2
S0	1.7k	4.6k	0.9k	5.8
S1	28.0k	74.2k	15.5k	6.5
S2	15.9k	42.2k	4.2k	3.1
S3	16.5k	43.6k	6.1k	4.4
S4	24.0k	63.6k	19.1k	9.3
S5	14.5k	38.4k	9.6k	7.8
S6	29.2k	77.3k	31.8k	12.8
S7	15.6k	41.3k	4.7k	3.6
S8	4.2k	11.2k	1.2k	3.5
S11	0.9k	2.4k	0.2k	3.1
S13	0.5k	1.4k	0.2k	3.6

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

Table 14-10: Details of resources by crest/limb domain.

	Volume (m3)	Tonnes	Ounces	Average Grade (g/tonne)
Crest	75.2k	199.2k	55.3k	8.6
Ind	14.8k	39.3k	15.8k	12.5
S1	8.8k	23.3k	8.8k	11.8
S2	5.3k	14.2k	5.7k	12.5
S3	0.1k	0.4k	0.0k	2.2
S4	0.5k	1.4k	1.3k	29.3
Inf	60.4k	159.9k	39.5k	7.7
SM2	3.8k	10.0k	1.0k	3.0
SM1	1.2k	3.2k	0.2k	2.3
S0	0.0k	0.1k	0.0k	3.8
S1	8.7k	23.1k	5.4k	7.3
S2	3.7k	9.9k	0.9k	2.8
S3	4.3k	11.4k	1.4k	3.8
S4	10.2k	27.0k	6.3k	7.3
S5	5.1k	13.4k	3.7k	8.5
S6	13.8k	36.5k	17.6k	15.0
S7	7.0k	18.4k	2.1k	3.6
S8	2.2k	5.8k	0.7k	3.6
S11	0.3k	0.8k	0.1k	3.1
S13	0.1k	0.3k	0.0k	3.6
Limb	145.8k	386.4k	99.5k	8.0
Ind	42.4k	112.3k	42.1k	11.7
S1	22.2k	58.8k	21.3k	11.3
S2	19.4k	51.4k	19.3k	11.7
S3	0.2k	0.5k	0.0k	2.2
S4	0.6k	1.6k	1.5k	29.0
Inf	103.5k	274.2k	57.3k	6.5
SM2	3.8k	10.1k	0.9k	2.8
SM1	4.0k	10.6k	1.2k	3.4
S0	1.7k	4.5k	0.9k	5.9
S1	19.3k	51.1k	10.2k	6.2
S2	12.2k	32.3k	3.3k	3.2
S3	12.2k	32.2k	4.7k	4.5
S4	13.8k	36.6k	12.8k	10.9
S5	9.4k	25.0k	5.9k	7.3
S6	15.4k	40.8k	14.1k	10.7
S7	8.6k	22.9k	2.6k	3.5
S8	2.0k	5.4k	0.6k	3.4
S11	0.6k	1.6k	0.2k	3.1
S13	0.4k	1.1k	0.1k	3.6

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

Table 14-11 - Proportion of mineral resources by crest/limb domain[1].

Domain	Proportion of Tonnes	Proportion of Metal
Crest	34%	36%
Limbs	66%	64%
Total	100%	100%

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

Zero Block-Cut-off:

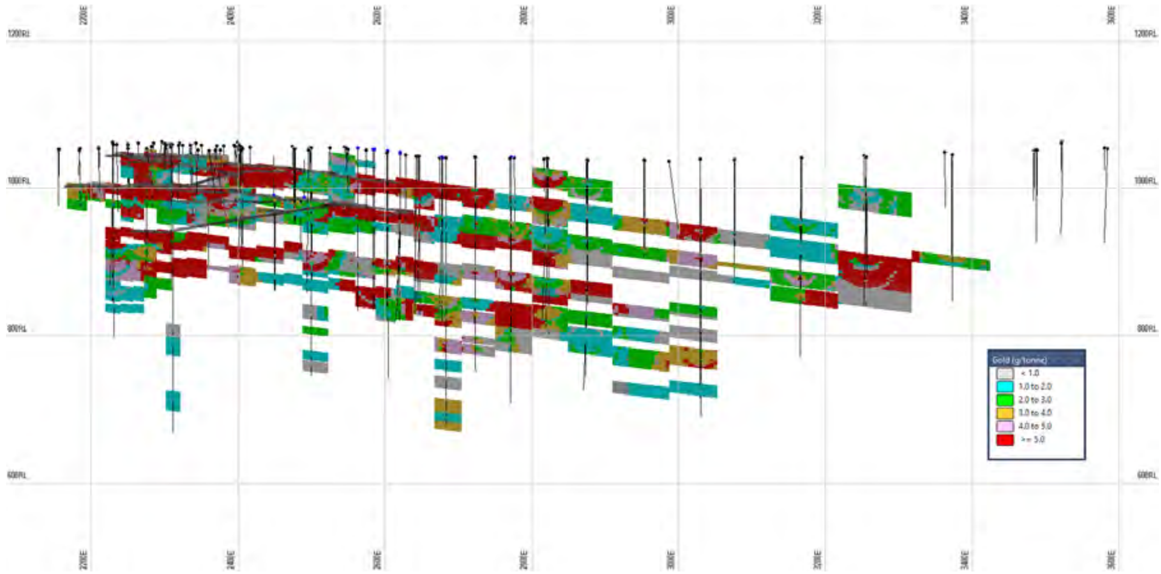


Figure 14-14 - Vertical longitudinal sections showing all blocks.

2 g/tonne Block-Cut-off:

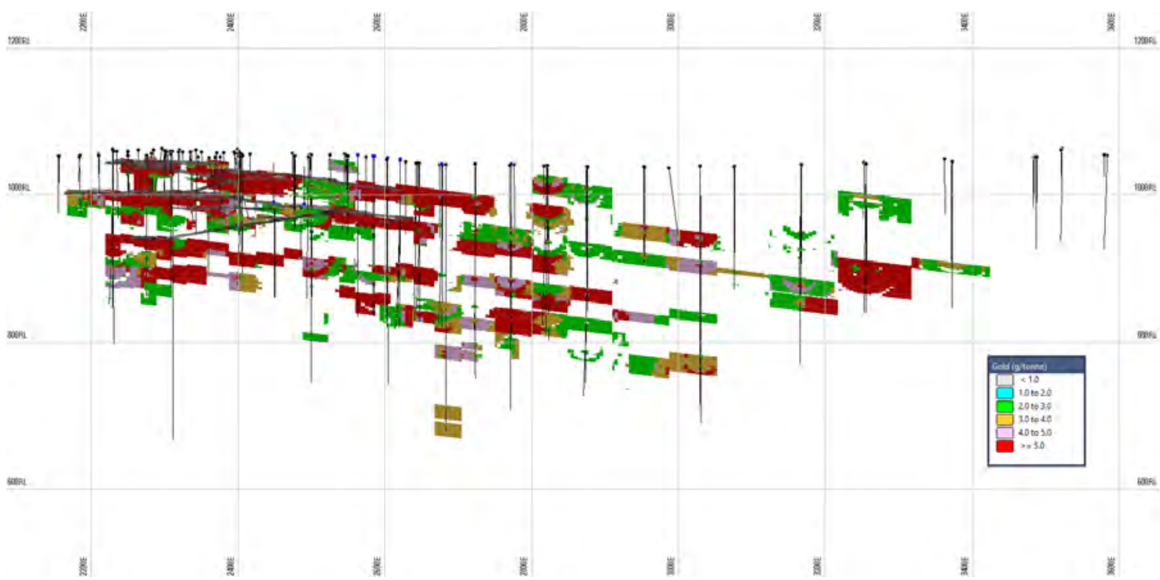


Figure 14-15 - Vertical longitudinal sections showing blocks at 2 g/tonne block cut-off grade.

3 g/tonne Block-Cut-off:

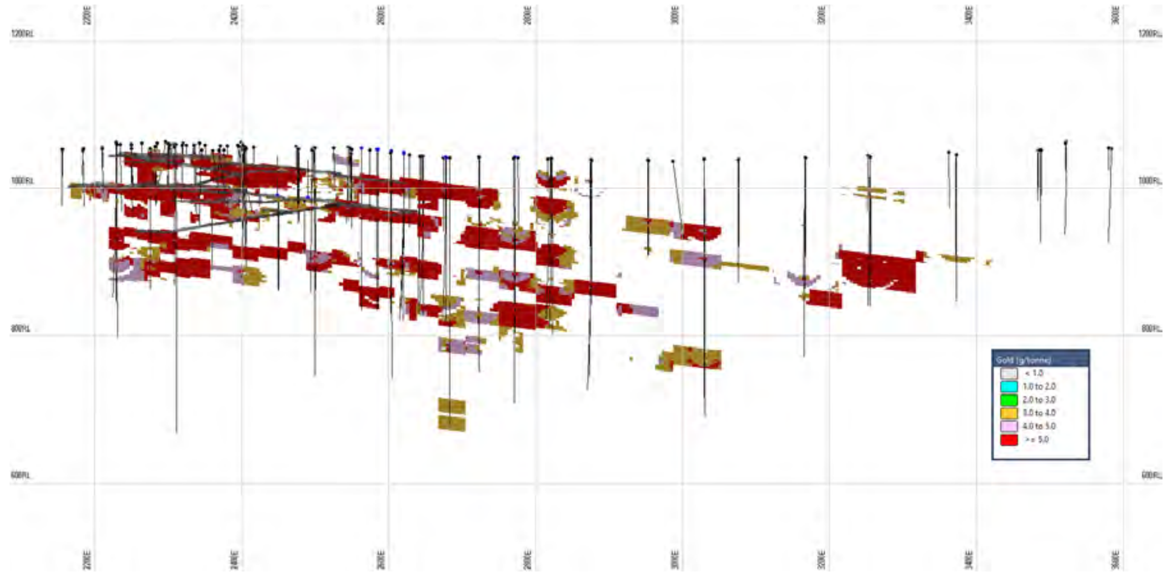


Figure 14-16 - Vertical longitudinal sections showing blocks at 3 g/tonne block cut-off grade.

4 g/tonne Block-Cut-off:

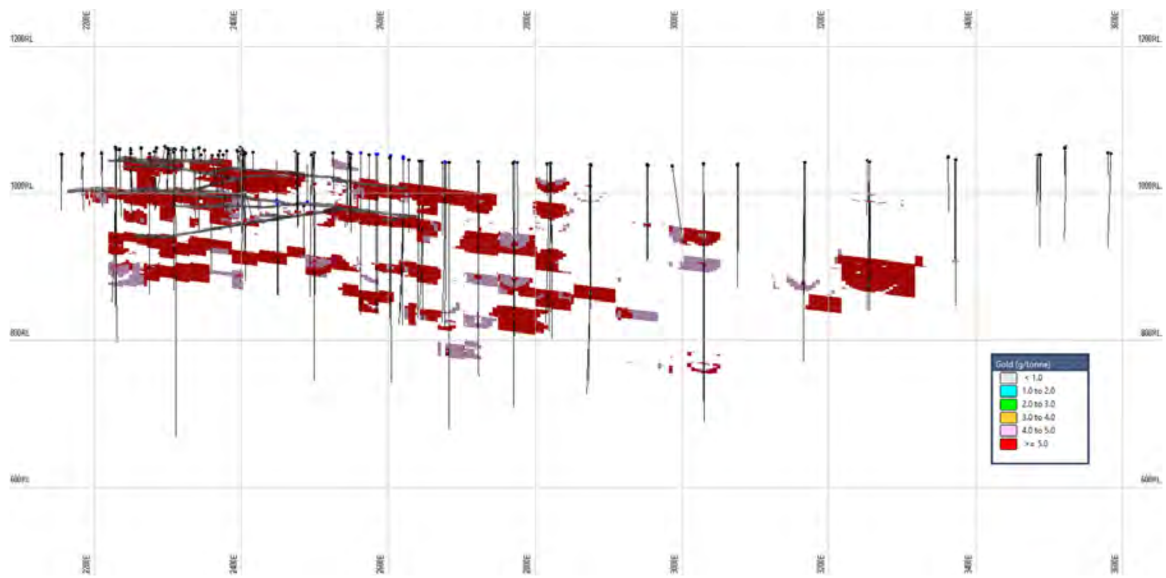


Figure 14-17 - Vertical longitudinal sections showing blocks at 4 g/tonne block cut-off grade.

Figure 14-18 - Section 2250E, Showing Block Grades.

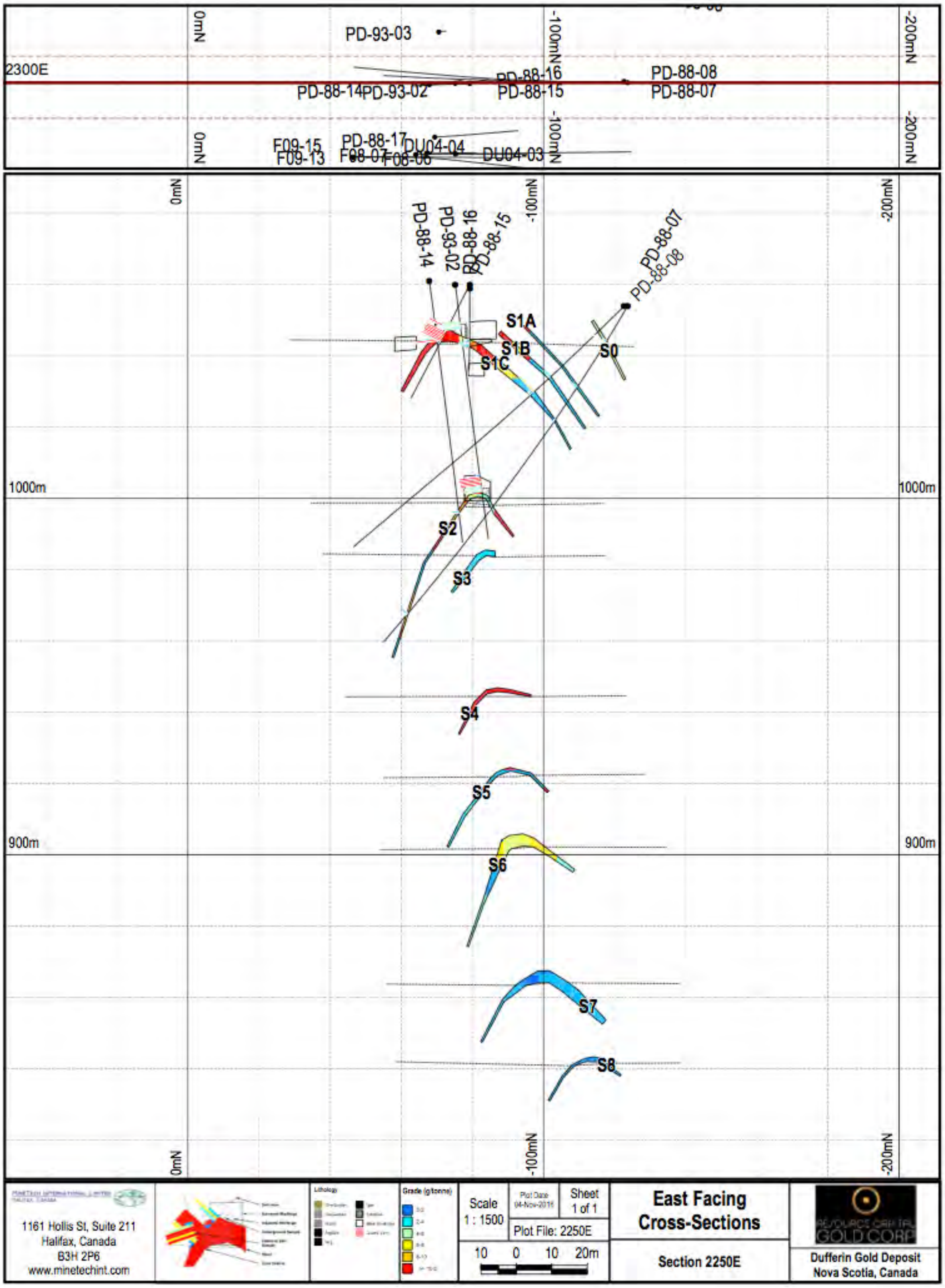


Figure 14-19 - Section 2500E, Showing Block Grades

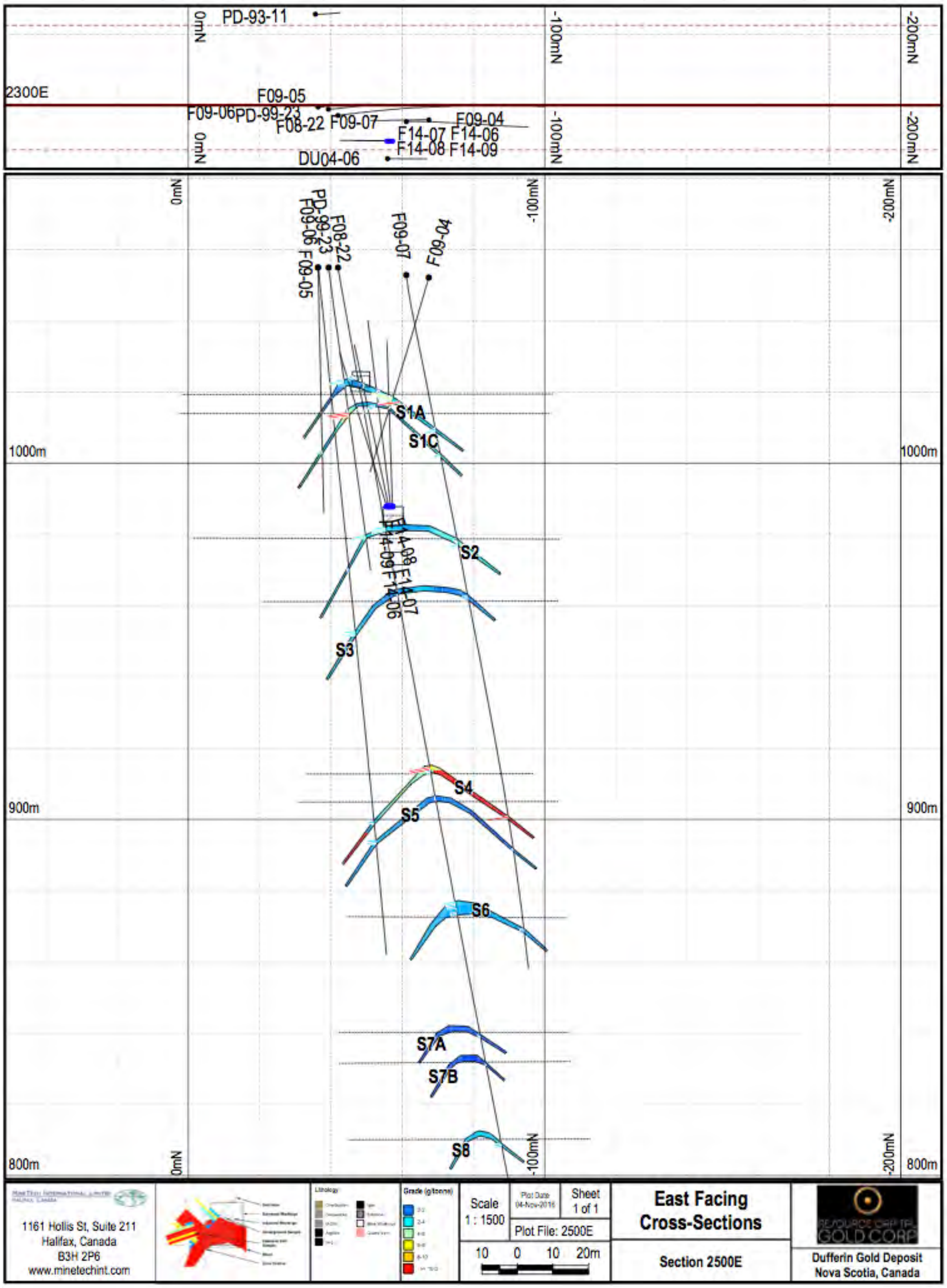
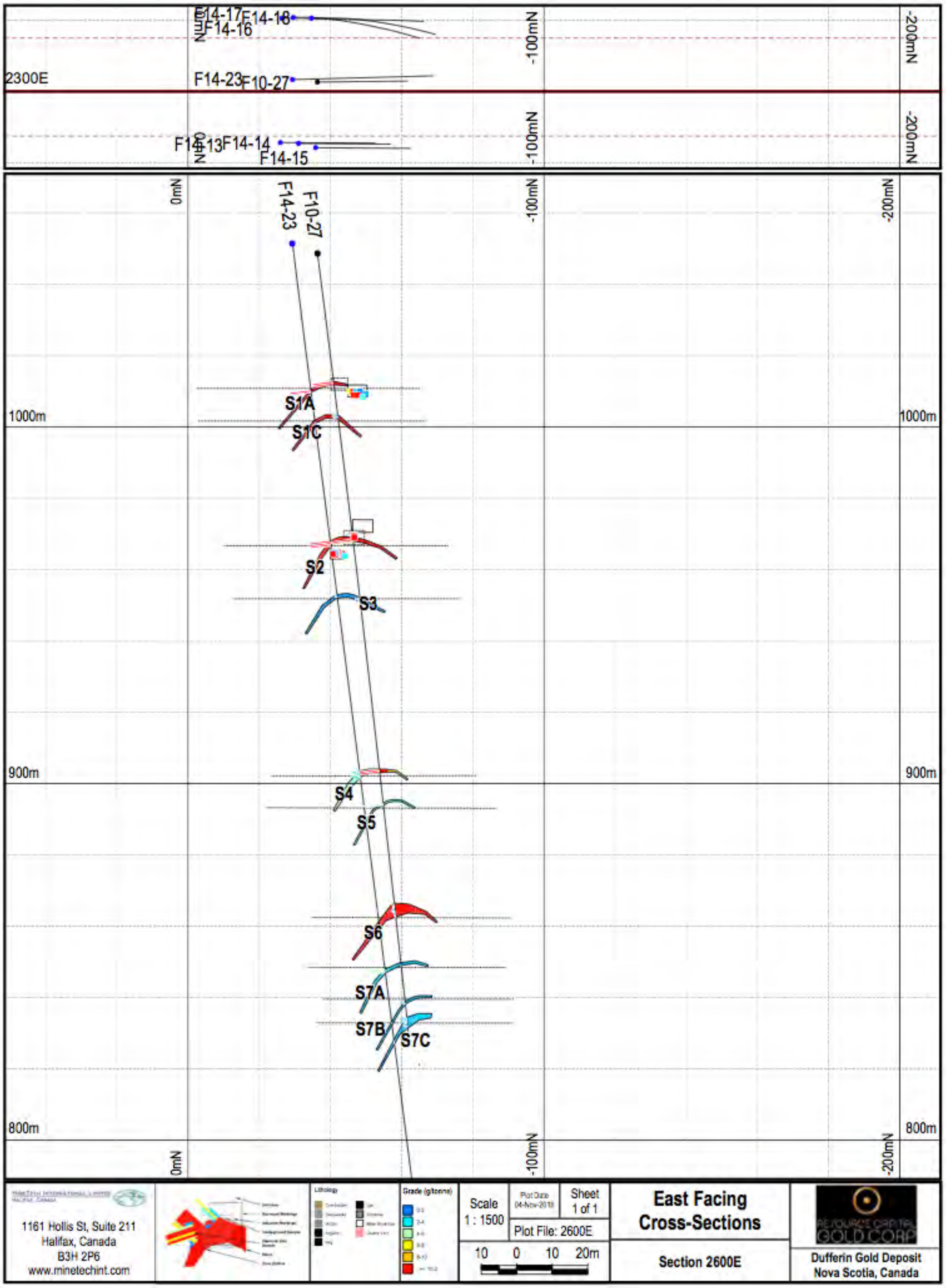


Figure 14-20 - Section 2600E, Showing Block Grades.



14.1.15 Non-Top-Cut Results

For interest sake, block grades were re-estimated without using a top-cut grade (Table 14-12). The results are not considered to be a "Mineral Resource" and are presented here for information purposes only.

Table 14-12: Non-top-cut grade estimation results.

	Volume (m3)	Tonnes	Ounces	Average Grade (g/tonne)
Ind	57.2k	151.5k	59.5k	12.21
S1	31.0k	82.1k	30.7k	11.63
S2	24.7k	65.5k	25.9k	12.30
S3	0.3k	0.9k	0.1k	2.19
S4	1.1k	3.0k	2.8k	29.47
Inf	163.8k	434.1k	100.9k	7.23
SM1	5.2k	13.8k	1.4k	3.16
SM2	7.6k	20.1k	1.9k	2.87
S0	1.7k	4.6k	0.9k	5.82
S1	28.0k	74.2k	15.6k	6.54
S2	15.9k	42.2k	4.2k	3.09
S3	16.5k	43.6k	6.1k	4.37
S4	24.0k	63.6k	19.1k	9.35
S5	14.5k	38.4k	9.6k	7.76
S6	29.2k	77.3k	35.8k	14.39
S7	15.6k	41.3k	4.7k	3.57
S8	4.2k	11.2k	1.2k	3.48
S11	0.9k	2.4k	0.2k	3.08
S13	0.5k	1.4k	0.2k	3.57

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65;
[3] Gold price = $US 1250 per ounce.
[4] No top-cut grade was used.

14.1.16 Block Model Validation

14.1.16.1 Global Statistics

The global block mean *should* closely approximate the declustered sample mean. For Dufferin, the global block mean was within 6% of the declustered mean, which is an acceptable fit.

Table 14-13 - Comparison of sample and block statistics.

Item	Mean	Variance	Standard Deviation
Length-Regularised Samples	6.95	443	21.0
Declustered Samples	**5.28**	233	15.2
Block Model	**5.56**	45.0	6.71
Length-Regularised Samples Versus Model	25%	884%	213%
Declustered Samples Versus Model	5%	418%	127%

14.1.16.1.1Degree of Smoothing

The block model is much smoother compared to the declustered samples, with standard deviation values of 15.2 g/tonne and 6.71 g/tonne, respectively (Table 14-13). The standard deviation of the block grades is only 44% of that of the declustered samples.

14.1.16.1.2Volume Comparison

A volume comparison was undertaken to ensure that block volumes agreed with the wireframe volumes. There was only a 0.2% difference between the two volumes, which is acceptable.

Table 14-14 - Comparison between wireframe and block model volumes.

Object	Volume
Wireframe	375,800 m^3
Block Model	375,000 m^3
Difference	0.2%

14.1.16.2 Swath Plot Analysis

A swath plot was constructed to provide a semi-local validation of the average grade conformance across the model (Figure 14-21). Generally, the block grades followed the same trend as the sample grades, but, as one would expect, more smoothly.

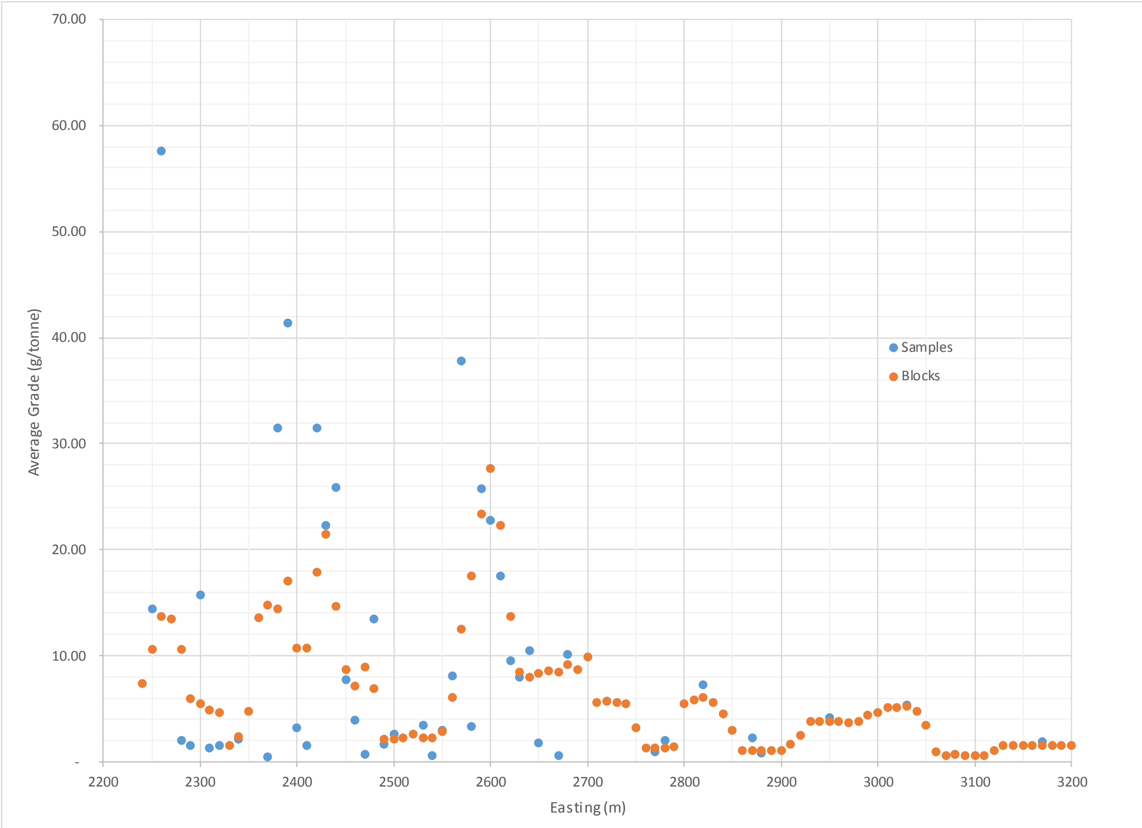


Figure 14-21 - Easting swath plot for Saddle 1 showing sample assays and estimated block grades (facing north).

14.1.16.3 Grade Interpolation Performance

The majority of blocks were estimated in the first pass (Figure 14-22). The vast majority of blocks were estimated using two holes (Figure 14-23) and the maximum number of 36 samples (Figure 14-24). The average distance between sample and block was 24 metres and the average distance between the closest sample and block was 16 metres.

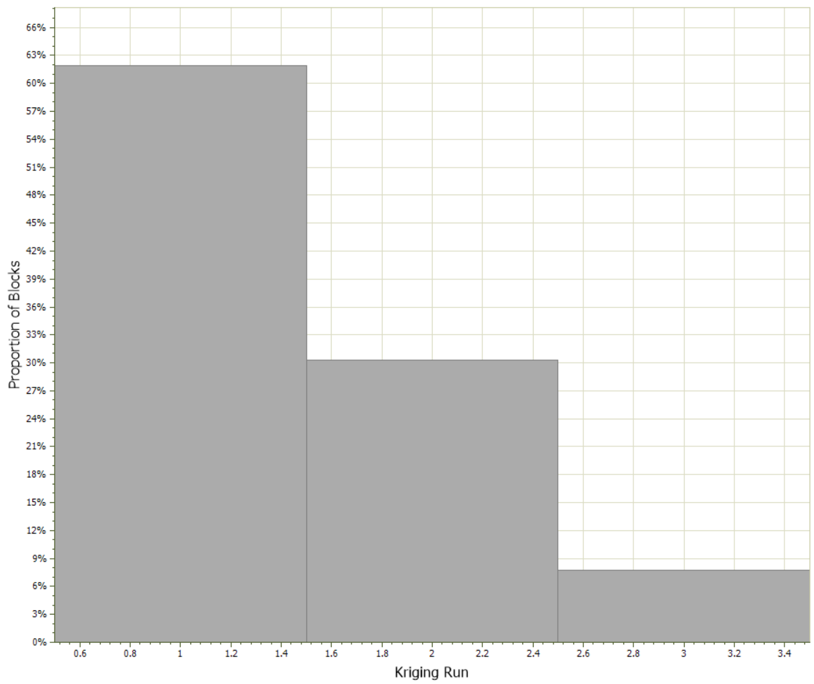
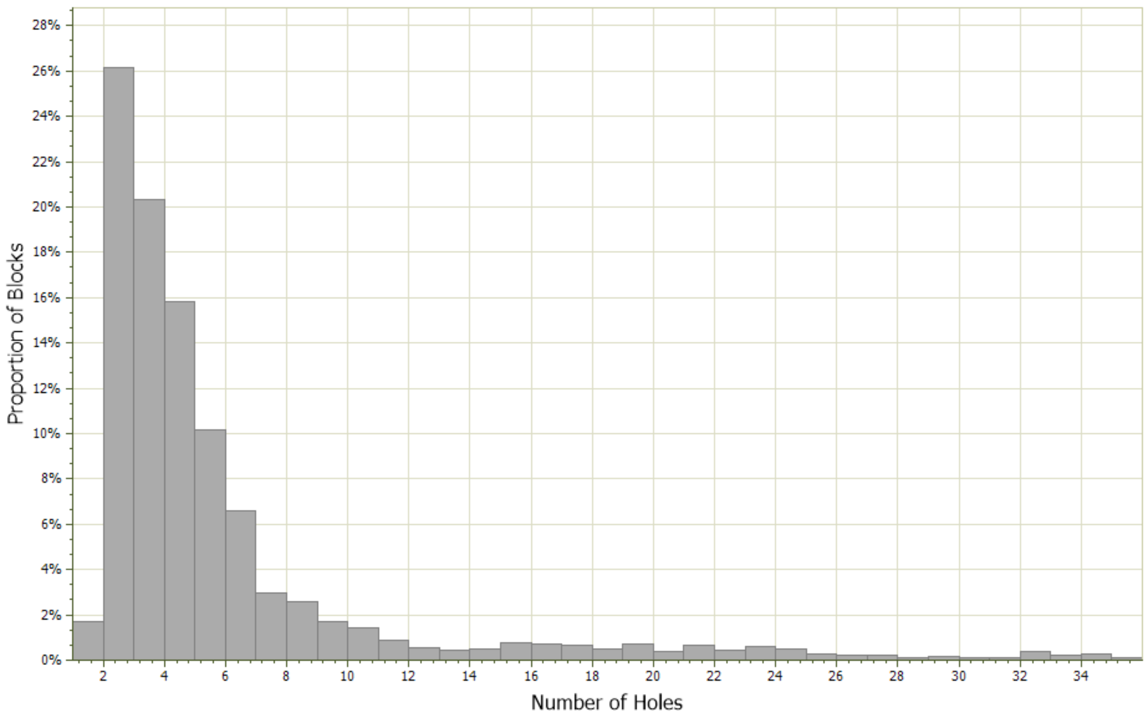
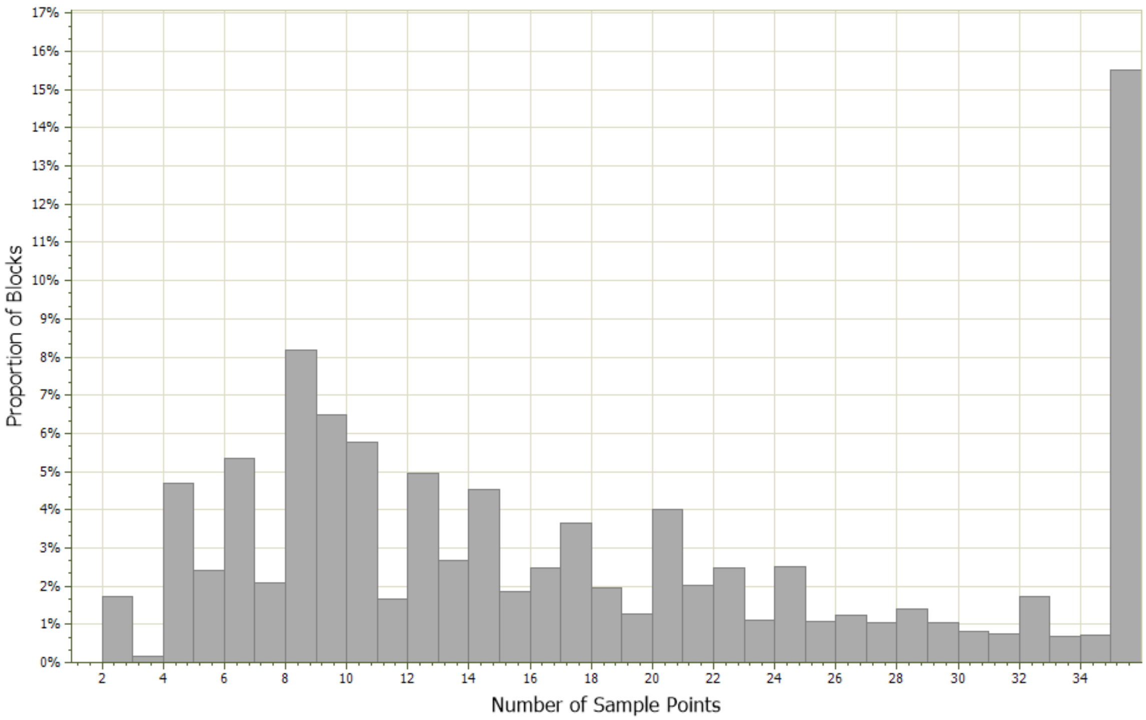


Figure 14-22 - Kriging run histogram showing blocks that were estimated during the 1st, 2nd, and 3rd passes.



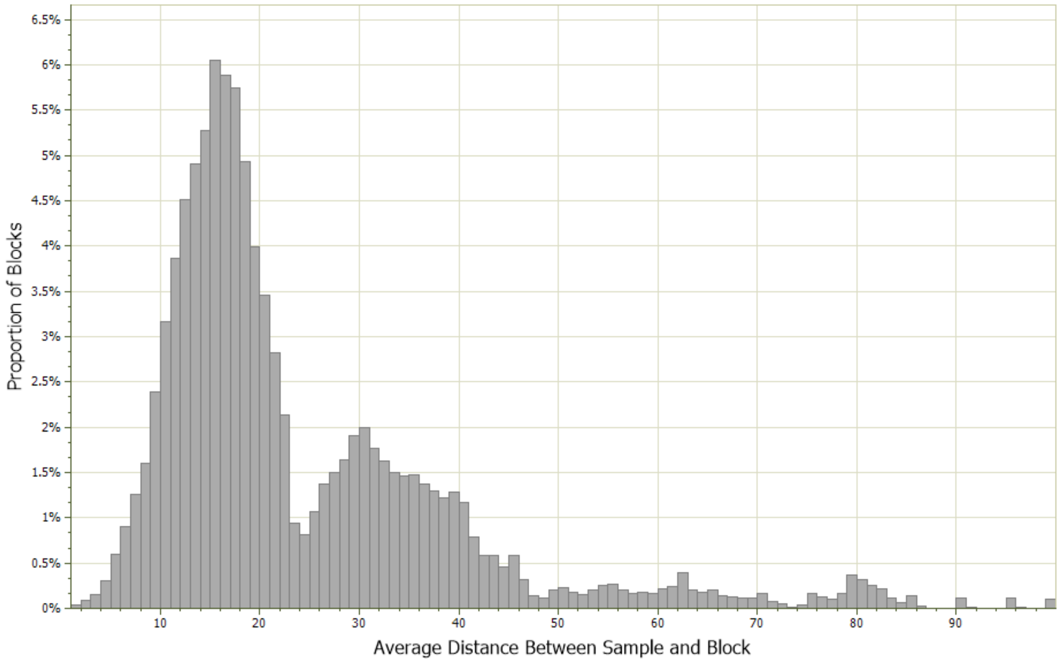
Minimum value	1	Variance	32	Geometric std dev	2
Maximum value	36	Standard deviation	6	Sichel's estimator	5
2nd highest	36	Coeff. of variation	1	Sichel's V	1
3rd highest	36	Median	4	Sichel's gamma	1
4th highest	36	Ln mean	1	Chi-Square Test	95905025
N	573135	Ln std deviation	1	Degrees of freedom	29
Mean	5	Geometric mean	4		

Figure 14-23 - Histogram showing number of holes used for block grade estimation.



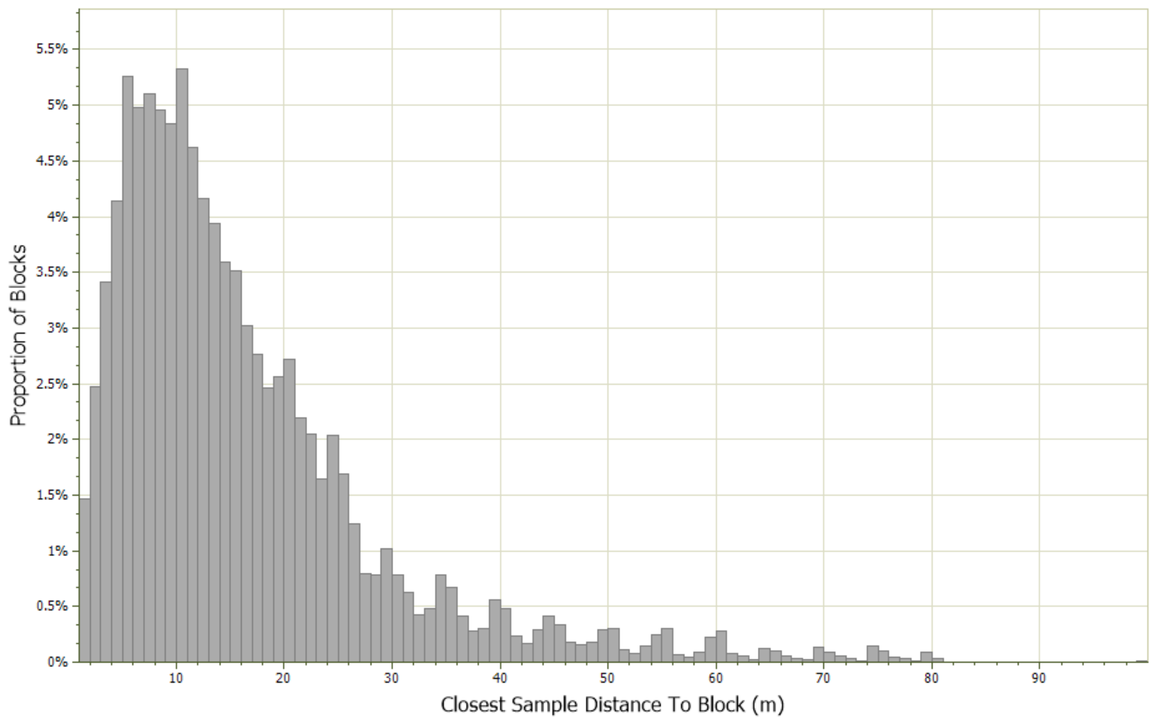
Minimum value	2	Variance	116	Geometric std dev	2
Maximum value	36	Standard deviation	11	Sichel's estimator	18
2nd highest	36	Coeff. of variation	1	Sichel's V	0
3rd highest	36	Median	14	Sichel's gamma	1
4th highest	36	Ln mean	3	Chi-Square Test	1803381
N	573135	Ln std deviation	1	Degrees of freedom	34
Mean	18	Geometric mean	14		

Figure 14-24 - Histogram showing number of sample points used for block grade estimation.



Minimum value	1.031	Variance	240.292	Geometric std dev	1.755
Maximum value	99.996	Standard deviation	15.501	Sichel's estimator	23.996
2nd highest	99.996	Coeff. of variation	0.642	Sichel's V	0.317
3rd highest	99.996	Median	18.624	Sichel's gamma	1.172
4th highest	99.996	Ln mean	3.020	Chi-Square Test	4010962.900
N	935875	Ln std deviation	0.563	Degrees of freedom	89
Mean	24.159	Geometric mean	20.483		

Figure 14-25 - Histogram showing the average distance between sample and block.



Minimum value	1.001	Variance	162.354	Geometric std dev	2.138
Maximum value	99.912	Standard deviation	12.742	Sichel's estimator	16.572
2nd highest	99.912	Coeff. of variation	0.782	Sichel's V	0.577
3rd highest	99.912	Median	12.826	Sichel's gamma	1.335
4th highest	99.912	Ln mean	2.519	Chi-Square Test	5920580.041
N	570564	Ln std deviation	0.760	Degrees of freedom	69
Mean	16.293	Geometric mean	12.417		

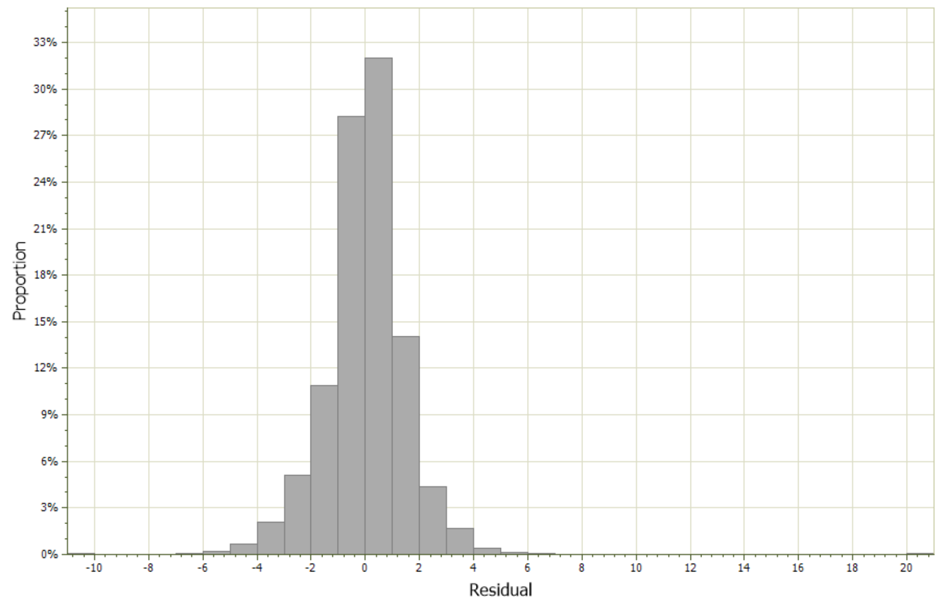
Figure 14-26 - Histogram showing the closest distance between sample and block.

14.1.16.4 Cross-Validation

Cross-validation of the semi-variogram model and search parameters was carried out (Figure 14-27). The mean residual was +0.0067, indicating almost no estimation bias and a fair balance between under- and over-estimation. A longitudinal view of the cross-validation residuals did not reveal any obvious spatial trend(s) for under- or over-estimation.

Table 14-15 - Cross-validation statistics.

	Mean	Std Dev
Raw Data :	7.5926	18.167
Transformed :	0.26645	2.0580
Estimate :	0.26403	1.5573
Back transform :	8.6576	15.181
Standard error :	1.6191	0.92194
Error statistic :	0.001966	0.92219



Minimum value	-10.0416	Variance	2.2951	Geometric std dev	3.4564
Maximum value	20.5161	Standard deviation	1.5150	Sichel's estimator	1.2955
2nd highest	6.7194	Coeff. of variation	225.3577	Sichel's V	1.5381
3rd highest	5.4119	Median	0.0407	Sichel's gamma	2.1566
4th highest	5.2717	Ln mean	-0.5097	Chi-Square Test	1239.3156
N	2183	Ln std deviation	1.2402	Degrees of freedom	10
Mean	0.0067	Geometric mean	0.6007		

Figure 14-27 - Histogram of cross-validation residuals.

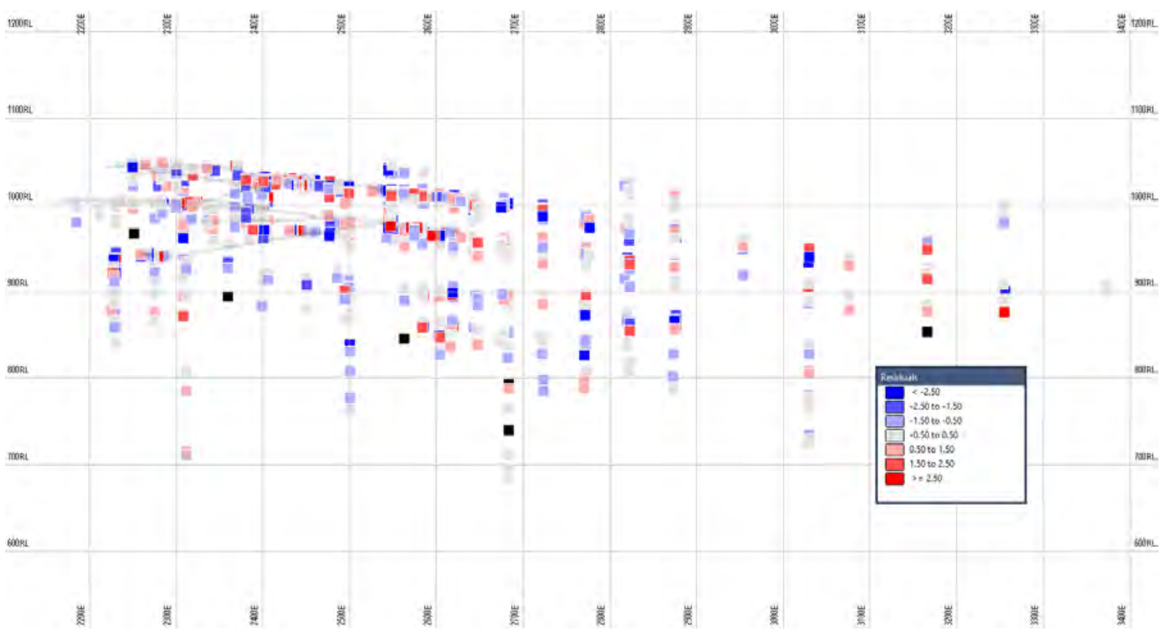


Figure 14-28 - Longitudinal view of cross-validation residuals, facing north (mine grid).

14.2 West Dufferin Mineral Resources

14.2.1 Introduction

The resource estimate for the West Dufferin portion of the Property was carried out by Greg Mosher, P.Geo., Principal Geologist with Global Mineral Resource Services. (GMRS). GMRS received collar location, downhole-survey and assay (gold) files from Resource Capital Gold Corp. (RCG) as csv files. GMRS also received from RGC, wireframe geological models for 14 of the veins present at West Dufferin in dxf format. Several of the modelled domains overlap by small amounts. Rather than re-model them, the domains were assigned estimation priorities so that the overlapping volumes were attributed to only one of the domains and were subtracted from the second.

All data and wireframes are located in local metric grid coordinates. The veins that comprise the Property strike approximately east-west and the local grid is oriented so that the baseline (Y=0) coincides with the principal anticlinal fold axis. The elevation datum has been increased by 1000 meters (m) so that all drillhole depths are positive. Most surface drillholes have collar elevations between 1,600 and 1,700m. The Property was tested by drillholes over a strike length of 1,900m, between local grid 320E and 2120E but the majority of mineralized intercepts were encountered between local grid coordinates 1,200E and 1,800E. Therefore, the veins have been modelled and the resource estimated within this 600 m portion of the strike length of the vein trend. Within this interval of the grid, fences of holes were drilled on approximately 25-metre intervals.

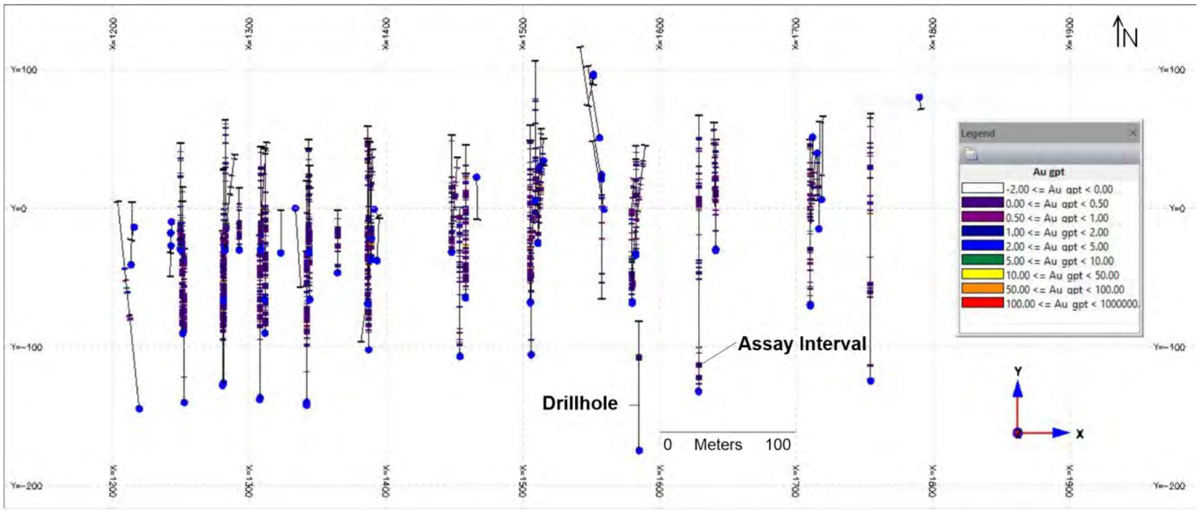
Drill data suggests that many more veins are present within the Property but there is insufficient data to permit a confident interpretation of their shape and orientation in space. Therefore, the following estimate was restricted to the resource contained within 14 veins that could be modelled with sufficient confidence to extrapolate the veins between at least two drill sections along strike.

14.2.2 Exploratory Data Analysis

14.2.2.1 Assays

GMRS received drill collar data for 159 holes of which 79 were located between local grid coordinates 1,200 and 1,800E, the grid interval for which the following resource was estimated. The total dataset contains 5,094 gold assays; the veins in the central portion of the Property that form the basis of the resource estimate contain 856 assays. The full assay dataset has an arithmetic average gold grade of 2.23 grams / tonne (g/tonne); the sub-set used for the resource estimate has an arithmetic average gold grade of 2.18 g/tonne. A plan of the drillholes with assays is shown in Figure 14-29.

Figure 14-29: West Dufferin Resource Estimation Drillholes Plan View



14.2.2.2 Capping

The West Dufferin gold assays dataset contains a small number of very high grades, with a maximum of 3,168 g/tonne gold compared to an average of 2.23 g/tonne gold. The dataset was assessed for the need for capping and if necessary, for an appropriate capping level. The log-normal cumulative frequency plot of the total dataset is shown in Figure 14-30 with a prominent break visible at 100 g/tonne. This level was adopted as the appropriate capping level. At a capping level of 100 g/tonne gold, 18 samples were affected and the capped dataset had a mean gold grade of 1.33 g/tonne. When this capping level was applied to the sub-set used for the resource estimate, only one assay was affected so that the capped and uncapped estimation results are very similar.

Figure 14-30: West Dufferin Capping Curve Au g/tonne

14.2.2.3 Composites

Conceptual mine planning for the East Dufferin deposit has indicated that the minimum practical mining width is likely to be 0.5m. For that reason, and because a majority (79%) of the sample lengths for West Dufferin are 0.5m or less, that length was chosen as the most appropriate composite length. Composites less than 0.1m in length were discarded. However, the samples within the modelled wireframes are all between 0.5 and 0.1m in length so the composites retained the length of the underlying assay sample set.

14.2.2.4 Bulk Density

There are no bulk density measurements for West Dufferin. Previous estimation work conducted on the East Dufferin deposit used bulk density values of 2.65 and 2.7 g/cm^3. The slightly conservative bulk density value of 2.65 g/cm^3 was chosen for West Dufferin and was used to convert volumes in the block model to tonnages.

14.2.3 Geological Interpretation

Gold mineralization at West Dufferin is contained in a series of stacked anticlinally- and synformally-folded quartz veins. Drilling has indicated that there are numerous veins but the spatial distribution and orientation of holes drilled to date, coupled with the complicating effects of surface erosion that has removed the apical portion of many of these veins, leaving only the limbs, makes confident interpretation and correlation of these many veins, both within a given vertical section, and between sections along strike extremely difficult. As a consequence, many of the veins for which there is drill evidence have not been modelled because that evidence is insufficient. Those veins that have been modelled are a combination of anticlinal and synclinal fold crests and limbs. The anticlinal crests can be up to several meters thick; most limbs are on average, less than one meter thick and drill data suggests that the limbs attenuate away from the fold crests. Because of their superposition and close proximity, it is difficult to demonstrate the distribution of all 14 vein domains that have been used in this resource estimate;

Figure 14-31 shows the three main vein domains (S1, S2a and S3) in plan view; Figure 14-32 shows the same three veins in perspective, looking slightly north of west.

Figure 14-31: Plan View Veins S1, S2a and S3.

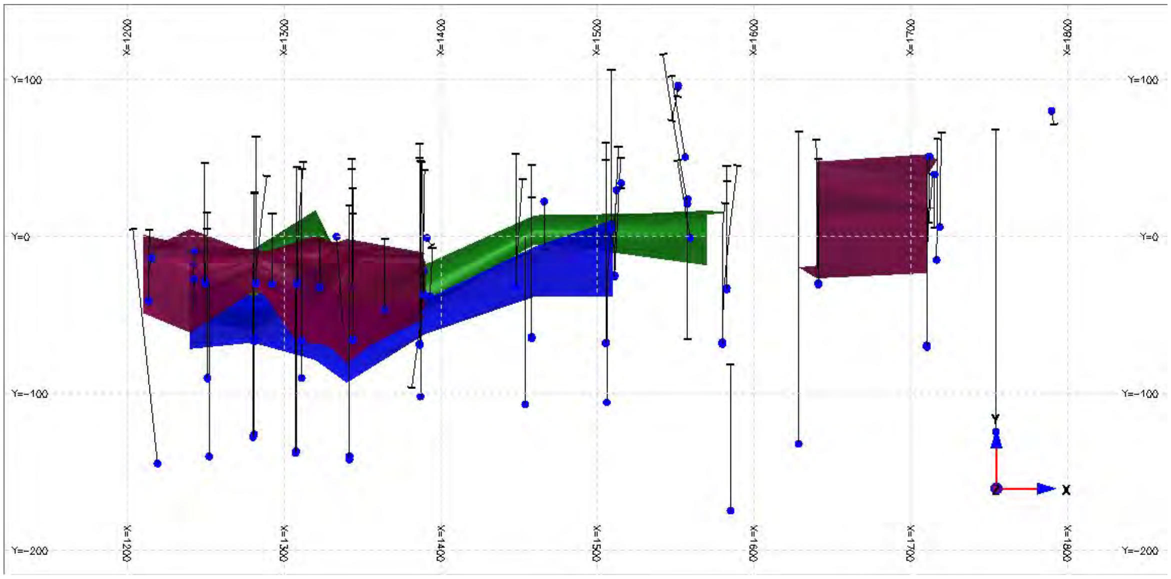
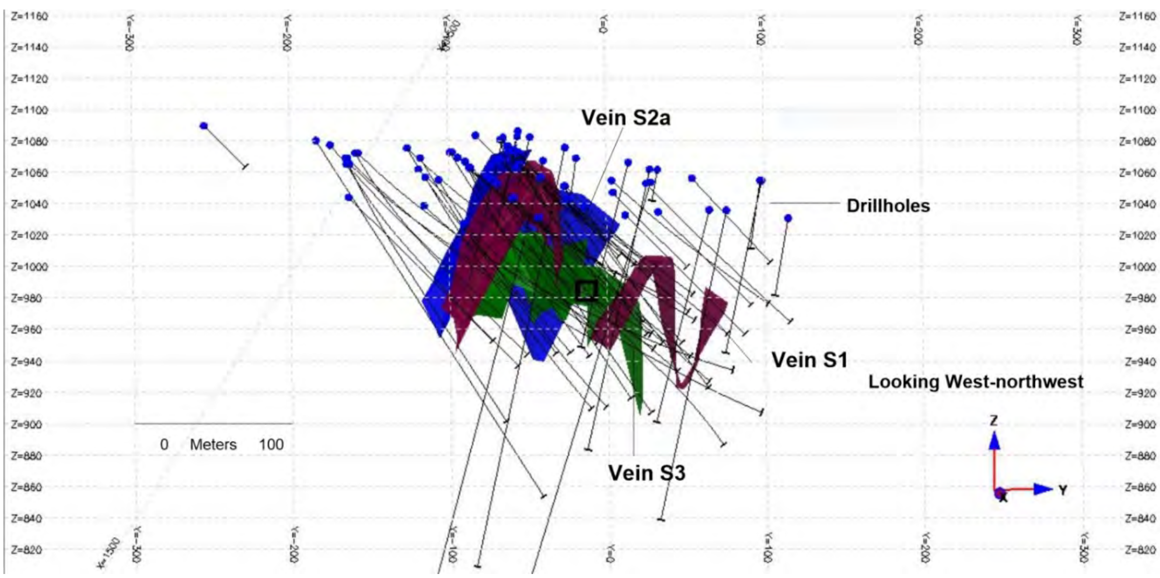


Figure 14-32: Perspective View Veins S1, S2a and S3 Looking West-northwest



14.2.4 Spatial Analysis

Composites are not distributed equally among the modelled veins: three of the modelled veins contain 63% of the composites and vein (S1) contains 230 composites (27%), the most of any of the veins. The remaining eleven vein domains contain between 67 and 3 composites each. The composites in Vein S1 were used to model the variography and those parameters were applied to all 14 vein domains. The variographic analysis was carried out using Sage 2001 software. A spherical model with two structures was used. Results are presented in Table 14-16.

Table 14-16: West Dufferin Variography

DOMAIN	ELEMENT	NUGGET	C1	FIRST STRUCTURE ROTATION (°)			FIRST STRUCTURE RANGE (m)		
Vein C_1	Gold	0.727	0.254	40	-43	52	2	22	37
			C2	SECOND STRUCTURE ROTATION (°)			SECOND STRUCTURE RANGE (m)		
			0.018	58	23	53	100	62	6

A search ellipse was constructed to conform to the assumed easterly plunge of the veins, and to the drillhole spacing to ensure that the search ellipse was long enough to span at least two fences of drillholes.

Table 14-17: West Dufferin Search Ellipse

DOMAIN	AZIMUTH (°)	DIP (°)	SPIN (°)	MAJOR (m)	MEDIAN (m)	MINOR (m)
Vein C_1	90	-5	0	75	50	50

14.2.5 Block Model

Block Model parameters are presented in Table 14-18.

Table 14-18: West Dufferin Block Model

Block Model Origin*		Block Discretization	
Origin X	1100	X	2
Origin Y	-200	Y	2
Origin Z	700	Z	2
Block Size (m)		Number	
x	10	Columns	71
y	2.5	Rows	161
z	2.5	Levels	161
* Block Centroid		Rotation: No rotation	

14.2.6 Interpolation Plan

The resource was estimated using SGS Geostat Genesis software. Blocks were interpolated in a single pass using Ordinary Kriging (OK). In order for a grade to be interpolated into a block it was necessary that at least two composites were located within the volume of the search ellipse. A minimum of two and maximum of 12 composites were allowed, with a maximum of one composite allowed from a single drillhole. These constraints mean that a minimum of two drillholes were necessary for a grade to be interpolated, and a maximum of 12 drillholes were allowed to support that estimate.

Estimates for both capped and uncapped gold were made and are presented in Section 14.7. It should be noted that, as only one assay was capped for the subset of assay values used, the capped and uncapped grades are essentially the same. Grades were also estimated using Inverse Distance Squared (ID^2) as a check on the OK results as the two outcomes are generally similar.

The number of samples and the average distance of those samples to the block centroid, for estimates within the three principal veins, S1, S2a, and S3, that contain the largest number of composites (63%) and the largest tonnage of resource (79%) are summarized in Table 14-19. The average number of composites indicates that the estimate is in general well-supported and that in general supporting composites were drawn from two adjacent fences of drillholes.

Table 14-19: Average Number of Composites Used and Average Distance to Block Centroid

Domain	Average # Composites Used	Average Distance of Composites (m)
S_1	7	46
S_2a	5	45
S_3	6	43

14.2.7 Mineral Resource Classification

All resources that have been estimated for West Dufferin are classified as Inferred. This classification is based on the fact that the majority of samples used for the estimate come from drill programs conducted prior to 2000 and no core remains from any of these holes with which to verify the assay results obtained from them, and secondly, there is no QA/QC data for any of the assays, including the latest (2010) holes. A third, although lesser, deficiency is the absence of any bulk density data, but given that the host rock for the mineralization is quartz vein, an assumed average value of 2.65 g/cm^3 is not likely to be significantly in error.

14.2.8 Mineral Resource Tabulation

The West Dufferin resource was estimated at a range of cut-off grades for capped gold grades; these are summarized in Table 14-20. The resource at a capped gold cut-off grade of 2 g/tonne is taken as the base case. Gold grades have been rounded to the nearest 0.1 grams; tonnes, short tons and Troy ounces have been rounded to the nearest 100.

Table 14-20: West Dufferin Inferred Resource Summary using Ordinary Kriging

West Dufferin Resource Summary Ordinary Kriging					
Cut-off Au_Capped g/tonne	**Au_Cap g/tonne**	**Au Uncap g/tonne**	**Tonnes**	**Troy oz Cap**	**Troy oz Uncap**
Au_Cap 5 g/tonne	11.19	11.24	107,500	38,700	38,800
Au_Cap 4 g/tonne	10.46	10.50	120,700	40,600	40,800
Au_Cap 3 g/tonne	9.39	9.42	142,500	43,000	43,100
Au_Cap 2 g/tonne	6.13	6.15	269,800	53,200	53,300
Au_Cap 1 g/tonne	4.52	4.53	411,500	59,800	59,900

Conversion: Grams to Troy ounces = grams*0.0321507

The OK resource estimate for each vein domain at a cut-off grade of 2 g/tonne for capped gold is presented in Table 14-21. Statistics for the uncapped gold resource are omitted because they are the same as those for the capped grades. It should be noted that there was insufficient data to obtain estimates for two zones and three zones did not contain any resource at a lower threshold of 2 g/tonne. Gold grades have been rounded to the nearest 0.1 grams; tonnes, short tons and Troy ounces have been rounded to the nearest 100.

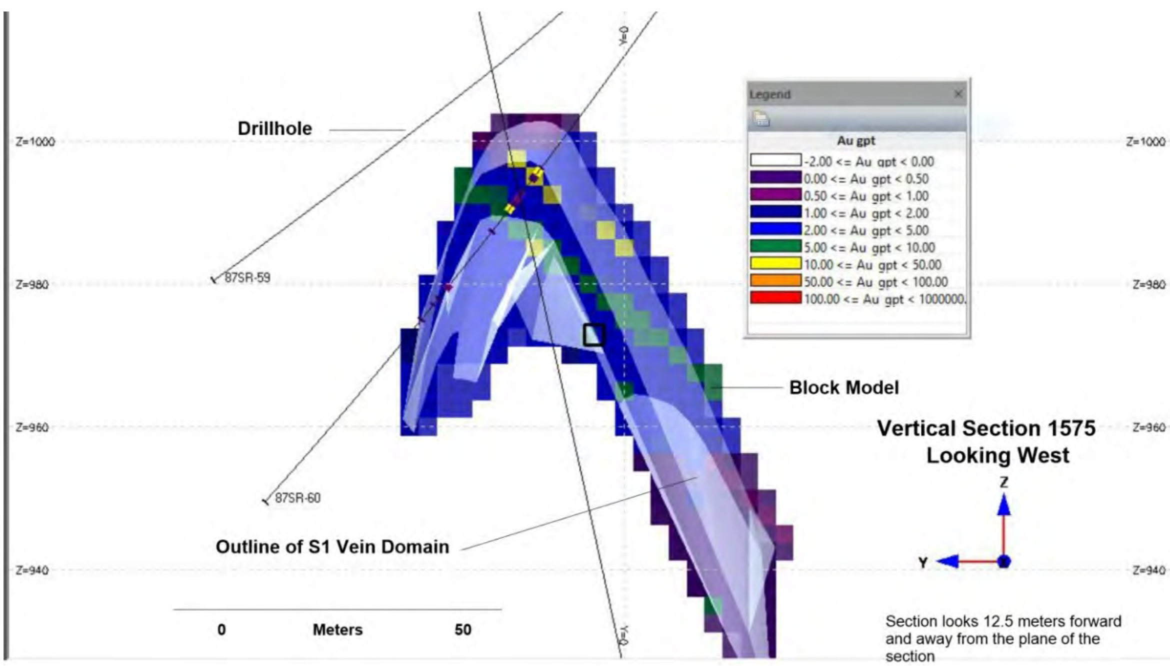
Table 14-21: West Dufferin Ordinary Kriging Inferred Resource at Cut-off of 2 g/tonne Capped Au

Domain	Au_Cap g/tonne	Au Uncap g/tonne	Tonnes	Troy oz Cap	Troy oz Uncap
Vein S_1	21.2	21.5	14,600	10,000	10,100
Vein S_2	2.2	2.2	2,800	200	200
Vein S_2a	4.9	4.9	193,600	30,500	30,500
Vein S_3	4.7	4.7	2,300	400	400
Vein S_5	11.0	11.0	5,600	2,000	2,000
Vein S_5a	2.8	2.8	34,100	3,100	3,100
Vein S_6a	2.8	2.8	3,300	300	300
Vein S_7	16.3	16.3	13,400	7,000	7,000
Average / Total	**6.1**	**6.1**	**269,800**	**53,500**	**53,600**

14.2.9 Block Model Validation

The block model was validated in three ways: 1) visually, by comparing the block grades with underlying composites (Figure 14-33), 2) statistically by comparing arithmetic averages of composites within each domain with the corresponding OK and ID^2 results (Table 14-22), and 3) by swath plots of block grades versus composite grades in the x, y and z directions through the block model. Swath plots are presented in Figure 14-34 for the S1 domain**.**

Figure 14-33: Vertical Cross-Sectional View Vein S1 Showing Block Model and Drillhole Composites

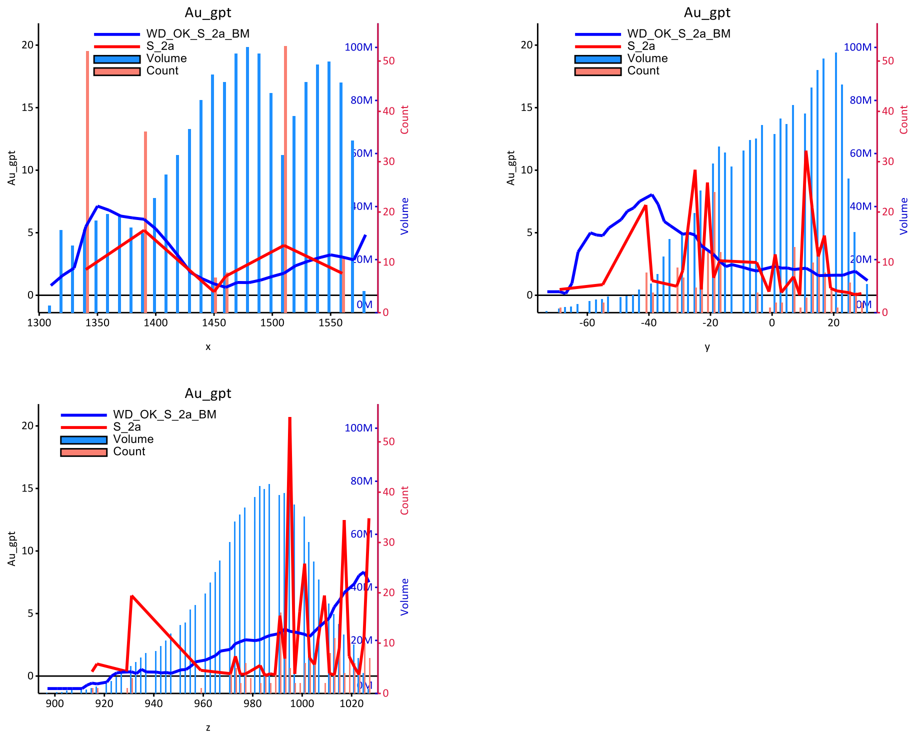


Note that Table 14.7 compares tonnes and gold grades at a zero grade cut-off. This cut-off is not meaningful in the context of "reasonable prospects for economic extraction," but is useful to compare unfiltered results of various estimation techniques and is used here for that purpose only.

Table 14-22: Comparison of Arithmetic Average, OK and ID^2 Estimates at Zero Cut-off

Arithmetic Averages			OK Estimate		ID2 Estimate	
Vein	Tonnes	Au_Cap g/tonne	Tonnes	Au_Cap g/tonne	Tonnes	Au_Cap g/tonne
S_1	270,133	1.73	217,045	1.89	217,045	1.71
S_2	60,796	1.36	51,350	1.12	51,350	1.12
S_2a	340,300	3.23	334,758	3.17	334,758	3.17
S_3	123,766	1.03	129,265	0.69	129,265	0.69
S_3_4	25,694	0.00	19,934	0.11	19,934	0.11
S_4	102,476	0.63	73,481	0.63	73,481	0.63
S_5	21,783	1.41	5,976	10.34	5,976	10.34
S_5a	55,846	1.02	52,579	1.98	52,579	1.98
S_6	39,583	0.47	Insufficient data - no estimate			
S_6a	52,412	1.39	42,006	0.71	42,006	0.71
S_7	22,708	9.04	22,513	9.98	22,513	9.98
S_8_9	17,373	0.30	13,654	0.30	13,654	0.30
S_10_11	9,680	0.23	10,255	0.23	10,255	0.23
S_12	41,703	1.31	Insufficient data - no estimate			
	1,184,253	1.93	972,815	2.49	972,815	2.43

Figure 14-34: Swath Plots Vein S1 Domain



14.2.10 Comparison With Previous Estimates

There are no previous resource estimates on the West Dufferin portion of the project.

14.3 Mineral Resources for the Entire Dufferin Property

Mineral resources for both parts of the deposit – East and West Dufferin – are presented in Table 14-23.

Table 14-23: Mineral resources for the entire Dufferin Property.

	Volume (m^3)	Tonnes	Ounces	Average Grade (g/tonne)
East Dufferin				
Indicated	57,200	151,500	58,000	11.9
Inferred	163,800	434,100	96,800	6.9
West Dufferin				
Inferred	101,800	269,800	53,200	6.1
Combined				
Indicated	57,200	151,500	58,000	11.9
Inferred	265,600	703,900	150,000	6.6

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65;
[3] Gold price = $US 1250 per ounce.
[4] West Dufferin top-cut: 100 g/tonne; East Dufferin top-cut: 200 g/tonne.

15 Mineral Reserve Estimates

No mineral reserves are being reported in this document.

16 Mining Methods

This section outlines proposed mining methods for the Dufferin Property.

> *Underground mining is a dynamic art and science with a diverse set of design, production, and economic criteria that must be considered before selecting an optimal method to extract the mineralised rock. While all methods can be technically classified as either self-supported, supported, or caving, the multitude of subsystems and existing variations bear witness to the fact that each ore body is unique and, as such, justifies an individual approach to optimize mineral extraction. In this sense, underground mining is an art. The ability to evaluate the unique characteristics of each ore body and utilize sound engineering principles to design the optimal system constitutes effective method selection. (Orr, S.A., Mining Engineering Handbook, p. 1837)*

16.7 Selection of Mining Method

16.7.1 Overview

Dufferin's saddle veins are stacked around the crest of the Crown Reserve Anticline. The anticline strikes northeast-southwest (about 70° azimuth) and plunges about 7° (12.5%) to the east.

Diamond drilling would be done from the underground workings to fill in gaps in the data and to better define the geometry of the limbs.

The crest area of the saddle vein typically measures about 10m across. This area would be mined by drifting and slashing. A thick vein at the crest would be mined in lifts from the top down. The centre portion of a wide crest area should be supported by backfill or long bolts (width divided by 3).

The south limb dips approximately 45° to 55° to the south while the north limb dips approximately 57° to 75° north. The crest area can be 1m to 5m thick. Leg areas with a dip of less than the angle of repose of about 45° will require mechanical assistance to move the rock downhill.

Diamond drilling will be used to determine the structure of the saddles. The experience in the Australian Bendigo mines was that the grade could be determined after bulk sampling 20 m of the vein. When the leg vein geometry has been determined by underground diamond drilling, a bottom sill and raise are excavated in the vein. A vein width greater than 1 metre can be mined using a longhole drill. A vein with a width less than 1 m could be mined using the thermal fragmentation method. The fold limb veins are generally thin, less than 1m true thickness, and can extend between 10m and 30m down dip south and north. Both the vein and waste walls are generally strong.

Underground workings exist to 100m below surface. These workings are currently dewatered to Saddle 4 at an elevation of 944 m. The mine datum is Mean Sea Level + 1000 metres. Most of the mine has been rehabilitated, but there are issues with several key infrastructure areas that should be addressed prior to restarting operations.

16.7.2 Geological and Geometric Considerations

The Dufferin deposit presents an interesting challenge when attempting to determine the type of mining method to use to exploit the deposit. From diamond drilling and past production, it appears that the main mineralized portions of the anticline structure are concentrated immediately (10m-20m) around the crest (saddle). Given the spacing between the crests, saddles must be mined separately, some using non-mechanized methods. Mining methods for the deposit are further complicated by the fact that the deposit has a plunge of approximately 8 degrees to the East.

There are several different mining methods that could be used to exploit both the crests and the limbs of the Dufferin deposit. The crests have a relatively shallow (or no) plunge, and can be mined using drifting, benching, and slashing walls. The limbs, which are much steeper in dip (60-70 degrees, although some as shallow as 45 degrees. The limbs are 10-25m in height and can be mined using a narrow vein mining method such as shrinkage, cut and fill, long hole, or an open stoping method.

The mining method should minimize dilution from the walls of the vein. Prior to mining, the vein should be outlined by underground diamond drilling. Once the vein is clearly defined, a cross cut can be driven to both the north and south limbs of the saddle vein.

The final choice of the mining method will depend upon the dip and continuity of the limb veins. A very irregular leg vein would be a challenge for longhole mining or thermal fragmentation. In the case of a rich but patchy mineralization, cut and fill should be considered as the method is selective but labour intensive.

Stull mining and take down back mining can be used if the mineralized leg has less that 15m vertical extent.

All narrow vein mining system requires a lower sill level which defines the base of the stope to be mined.

From this sill level a raise is driven in the vein to the crest area of the saddle vein. The sill, raise and the crest mining outlines the leg vein stope on three sides.

Diamond drilling and the development workings will provide the engineer with enough information to decide how and if to mine the area.

Crest mining will be completed prior to mining the limbs. Longhole drilling would be completed from the saddle crest to the bottom sill. A straight vein could be blasted out two or three rings at a time. The blast holes would be surveyed top and bottom. Some holes may have to be re-drilled to achieve proper fragmentation and minimal dilution from the walls.

If the leg vein is straight and high grade, thermal fragmentation (REF) should be seriously considered.

16.7.3 Mass Mining

After careful examination of the deposit, considering the geometry and spacing of the veins, it was decided to continue with a high grade underground mining method for the PEA, as opposed to a low grade mass mining option. By mass mining, we mean taking all rock within the crest and then sorting this rock so 95% of the gold is captured in 40% of the muck. The waste rock stays underground as backfill. At this time there is insufficient data to make any prediction on the results of pre-concentration. A test program is recommended to determine what sorting technology might work at Dufferin.

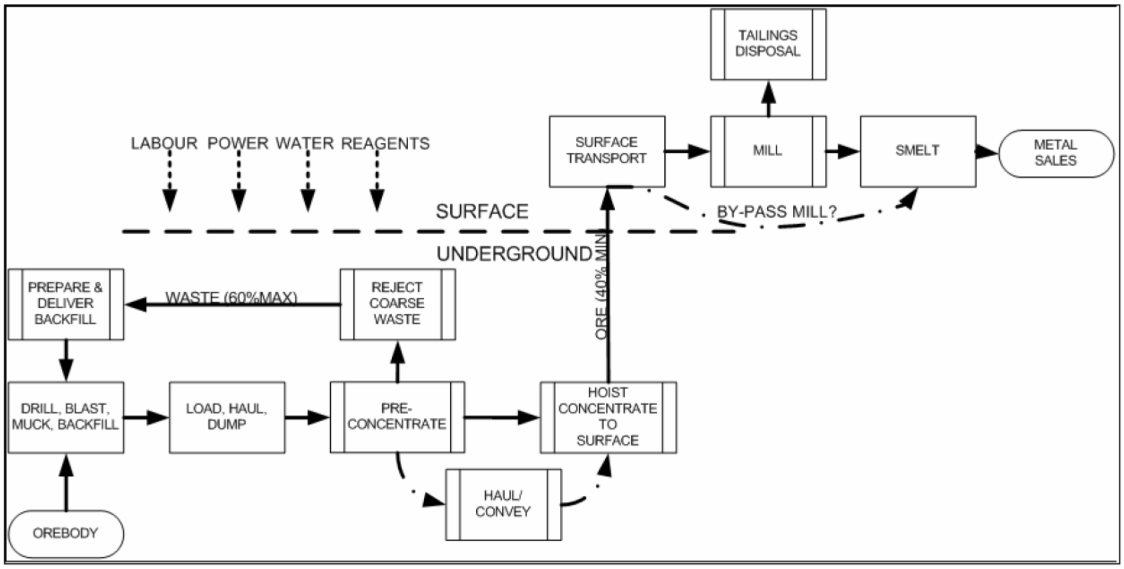


Figure 16-1: A flow chart for mass mining, integrating an underground pre-concentration plant (after Figure 4, Bamber et.al. CIMM Oct 2004).

16.7.4 Narrow vein mining

Narrow vein mining is used to describe mining widths of 2 metres or less. The Dufferin deposit is a saddle vein deposit hosted by metasediments that have been folded into anticlines and synclines. Quartz veins draped over the anticlines have a saddle shape. The crest area of the fold is thicker than the limbs, which pinch out down dip, parallel to the bedding. This type of deposit was mined in the Bendigo gold fields of Australia between 1851 and 1954 (Laidlaw, 1993).

Saddle vein mining in the Bendigo goldfields of Australia outlined the saddles by crosscutting every 30m from a vertical shaft. When a vein was encountered by the level crosscut, the miners would drive a raise on the vein to the top of the saddle. A mining width of 60 centimetres (2 feet) was common on the limbs. The rock was sorted underground so waste rock was left underground in old stopes where possible.

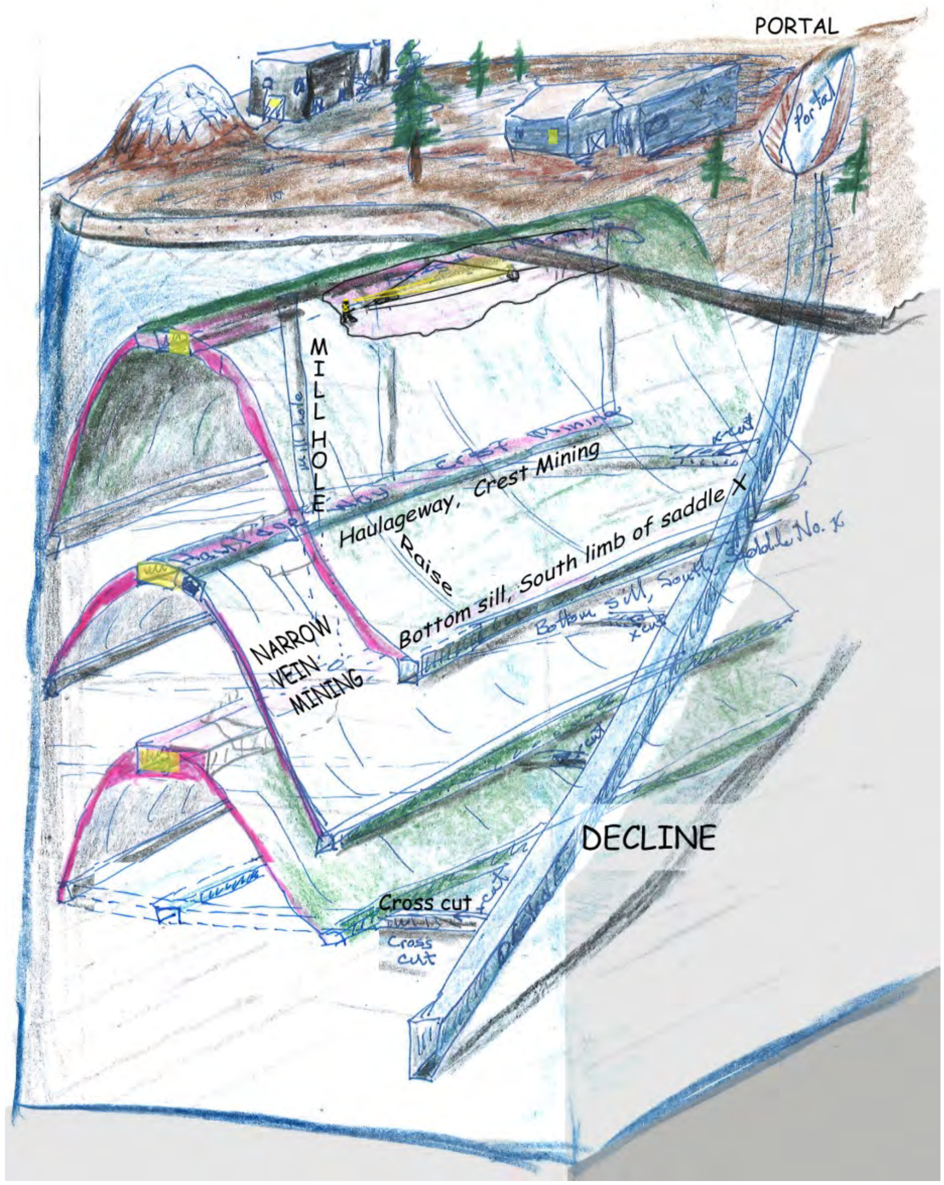


Figure 16-2: An illustration of the proposed mining method.

The success of the narrow vein mining will be determined by how consistent the geometry of the vein is between levels and along strike. If the strike and dip continuity is consistent for 10m or more down dip and along strike, stope mining widths can be less than one metre. If the vein geometry is not consistent then a cut and fill method might be used.

At Dufferin, the vein limbs pinch out down dip. Diamond drilling can help define the base of the limb vein, then a crosscut can be driven from the decline and the base of the limb defined with a bottom sill. The sill would be driven along strike at the same angle as the saddle crest's plunge. From the sill, a raise would be driven every 25m to the top of the saddle. Approximately 25 cm of waste would be mined adjacent to the top sill so that the longhole drill can be set up to drill off the hanging wall of the vein. Once the geometry of the limb is known mining can commence.

16.7.5 Crest Area Mining

The mining method chosen for the crest area is drift and slash. The decline access would continue to be advanced to a depth of about 400m or more if additional saddles are outlined at depth.

From the decline, haulage drifts will be excavated beneath sets of saddle veins (eg, saddle 2 - saddle 6) at the same angle as the plunge of the saddle, for the full length of the deposit. From the haulage drift, cross cuts will be driven as described above. From the raise driven up from the bottom sill, two rounds are driven on each side of the raise, in the crest area, to provide room for an electric slusher setup. Once the slusher is set up, mining proceeds in the crest area of the saddle, working the ground for a distance of about 50m along strike in each direction. Mineralized rock would be pulled to a mill hole or to a compartment in the raise if the dip is steep enough for the rock to flow.

16.7.6 Limb Mining

In the Bendigo mines, limbs were traditionally mined using shrinkage mining (overhand and underhand), cut-and-fill mining or resue mining (Laidlaw, 1993). These methods are labour intensive and dilution is high since the mined width has to be sufficient for people, at least 60cm. A method that does not require persons in the stope is preferred. Two methods have been considered, longhole mining and thermal fragmentation.

Longhole mining is well known, while underground mining using thermal fragmentation is a relatively new technology developed in Canada in the late 1990s using Russian open pit mine drilling technology.

16.7.7 Longhole Stoping

With longhole mining, accurate drilling and a high powder factor are required to pulverize the rock so it will flow easier. Drill holes are surveyed top and bottom and if necessary, re-drilled to ensure the entire vein is mined. The vein would be mined by blasting several times along the strike of the vein and mucking out the blasted rock between blasts.

16.7.8 Thermal Fragmentation

With thermal fragmentation a longhole drill drills a pilot hole for a burner head which is inserted into the hole and slowly hoisted out to enlarge the cavity to the limits of the vein. Quartz spalls at a different rate than some wall rocks so dilution can be zero if the vein is 30cm or more wide.

Thermal fragmentation involves drilling a hole into which heated air is injected at very high temperatures. The sudden change in air temperature causes stress in the rock causing it to spall from the walls. The rock fragments are collected by air suction and from the bottom of the hole. A wooden plug is fitted into the pilot hole breakthrough on the bottom sill. Once the hole has been enlarged to the desired diameter, the wooden plug is removed and the cuttings are collected. In this way, the process can recover virtually just the mineralised rock, without generating waste rock.

Thermal fragmentation has been used in mining for many centuries. Agricola described the thermal fragmentation process in *De Re Metallica*:

> "Even if a vein is a very wide one, as tin veins usually are, miners excavate into the small streaks, and into those hollows they put dry wood and place amongst them at frequent intervals sticks, all sides of which are shaved down fan-shaped, which easily take light, and when once they have taken fire communicate it to the other bundles of wood, which easily ignite."

A—Kindled logs. B—Sticks shaved down fan-shaped. C—Tunnel.

> "While the heated veins and rock are giving forth a foetid vapour and the shafts or tunnels are emitting fumes, the miners and other workmen do not go down in the mines lest the stench affect their health or actually kill them, as I will explain in greater detail when I come to speak of the evils which affect

> miners. The Berg1'lleister, in order to prevent workmen from being suffocated, gives no one permission to break veins or rock by fire in shafts or tunnels where it is possible for the poisonous vapour and smoke to permeate the veins or stringers and pass through into the neighbouring mines, which have no hard veins or rock. As for that part of a vein or the surface of the rock which the fire has separated from the remaining mass, if it is overhead, the miners dislodge it with a crowbar, or …" (Agricola, G. , 1556, Book IV).

Thermal fragmentation was used to break rock in ancient times and up to the introduction of explosives in mines[14]. Jet Piercing, which was used in Minnesota's taconite iron mines after 1947, used a rocket type burner to drill blast holes. The method was dangerous and holes drilled had a variable diameter which created problems with blasting. The method fell into disuse in North American mines but the idea was further developed in Russia where they used a pilot hole and inserted a burner using diesel fuel and air. This eliminated the use of oxygen which was expensive and difficult to handle (Poirier et.al., 2002).

The advantages to the method include: less dilution, lower transportation and processing costs, less explosive use and less tailings. In addition, the Bond Work Index[15] for grinding is lowered by the thermal stress applied to the rock.

Stoping requires the same development as other narrow-vein mining methods. A sub-level bottom sill drift would be established and then a pilot hole is drilled with a longhole drill. The hole is enlarged using the Nippon Dragon thermal fragmentation unit. The Nippon Dragon is designed to operate in a drift no smaller than 1.8 m wide by 2.8 m high.

Brisebois et.al (2010) presented a CIMM paper describing a comparison between the mining using thermal fragmentation and shrinkage stoping. A six inch (152mm) pilot hole was used for the thermal fragmentation and a mining width of 1.8m used for the shrinkage. The result was the thermal fragmentation method mined 75% less rock in less time, for the same amount of gold. Thermal fragmentation work was supported by CANMET who compared the method with longhole stoping of a 30cm wide vein. The results were reported by Poirier at a CIM AGM in Vancouver in 2002 and are reproduced below, inflated to 2016 dollars.

A three person crew is used per shift – one operator, 1 helper and 1 longhole driller. Two weeks training is required for the operators. The unit consumes 50 litres of fuel per hour and produces between 3 tonnes per hour and six tonnes per hour. Holes are drilled into the vein. Nippon has recently developed a unit for a 4-1/2 inch hole.

[14] Gunpowder for blasting was introduced in 1627 but fire-setting continued for nearly 75 years afterward. With fire-setting, an advance between 5 and 20 feet per month was achieved. (Footnote in Translation of Georgious Agricola's De Re Metallica, 1556 by Hebert Clark Hoover and Lou Henry Hoover 1950, page 119, Book IV)

[15] Bond Work Index is used to estimate the power required for grinding a material, in kW/tonne.

Table 16-1: Brisebois Comparison between Thermal Fragmentation and Shrinkage (Brisebois, 2010).

Tonnage Calculation (40 m by 20 m Block)	Thermal Fragmentation	Shrinkage
Width in situ (m)	0.5	0.5
Mining Width – Final Result	0.5	1.8
Planned Dilution	0%	260%
Height (m)	20	20
Length (m)	40	40
Density	2.8	2.8
Total stope tonnage (t)	1120	4032

Figure 16-3: Nippon Dragon patented Thermal Fragmentation unit displayed in November 2014 (photo from Nippon Dragon Resources Press Release).

A paper presented by Sylvie Poirier, Jean-Marie Fecteau and Marcel Laflamme of Natural Resources Canada (CANMET) and Donald Brisebois of Rocmec describes the technology and the history of thermal fragmentation in mining and reports on the changes to the equipment to adapt it the underground environment. They summarize the tests undertaken on surface and underground since 1995. The test for narrow vein mining extraction analysed a total of 29 holes with depths from 6 to 22 metres were reamed. From a pilot hole, the holes were reamed to diameters which varied from 254 mm to 455 mm.

“An oval hole was obtained in a quartz vein contained within an andesite wall. The quartz proved to have better spalling properties than the andesite and broke more easily. From the data gathered following the last series of surface tests, a preliminary underground year 2000 mining cost of $113/tonne ($182/t in 2016) has been estimated. This amount is used in the simulation shown in Table 3 which compares the thermal fragmentation mining concept and the long-hole mining method for the extraction of a 30-cm vein."

Table 16-2: Productivity comparison between thermal fragmentation and shrinkage methods.

Tonnage Calculation (40 m by 20 m Block)	Thermal Fragmentation	Shrinkage
Width in situ (m)	0.5	0.5
Mining Width – Final Result	0.5	1.8
Planned Dilution	0%	260%
Height (m)	20	20
Length (m)	40	40
Density	2.8	2.8
Total stope tonnage (t)	1120	4032
Number of Personnel	2	2
Productivity per 12 hrs Shift (t)	30	60
Tonnes Extracted per 24 hrs	60	120
Days Required to Extract Block	18.7	33.6

Table 16-3: Cost comparison between thermal fragmentation and longhole methods.

Estimate cost comparison between underground thermal fragmentation and long hole. (Poirier S. et.al., 2002, Table 3, Modified by inflating costs by 1.6)				
	Thermal Drilling Units 1960 tonnes		Long-hole Drilling units, 14,600 tonnes	
Construction cost 2016/2000	Total Cost $ in 2016$		Total Cost $ in 2016$	
Development				
Drifts	1600	$288,000	1600	$288,000
Subdrifts	1600	$192,000	1600	$192,000
Raises	1600	$96,000	1600	$192,000
Drawpoint			1600	$96,000
Mining cost ($/t)	$181.60	$355,936	$30.40	$443,840
Mucking	$12.80	$25,088	$6.40	$93,440
Transportation	$19.20	$37,632	$9.60	$140,160
Milling	$25.60	$50,176	$32.00	$467,200
Environment	$3.20	$6,272	$3.20	$46,720
Backfilling			$8.00	$116,800
Total		**$1,051,104**		**$1,959,360**
$ per tonne		$536.3		$134
Grams		62611		68365
Ounces		2013		2198
Cdn$ per ounce		$522.2		$891.43
US$/ounce ($C1.33/US$)		$391.62		$668.57

16.7.9 Mine Mobile Equipment

Mobile mining equipment can be powered by diesel, battery or trailing cable electric motors and by compressed air. Diesel equipment has been used in this PEA as it is readily available. The down side of diesel is the air quality, even with the EPA Tier 3 diesel engine which impacts ventilation cost. Electric powered machines are becoming better with longer life, more powerful batteries and should be considered for the long term. More mining operations are converting to the use of battery powered mobile equipment because of the improved working environment, reduced ventilation requirements and lower operating costs of the equipment. Electric motors last longer and mine ventilation requirements are much less than with diesel. Air quality would be better.

16.8 Existing and Proposed Development

Access to the deposit is via a decline which is driven in waste. The decline typically measures 5-6 metres wide by 3.5-4.0 metres high and descends at an average slope of 8°, about 14 metres vertical for every 100 horizontal metres. The decline is approximately 1,300 metres long, descending with varying slope to 100 vertical metres to access Saddles 1-4. Stoping (mining) was carried out on Saddles 1-3, but those saddles were not completely “mined out.” A limited amount of mining was carried out on Saddle 4.

A production/ventilation shaft should be seriously considered if the resources are found below a depth of 300 metres.

Fold limb (leg) mining will require a bottom sill, top sill and a raise that can be used as an ore pass. The mining sequence will include definition drilling to outline the veins. This will be followed by horizontal sills and an Alimak raise or a raise bored mill hole.

Both the bottom sill and the top of the leg vein would be developed in mineralization – in the case of multiple stacked veins, the sequence would be from the bottom up.

The crest areas can be mined using standard rounds to advance down the plunge of the anticline. A wide area would have the ability to take a round and two slashes. The opening at the top of the north and south legs would have to be high enough to accommodate a long-hole drill.

Resue mining using a blasting plan that employs millisecond delays to sort waste and ore grade material as illustrated in Figure 16-4. They report that “Orica Mining Services eDev™ Electronic Blasting System for Tunnelling, the waste is thrown up to 40 m from the face, while majority of the ore is left 5 - 10 m from the face, effectively segregating ore and waste” (Hamilton and Degay, 2011).

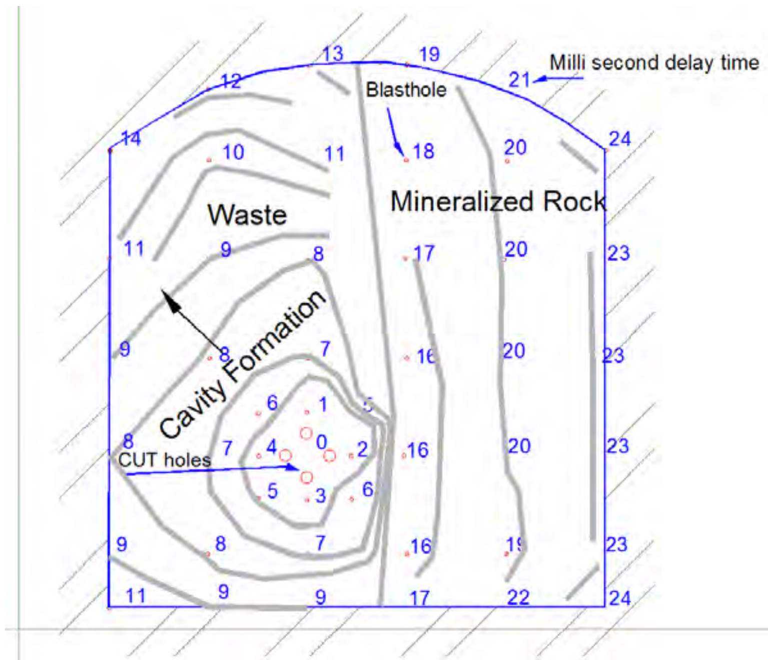


Figure 16-4: Resue blasting pattern. The waste is initiated first and the vein second. The waste is thrown away from the face and vein material is kept closer to the face. (Modified after Hamilton and Degay 2011)

Each year the decline should be advanced in waste about 35m vertical metres. This means about 220 m decline advance per year with the -15% ramp. Another 50m should be added for remucks, manholes and substation bays.

Cross Cuts to the base of the leg would also be in waste. Mill holes and ore passes and ventilation raises would be as well so we will assume that approximately 100,000 tonnes of waste development is required per year. The total rock broken would approximately 80,000 m^3 per year. This would require approximately 240 tonnes of ANFO explosives.

Table 16-4 - Mining Activity Prior to 2014.

Segment	Approximate Length (m)
Ramp	1,300
Ventilation Raises	100
1st Saddle Stope (Level 1)	145
1st Saddle Stope (Level 2)	94
2nd Saddle Stope (Level 3)	213
2nd Saddle Stope (Level 4)	113
2nd Saddle Stope (Level 6)	138
3rd Saddle Stope (Level 5)	150
Total metres	**2,253**

16.9 Historical Mining Method

Historically, the preferred method was to drift along the crest of the saddle. At some point along strike, a ramp was driven in waste back to the north, in a counter-clockwise direction, to access the saddle below.

This method seems attractive because almost all development is in ore, and thus provides immediate production to generate revenue at the start of the project. However, this method has a slow rate of production, as only 1 or 2 faces are available per saddle. Also, the main ramp and haulage way run through a portion of the saddle crest. This delays development of access to the bottom of the mine, and it sterilizes the leg portion of the saddle within the main ramp, which can only later be mined at the end of the mine life when the ramp is no longer needed.

In past mining operations, some areas were large enough to warrant benching the floor to access either the bottom of the crest or the leg(s). This was done in areas outside the main ramp, and have since been backfilled with waste rock. There were also three areas where drifting along the bottom of the leg of the saddle was done. One of these leg drifts was also used to experiment using the longhole mining method on the leg portion of the saddle.

During 2014, Ressources Appalaches completed approximately 1,060 m of drifting along saddle veins and also completed a limited amount of leg mining. The total volume excavated was approximately 17,200 m^3 (45,600 tonnes).

Table 16-5 - Drift Mining during the period May-November 2014.

	Drifting					
	from	to	metres	Face Area	volume	tonnes
Saddle 1	2215	2630	415	15	6225	16,496
Saddle 2	2265	2685	420	15	6300	16,695
Saddle 3	2305	2470	165	15	2475	6,559
Saddle 4	2250	2310	60	15	900	2,385
Totals			1,060		15,900	42,135

Table 16-6 - Leg Mining during the period May-November 2014.

	Leg Mining					
	from	to	metres	Face Area	volume	tonnes
Saddle 1	2455	2465	10	12	120	318
Saddle 2	2300	2400	100	12	1200	3180
Saddle 3						
Saddle 4						
Totals			110		1,320	3,498

16.10 Proposed Mining Method

A combination of mining methods was selected for the purpose of this study.

The general idea of the proposed plan is to drive the ramp down in the waste rock and mine the saddles from the bottom up. Main levels would be accessed off the decline at vertical intervals ranging from 50 m-100 m, targeted at the bottom of a group (or panel) of saddles. These levels would contain a main haulage drift that would run within the saddle along strike, providing access for multiple stopes to be in production at one time through a series of crosscuts and raises.

From the haulage drifts, crosscuts would be driven to intersect the bottom of the limb of the saddle at approximately 100 m intervals along strike. From the crosscut, an alimak or "open" raise would be driven up to provide access to the saddles for that level (2-4 saddles depending on the location). This raise would be used as an orepass and/or mill hole to deliver production muck to the haulage level below.

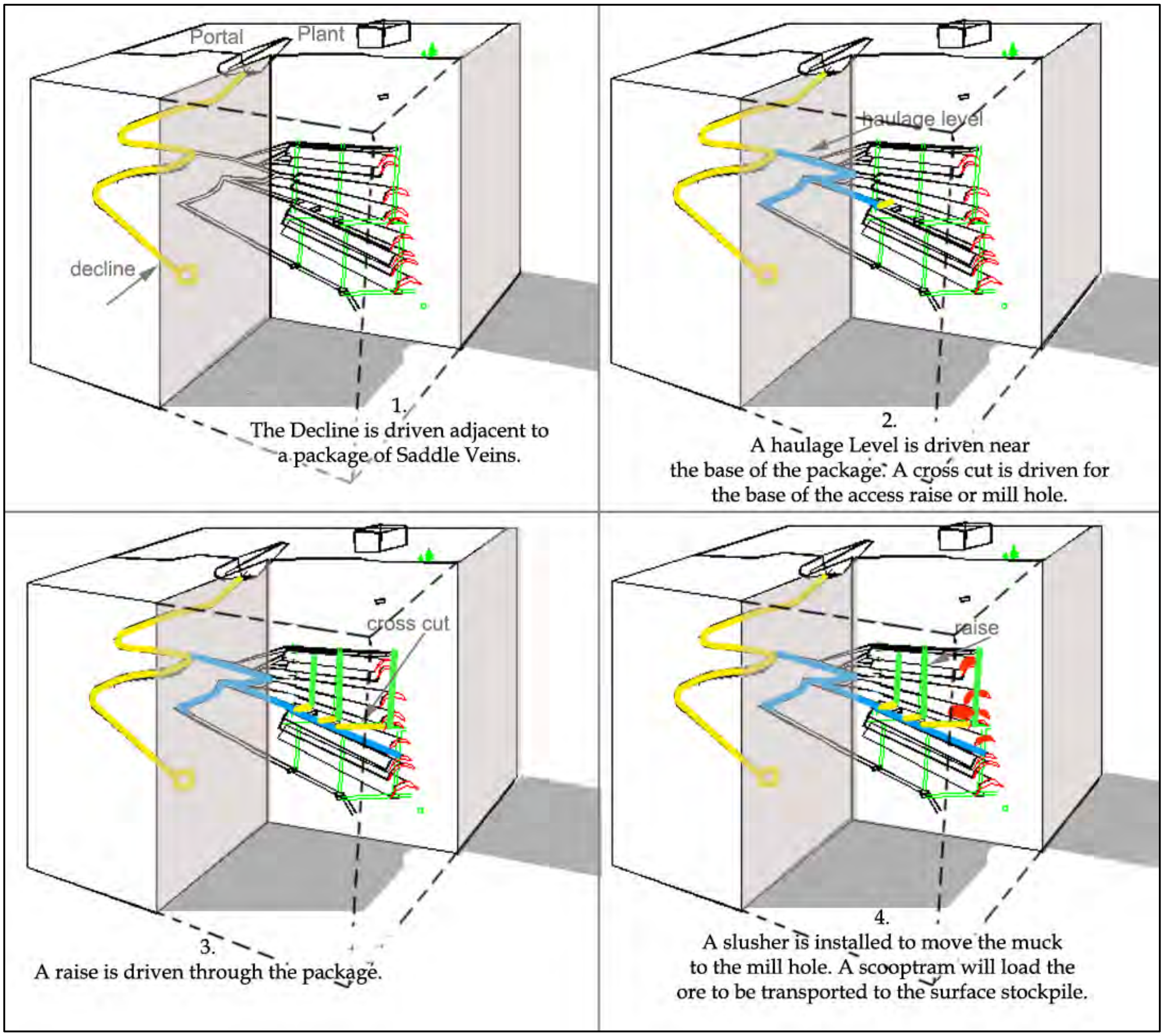


Figure 16-5 – A step by step illustration of access to the mining stopes.

16.10.1 Production Rate

A nominal processing rate of 100,000 tonnes per annum is anticipated. To provide this tonnage approximately 40,000 m^3 of mineralised rock would be delivered to the stockpile. The mill would work 2 shifts per day each treating 150 tonnes. One shift a week would be used for maintenance, so there would be 678 working shifts ((365 x 2) – 52) available.

Resources were estimated to be approximately 3,000 tonnes per vertical metre (1,150 m^3). Approximately 30% (350m^3) of the resource is in the crest and 70% (800m^3) in the legs.

To deliver 100,000 tonnes to the plant, a vertical distance of approximately 35 metres has to be mined each year. If the strike length of the mineralization is increased then the vertical distance to be mined would be less.

16.10.2 Sorting

Sorting of the material prior to, or after removal from the mine is a possibility. There are many methods that might work (optical, electromagnetic, gravity, x-ray) that could reduce the tonnage and increase the grade of material sent to the mill.

16.11 Potentially Recoverable Resources

The author decided to use a resource cut-off grade of 2 g/tonne to identify and outline potentially mineable stopes, with a goal to have a minimum target mining grade of 3 g/tonne. A grade and tonnage for each 10 m section was calculated for all blocks greater than 2 g/tonne. Where the resulting average wireframe grade was less than 3 g/tonne, the grade and tonnage used for that 10 m section was from blocks greater than 3 g/tonne, if available. If there were no blocks greater than 3 g/tonne, the 2 g/tonne cut value was used. The 2 g/tonne cut was used for the vast majority of the stope wireframes.

Stopes were created by combining 10 m sections of each saddle so that each stope was approximately 100 m in strike length, with flexibility allowed for slightly longer or shorter strike lengths depending on the grade. With the plan to access the stope in the approximate middle of the strike length for each stope, a contiguous group of 10 m sections was created for each potential stope. This caused some lower grade resources, on the fringes of each stope, to be left out of stope plans.

16.12 Unplanned Mining Dilution and Recovery

It is expected that during regular mining operations there will be a small quantity of unplanned dilution, mainly caused by blasting overbreak or peeling hanging walls. Overbreak caused due to blast damage can be minimized by utilizing perimeter blasting techniques. A separate unplanned dilution was calculated for saddle crests and saddle limbs.

For all stope mining, dilution was assumed to be 20% of the stope tonnes at zero grade. For development in mineralization, a 2 g/tonne grade was assumed.

Limb mining is planned to have a minimum mining width of 0.5 m. Twenty percent dilution was added to stope tonnes.

A stope recovery of 85% of the resource tonnes was assumed.

16.13 Stope Mining Cut-off Grade

To consider if a particular stope would be scheduled to be mined in the production schedule, a formula was created to determine the economics of each stope. The basic form of the formula is profit = revenue - cost. Potential revenue was estimated by calculating the contained metal in the stope with the gold price and average overall recovery. The cost to mine the stope was estimated by totalling the cost to develop the stope (access raise and crest crosscut) and the total cash cost to mine the potentially mineable rock. Stopes that indicated a positive net value were scheduled to be mined; otherwise they were not scheduled to be mined.

16.14 Development Mining

16.14.1 Development Plan

A certain amount of pre-production development is required. After diamond drilling for saddle structure, cross cut is driven to intersect the vein and then establish a lower sill for the stope.

While developing the mine deeper, development to the east would also be required. In order to provide sustainable production, it is recommended a minimum of 6 stopes be available at all times. Pre-Production development includes the work to put in the decline, ventilation raises, haulage drifts, cross cuts and stope access raises.

Each group of saddles will be developed from the decline. The general plan to develop each stope is that cross cuts will be excavated from a haulage level or from the decline. From the cross cut, raises would be driven to access the group of saddles. An open raise or two-compartment Alimak raise would be driven initially. One compartment is a manway for stope access. A partition would separate this compartment and the second compartment would be used for muck removal. A slusher can be dismantled and pulled into the stope once the stope has advanced horizontally.

16.14.2 Existing Underground Infrastructure

All past mining at the Dufferin deposit was exploited using mechanized mining methods, with access to the underground via a decline ramp.

During 2014, the underground mine workings were dewatered and the mine was partially rehabilitated, although there remains some work to be done to optimize distribution of services for the mine. Mine services were rehabilitated and upgraded where necessary; including electrical, compressed air, ventilation, water supply, and water discharge. There is a small refuge station located along the ramp approximately 500m from the portal entrance.

The decline will be driven to access deeper saddles. Approximately 400 m of ramp development would be required to access the Saddle 6 haulage drift from the bottom of the existing main ramp. This section of ramp should be given a high priority. It will take approximately 3 months get to Saddle 6 Haulage Drift, as well as the first stope on the Saddle 6 Haulage Drift.

Table 16-7 - Ramp development required to reach the Saddle 6 haulage drift.

Ramp Section	**Section Length**	**Width**	**Height**	**Round Length**	**Rounds Needed**
units	*m*	*m*	*m*	*m*	*#*
Ramp S4 Sump to 924 VR	102	4	4	3	34
VR 924 Xcut	12	3	4	3	4
Ramp 924VR to Ramp S6	268	4	4	3	90
Ramp to S6 Total	**382**				**128**

16.14.3 Ventilation Raises

Ventilation is critical for the operation of any underground mine. The existing ventilation raise system is functional, although it has several inefficiencies that should be mitigated prior to restarting production. The first section of the raise from the surface is very narrow – 1.2 m in diameter – and could benefit from a larger cross section. Also, the existing ventilation raise system is not aligned very well, and has numerous jogs and constructed bulkheads. This inefficiency could be mitigated by excavating a new section of vent raise from the 1006 Level, straight to surface.

As the ramp is deepened, the ventilation system will also be extended. An additional ventilation raise would be required as the mine is developed to the east.

Table 16-8 - Ventilation Raise Details

Ventilation Raise Section	**Section Length**	**Width**	**Height**	**Round Length**	**Rounds Needed**
	m	m	m	m	#
Main System VR 924	36	3	3	2.4	15
Main System VR 865	60	3	3	2.4	25
Main System Total	**96**				**40**
East System VR 914	133	3	3	2.4	56
East System VR 909	111	3	3	2.4	47
East System Total	**244**				**103**

16.14.4 Production Shaft

A production / ventilation shaft was examined and should be considered for the deeper resources. A shaft would the most efficient method of transporting muck to the surface. When the vertical distance from surface is between 200 m and 300 m it is generally less cost to use a production shaft. This shaft would double as a ventilation shaft.

16.15 Limb Mining

Dynatec Corporation presented their approach to mining narrow veins using longwall stoping at Newmont's Ken Snyder gold Mine in Nevada. All stoping was carried out using 60mm x 15m long blastholes drilled from the upper sill drift. Geologists worked with the development crews and mapped and sampled each face. The backs of the sill were mapped to determine the stope outline. Blastholes were surveyed-in and were all drilled to break through the lower sill. Holes were drilled at about 1m spacing. Internal holes are staggered and timed to form a "V" cut, throwing the broken muck away into the void space. Three to four rings were blasted, then mucked by a remotely controlled 1.5 yard scoop. Makuch cautions to "make sure the ore vein is placed no closer than 0.5 metres to the hanging wall side of the drill drift to allow the longhole drill to be able to drill parallel and take all of the ore" (Makuch, A.P., 2001).

After development is complete within the stope, the production cycle will consist of the standard unit operations: drill and blast, mucking the material, then installation of any required ground support. Mining production would be carried out at a rate of approximately one blast per working shift.

16.16 Backfilling

For this deposit, backfilling is not generally required for ground support; however, It may be possible to place waste rock material in parts of some stopes after mining of stopes are complete.

16.17 Mining Operations Personnel

Hiring highly skilled hard rock miners with non-mechanized mining experience would be a critical part of the success of the Dufferin project, although some positions could be filled using less experienced miners. Lead hands, jumbo operators, scoop operators, production miners, and bolters would need to be highly skilled, while truck drivers and nippers could be less experienced.

						Year 1 to	Year 7
Classification	**$/year**	**Number**		**Total $/yr 1**	**Total $/yr 2**		
Mine and Administration Staff		Year 1	Year 2 +				
Mine manager and General Superintender	$150,000	1	1	$150,000	$150,000	$150,000	$150,000
Mine Superintendent	$120,000	0.5	1	$60,000	$120,000	$60,000	$120,000
HR Administrator / Safety Coordinator	$80,000	1	1	$80,000	$80,000	$80,000	$80,000
Reception	$40,000	1	1	$40,000	$40,000	$40,000	$40,000
Accounting/Purchasing	$50,000	2	2	$100,000	$100,000	$100,000	$100,000
Trainer	$50,000	1	1	$50,000	$50,000	$50,000	$50,000
Surface Facilities							
Surface Supervisor	$60,000	1	1	$60,000	$60,000	$60,000	$60,000
Carpenter / surface maintenance	$50,000	1	1	$50,000	$50,000	$50,000	$50,000
Yard Maintenance Loader/Truck Operator	$40,000	1	1	$40,000	$40,000	$40,000	$40,000
Security/First Aid	$40,000	4	4	$160,000	$160,000	$160,000	$160,000
Maintenance Facilities							
Maintenance Superintendent	$75,000	1	1	$75,000	$75,000	$75,000	$75,000
Lead Mechanic	$80,000	2	4	$160,000	$320,000	$160,000	$320,000
Mechanical	$60,000	2	4	$120,000	$240,000	$120,000	$240,000
Electrical	$60,000	2	4	$120,000	$240,000	$120,000	$240,000
Warehouse Worker	$45,000	1	1	$45,000	$45,000	$45,000	$45,000
Maintenance Clerk	$35,000	0.5	1	$17,500	$35,000	$17,500	$35,000
Engineering Staff							
Chief Mine Engineer	$100,000	1	1	$100,000	$100,000	$100,000	$100,000
Mine Engineer/Projects Engineer	$80,000	1	1	$80,000	$80,000	$80,000	$80,000
Survey Tech/Mine Planner	$60,000	0.5	1	$30,000	$60,000	$30,000	$60,000
Environmental /Ventilation Technician	$60,000	0.5	1	$30,000	$60,000	$30,000	$60,000
Geology Staff							
Chief Geologist	$75,000	1	1	$75,000	$75,000	$75,000	$75,000
Senior Geologist	$65,000	1	1	$65,000	$65,000	$65,000	$65,000
Samplers	$45,000	2	2	$90,000	$90,000	$90,000	$90,000
Mine Operations Staff							
Shift Supervisor	$100,000	2	4	$200,000	$400,000	$200,000	$400,000
Underground Labourer	$60,000	2	4	$120,000	$240,000	$120,000	$240,000
Jumbo Operator	$100,000	2	4	$200,000	$400,000	$200,000	$400,000
Truck Drivers	$60,000	2	8	$120,000	$480,000	$120,000	$480,000
Mine Clerk	$40,000	0.5	1	$20,000	$40,000	$20,000	$40,000
Dry/Janitorial	$40,000	1	2	$40,000	$80,000	$40,000	$80,000
Mine Development							
Lead Development Miner	$110,000	2	4	$220,000	$440,000	$220,000	
Development Miner 1	$90,000	6	12	$540,000	$1,080,000	$540,000	
Processing Plant							
Mill Manager	$100,000	1	1	$100,000	$100,000	$100,000	$100,000
Mill Superintendent	$80,000	1	1	$80,000	$80,000	$80,000	$80,000
Assayor	$60,000	1	1	$60,000	$60,000	$60,000	$60,000
Operators	$50,000	4	8	$200,000	$400,000	$200,000	$400,000
Loader operator	$45,000	2	4	$90,000	$180,000	$90,000	$180,000
total personnel		55.5	91	$3,787,500	$6,315,000	$3,787,500	$4,795,000

Figure 16-6: Dufferin Mine Personnel.

16.18 Equipment

Initially, all mobile equipment would be diesel powered, equipped with appropriate exhaust scrubbers as required as this equipment is readily available. A single brand with interchangeable parts is recommended. Also it must be ensured that the equipment will fit into the mine openings.

Longhole drilling and Thermal Fragmentation require at least 2.8m head room so the crest at the top of the limb has to be excavated to a 3m height and also into the hanging wall 25cm to 30 cm.

As additional resources are identified the option to lease electrically powered equipment should be examined as electric motors have higher overall energy conversion efficiency. Electric motors produce no harmful gasses so ventilation costs are reduced.

16.18.1 Drilling Equipment

Most development drilling would be done using diesel/electric jumbo drill rigs. Raises would be “open” or developed using an Alimak-type rig.

A longhole drill rig would be used for mining longhole stopes when needed. This drill rig can also be used to drill drain holes or utility holes, and to drill holes for cable bolts if necessary. Longhole drill rigs can be either pneumatic or electric powered.

Longhole Drill (Drillmaster 100 Longhole)	$650,000 new, $13,000/month lease cost
Penetration Rate (m/min)	1.2
Tonnes Per Metre Drilled	2
Item	**Description**
Bits	300 m/bit
Steel	1200 metres per Steel
Drilling Oil	1 L/hr
Hydraulic Oil	1 L/hr
Air Hose	10 rounds per hose
Water Hose	10 rounds per hose
Electricity	80 kWatt
Diesel (Tramming)	5 L per Shift
Maintenance Supplies	
Operating Cost, approximately $1.60/m drilled.	

16.18.2 Mucking & Loading Equipment

In this PEA, diesel powered scoops will be used within the mine primarily to muck development material and to load trucks.

If saddle orientations are favourable (thicker than 3 m), medium sized scoops (2 yd to 3.5 yd class capacity) could be used to muck production mining.

16.18.3 Rock Haulage

Diesel powered articulated underground haul trucks would be used to carry both production and development materials from the underground mine to the surface. It is planned to use two trucks with a rated capacity of 20 tonnes each for this study. It is anticipated that the majority of development material would need to be removed from the mine and stored on surface.

If additional resources are developed from new exploration or definition drilling, a production hoisting shaft should be considered. Access at the bottom of the mine from the ramp would allow development of the shaft to be excavated from the bottom up, saving considerable capital cost compared to conventional shaft sinking from the top down.

16.18.4 Support Equipment

Scissor lift vehicles are used for several regular activities at the mine. Scissor lifts would be used regularly to install ground support and services. They can be also be used to transport materials prior to the purchase of a boom truck.

A boom truck is used to transport materials and supplies from surface to locations underground. It consists of a flatbed truck equipped with a boom, planned for purchase in month 1.

A grader is a useful piece of equipment to maintain a smooth roadway, which improves tire life as well as haulage efficiency. As an alternative to the purchase of a grader, the roadway can be smoothed using a weighted scraper pulled behind a truck.

An Alimak raise climber will be used for all raise mining, including all orepass raises, personnel raises, and ventilation raises.

16.18.5 Auxiliary Equipment

A man carrier capable of carrying 8-10 people would be purchased at the start of production. Small mine utility vehicles would also be purchased for non-production activities.

16.19 Ground Support

Primary ground support in permanent areas would consist of rock bolts combined with wire mesh screen. Secondary ground support may be required for certain unfavourable conditions, and may consist of cable bolts and/or shotcrete. Ground support within stopes would use rock bolts but no screen, and pillars may be created (from low grade rock material or artificial material) to increase stope stability. Consideration has been given to using stull type of supports for limb mining methods.

16.20 Ventilation

Ventilation requirements have been estimated based on providing 0.06 m^3/s of fresh air per rated kW of diesel powered equipment. An anticipated utilization rate was factored in for each piece of mobile equipment. The Nova Scotia Underground Mining Regulations specify that[16]

"A ventilation system must meet all of the following criteria:

a) it must be constructed, operated, tested, calibrated, inspected and maintained in accordance with the manufacturer's specifications or specifications certified by an engineer;
b) it must supply ventilating air to underground areas where persons are working or travelling or might work or travel;
c) it must be adequate to ensure the health and safety of persons underground."

No specific amount of air is mandated in the 2013 Regulations, rather the operator must "ensure sufficient air quality is provided to all active workings underground to ensure that any contaminants are diluted below the occupational exposure limit."

An air quality monitoring program must be developed (s. 211) and air quality and air quantity tests must be completed at least once a month.

16.20.1 Fresh Air Supply

The quantity of air supplied at the working face to dilute diesel emission is 6m^3/second/100 kW, adding and 0.05m^3/second for each person underground.

Table 16-9 An estimate of the ventilation requirements based on vehicle utilization.

HP	KW	Equipment	Approximate Utilisation (%)	CFM*	m^3/second
165	123	Boom Truck	25%	5,240	1.85
52	39	1 Yd Scoop - Wagner	25%	1,650	0.58
82	61	Scissor Lift	25%	2,600	0.92
165	123	3 yd. Scoop	75%	15,720	5.54
85	63	Southbay Man Carrier	25%	2,700	0.95
270	201	Haul Truck	75%	25,720	9.06

[16] Underground Mining Regulations made under Section 82 of the Occupational Health and Safety Act S.N.S. 1996, c. 7 O.I.C. as amended up to O.I.C. 2013-105 (March 28, 2013, effective April 1, 2013), N.S. Reg. 141/2013

HP	KW	Equipment	Approximate Utilisation (%)	CFM*	m^3/second
270	201	Haul Truck	50%	17,150	6.04
40	30	Toyota Man Carrier	25%	1,270	0.45
165	123	3 yd Scoop - Remote	25%	5,240	1.85
80	60	M-30 Scissor Lift	25%	2,540	0.89
55	41	1-Boom Jumbo	25%	1,750	0.62
55	41	1-Boom Jumbo	25%	1,750	0.62
Assume 12 persons underground:				1,271	0.60
** CFM = cubic feet per minute.*			Totals	84,600	30

Fresh air enters the mine through the ventilation raise, currently powered by a 75 HP fan. This fan will be upgraded to a larger 150 HP - 200 HP fan. A new ventilation raise be developed from the bottom of the mine directly to the surface.

The ventilation raise delivers fresh air to the bottom of the mine, and from there it travels back up the ramp to exhaust at the portal to the surface. As air returns up the ramp, air is collected and delivered to the haulage drift level by a single fan (100 HP) that is hung in the ramp. Air will be delivered to the end of the haulage drifts using flexible ducting, when it returns back to the ramp along the haulage drift.

16.20.2 Exhaust Air Return

Exhaust air returns from working areas to the haulage drift and then to the ramp. The ramp serves as an exhaust air return at this time, which makes it necessary to insure adequate ventilation in the mine. Once a shaft is in place, the flow can be intake at the portal and exhaust for the raise.

16.20.3 Secondary Egress

Secondary egress from the underground to surface would be via the existing and planned ventilation raises. The ventilation raises would be equipped with ladders and landings and function as emergency escape ways.

16.20.4 Mine Air Heating

During the colder winter months the intake air would require heating, for reasons of safety and productivity. A propane or natural gas heater would be installed in the air intake raise, and will be used to keep the mine intake air above freezing.

16.21 Mine Services

16.21.1 Electrical

The main electrical trunk would run down the ventilation raise, branching off to a substation at each level. Electricity would be used to run power to pumps, fans, electric jumbos, and refuge stations. It may be possible to also use electric-hydraulic jackleg drills.

Electrical substations would be located underground, with one substation serving each stope. After stopes are completed, it would be possible to relocate electrical substations for reuse in new stopes.

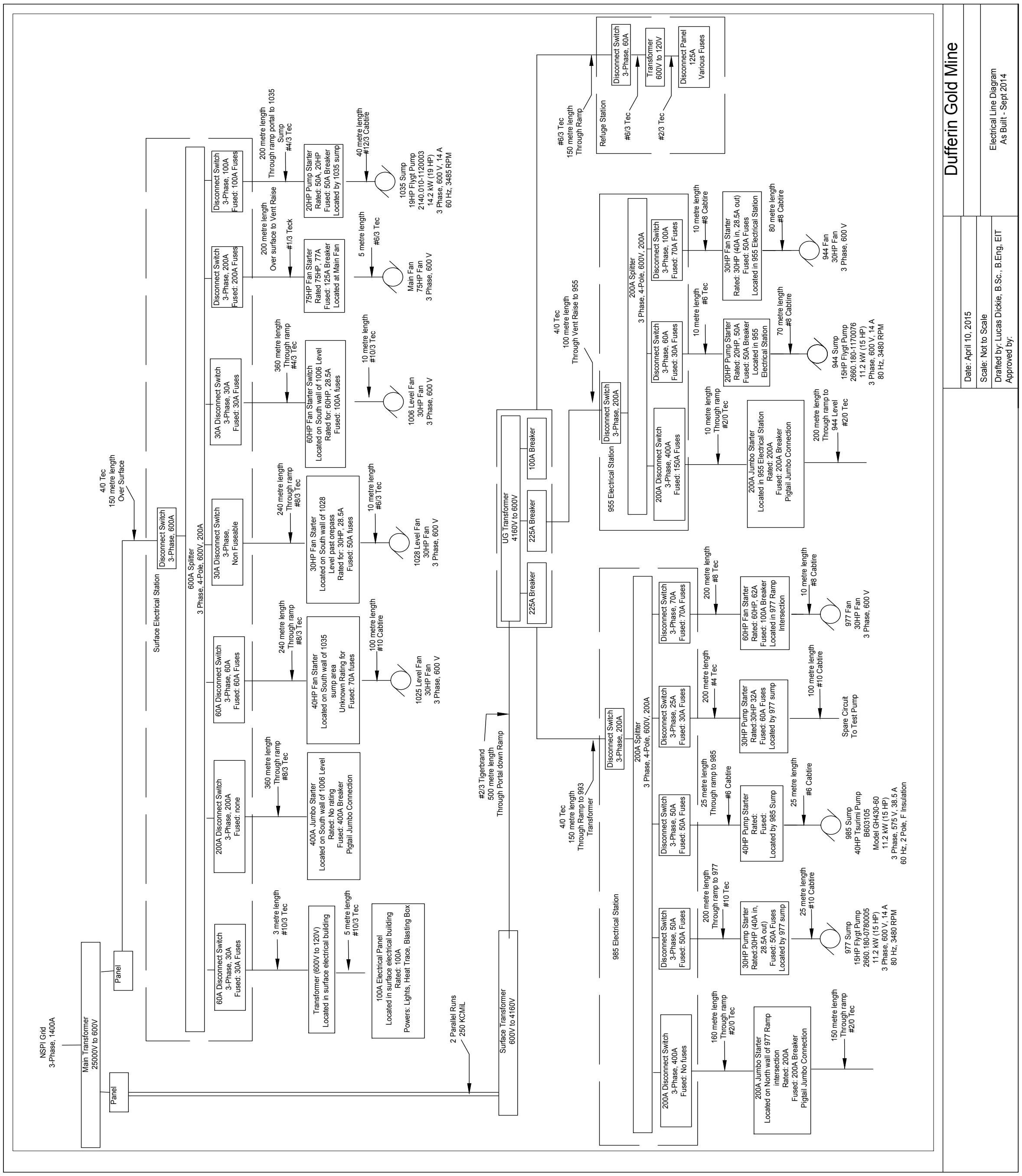


Figure 16-7 Dufferin Gold Mine Electrical Layout

16.21.2 Compressed Air Supply

Compressed air will be delivered to the underground using a main line of 152 mm (6") steel pipe, run in both the ramps and the ventilation raises. Branches from the main line will provide compressed air to the stopes by 102 mm (4") steel pipe, and 51 mm (2") steel and PVC pipes would deliver compressed air to the mining face. Equipment that requires compressed air includes:

- Jacklegs and stopers.
- Cleaning blastholes.
- Loading explosives.
- Longhole drill setups.
- Refuge station pressurization.
- Pneumatic pumps for dewatering the active face.

16.21.3 Production Water Supply

Production water would be used underground for drilling operations, dust suppression during mucking, and washing walls and backs for scaling and sampling purposes. Production water would be provided by water pumped from the polishing pond, gravity fed to the mine through the ventilation raise using a 102 mm (4") steel pipe. Connections at each level would provide production water to each stope using 102 mm (4") steel pipe, with 51 mm (2") steel or PVC pipes delivering water to the active face.

16.21.4 Mine and Mill Water Discharge

Water that accumulates at the active face is pumped away using a small pneumatic diaphragm pump (Wilden type pump) or a small electric pump, using a 51 mm (2") steel or PVC pipe from the face to the sump.

Sumps could be connected by a system of drain holes, which would drain all water to the bottom sump in the mine.

A permanent pumping station will be constructed at the bottom of the mine that would pump water to the surface settling pond system. The pumping arrangement would be setup as a redundant parallel system, with either side capable of providing mine dewatering without the other.

16.21.5 Communications

Current communication to underground locations is by wired telephone system. A leaky feeder type radio communication system is envisioned.

16.22 Maintenance

The maintenance department would provide an essential service to mining operations. An experienced maintenance planner would run the department to ensure compatible equipment is acquired for the mine.

16.22.1 Mobile Maintenance

A maintenance shop is on site, near to the location of the underground portals. It could be expanded to have 3 bays and ample space for laydown in between bays.

16.22.2 Ramp Roadbed

Proper maintenance of roadways will increase both tire life and haulage efficiency. A drainage ditch should be maintained on one side of the ramp to divert water off of the roadway. Crushed rock should be spread on the roadway as required.

17 Recovery Methods

17.1 Flowsheet Development

Gravity and flotation are used at Dufferin to recover a doré brick and a flotation concentrate from the mill feed.

Ressources Appalaches upgraded the plant to process 300 tonnes per 24 hour day before and during the processing of approximately 18,000 tonnes of mainly low grade muck from underground.

Figure 17-1 illustrates the physical set up of the plant. Figure 17-3 is a detailed view of the refinery.

We have used an overall mine gold recovery of 95% in this study. It is assumed that 85% of the resource can be mined by stoping the crest and limbs of the saddles. Another 10% of the resource would be mined during development. In the mill, 95% of the gold would be recovered from the rock. The remaining 5% would be lost to the tailings. Some of the tests using cyanide were conducted by Ressources Appalaches. Recovery using cyanide was close to a 97%.

Most of the gold was recovered in the gravity section of the Dufferin plant. Flotation concentrate was collected; however, Ressources Appalaches never had much luck producing a saleable product.

Table 17-1 lists the various power requirements for the mill motors.

The bottlenecks in the mill are the fine rock bin and the grinding circuit. The rock bin holds about 10 tonnes of crushed rock, too small to provide a constant feed to the mills. The bin should be replaced with a 1,000 tonne bin that can hold 3 days crushed mineralised rock.

The grinding mill also has to be optimized to reduce the amount recycled and the refinery furnace has to be upgraded to provide a higher temperature so that impurities will be volatized.

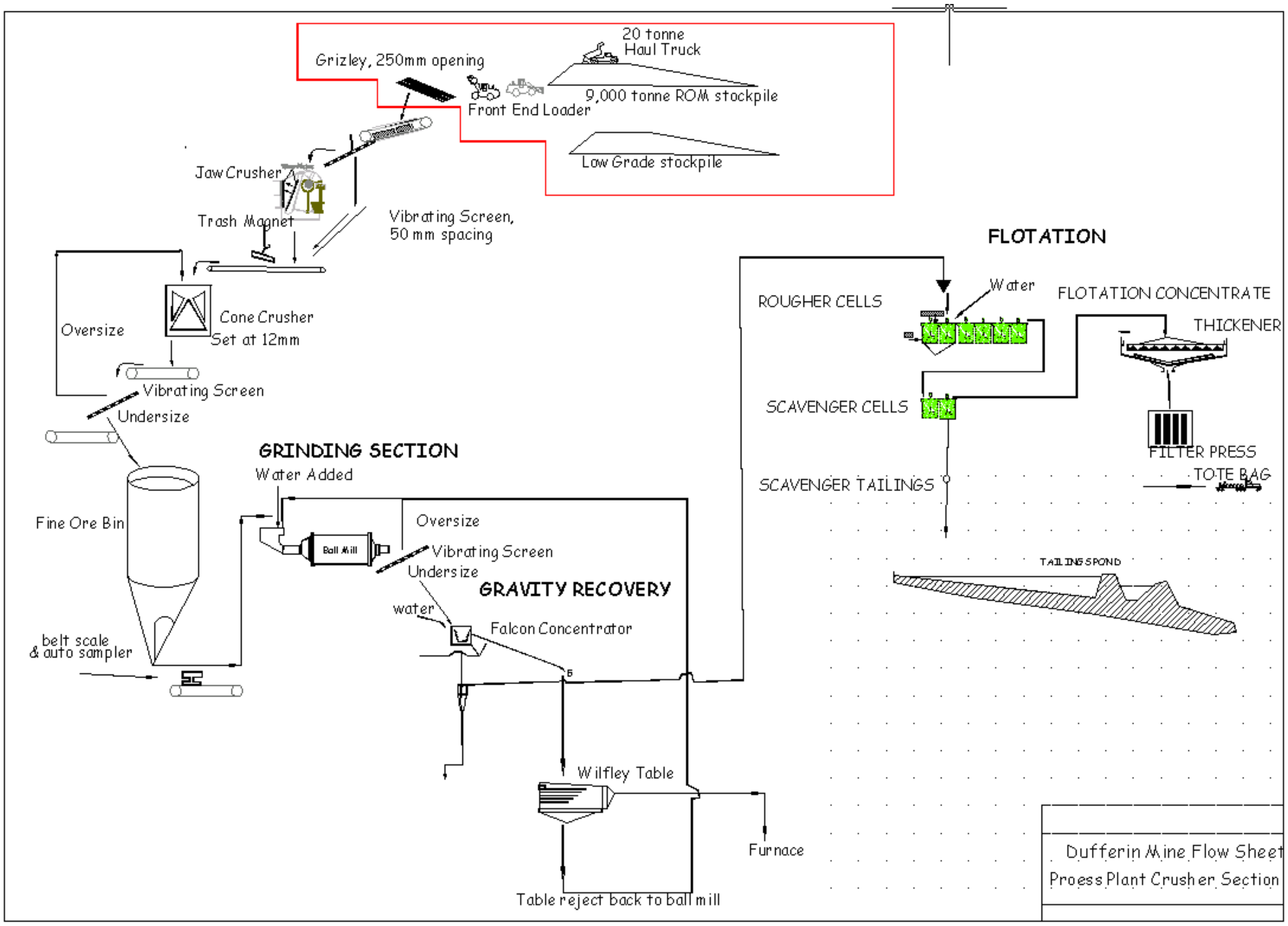


Figure 17-1 - Flow sheet illustrating the flow of material through the Dufferin Mill.

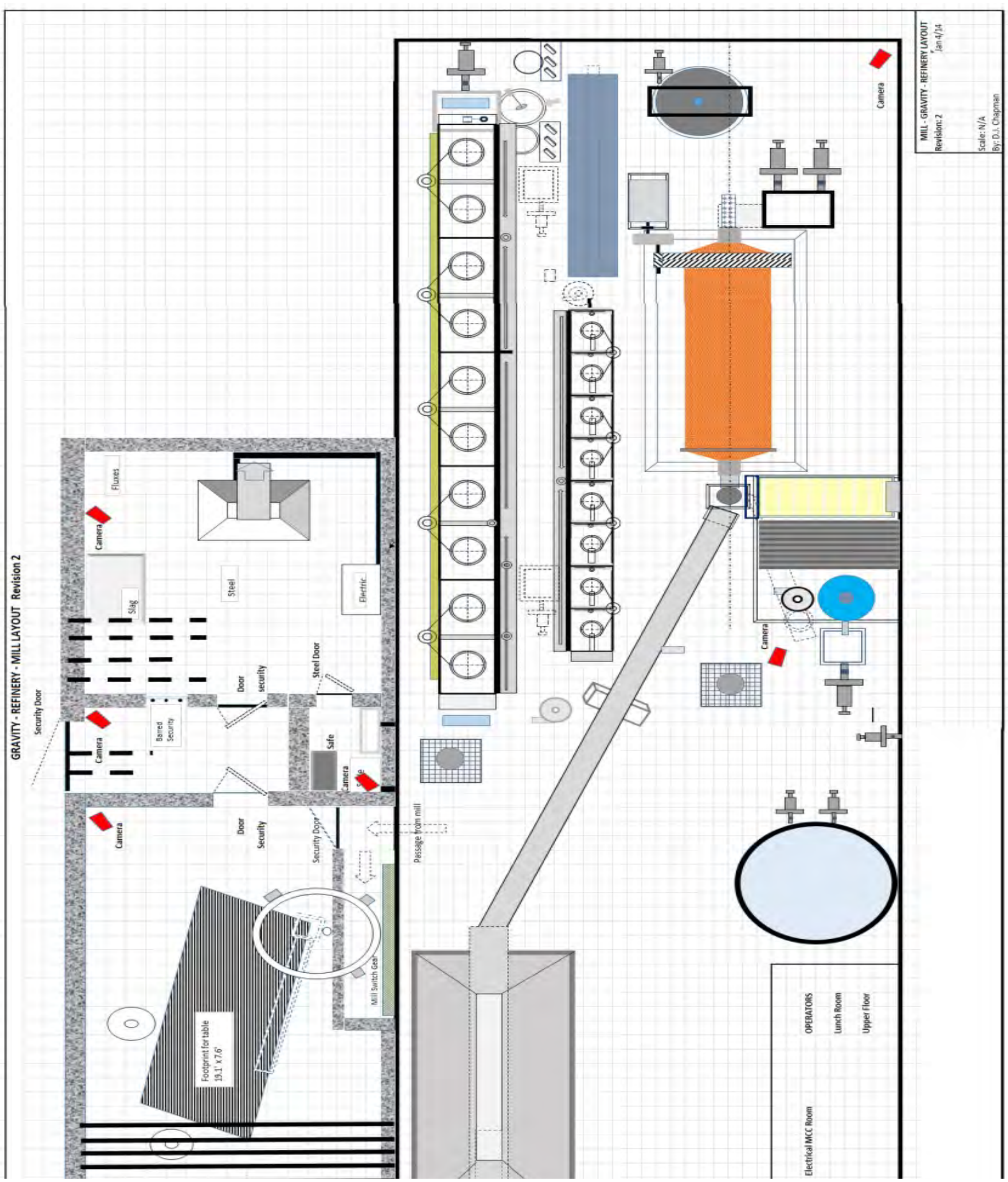


Figure 17-2 - Mill Layout

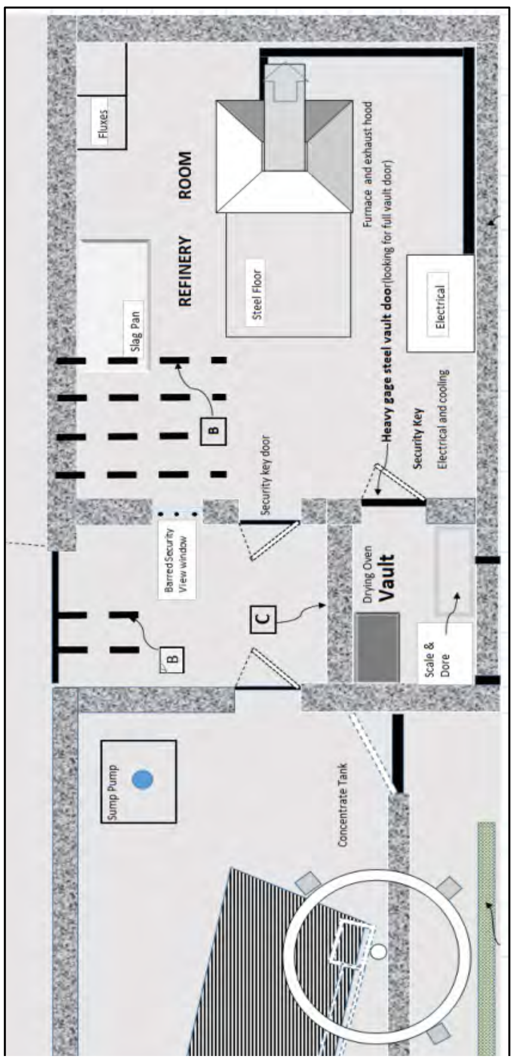


Figure 17-3 - Refinery Room Layout

Table 17-1 - Motor Power Requirements for the Mill.

Power		Unit Power		Units		Total Power	Utilization	Per Day	Day	Month	Year
		HP	kW	#	HP	kW		kWh	$/day	$/month	$/year
Crusher											
Jaw feed conveyor		5	3.7	1	5	3.7	29%	26	$3	$80	$957
Jaw Crusher		125	93.2	1	125	93.2	29%	639	$66	$1,994	$23,931
Screening Plant Conveyors		5	3.7	6	30	22.4	29%	153	$16	$479	$5,743
Screening Plant		25	18.6	1	25	18.6	29%	128	$13	$399	$4,786
Cone Crusher		100	74.6	1	100	74.6	29%	511	$53	$1,595	$19,144
Mill											
Mill feed conveyors		5	3.7	4	20	14.9	93%	332	$35	$1,037	$12,444
Ball Mill		250	186.4	1	250	186.4	93%	4,155	$432	$12,962	$155,549
Ball Mill Recirc Pumps		25	18.6	2	50	37.3	93%	831	$86	$2,592	$31,110
Sump		7.5	5.6	1	7.5	5.6	10%	13	$1	$42	$503
Falcon											
Falcon Screen		1.3	1.0	2	2.6	1.9	93%	43	$4	$135	$1,618
Falcon Concentrator		30	22.4	1	30	22.4	93%	499	$52	$1,555	$18,666
Falcon Concentrator Aux		7.5	5.6	1	7.5	5.6	93%	125	$13	$389	$4,666
Gold Room											
Wifley Table		3	2.2	3	9	6.7	93%	150	$16	$467	$5,600
Sump		7.5	5.6	1	7.5	5.6	10%	13	$1	$42	$503
Furnace		65A, 460V									
Flotation											
Agitator Pumps - Rougher		15	11.2	5	75	55.9	93%	1,246	$130	$3,889	$46,665
Rougher Pumps		5	3.7	2	10	7.5	93%	166	$17	$518	$6,222
Agitator Pumps - Cleaner		5	3.7	4	20	14.9	93%	332	$35	$1,037	$12,444
Cleaner Pumps		5	3.7	1	5	3.7	93%	83	$9	$259	$3,111
Thickener											
Thickener Tank		1.5	1.1	3	4.5	3.4	93%	75	$8	$233	$2,800
Filter											
Filter Press		8A, 600V									
Filter Conveyor		2	1.5	1	2	1.5	50%	18	$2	$56	$670
Other											
Tailings Pump		30	22.4	1	30	22.4	93%	499	$52	$1,555	$18,666
Mill water feed from Polishing		20	14.9	1	20	14.9	93%	332	$35	$1,037	$12,444
Water Tank Pumps		7.5	5.6	2	15	11.2	50%	134	$14	$419	$5,025
Mill Total				46	851	634		10,504	$1,092	$32,772	$393,265
								315,117			
Load Factor						69%					

17.2 Mineralised rock and Low Grade Stockpile

A stockpile will be built for about 1 month's mill supply adjacent to the Jaw crusher building. There should also be a low grade stockpile once the mine can steadily supply 300 tonnes per day mineralized rock to the mill. The mill should be tuned up on low grade development muck, but once the mill is running smoothly, low grade should be stockpiled separately and processed at the end of the mine life.

17.3 Crushing and Grinding

From the stockpile, mineralized rock is first crushed to about 12mm by a jaw crusher and cone crusher. Crushed mineralized rock is then sent to a fine mineralized rock bin and on to a grinding section. Vibrating screens are used to separate feed streams and a series of conveyors deliver coarse feed to the various units. After grinding, cyclones are used to separate the feed streams, with coarse material now going to the gravity separation section and finer material through the flotation circuit.

Recovery of the gold is dependent upon grind size; however, as fine material has more surface area, a finer grind means additional reagents must be used in the flotation process as the surface area increases.

Prior to going through the jaw crusher, the mineralized rock passes through a grizzly with bars spaced at 250 mm. Any oversize blocks are set aside for eventual breakup using a hydraulic hammer.

The mineralized rock is crushed in a jaw crusher, then on to a cone crusher. From the cone, material goes over a vibrating screen. Oversize is sent back to the cone while undersize goes to the grinding mill where water is added.

17.4 Concentration

After grinding, the material passes through a cyclone which separates the course material for processing through the gravity circuit. A Falcon concentrator is used to make a concentrate which then passes over a table where high grade gold is collected and stored in the vault for later melting into a doré brick.

Reject from the table goes back to the Falcon concentrator for reprocessing. Tails from the Falcon concentrator and the undersize from the cyclone are directed to the flotation circuit. Tailings from the flotation circuit are sent to the tailings pond. The concentrate from the flotation circuit is thickened, dewatered and bagged in one tonne tote bags for shipment to a smelter.

17.5 Tailings

Essentially all of the feed will be directed to the tailings facility using slurry pumps.

Some work remains to be done to shore up the walls of the tailings pond and to raise the dams to about a 10m height above the ground surface.

The facility can handle the resources outlined to date; however, alternatives are required to accommodate any additional mineralized rock. At the end of mine life, the material in the pond could be reprocessed using cyanide to recover any remaining gold.

17.6 Milling, Grinding, Gravity, and Flotation

Table 17-2 – Estimated throughput (tph) for major mill equipment.

Item	Base Units (Solids, Slurry)	Rated Capacity in Base Units (tonnes/hour)	Base Thruput in Base Units (tonnes/hour)	Peak Thruput in Base Units (tonnes/hour)
Jaw Crusher 125 HP		100-150	100-150	100-150
Vibrating Screen 16'x4'		75-100	75-100	75-100
Cone Crusher		75-100	75-100	75-100
Ball Mill 6'x14'	Solids	13	12.5	13
Vibrating Trash Screen 40"x84"		50	44	50
Falcon Concentrator		60	44	60
Flotation Concentrate Thickener		0.15	0.09	0.15
Flotation Concentrate Filter Press		0.2	0.09	0.2
Shaking Table 6'x14'		1.5	1	1.5

The ball mill is the bottleneck in the present setup; however, a regrind mill may be added to ensure smooth production.

17.7 Mass Balance

17.7.1 Summary

In order to get an indication of the head grade of the material that was milled, a mass balance of the gold was completed by adding the gold in the doré bricks, the flotation concentrate, the tailings and the gold found in the "clean up" of the mill.

During 2014, Ressources Appalaches' processing plant treated approximately 17 thousand tonnes of underground development rock. The average grade of the material was 1.89 g/tonne. About 2,685 tonnes had a grade above the cut-off of 3 g/tonne in the corrected mass balance. The remainder was low-grade development rock.

The plant recovered 28.6 kg of gold (921 troy ounces), and the operators produced another 1kg of gold (32 troy ounces) from slags for a total recovery of 29.6 kg of gold (953 troy ounces). The doré brick averaged about 73.6% gold with the remainder being silver. 93.2% of the gold was collected using gravity separation. 6.8% was recovered in the floatation circuit.

The overall recovery was 93.4%. Of this amount, about 94.4% of the gold was recovered by gravity and only 5.6% by flotation. This may be because the head grade was low or it could be that the gravity circuit is very efficient, leaving little gold for flotation. Flotation is supposed to pick up the very fine gold particles that are too small for gravity collection. A higher head grade, it is expected that 85% of the gold recovery will be by gravity and the other 15% recovery by using flotation.

17.7.2 Head Grade

For 2014 production, the head grade of the material processed was estimated by the mine staff to be 1.49 grams per tonne. After adding up all of the gold and then dividing this amount by the tonnage milled, the grade was found to be 1.94 g/tonne. The grade of the individual lots were adjusted by the ratio 1.94/1.49 or 131%.

Of the tonnage milled, about 2,200 tonnes were estimated to be above 3 g/tonne. The average grade of this material was about 7 g/tonne.

Table 17-3 Head Grade of material processed in the Dufferin Mill during 2014. The head grade has been adjusted on the basis of the mass balance.

DMT Milled	Mine est. Grade, g/tonne	Est. Grams	Corrected Head Grade, g/tonne	
5,109	0.56	2,860	0.73	
3,598	0.52	1,871	0.68	
2,233	1.76	3,938	2.31	
1,494	1.99	2,965	2.60	
1,498	3.21	4,806	4.20	
1,310	0.90	1,183	1.18	
1,255	1.64	2,054	2.14	
496	2.71	1,343	3.54	
269	6.05	1,625	7.91	
275	6.89	1,896	9.01	
148	12.0	1,767	15.63	
17,684	1.49	26,310	1.94	
2,685	4.26	11,438	5.57	above 3 g/tonne cut off
691	7.65	5,288	10.00	high grade

Table 17-4 - Flotation Concentrate tonnage and grade.

Flotation concentrate		
tonnes	g/tonne	grams
140	15	2100

Table 17-5 - Mass balance.

Grams Au	% of total gold	Source
28,646	83.29%	sent to JM (dore brick assay).
995	2.89%	Gold in slag
2,100	6.11%	grams estimated to be in flotation concentrate.
2,653	7.71%	Au estimated in tailings
34,394	100.0%	total
92.3%	overall gold recovery	

Table 17-6 – Comparison of mine estimate and calculated head grade.

17,684	DMT milled
1.94	head grade of tonnes milled
131%	of mine estimate

Doré Bricks

Twenty doré bricks containing about 31.5 kg of gold was sent to Johnson Matthey at 130 Glidden Road, Brampton, Ontario (JM).

Prior to shipping, the mine assayed the doré brick. Upon receipt, JM also completed an assay. The mine assays were generally about 1.3% higher than the JM assay, except for DM-003. This brick assayed 47.2% at the mine but only 8.55% when JM did the assay.

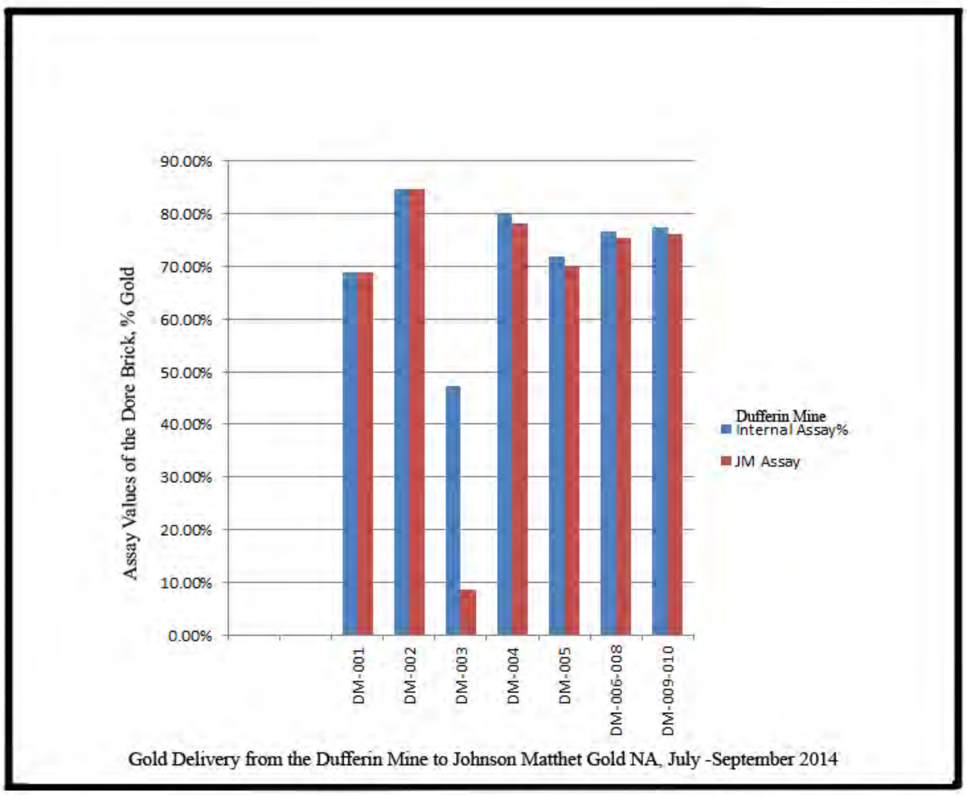


Figure 17-4 - Dufferin Mine Assays of bricks vs. JM assay of doré bricks.

Table 17-7 - Dufferin Mine Tabulation of sales, 2014.

Date Smelted	Lot	Date Settled - JM	Weight KG	gms*Internal assay	gms*JM assay	Assay% Internal	JM Assay	Grams recovered
to	DM-001	25/7/14	4.1571	2,860	2,860	68.79%	68.79%	2,859.67
31/7/14	DM-002	20/8/14	1.8930	1,602	1,604	84.64%	84.72%	1,603.75
	DM-003		3.1250	1,475	267	47.20%	8.55%	267.19
	DM-002-00:	TOTAL	5.0180					1,870.9371
8/8/14	DM-004	27/8/14	4.9220	3,938	3,848	80.02%	78.18%	3,938.34
18/8/14	DM-005	5/9/14	4.1208	2,965	2,894	71.96%	70.22%	2,965.35
27/8/14	DM-006	12/9/14	2.3498	1,795		76.37%		1,794.57
	DM- 007	12/9/14	1.2430	858		69.01%		857.82
	DM-008	12/9/14	2.6819	2,351		87.67%		2,351.18
			6.2747	-				5,003.5657
	DM-006-008		6.272		4,733	76.63%	75.46%	4,806.23
10/9/14	DM-009	26/9/14	1.2051	946		78.48%		945.75
	DM-010	26/9/14	0.3365	237		70.54%		237.39
	Total		1.5416	-				1,183.1427
	DM-009-010		1.5420		1,174	77.41%	76.14%	1,193.66
17/9/14	DM-011	3/10/14	1.6618	1,146		68.97%		1,146.17
	DM-012		1.1867	908		76.53%		908.21
	DM-011-01:	TOTAL	2.8486	-				2,054.37
24/9/14	DM-013	10/10/14	0.6222	458		73.61%		458.01
	DM-014	10/10/14	1.4035	885		63.07%		885.20
	DM-013-01	TOTAL	2.0257	-				1,343.21
1/10/14	DM-015	17/10/14	1.1126	844		75.88%		844.21
	DM-016	17/10/14	0.9832	781		79.39%		780.54
	DM-015-01(	TOTAL	2.0957	-				1,624.74
8/10/14	DM-017	24/10/14	2.5761	1,896		73.61%		1,896.27
29/10/14	DM-018	14/11/14	2.8790	2,119		73.61%		2,119.27
	DM-019	14/11/14	2.2329	1,644		73.61%		1,643.63
	DM-020	14/11/14	2.4008	1,767		73.61%		1,767.26
	DM-019-02(	TOTAL	5.1119	-				3,762.89
TOTAL		grams sent tc	(Mine assay)	31,476			Grams JM recovered	28,316
							% recovered of gold sent	90.0%

17.7.3 Flotation Concentrate

Flotation concentrates from 2014's mine production are stored in 1 tonne tote bags on site – none was sent for gold recovery. The grade was estimated by taking a 0.25 kg sample from 10 of the 140 bags. This sample was then fire assayed at the Dalhousie Mineral Engineering Centre. A multi element analysis was completed on one sample.

Table 17-8 – Gold assay of flotation concentrate.

Re: Results of analysis on submitted samples.

30g fire assay-lead collection, AAS finish.

	Original Sample	**Duplicate Sample**
Sample Number	**Au (mg/kg)**	**Au (mg/kg)**
Sample 83	9.892	9.708
Sample 87	13.56	13.17
Sample 91	13.45	13.59
Sample 92	10.86	10.80
Sample 97	11.21	11.57
Sample 107	10.25	10.30
Sample 113	19.23	18.27
Sample 123	18.11	18.19
Sample 1000	9.836	9.359
Sample 1100	32.54	15.63

Certified Reference Samples	Au (mg/kg)
SF67 (0.835±0.006)	0.902
OXP116 (14.92±0.11)	14.87

Table 17-9 – Multi Element analysis of sample 1100.

Near total acid digestion, ICP OES finish. Some refractory elements may							
only be partially digested. Some volatile elements may be partially lost.							
*Aqua regia digestion.							
Sample ID	**mg/kg**						
	Ag*	**Al**	**As***	**Ba**	**Be**	**Bi**	**Ca**
MineTech 1100	7.5	49552	175062	592	2.0	31	4928
CCU-1d		1894		16	0.1	<10	2631
MP-1b	48	21506	22008	8.3	0.6	1120	26134
Sample ID	**mg/kg**						
	Cd	**Ce**	**Co**	**Cr**	**Cu**	**Fe**	**Ga**
MineTech 1100	<10	63	462	54	1035	185094	<50
CCU-1d	286	<10	338	10	239465	294843	<50
MP-1b	603	187	5	9.4	31879	83546	123

Sample ID	**mg/kg**						
	Ge	**In**	**K**	**La**	**Li**	**Mg**	**Mn**
MineTech 1100	<50	<100	20354	33	34	6256	270
CCU-1d	<50	<100	523	<5	<10	5855	93
MP-1b	<50	484	2189	75	33	357	505
Sample ID	**mg/kg**						
	Mo	**Na**	**Nb**	**Ni**	**P**	**Pb**	**S**
MineTech 1100	9	4779	<50	1027	325	2286	117667
CCU-1d	11	2119	<50	23	<100	2578	329229
MP-1b	291	2270	<50	7	<100	21136	136544
Sample ID	**mg/kg**						
	Sb	**Se**	**Sn**	**Sr**	**Ta**	**Te**	**Ti**
MineTech 1100	157	<50	<50	78	<50	<100	1315
CCU-1d	70	240	<50	14	<50	<100	72
MP-1b	<50	<50	4431	16	<50	<100	607
Sample ID	**mg/kg**						
	Tl	**V**	**W**	**Zn**	**Zr**		
MineTech 1100	<100	56	<50	1732	124		
CCU-1d	<100	5	<50	26454	10		
MP-1b	<100	<5	956	165856	100		

Table 17-10 Silver and Arsenic analysis of the flotation concentrate.

Re: Results of analysis on submitted samples.			
Aqua regia acid digestion, ICP OES finish.			
Sample	**mg/kg**		
	Ag	**As**	
Sample 83	5.0	129900	
Sample 83 Dup.	4.8	128904	
Sample 87	5.8	158223	
Sample 91	5.6	156351	
Sample 92	4.7	146207	
Sample 97	4.0	175821	
Sample 107	4.0	177536	
Sample 113	7.2	207263	
Sample 123	6.6	176467	
Sample 1000	4.3	111157	
Sample 1100	7.5	175062	
Certified Reference Samples	*Ag (mg/kg)*	*As (mg/kg)*	*Recommended Value*
CANMET MP-1b	47.9		47
CANMET MP-1b		22008	23000

18 Project Infrastructure

There exists significant infrastructure on the Dufferin site. Currently, the site's infrastructure consists of:

- Underground workings, in good condition, including 2,253 m of underground development over 5 levels reaching Saddles 1 through 4, with a vertical depth of 100 m and an east-west extent of 470 m (see **Table 14-3**). A very small section is currently flooded, as a sump. Some areas need rehab.
- A 300 tpd gravity/flotation mineral processing facility, including a crusher and a grinding mill, currently under care and maintenance;
- An assay laboratory, capable of fire assay with gravity finish;
- Core racks;
- A powder magazine and cap magazine (rented; property of Atlantic Explosives Ltd.);
- A tailings management facility;
- Three-phase, grid-connected power;
- Diesel fuel tank with pump;
- Water wells (non-potable);
- A Quonset-style, steel clad workshop in good condition;
- A security/first aid trailer in fair condition;
- A mine dry trailer, in fair condition;
- Several office trailers in fair-to-good condition; and,

Figure 18-1 through Figure 18-17 show the Property as it was in January, 2015. They are an adequate representation of the current site conditions.

Figure 18-1 - Office Buildings

Figure 18-2 - Technical Offices

Figure 18-3 - Crusher Building

Figure 18-4 - Mill Building

Figure 18-5 - Compressor House & Rear of Mill

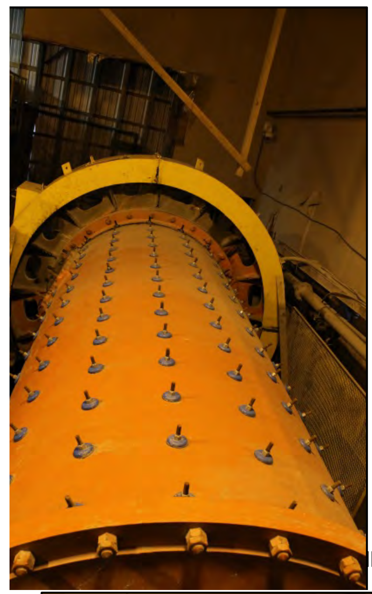
Figure 18-6 - Powder Magazine

Figure 18-7 - Cap Magazine

Figure 18-8 - Assay Lab

Figure 18-9 - Workshop

Figure 18-10 - Jaw Crusher



l

Figure 18-12 - Filter Press

Figure 18-13 - Flotation Tanks

Figure 18-14 - Knelson Concentrator

Figure 18-15 - Gold Room Furnace

Figure 18-16 - Gold Room Table

Figure 18-17 - Gold Room Vault

Figure 18-18 - Portal

Figure 18-19 – Decline (looking up)

18.1 Underground Workings

The underground workings are accessed through a portal and decline. The decline typically measures 5-6 metres wide by 3.5-4.0 metres high and descends at 8 ° (-14 %).

The decline is approximately 1,300 metres long and descends 100 vertical metres to access Saddles 1 through 4. Stoping (mining) was carried out on Saddles 1 through 3. Those saddles were not mined out. Limited mining has been carried out on Saddle 4.

The existing underground workings are described in Section 14.1.

18.2 Milling Facility

The mill originally used a gravity process consisting of crushing, grinding, jigging and tabling. Its capacity was in the neighbourhood of 200-300 tonnes per day of coarse mineralised rock. The mill was reconfigured for flotation recovery in 2006.

The mill was refurbished and upgraded prior to the restart of production in 2014, with a planned capacity of 300 tonnes per day (tpd). A milling rate of 245 tpd was achieved by August 20th, 2014.[17] The milling facility is further described in Section 17.

18.3 West Dufferin

West Dufferin's infrastructure is described in Section 4.5.

[17] Ressources Appalaches, Press Release, August 20th, 2014, "Production of 245 tons per day at the Dufferin Mine"

19 Market Studies and Contracts

19.1 Market Studies

Gold can be traded freely, through dealers or major banks.

19.1.1 Gold Price for PEA

For this PEA a gold price of $ US 1250 per troy ounce has been used. As illustrated in the kitco.com chart below, the price of gold has been between $US 1 000 and $US 1 400 per troy ounce for past three years.

Figure 19-1 Gold Price in US$/troy ounce, October 2006 - September 2016.

World mine production was estimated to be approximately 3 000 tonnes (96.45 million ounces) in 2015 (George, 2016).

19.1.2 Exchange Rate

For this PEA study we will use an exchange rate of $US0.75 to the Canadian dollar ($C1.33/$US). Table 23-1 and Figure 23-1 Gold Price in US$/troy ounce, October 2006 - September 2016. Below is a table showing the exchange rate between Canada and the USA over the last ten years. A troy ounce of gold is priced at $C1 733 or $55.7 per gram. A constant exchange rate was used for the project.

Table 19-1-1 Exchange rate between US and Canadian dollars, 2006 – 2016 (Canadian Forex).

Year	$US/$Cdn	$Cdn/$US
2006	$0.882	$1.134
2007	$0.935	$1.069
2008	$0.944	$1.059
2009	$0.880	$1.136
2010	$0.971	$1.030
2011	$1.011	$0.989
2012	$1.000	$1.000
2013	$0.971	$1.030
2014	$0.906	$1.104
2015	$0.783	$1.277
2016	$0.757	$1.321

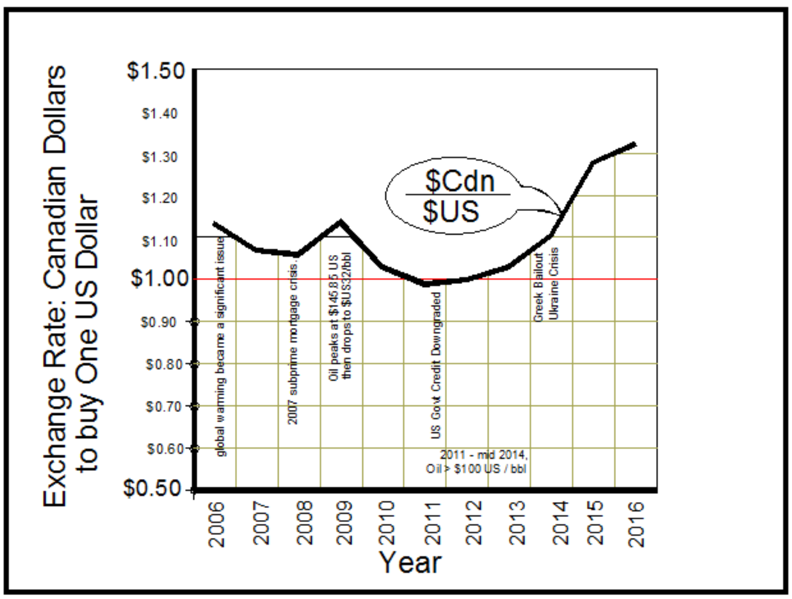


Figure 19-2 A graph of the Canadian - American exchange rate over the period 2006 – 2016.

Credit Suisse and BofA Merrill Lynch have predicted gold will be $1,500 going into 2017. Dutch bank ABN Amro forecasts gold to average $1,425 (Mining.com, August 2016).

The authors are not aware of any market studies or analyses completed by the issuer.

19.2 Contracts

The Property has two principle products, a gold doré bar and a gold concentrate. Any new production must establish contracts for transportation of these products to a refinery, and contracts for refining. During the 2014 production, doré bars were transported to the refinery by Brinks Global Services. The refinery, Johnson-Matthey, used silver credits from the doré bars as payment for refining. The existing contracts for transport and refining from 2014 are no longer in force.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental Assessment

An Environmental Assessment (EA) is required by the Nova Scotia Department of Environment (NSDE) for any project that applies for an Industrial Approval (IA) to operate. The EA is a comprehensive study that evaluates the potential and real environmental impacts that a proposed project must consider, and demonstrates a strategy to mitigate any such impacts. This study can take more than one year to complete it, as has to take into account seasonal conditions, migratory patterns and public consultation. This section of the report summarizes the available environmental studies related to the Project and discusses any material impacts that these studies identified.

20.1.1 1986 to 1994 – Seabright Resources / Seabright Explorations

The earliest formal environmental reports were submitted to government in January 1989 after Corner Bay Minerals Inc. (formerly Seabright Exploration Inc.) initiated exploration activity and had Porter Dillon Consulting apply for an Industrial Waste Permit. At that time tests were conducted by Porter Dillon primarily to classify the waste rock which was used to construct the existing access road to site. The report presented a reasonable indication of the materials' acid consuming potential in the general area of the proposed workings.

A Feasibility Study was completed in November of 1993 by Peak Engineering Ltd., which did not identify any direct impact on the existing water courses in the area, and concluded that there would only be minimal effect on existing natural lands during construction and operation for the following reasons:

- Mineralized rock and waste rock is non-acid generating and contains low levels of heavy metals such as copper and mercury. It should be noted, though, that mineralized rock and waste rock have high arsenic values;
- A small project area (under 10 ha) in which no sensitive environmental issues were identified;
- A project site which has been disturbed by previous activities and the current work will mitigate this existing ground exposure;
- Complete isolation of the project area from the hydrologic basin due to topography, creating a single point discharge sub-basin which can easily be engineered and monitored;
- A gravity extraction circuit for milling which relies entirely on water for processing and gold separation;
- A process circuit which isolates sulphide material from the silica and provides for a separate low tonnage sulphide waste stream which can be handled by bagging within the mill complex;
- Power and road access currently available on site so any impacts from these services have been defined;

- A milling/tailings water balance circuit which limits the total site discharge to the minimum level, i.e. Once startup is completed, process makeup water will be by reclaim from tailings, thus creating a closed mill circuit; and,
- A small work staff (approximately 25 persons) and no camps on site.

20.1.2 1994 to 2008 – Jemma Resources

In May 1994 an Environmental Assessment was conducted by Jacques, Whitford and Associates Limited (JWA) for Dufferin Resources Inc. The project described in the assessment was for a proposed mine/mill operation for 100 tonnes per day. The proposed mining would be carried out by advancing a sloping incline to the underground mineralization and mineralized rock processing would be by gravity separation methods (water separation process). JWA concluded that the Dufferin project was "considered to pose no environmental threat due to long term surface alteration of the area"[18] for the following reasons:

- The Dufferin site is located approximately 1.5 km from human habitation, with forestry industry workers and hunters being the only visitors to the area. The site was cleared by Corner Bay Minerals in 1988. The project involved approximately 10 hectares, an area similar to that envisaged by Corner Bay minerals.
- No chemicals were planned to be used in the gravity milling process. All sulphide minerals feed in the milling process were to be mixed with the backfill and returned underground. Approximately eighty-six percent of the waste rock mined underground was to be returned underground as backfill. The remaining fourteen percent of the waste was to be used to upgrade the access road and the immediate mine site. The greywacke and slates of the Goldenville formation were tested in 1988 by Porter Dillon and in 1993 by JWA for their study and were found to be net acid consumers.
- Tailings were to be collected behind an eastern dam in a naturally occurring basin-shaped depression immediately north of the mill. Designed and approved sedimentation ponds were to be installed and will be equipped with overflows. The compact site would drain to the tailings pond via perimeter diches and site grading.
- As the mine was not in the immediate vicinity of old workings, water inflows to the mine were expected to be minimal, with the majority of the water discharge from the mine consisting of drill and wash water. The mill and mine were to operate on water reclaimed from the siltation pond. In the temperate climate of the eastern shore of Nova Scotia, the clarified overflow from the siltation pond was expected to be that caused solely by rainfall.
- With the small crew required to operate the mine and the mill, sewage could easily be handled by a conventional septic system.

---

[18] Registration Form, Dufferin Gold Mine, Section 2.5, page 22, by Peter Pheeney, MScE., P.Eng., JWA Ltd., May 9, 1994

20.1.3 2008 to 2015 – Ressources Appalaches

In April 2013, an Industrial Approval Application was prepared by Conestoga-Rovers & Associates (CRA) for submission to the Nova Scotia Department of Environment on behalf of Dufferin Ressources Inc. The Industrial Approval Application Supporting Documents included the JWA Environmental Assessment. Interviews with both the then-current General Manager and President of Ressources Appalaches (RA), who were the operators of the mine and who commissioned the Industrial approval, stated that the existing EA was accepted by the NSDE under 2 conditions:

1) The project would remain within the same footprint identified in the 1994 JWA report. The major concern was the construction and rehabilitation of the existing tailings management facility (TMF).
2) Prior to any approval for constructing/extending the existing TMF or amending the existing IA for 300 tonnes per day to 575 - 600 tonnes per day, there would be consultation and review with government to evaluate what additional environmental studies would be required, if any.

Ressources Appalaches commissioned a memo from Conestoga-Rovers & Associates, outlining the permitting steps needed to increase the mining rate beyond the already-permitted 300 tpd rate. Such an increase is outside the scope of this report.

20.2 Waste and Tailings Disposal

20.2.1 Tailings Management Facility

The Tailings Management Facility (TMF) is designed for the permanent storage of tailings/waste rock, collection/ recycling of process water and settlement of tailings slurry prior to discharge to the environment if required.

The facility is constructed of several dams (North, South and Internal) within the current approved footprint of 4.2 ha. It was designed to retain 115 000m^3 of material, consisting of 118,000 tonnes of tailings and approximately 53,000 tonnes of waste rock (latest available survey). The dams were designed to be built to an elevation of 66 m, with tailings to a maximum elevation of 65.5 m and waste rock to a maximum elevation of 69 m.

The AMEC Environmental & Infrastructure Ltd. (AMEC) Industrial Approval Document for the mine states that the facility has a demonstrated ability to operate effectively and produce effluent that meets applicable guidelines. The completion of two years of operation is an important milestone with respect to funding for this project, and will mark the deposition of 119,000 tonnes of tailings within the current approved footprint (4.2 ha) of the tailings facility, with another 91,000 tonnes to be placed underground.

The original submission to construct a TMF was part of the JWA 1994 Environmental Assessment. There are no as-built drawings to confirm the construction of these dams. The design was not strictly adhered to in regards to the construction of the north berm and final elevations, which resulted in less storage capacity than the design called for.

In May 2013, AMEC submitted a Tailings Management Plan to government on behalf of RA. Under Section 3.2 Tailings facility Configuration for Initial Two Years, AMEC address this by confirming that the final construction of the TMF would be in configuration with the 1994 plan.

Construction works were carried from January 2014 until May 2014 for the construction of the principal tailing dam and related structures. Work was stopped at that point in order to begin the mill operation, and the dam modifications were not completed. Additional work on the tailings management facility would need to be carried out prior to restarting the mill.

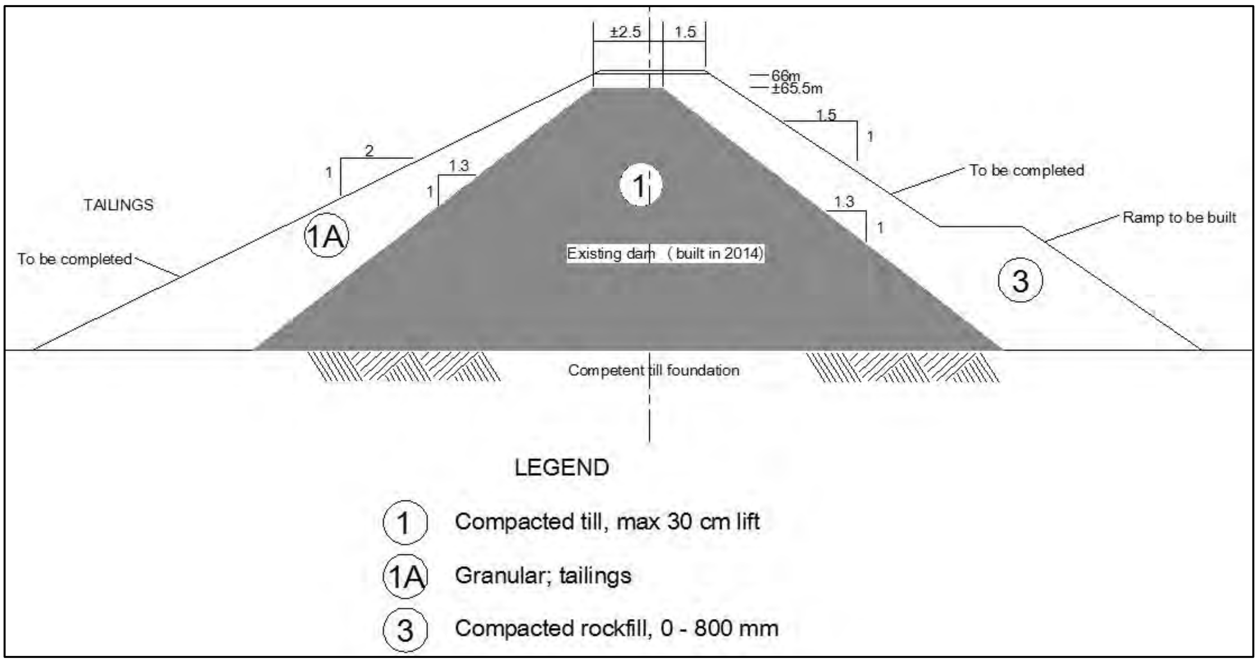


Figure 20-1 - Principal Dam cross-section

Figure 20-1 presents a cross-section of the principal existing dam. The grey area shows the existing construction and the lines are showing what is left to be done.

- The dam crest needs to be completed to elevation 66 m and widened to 4 m wide. The toe of the dam is at about 55 m elevation.
- The slopes need to have a 1.5H: 1V gradient.
- Tailing planned upstream will be transported and dumped from the crest.
- The spillway needs to be corrected. This includes the slope correction, installation of the geomembrane and the geogrid with the concrete in the cells.
- When the dam is completed to final elevation and slopes, it is recommended to install monitoring wells to obtain in place density information and to follow the water levels within the dam.
- Survey of the final geometry is required for the as built drawing production.

There was no current Tailings Deposition Plan or Tailings Operations Maintenance and Surveillance Manual available for review and both are a requirement of the Environmental Assessment and Canadian Dam Association Guidelines (CDAG). Also it is a requirement of the CDAG that an inspection be done on any TMF within one year of new construction or significant alterations.

20.2.2 Water Balance

The mine is designated as a zero discharge mine (< 50m^3/day). The application for approval (CRA-2013) made the following statements:

- Water for process start-up will come from a number of sources, such as drilled wells and from underground workings.
- It is anticipated that reclaimed tailings water, mine water and captured as rain water in the siltation/polishing pond will be sufficient to provide a closed circuit process loop.
- During the periods of high precipitation, any surplus water in the tailings area will result in the excess water being stored for reuse or released through the Polishing Pond.
- Once the mill is fully charged, reclaimed water from the tailings run and the majority of water from the thickener process will provide stability in the mill's daily water requirements. However, water loss in the mill process and due to evaporation in the tailings area will require that some water be required from sources other than those indicated here.
- The water storage tank on site will assist with providing water through a controlled header system allowing feed to all areas requiring water without interruption in flow. These tanks are required to have high and low level indicators to control the discharge and inflow to provide the water required to operate the circuits.

There is no definitive water balance for this site to date to validate the above statements. There were plans to drill a well and apply for a Water Withdrawal Permit but neither was completed. The current TMF has a sufficient volume of water stored in it to start up the mill but if the water balance is calculated and found to be negative then one or both of these additional water sources will be required. If the water balance is calculated to be positive, then a plan for water management and tailings deposition will need to be followed to ensure long term supply of process water and discharge does not exceed 50m^3 per day (triggering point for Metal Mining Effluent Regulations).

A preliminary water balance was carried out for this report.

Climate data was obtained for the Halifax International Airport[19]. Total yearly precipitation over the 20 year period was about 1200 mm of rain and 220cm of snow for a total of about 1400mm rain equivalent. The tailings pond catchment area is approximately 5 ha or 50,000 square metres. Precipitation over the year would add about 70,000 cubic metres of water.

---

[19] http://climate.weather.gc.ca/climate_normals/index_e.html#1971

From October to April, the most frequent winds are from the West, Northwest and North. From May to September, the most frequent wind direction is from the South.

Some water comes into the plant with the feed; from 2% to 5% is expected. Most of the water for the process is added to the ball mill. In the plant, water is extracted from the flotation concentrate in the filter press. The mine requires approximately 4 cubic metres of water for every tonne of rock processed. At least 60% of the water can be recycled.

Water will be lost to evaporation, infiltration to the ground, and in the pore space of the tailings material.

Table 20-1 - Estimated water balance.

300 tpd solids	**Range**	
Daily Water Balance	**m^3**	**m^3**
Water in ROM Rock	6	15
Water with tails, 50% solids	300	300
Surface Water, Average	5	8
Mine Discharge Water	200	400
Subtotal, Water into TSF	*511*	*723*
Water Evaporation & Infiltration	80	110
Water Retained with Tails	150	200
Subtotal, Water Loss	*230*	*310*
Water available for Recycling	*281*	*413*

The mine makes between 200 and 400 m^3 of water per day. During operations, some of this water will be used in the operation. Past experience indicates the mine will be drier with depth and time. The amount of water that will discharge from the site has not yet been measured; however, we believe there is adequate water to make up for evaporation and infiltration losses.

20.2.3 Surface and Groundwater Chemistry

Groundwater samples were taken in 1994 after the wells were drilled and were again sampled in January 2004. All values were well within Metal Mining Liquid Effluent Regulation limits.

Table 20-2 - Results of groundwater analyses from January, 2004. Samples M1 and M2 correspond to the camp potable water well and the process make-up water well, respectively.

		Sample			
Determination	**Unit**	**M-1**	**M-2**	**M-3A**	**M-3B**
pH	units	7.45	7.40	5.52	6.25
Colour	CU	2.5	5.0	7.5	6.0
Conductivity	umho/cm	162	295	138	228
Turbidity	NTU	0.68	0.20	5.40	1.65

		Sample			
Total Suspended Solids	mg/L	0.67	0.67	11460	712
Sodium	mg/L	8.66	13.70	7.27	11.60
Potassium	mg/L	0.93	3.11	13.20	6.72
Calcium	mg/L	14.51	19.60	11.73	17.00
Magnesium	mg/L	6.56	16.50	4.71	9.43
Hardness	mg/L	63.2	116.8	48.7	81.2
Alkalinity	mg/L	62.60	135.0	8.10	56.60
Sulfate	mg/L	9.46	35.80	30.45	37.04
Chloride	mg/L	3.54	13.47	3.54	9.93
Total Phosphorus (P)	mg/L	<0.02	<0.02	<0.02	<0.02
Nitrate+Nitrite (N)	mg/L	0.15	3.80	3.00	6.10
Ammonia(N)	mg/L	0.17	0.29	0.10	0.22
Aluminum	mg/L	0.18	0.06	0.03	0.06
Arsenic	mg/L	0.006	0.036	<0.002	<0.002
Barium	mg/L	<0.02	<0.02	<0.02	0.03
Beryllium	mg/L	<0.005	<0.005	<0.005	<0.005
Boron	mg/L	<0.5	<0.5	<0.5	<0.5
Cadmium	mg/L	<0.001	<0.001	<0.001	<0.001
Chromium	mg/L	<0.01	<0.01	<0.01	<0.01
Cobalt	mg/L	<0.002	<0.002	<0.002	0.020
Copper	mg/L	0.005	0.004	0.004	0.006
Fluorine	mg/L	0.08	0.11	0.09	0.09
Iron	mg/L	<0.01	<0.01	0.16	0.09
Lead	mg/L	<0.002	0.002	<0.002	<0.002
Manganese	mg/L	<0.002	0.011	1.510	5.648
Nickel	mg/L	<0.002	0.011	0.004	0.015
Mercury	mg/L	<0.001	<0.001	<0.001	<0.001
Silica	mg/L	2.0	3.9	1.1	1.6
Tin	mg/L	<0.02	<0.02	<0.02	<0.02
Vanadium	mg/L	<0.02	<0.02	<0.02	<0.02
Zinc	mg/L	0.01	<0.01	<0.01	0.05
Total Oil & Grease	mg/L	<5	<5	<5	<5

Table 20-3 - Metal mining liquid effluent regulations.

Substance	Maximum Monthly Arithmetic Mean Concentration	Maximum Concentration in a Composite Sample	Maximum Concentration in a Grab Sample
Arsenic	0.5 mg/L	0.75 mg/L	1.0 mg/L
Copper	0.3 mg/L	0.45 mg/L	0.6 mg/L
Lead	0.2 mg/L	0.3 mg/L	0.4 mg/L
Nickel	0.5 mg/L	0.75 mg/L	1.0 mg/L
Zinc	0.5 mg/L	0.75 mg/L	1.0 mg/L
TSM	25.0 mg/L	37.5 mg/L	50.0 mg/L
Radium-226	10.0 pCi/L	20.0 pCi/L	30.0 pCi/L

A surface and groundwater monitoring program has been established for the site as part of the Industrial Approval No. 2013.086059. Various monitoring well and surface water sampling points are being used to provide long-term surveillance of water chemistry for the life of mine and closure of the operation. All stations are monitored for water quality, general chemistry and heavy metal content on a quarterly basis. There has also been a Compliance Monitoring Station established which is located at the effluent discharge of the polishing pond. In addition to the quarterly sampling program mentioned above, toxicity testing is conducted using the 96-hour Acute Lethality Test of Effluent to Rainbow Trout.

To date all sampling reports are up to date and there have been no exceedances of any of the monitored parameters.

20.3 Permitting Requirements

This section discusses approval requirements for the current and possible future operations. Table 20.1 shows in chronological order the major environmental related undertakings the Dufferin Mine has been granted and the approximate costing for each item.

Table 20-4: Completed Environmental Permitting Items (budgetary estimate) (Parks, 2014).

Item	Cost	Date of Approval
Environmental Baseline Studies	$345,000	Mar. 28th, 2013
Environmental Assessment Registration	$75,000	
Industrial Approval	$125,000	Nov. 26th, 2013
Subtotal	*$545,000*	
Contingency (20%)	$110,000	
Total	**$650,000**	

The Environmental Baseline Studies were performed by JWA and the subsequent Environmental Assessment Registration followed in May 1994. This EA was transferred to Dufferin Ressources Inc. in a letter from the Minister of the Environment on March 28, 2013. An Environmental Assessment is necessary component of an Industrial Approval Application. Once given, Environmental Assessments do not expire.

The Environmental Assessment approval would need to be transferred from Dufferin Resources Inc. to the new corporation, Maritime Dufferin Gold Corp ("Maritime Dufferin"). This transfer process was ongoing at the time of report writing. Nova Scotia Environment's major concern surrounded the state of the tailings facility and Maritime Dufferin's plans for it. In response, Maritime Dufferin submitted an updated tailings plan in December 2016.

An application for an Industrial Approval (IA) was submitted by CRA on April 2013. RA received the Industrial Approval from the Nova Scotia Environment (NSE) on November 26, 2013 (Industrial Approval No. 2013.08059). The IA is currently valid, and expires on September 1, 2018. This is the approval to begin mining and milling operations.

Ressources Appalaches has a five year written lease agreement for 6 acres of land to the east of the TMF's principal dam on which the pre- settling and polishing ponds are situated. These 6 acres sits on a larger parcel of land PID 40246472 owned by John McLellan. RA has agreed to minimize environmental disturbances during operations and rehabilitation of this land subsequent to the mine closure.

A Mining Lease is a requirement of DNR before any mining activities can take place on site. The Mining Lease # 94-2 for this property expired in September 2014 and has since been converted into an Exploration License.

A closure plan was accepted by the Department of Natural Ressources in a letter dated November 18, 2014 from Tom Lamb, Mining Engineer, Mineral Development and Policy Section. This will be discussed in more detail in the next section.

A reclamation bond is financial assurance that a company puts in trust with Nova Scotia Government to ensure taxpayers of the province do not have to pay for the rehabilitation of the project and associated costs in the event the company has financial troubles. This amount is calculated from the approved closure plan.

A Water Withdrawal Permit may be required for mill process water as stated in Section 20.2. CRA was approached to do preliminary investigation and consultation with NSE. The associated consulting and submission was estimated at approximately $ 18,000 but that also included a submission of an Environmental Assessment for the mine expansion activities. This process is expected to take several weeks. There has been past approval for water withdrawal for the previous operators of the mine from Eagle Lake. In addition to the water withdrawal permitting will be the infrastructure costs, such as setting up a pump and pump shed and run electrical and piping to Eagle Lake, the closest available surface water source.

An amendment to the IA to increase production may require an amendment to the existing EA. As discussed in Section 20.1, CRA was asked to develop a program that supplements existing environmental work and additional work the government may require. At this point it would be premature to estimate what the cost would be as it is unclear what will be required by the government.

Ressources Appalaches, through Dufferin Resources, was required to develop a Code of Practice before starting dewatering and rehabilitation of the old workings. This was submitted on behalf of RA by MineTech International Ltd. in April 2012. Once the mine was in the stage of production the Code of Practice was no longer valid and the operation then needed to comply with the Nova Scotia Mining Regulations. The mine has been on temporary Care & Maintenance since November 15, 2014 but still operates under the mining regulations. In addition Dufferin Mine staff and MineTech International have developed a Care & Maintenance Manual that supplements the mining regulations and protects the safety of workers, the environment and the security of the mine and all associated assets.

20.4 Social and Community Related Requirements

For any mining company to be successful in today's mining industry, it must be committed to environmental protection through an open information policy and a willingness to meet and address local concerns at any stage of the project. The 1994 JWA EA describes the outreach policy to the community at that time was the formation of a Community Liaison Committee (CLC). This committee was comprised of interested citizens from the local area and empowers the public to have a voice on any concerns.

The Industrial Approval Application prepared by CRA in April 2013 outlined the proposed public engagement plan for the project. The plan submitted was to again form a CLC for the mine before mineralized rock was extracted. The protocol that was to be used was developed in part by NSDE.

Individuals and businesses in the communities surrounding the Property were among Ressources Appalaches' creditors when Ressources Appalaches entered receivership. It should be expected that these unpaid bills will strain the community-company relationship.

Under Nova Scotia Environmental Legislation, a CLC can be dissolved with government approval if the company demonstrates there are no issues or concerns or the frequency of the meetings can be reduced. It should be noted that the 2013 CRA IA does not make any reference specifically to the CLC being a condition of the approval.

In full production, the mine is a significant regional source of employment. Due to the small population and historical industry in Sheet Harbour, there will be a need to hire from outside the area for some positions and therefore housing, meals and travel has and will remain a consideration. There are no plans to build infrastructure on site to accommodate these considerations. The past practice was to provide room and board to employees who are not from the local area by renting dwellings and hiring staff to cook and maintain these dwellings.

20.5 Mine Closure/Reclamation

The final stage of a mining project is closure of the site. This is achieved by returning the property back to a state as it was before production in regards to land and water that may have been disturbed by development. These sites are also monitored after completion of the rehabilitation activities to ensure long term stable conditions exist and no issues with the closure work have developed.

In the province of Nova Scotia, all mining projects are required to submit a reclamation bond, either in cash or equivalent security, equivalent to the full estimated cost of reclamation. The bond is returned as reclamation is carried out. See Section 20.3 for more details on the bond.

20.5.1 Ressources Appalaches (2013 to 2015)

A preliminary reclamation plan for mine site closure was prepared by AMEC in May 2013, and approved by the provincial government in November 2014. There was considerable correspondence between RA and DNR in which points were clarified and DNR asked for additional considerations and revisions. For this mine, the major reclamation items include:

- Removing of buildings and other surface structures;
- Removing the milling equipment from the site to be reused on another site or sold;
- Decommissioning and sealing of the portal, vent raise and other access points;
- Chemicals, fuels and explosives will be removed from site and disposed in accordance with all provincial and federal regulations;
- Tailings will be capped and vegetated;
- Surface contouring of the mine and mill site, including the area of the tailings impoundment;
- Monitoring for 3 years after reclamation is complete; and,
- The site will be fully reclaimed 2 years after production ends.

The government still requires a report to be submitted by the site owner on the characterization of the tailings and waste rock to the NSE and DNR. The final design of the tailings cover shall include considerations of the results of the tailings characteristics.

Total closure costs were estimated in the 2013 AMEC report to be approximately $1,150,000. However, as those costs were based on an assumption that more than 1m tonnes would be mined, the authors of this report do not consider them relevant to mine life under consideration in this report.

21 Capital and Operating Costs

21.1 Capital Costs

All capital items will be purchased during the pre-production period and the first three months of production. It is assumed that the majority of equipment and facilities will be acquired through lease agreements. Capital equipment costs used in this model are based on leasing of good quality used equipment. Capital costs are summarized in Table 21-1**.**

21.1.1 General and Administration (Overhead) Capital

21.1.2 Mine Development

A significant portion (approximately 80%) of the underground development would be considered a capital expenditure for accounting purposes. Main ramps, haulage drifts, and ventilation raises make up the majority of this type of development, with refuge stations and sumps/pumping stations making up the remainder.

21.1.3 Capital Labour Allotment

In the PEA, the first half years' work is essentially all mine development and exploration. After the first year of development, the mine should be in a position to go into continuous production. Mine labour and maintenance labour were separated out from the mine development costs. Activity costs were allocated to either capital or operating depending on the activity. When an activity was used for capital and operating activities, a ratio was calculated (on a monthly basis) between development and operating volumes, and applied to the total labour costs to divide into the appropriate section.

21.1.4 Mine Equipment

A small fleet of mobile equipment will be required to carry out the proposed mine plan. Costs for mine equipment were based on quotations from equipment suppliers, mine engineering handbooks, industry surveys, past experience at the Dufferin property, and MineTech experience. Some major equipment can be leased or rented as mine development and operations require.

Table 21-1: Capital costs.

Major Items and startup inventory		cap cost	total Year 1	Year 2
Pre-Production Mine Exploration & Development Year			$3,265,000	
Upgrade Tailings Facility	1	$250,000	$250,000	
crusher and conveyor distribution system	1	$25,000	$25,000	
Plant Inspection and Rehabilitation Planning	0	$50,000	$0	
Jackleg drills	12	$4,000	$48,000	
Stoper Drills	12	$4,000	$48,000	
Bit Sharpener	1	$20,000	$20,000	
Slusher buckets & blocks	12	$1,000		$12,000
Scoop Tires	4	$5,000	$20,000	
Truck Tires	12	$2,000	$24,000	
Main Fan and spare main fan plus motor	2	$61,000	$122,000	
Level fans	2	$17,000	$34,000	
Stope Fans	10	$5,000		$50,000
Vent monitoring equipment	1	$8,000	$8,000	$8,000
Air Heating System	1	$81,000	$81,000	
Small Face Pumps	12	$3,000	$36,000	
Tugger Hoist	6	$11,000		$66,000
Cap lamps	50	$200	$10,000	
Transformers	1	$34,000	$34,000	
First Aid	8	$1,000	$8,000	
Communications System	1	$25,000	$25,000	
Grout Pump	1	$19,000	$19,000	
Grizzly, Rock Breaker	1	$25,375	$25,375	
Furnace for doré	0	$60,900	$0	
Filter Press	1	$20,000	$20,000	
Upgrade ball mill	1	$20,000	$20,000	
Overhaul Thickener	1	$20,300	$20,300	
Cone Crusher rebuild	1	$30,450	$30,450	
Fine Mineralized Rock bin	1	$50,750	$50,750	
Cover all outside conveyors	1	$12,800	$12,800	
Falcon Concentrator	1	$101,500	$101,500	
New building for laboratory	1	$20,300	$20,300	
Self-Contained Breathing Apparatus	1	$50,000	$108,000	
Overhead crane / shop	1	$203,000		$203,000
Bobcat & spare parts	1	$15,000	$15,000	
Fire-Extinguishing Equipment	1	$5,000	$5,000	
Refuge Station Equipment	1	$5,000	$5,000	
skid mounted shotcrete pump	0	$50,000	$0	
Mine Drainage Pumps, starters, electrical Equipment	1	$150,000	$150,000	
Armoured Electical Cable & Electical Equipment	1	$40,000	$40,000	
Shop and Office Equipment	1	$250,000	$250,000	
Substation upgrade	0	$100,000	$0	
Personal Safety Equipment	1	$25,000	$25,000	
Battry Charging Equipment	1	$50,000	$50,000	
Initial Mine Ground Control Inventory	1	$50,000	$50,000	
Initial Inventory Fuel & Lubricants	1	$25,000	$25,000	
conveyor magnet, 1.2m long, 65 kg.,	0	15000	$0	
Fuel tank, 80,000 Litre, 4.57m diam., 4.87m high, 5.5 tonnes	0	63700	$0	
Plant & Plumbing	1	50000	$50,000	
Laboratory	1	100000	$100,000	
Engineering & Constr. Supervision	1	40000	$40,000	
Diamond Drilling				$320,000
Subtotal		Subtotal Rounded	$5,300,000	$700,000
Miscellaneous equipment and Contingency, 20%			$1,060,000	$140,000
Total			$6,360,000	$840,000

21.1.5 Reclamation Bond

A placeholder has been used in the economic model for the reclamation bond. A value of $1,200,000 was used, spread over the first year of production.

21.2 Operating Costs

21.2.1 Mine Operating Costs

Mine operating costs were broken down by personnel, materials, general and administration and surface expenses. The average operating cost was estimated to be $C108/tonne.

21.2.2 Processing Operating Costs

Process operating costs were estimated to be $C25 per tonne.

22 Economic Analysis

CAUTION TO READER
This preliminary economic assessment (PEA) is preliminary in nature, in that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. A PEA is a conceptual study of the potential viability of mineral resources. The economic viability of the inferred mineral resources has not been demonstrated.

22.1 Assumptions

This economic analysis is for the Dufferin Mine Project and Dufferin West (Salmon River) project. It does not include considerations for outstanding debts from previous operations, nor does it take in to account any possible corporate tax credits that a potential buyer may hold from other operations.

Mineral resources for the Dufferin property are discussed in Section 14 and tabulated in Table 14-23. A six-month pre-production development is followed by six months of stope mining, ramping up to full 300-tpd production at the end of Year 1. Mining dilution was 20.0% and the metallurgical recovery in the gravity-flotation mill was estimated at 95.0% of the head grade. The remaining 5 % is lost to the tailings in this analysis.

The average operating cost per tonne through the mill was $108/tonne.

Table 22-1: Summary of Cash Flow items

(NOTE ******** THIS ORDER OF MAGNITUDE STUDY USES INFERRED RESOURCES. ********)									
Base Case, $US 1,250/oz. Au. Dufferin Underground Gold Mine, Gravity-Leach Plant.		"MR" is Mineralized rock going into the mill from underground stoping.							
Capital Plant and Equipment	$9,848,000								
Revenue									
Column1	**development, 6 month ramp up to**	**Full Production Year 1**	**Production Year 2**	**… …**	**Production Year 7**	**Production Year 8**	**Production Year 10**	**Production Year 11**	**Production Year 12**
6 mo. Develop & build stockpile				… …					
East Dufferin "MR"Mined	40,000	80,000	88,000	… …	80,000				
Grade of material sent to mill	8.20	8.20	8.20	… …	8.20				
Stope Dilution (20%)	8,000	16,000	17,500	… …	16,000				
diluted Grade	6.83	6.83	6.83	… …	6.83				
Low Grade stockpile	34,500	51,800	69,100		155,600	138,300	138,300	138,300	
Low Grade Processed				… …				105,000	33,300
East Dufferin Tonnes sent to Mill	48,000	95,600	122,500	… …	113,000				
Gold in muck sent to mill stockpiles (ounces)	10,550	21,000	23,100	… …	21,000			6,752	2,141
West Dufferin "MR"Mined					**10,000**	**50,000**	**104,000**		
Sill Level, raises Development Muck (Low Grade)				… …		10,100	10,100	10,100	
Grade of development Muck, g/t	2.00	2.00	2.00		2.00	2.00	2.00	2.00	2.00
Grade of Stope "MR"	6.13	6.13	6.13		6.13	6.13	6.13	6.13	6.13
Stope Dilution (20%)					2,000	10,000	20,866		
West Dufferin Tonnes sent to Mill stockpiles					12,000	70,118	135,314	10,118	0
West Dufferin Gold in muck sent to mill stockpiles (ounces)						10,505	21,212	651	2,602
Total tonnes to Mill stockpiles East & West Dufferin	**48,000**	**95,600**	**105,000**	… …	**108,000**	**60,000**	**125,000**	**105,000**	**74,000**
East Dufferin Decline Advance, X-Cuts, tonnes Waste Mined	26,400	13,200	13,200						
Total Grams into mill	328,000	653,500	718,900	… …	653,500	326,700	659,800	230,200	147,500
Metallurgical Recovery									
Gravity-Flotation	95.0%	95.0%	95.0%	… …	95.0%	95.0%	95.0%	95.0%	95.0%
gold Recovery from Gravity-Flotation	311,600	620,800	682,900	… …	620,800	310,400	626,800	218,700	140,200
	10,000	20,000	22,000	… …	20,000	10,000	20,200	7,000	4,500
gold value, per troy ounce, US$	$1,250	$1,250	$1,250	… …	$1,250	$1,250	$1,250	$1,250	$1,250
Revenue bef"MR"insurance, freight, refining, US$	$12,522,700	$24,950,300	$27,445,300	… …	$24,950,300	$12,474,400	$25,189,700	$8,790,100	$5,632,900
Sales, Insurance, freight US$	$250,500	$499,000	$548,900	… …	$499,000	$249,500	$503,800	$175,800	$112,700
Gross Revenue (US$)	$12,272,300	$24,451,300	$26,896,400	… …	$24,451,300	$12,224,900	$24,685,900	$8,614,300	$5,520,300
Canadian $ / US $	$1.33	$1.33	$1.33	… …	$1.33	$1.33	$1.33	$1.33	$1.33
Gross Revenue (CDN $)	$16,322,100	$32,520,200	$35,772,200	… …	$32,520,200	$16,259,200	$32,832,200	$11,457,100	$7,342,000
Royalties (Approximate totals only, $CDN)									
1% NS Provincial Royalty ($Cdn)	$3,520,000								
Nycon Resources Royalty (Dufferin West, $Cdn)	$2,925,400								
Lascaux 1% Vendor Royalty, Dufferin East($Cdn)	$13,504,000								
Gross Revenue ($CDN)	**$13,758,900**	**$29,595,000**	**$32,814,500**	… …	**$31,863,100**	**$15,764,700**	**$31,833,800**	**$11,215,300**	**$7,152,500**
Operating Costs									
Total Operating Costs	**$6,159,200**	**$13,729,000**	**$14,170,000**	… …	**$14,083,600**	**$5,450,000**	**$7,264,600**	**$5,466,300**	**$3,883,200**
Operating Cost per troy ounce	$615.92	$688	$645		$706	$546	$360	$777.34	$862
Reclamation						$250,000	$250,000	$250,000	$250,000
Net Operating Income (Gross Margin)	**$7,599,600**	**$15,866,000**	**$18,644,400**	… …	**$17,779,600**	**$10,564,700**	**$24,819,200**	**$5,999,000**	**$3,519,400**
Net Income After Interest and Taxes									
Net Income Bef"MR"Interest and Taxes	7,599,600	15,866,000	18,644,400	… …	17,779,600	10,564,700	24,819,200	5,999,000	3,519,400
Less: Interest	378,000	756,000	756,000	… …	0	0	0	0	0
Taxes Payable	870,900	3,554,500	4,200,500	… …	4,233,700	2,556,300	5,870,500	1,494,800	918,300
Net Income After Interest	**0**	**11,555,500**	**13,688,000**	… …	**13,545,800**	**8,008,400**	**18,948,700**	**4,504,200**	**2,601,100**
Cash Flow Statement									
Net Income After Interest	0	11,555,500	13,688,000	… …	13,545,800	8,008,400	18,948,700	4,504,200	2,601,100
Add payback of Working Capital (no salvage value)									
Less Additions to Capital (After Year 1)			0		378,000	378,000	378,000	378,000	378,000
Less Borrowing	$9,848,000	$840,000	$492,400	… …	$2,000,000	$200,000	$200,000	$200,000	$0
Cash Flow	-9,848,000	10,715,499	13,195,575		11,167,820	7,430,439	18,370,749	3,926,216	2,223,120
Cash Flow per share									
Year	p up to Production	Production Year 1	Production Year 2	… …	Production Year 7	Production Year 8	roduction Year 10	roduction Year 11	roduction Year 12
Cumulative Cash Flow (CCF)	-9,848,000	10,715,500	13,195,600	… …	11,167,800	7,430,400	18,370,700	3,926,200	2,223,100
Capital Outstanding (beginning)	0	-9,848,000	867,500	… …	70,756,400	70,756,400	78,186,800	94,388,800	96,557,500
Capital Outstanding (end)	-9,848,000	867,500	14,063,100	… …	81,924,200	78,186,800	96,557,500	98,315,000	98,780,700
NPV05	$89,193,000	$67,062,400	US$						
NPV7.5	$75,852,000	$57,031,600	US$						
NPV10	$64,972,000	$48,851,100	US$						
Internal Rate of Return	121%								
Payback Period (Years)	1.08								

Table 22-2 Sensitivity Analysis Results (Millions of $Cdn.)

% Change	Exchange Rate	Operating Cost	Grade	Capital Cost
50%	$182	$48	$182	$83
40%	$163	$59	$164	$85
30%	$145	$67	$145	$86
20%	$126	$74	$126	$87
10%	$108	$82	$108	$88
base case	$89	$89	$89	$89
-10%	$71	$97	$71	$90
-20%	$52	$104	$52	$92
-30%	$33	$112	$33	$93
-40%	$15	$119	$15	$94
-50%	-$4	$127	-$4	$95

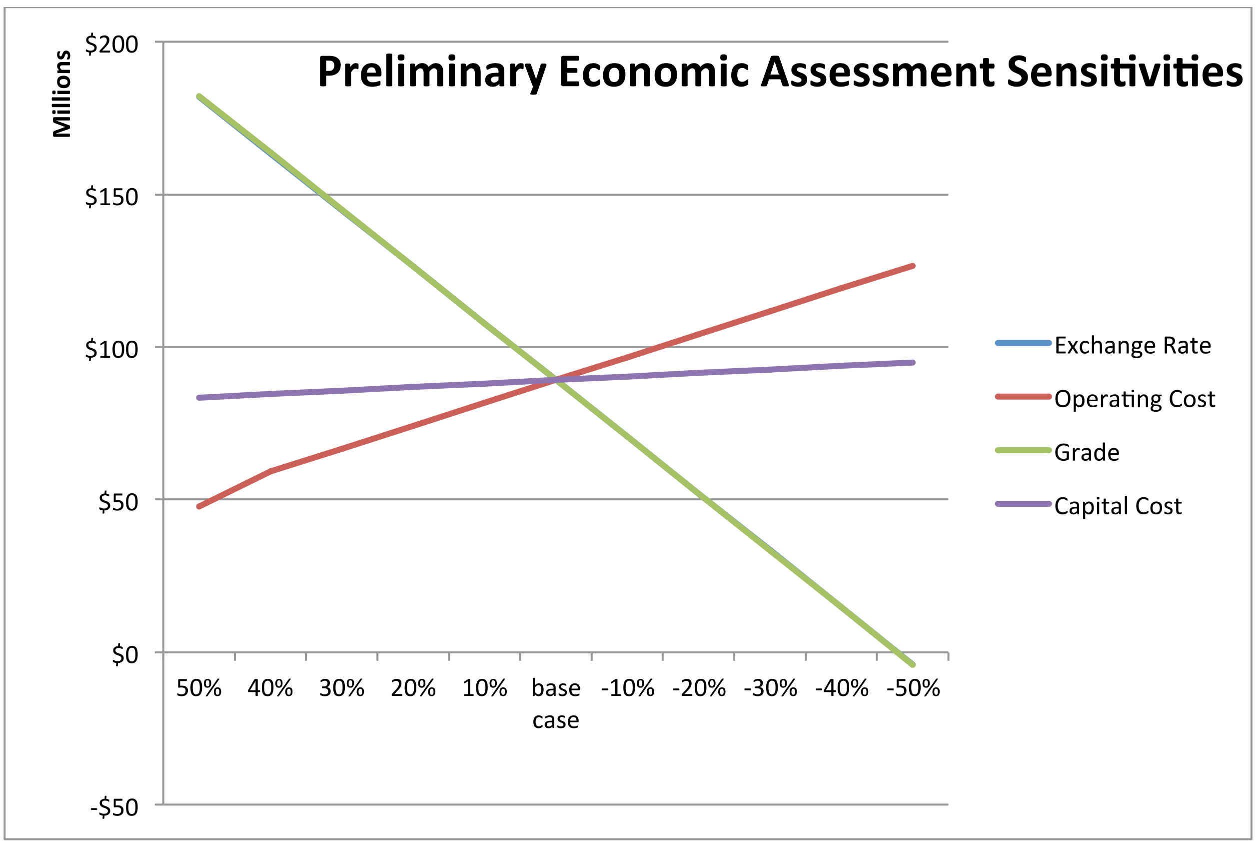


Figure 22-1 PEA Sensitivity to grade, exchange rate, operating cost and capital cost.

22.2 Taxes & Royalties

The province of Nova Scotia has a corporate tax rate of 3% for the first $350,000 per year, and 16% for anything greater. For this PEA we simply used a 16% tax plus $100,000 per year municipal taxes.

The net federal corporate tax rate is 15%. A Provincial royalty of 1% of NSR was also subtracted as well as the payments and royalties to the venders.

22.3 Net Present Value & Internal Rate of Return

An after tax cash flow of $89 million was estimated for the project. The pre-tax and post-tax cash flow estimates are summarized below.

Table 22-3 Preliminary Economic Assessment parameters.

Gold price	USD$1,250
Exchange rate USD-CAD	0.75
Total tonnes processed (includes dilution and development material)	1,231,000 tonnes
Average processing rate	290 tonnes per day
Diluted head grade East Dufferin	6.83 g/tonne Au
Diluted head grade West Dufferin	5.46 g/tonne Au
Gold recovery rate	95%
Average annual gold production	21,604 ounces Au
Capital, working capital, pre-production expense	CAD$9.85M

Table 22-4 Preliminary Economic Assessment results.

Pre-tax NPV_5	CAD$121.1M
Pre-tax IRR	158%
Post-tax NPV_5	CAD$89.2M
Post-tax IRR	121%
Payback	1.3 years
Average cash cost per ounce	CAD$617
Mine life	10 years
Total gold ounces recovered	216,050
Cumulative pre-tax cash flow	CAD$170.4M
Cumulative post-tax cash flow	CAD$126.3M

23 Adjacent Properties

The authors did not independently verify the information about adjacent properties contained in this section. Mineralization of the adjacent properties discussed below is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

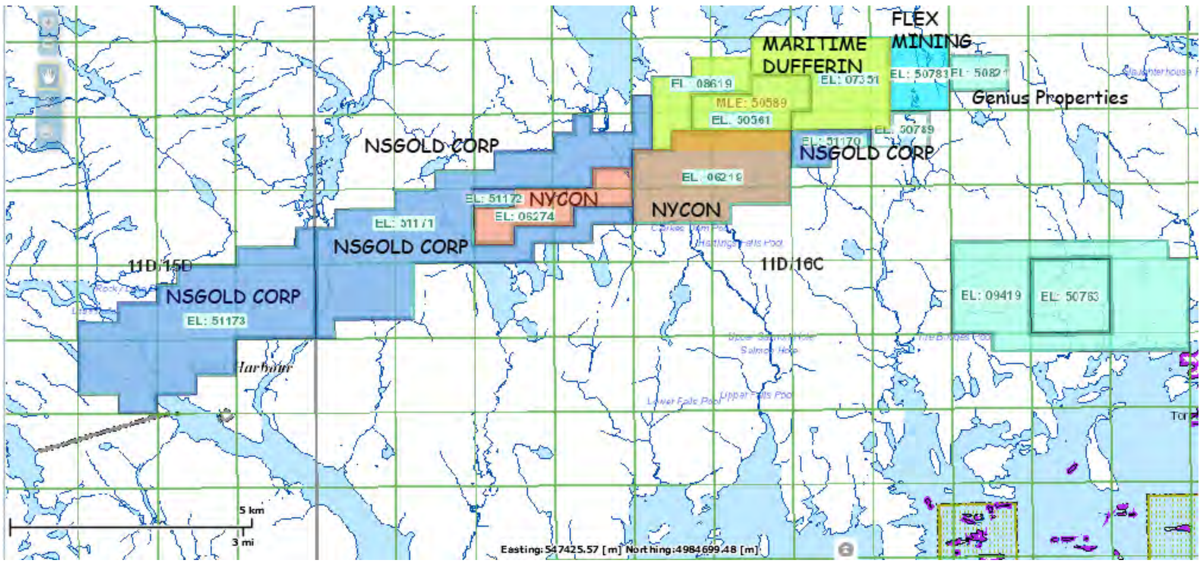


Figure 23-1: Adjacent Properties along trend with the RCGC controlled properties.

Table 23-1: Adjacent Property mineral rights Description.

Right Number	NovaROC ID	Holder Name	Right Type	Location	Issue Date	Expiry Date	Age	No. of Claims	Status
6274	201519	NYCON RESOURCES, INC.	Mineral Exploration Licence	MAP 11D16C TRACTS 63 CLAIMS E,F,J,K,L,M,P,Q MAP 11D16C TRACTS 64 CLAIMS M,N,O,P,Q MAP 11D16C TRACTS 81 CLAIMS A,B	28/09/2001	28/09/2016	15	15	Active
6219	201519	NYCON RESOURCES, INC.	Mineral Exploration Licence	MAP 11D16C TRACTS 65 CLAIMS J,K,L,M,N,O,P,Q MAP 11D16C TRACTS 66 CLAIMS M,N,O,P,Q MAP 11D16C TRACTS 79 CLAIMS A,B,C,D,E,F,G,H,J,K,L,M MAP 11D16C TRACTS 80 CLAIMS A,B,C,D,E,F,G,H,J,K	26/07/1979	26/07/2017	38	35	0
8619	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 80 CLAIMS L,O,P MAP 11D16C TRACTS 89 CLAIMS B,C,F,G,H,J MAP 11D16C TRACTS 90 CLAIMS E,F,L,M	20/05/2009	20/05/2017	8	13	Active
50561	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 79 CLAIMS N,O,P,Q MAP 11D16C TRACTS 80 CLAIMS Q MAP 11D16C TRACTS 89 CLAIMS A MAP 11D16C TRACTS 90 CLAIMS A,B,C,D,G,H MAP 11D16C TRACTS 91 CLAIMS D,E	02/09/1986	02/09/2017	30	14	Active
7351	564782	Maritime Dufferin Gold Corp.	Mineral Exploration Licence	MAP 11D16C TRACTS 77 CLAIMS N MAP 11D16C TRACTS 78 CLAIMS N,O,P,Q MAP 11D16C TRACTS 90 CLAIMS J,K,P,Q MAP 11D16C TRACTS 91 CLAIMS A,B,C,F,G,H,J,K,L,M,N,O,P,Q MAP 11D16C TRACTS 92 CLAIMS D,E,M,N	17/05/2007	17/05/2017	10	27	Active
50783	203130	FLEX MINING & EXPLORATION LIMITED	Mineral Exploration Licence	MAP 11D16C TRACTS 92 CLAIMS A,B,C,F,G,H,J,K,L,O,P,Q	17/11/2015	16/12/2016	1	12	Active
50821	564740	Genius Properties Ltd.	Mineral Exploration Licence	MAP 11D16C TRACTS 93 CLAIMS E,F,G,K,L,M	31/12/2015	31/12/2016	1	6	Active
51170	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D16C TRACTS 78 CLAIMS E,F,J,K,L,M	05/10/2016	05/10/2017	1	6	Active
51171	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D16C TRACTS 89 CLAIMS D MAP 11D16C TRACTS 80 CLAIMS M,N MAP 11D16C TRACTS 81 CLAIMS C,D,E,F,G,H,J,K,L,M,O MAP 11D16C TRACTS 82 CLAIMS A,B,C,D,F,G,H MAP 11D16C TRACTS 83 CLAIMS A,B MAP 11D16C TRACTS 62 CLAIMS ALL MAP 11D16C TRACTS 63 CLAIMS B,C,D,G,H MAP 11D16C TRACTS 64 CLAIMS E,F,J,K,L MAP 11D16C TRACTS 59 CLAIMS E,M,N MAP 11D16C TRACTS 60 CLAIMS D,E,F,G,H,J,K,L,M,N,O,P,Q MAP 11D16C TRACTS 61 CLAIMS A,B,C,D,E,F,G,H,J,K,L	05/10/2016	05/10/2017	1	76	Active
51172	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D16C TRACTS 63 CLAIMS N,O	05/10/2016	05/10/2017	1	2	Active
51173	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D15D TRACTS 49 CLAIMS ALL MAP 11D15D TRACTS 72 CLAIMS A,B,C,D,H MAP 11D15D TRACTS 50 CLAIMS A,B,C,D,E,F,G,H,J,K,L,M,O,P,Q MAP 11D15D TRACTS 47 CLAIMS E,F,J,K,L,M,N,O,P,Q MAP 11D15D TRACTS 46 CLAIMS A,B,E,F,G,H,J,K,L,M,N,O,P,Q MAP 11D15D TRACTS 51 CLAIMS A,B,C,D,G,H	05/10/2016	05/10/2017	1	66	Active
51182	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D15C TRACTS 72 CLAIMS N,O,P,Q	05/10/2016	05/10/2017	1	4	Active
51183	202546	NSGOLD CORPORATION CORPORATION NSGOLD	Mineral Exploration Licence	MAP 11D15D TRACTS 60 CLAIMS O,P MAP 11D15D TRACTS 61 CLAIMS B,F,G				5	Active

24 Other Relevant Data and Information

No additional information or explanation is necessary to make the technical report understandable and not misleading.

25 Interpretations and Conclusions

CAUTION TO READER

This preliminary economic assessment (PEA) is preliminary in nature, in that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. A PEA is a conceptual study of the potential viability of mineral resources. The economic viability of the inferred mineral resources has not been demonstrated.

Work since the 2012 Technical Report on the East Dufferin portion of the property has extended the known dimensions of the deposit, both along strike and down dip. The drilling to date at East Dufferin has outlined mineralized "Meguma Style" stacked saddle veins along strike about 1400 metres and vertically to about 400m below the surface, with detailed sampling along 700m of strike length and about 250m below surface.

Since the 2012 Technical Report, a total of 3,699 metres of drilling has been completed at East Dufferin. Nine holes totalling about 600 metres were drilled from underground and 14 holes totalling about 3,099 metres were drilled from the surface. All holes drilled encountered gold mineralization.

In addition to drilling, the processing plant was refurbished and processed about 18,000 tonnes of low grade material from the underground workings. The company also mined out approximately 28,000 tonnes of waste.

During 2014, Resource Appalaches ('RA') completed approximately 1060 m of drifting along saddle veins and also completed a limited amount of leg mining at East Dufferin. The total amount excavated was about 17,200 m^3 or 45,600 tonnes.

A number of elements contributed to the delays in getting the property into production. Stope development did not keep pace with production, causing production shortfalls. The lack of a mine air heater delayed work during the winter months. Equipment purchases were made without a thorough mechanical inspection.

Resource Capital Gold Corp. has also purchased additional mineral licenses covering the West Dufferin portion of the project, which contain additional historical mine workings, 16,453 meters of drilling in 153 holes, and additional mineral resources as described below. These mineral resources have been included in the overall project economics.

25.1 Current Mineral Resource Estimate

A mineral resource estimate was prepared for this report. Although there have been previous estimates of the amount and quality of the gold mineralization at the Dufferin deposit, this report is the first NI43-101-compliant Mineral Resource Estimate for the property.

At East Dufferin, Saddle 6 is the deepest saddle for this report's resource estimate that contains a significant amount of +2 g/tonne material. Deeper saddles have been identified through diamond drilling; however, the wide intercept spacing at depth meant that only a limited quantity of mineral resources were outlined at depth.

The extent of the main block of resources at east Dufferin at a 2 g/tonne block cut-off is 2200-3200 metres East and 800-1045 metres in Elevation, or 1000x245 metres in longitudinal area.

Dufferin Property Mineral Resources

	Volume (m^3)	Tonnes	Ounces	Average Grade (g/tonne)
East Dufferin				
Indicated	57,200	151,500	58,000	11.9
Inferred	163,800	434,100	96,800	6.9
West Dufferin				
Inferred	101,800	269,800	53,200	6.1
Combined				
Indicated	57,200	151,500	58,000	11.9
Inferred	265,600	703,900	150,000	6.6

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65;
[3] West Dufferin top-cut: 100 g/tonne; East Dufferin top-cut: 200 g/tonne.

East Dufferin's Mineral Resources, Broken Down by Crest and Limb Domains

Domain	Proportion of Tonnes	Proportion of Metal
Crest	34%	36%
Limbs	66%	64%
Total	100%	100%

[1] Planned dilution, at a 0.5 metre minimum mining width, was included. Neither unplanned dilution nor mining losses were incorporated.
[2] Block cut-off = 2 g/tonne; SG = 2.65; Top-cut = 200 g/tonne;
[3] Gold price = $US 1250 per ounce.

25.2 Mining Plan

A mining plan employing narrow vein mining techniques was prepared for this PEA. The crest areas would be mined using mechanised drifting methods. The limbs would be mined using narrow vein stoping methods such as longhole stoping or thermal fragmentation.

Trucks would haul mineralised rock to the stockpile adjacent the mill. Processing includes crushing, grinding gravity concentration and concentration by flotation.

Saleable gold products from the site are doré bricks and flotation concentrate. About 85% of the recovered gold will be in the form of doré bricks which will assay about 80% gold and 20% silver. The remaining 15% of the gold that is recovered will be in a flotation concentrate assaying between 30 grams/tonne and 300 grams/tonne.

25.3 Economic Analysis

An economic analysis was performed for the Dufferin Mine Project, including both East and West Dufferin.

The cash flow does not include considerations for outstanding debts from previous operations, nor does it take in to account any possible corporate tax credits that a potential buyer may hold from other operations. A base case gold price of US$1,250/oz. and a base case exchange rate of 0.75 CDN/US was used for economic modelling.

Sensitivities were performed; the project was found to be most sensitive to, in order, the Canadian dollar exchange rate, gold grade, the price of gold, operating cost and capital cost.

Economic results in the PEA show the production of 216,050 ounces of gold during a 10-year mine life, with a pre-tax IRR of 158% and a capital payback period of 1.3 years with a pre-tax net present value of CAD$121,100,000 at a 5% discount rate.

The post-tax economic results are present value of CAD$89,200,000 at a 5% discount rate and IRR of 121%.

These economic numbers do not include any of the planned production from the stockpiled materials described in the November 9 press release, which would be in addition to such amounts.

The PEA includes a 6-month pre-production period followed by 6 months of initial mining ramping up to full 300-tpd production at the end of Year 1. The pre-production period has already started, with Company staff onsite for the last two months in preparation for beginning mining activities.

The PEA anticipates at total of CAD$9.85M in capital expense, which includes $5.29M in plant, equipment, and pre-production development cost; $1.20M of reclamation bonding; $2.29M of working capital; and $1.06M in contingency.

26 Recommendations

Based on the results of this report, a two phase program is recommended for the East Dufferin portion of the project. We recommend that additional work at West Dufferin be conducted once mining is in operation at East Dufferin.

26.1 Phase 1

Phase 1 includes field work needed at East Dufferin for a Preliminary Feasibility Study so that Mineral Reserves may be identified.

26.1.1 Diamond Drilling

A 3,500 m diamond drilling program is recommended. Ten holes from surface for 3,000 metres and another 20 holes for 500 metres from underground. The drill holes are designed to quickly expand the available resources and to upgrade the inferred resources to indicated resources. Drilling would be concentrated around the existing workings and near the east end of Saddle 1. In addition to diamond drilling, channel samples and, where possible, muck samples, should be taken from the exposed veins underground.

26.1.2 Metallurgical Work

A 30 to 50 kilogram sample of broken mineralised rock should be collected and sent to a laboratory to test the material for sorting characterization.

Bench scale metallurgical tests should be conducted on a representative sample to determine the optimum grind size and the optimum split between gravity separation and flotation. The flotation concentrate from the previous operator’s processing work was very low grade and a review of the metallurgy is recommended. This work would consist of scanning electron microscope work, size analysis, magnetic separation, assays, and professional time. Combinations of sorting, gravity, cyanidation and flotation processing would be considered. An optimized flow sheet would be developed from the work.

26.1.3 Tailings Facility

Preliminary engineering design work should be carried out for (a) providing tailings capacity for next 1-2 years, and (b) expanding the tailings capacity for longer term use.

26.1.4 Mining Methods

Two mining methods appear to be viable for the limbs of the saddle veins - thermal fragmentation and longhole drilling. A visit to the mines using the methods should be arranged in order to better understand the pros and cons of each method.

Work from Phase 1 would be used to develop a Pre-Feasibility Study (PFS) in Phase 2.

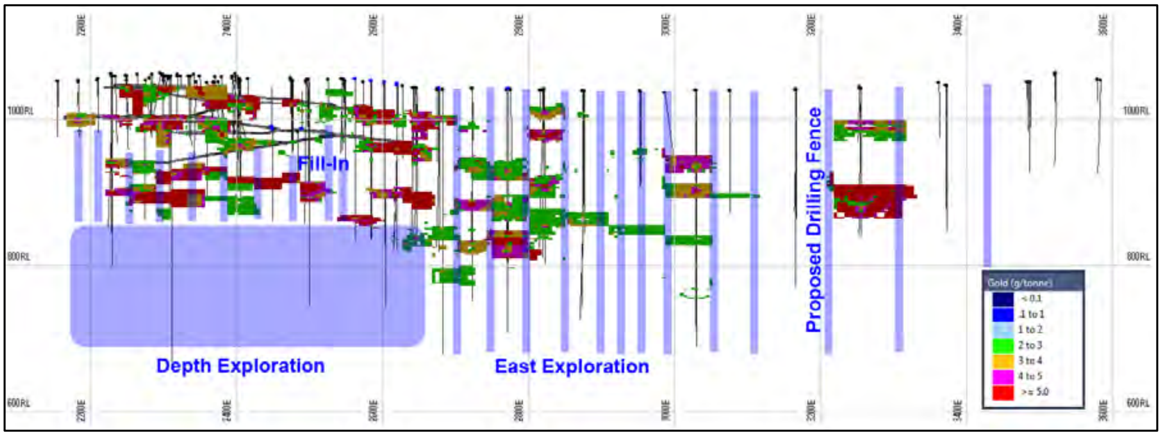


Figure 26-1: Drilling recommendations showing both required fill-in drilling and recommended exploration drilling.

26.2 Phase 2

In Phase 2, a preliminary feasibility study would be prepared, using results from Phase 1 field work. The study would include an analysis of various stoping methods, an updated estimate of the Mineral Resources and an estimate of Mineral Reserves.

The report would have the following scope:

1. Re-estimation of mineral resources.
2. Detailed mine planning and production scheduling.
3. Geotechnical study investigating maximum stope dimensions and minimum pillar distances between stopes.
4. Detailed tailings management facility planning and design.
5. Test stoping to evaluate planned stoping method(s).
6. Detailed capital and operating cost estimation.
7. Detailed economic evaluation.

Upon completion of Phase 2, a production decision could be made by the owner.

26.3 Other Recommendations

If the Pre-Feasibility Study from Phase 2 is positive, some surface features on the site should be re-surveyed. Proper survey monuments were installed for the first time at the Dufferin site in 2014. The author suggests that these monuments should be used for site survey control in future surveys and that map projections and the mine datum be clearly identified. A selection of important features, such as drill collars and mine openings, should be re-surveyed, using those monuments for control, to confirm the current survey.

26.4 Budget

Budget estimates for the work outlined above are shown below:

Table 26-1 - Budget for Recommendations (CAD)

Phase 1, Confirmation of Resources	**Unit Cost**	**Units**	**Item Cost**	**Total**
Diamond Drilling (metres)	$ 120/m	3,500 m	$ 420,000	
Test Stoping			$ 50,000	
Tailings Design			$ 50,000	
Metallurgical tests			$ 25,000	
			sub-total	*$ 450,000*
	Contingency			*$ 40,000*
	Total, Phase 1 (rounded)			**$ 590,000**

Phase 2, Pre-Feasibility Study			**Item Cost**	
Pre-Feasibility Study Report Writing			$ 100,000	
			sub-total	*$ 100,000*
	Contingency, +/- 30%			*$ 30,000*
	Total, Phase 2 (Rounded)			**$ 130,000**
	Total (incl. contingency)			**$ 720,000**

27 References

ACA Howe, (1988), Geological report on the Dufferin Mine property, Halifax, Nova Scotia, Exploration license 4579: Assessment report AR 89-206, prepared for Jascan Resources Inc., June, 1988, 844 p.

Agricola, G. (1556), "De Re Metallica", translated by Herbert Clark Hoover and Lou Henry Hoover, 1950, Dover Publications, Inc., New York, 638 pages

Alberstat , Joann (2015) "N.S. gold mine placed in receivership" Halifax Chronical Herald, Business Section January 20, 2015

Atkinson, Peter H, (1993): The Dufferin Project, Port Dufferin, Nova Scotia, A Feasibility Study (Incomplete), by Peak Engineering Ltd. for Dufferin Resources Inc.

Bamber A., B. Klein, M. Morin and M. Scoble, (2004), Reducing Selectivity in Narrow-Vein Mining through the Integration of Underground Pre-Concentration, CIMM, October 2004

Bates, Jennifer L.E., (1987): Gold in Nova Scotia; Department of Mines and Energy, Information Series No. 13.

BMO Capital Markets, The Goods, January 2015

Brisebois, D. (2010), "Thermal Fragmentation: Reducing Mining width when Extracting Narrow Precious Metal Veins" Canadian Institute of Mining, Metallurgy and Petroleum, Pages: 31, Publication Date: May 1, 2010

British Columbia Securities Commission, (2002): National Instrument 43-101, Companion Policy 43-101CP, Form 43-101Fl, January, 2002.

Bergmann, C. (2009) Developments in"ore"Sorting Technologies, Council for Mineral Technology, Brochure from Mintek, Johannesburg, South Africa,

CIBC Economics, Commodities Update, December 2014

Coles, W. (2002, 2003): Personal Communication.

Covey, Glen, (1988): Compilation of Data from the Mooseland West Zone, (including LP Survey data) Assessment Report 88 234, for Acadia Mineral Ventures Ltd, 25 p.

Douglas, G. V., (1948): Structure of the Gold Veins of Nova Scotia: in Structural Geology of Canadian"ore"Deposits, Can. Ins. Mining Met., Jubilee Vol., pp. 919-926, 1948.

Dufferin Resources Inc., (1994): Mining Permits and Registration documents.

D.R Duncan & Associates Ltd., (2000): An Application to Amend Milling Permit 0016, for Dufferin Resources Inc. (Incomplete).

D.R. Duncan & Associates Ltd., (2011), Assessment report on the Salmon River property, Halifax County, Nova Scotia, Exploration license 06219: report prepared for Nycon Resources, July 20, 2011, 183 p.

Duncan, David R. and Graves, R. Mark, (1993): Report on the 1993 Drilling Program, Geology and Geological Mineral Inventory, Dufferin Property by D.R. Duncan & Associates Ltd. for Dufferin Resources Inc. (Incomplete).

Duncan, DR, "Assessment Report on the Salmon River Property Halifax County, Nova Scotia Exploration License 06219 prepared for Nycon Resources Incorporated by D R Duncan and Associates" NSDNR Assessment Report ME 2011-070, 2011, 178 page(s), 13 map(s). ISN: 22860

Makuch A.P. (2001) " Dynatec's Approach to Narrow Vein Longhole Stoping at Midas, Nevada", presented at the CIM International Symposium on Mining Techniques of Narrow Vein Deposits Val D'Or, Quebec October 1 – 4 , 2001

Environmental Assessment Act Registration Report for the Proposed Underground Gold Mine at Dufferin Mines, Halifax County, by Jacques, Whitford and Associates Limited; Dufferin Resources Incorporated; Nova Scotia Department of Natural Resources; Nova Scotia Department of the Environment, Open File Report ME 1993-043, 1993, 144 page(s), 14 map(s). ISN: 18000

EnviroGold Technologies Inc., (2002): Memo re; Mill Production for Year 2001, Company correspondence.

Faribault, E.R., (1902): Section of Dufferin Mine, Salmon River Gold District, Halifax Co., Nova Scotia, Geol. Surv. Can., Mem 20.

Fowler, TJ. and Winsor, C.N., (1996): Evolution of chevron folds by profile shape changes: comparison between multilayer deformation experiments and folds of the Bendigo Castlemaine goldfields, Australia: in Tectonophysics, vol. 258, pp. 125-150.

Goldfarb, R.J., Baker, T., Dubé, B., Groves, D.I., Hart, C.J.R., and Gosselin, P., 2005, Distribution, character, and genesis of gold deposits in metamorphic terranes: Ecnomic Geology 100th Anniversary Volume, Society of Economic Geologists, Littleton, Colorado, USA, p. 407-450.

Gordon, Richard, (2003): Personal Communication.

Grewal, I. and Duncan, L., (2003): Knelson Metallurgical Test Report 20030, Tailings Samples.

Grewal, I. and Duncan, L., (2003): Knelson Metallurgical Test Report 20046, Coarse"ore"Sample.

Grewal, I. and Wilson, M., (2003): Knelson Metallurgical Test Report 20049, Azure Sands.

Gold, Salmon River, Halifax County, Nova Scotia. Report on Diamond Drilling [Assessment Report on the Salmon River Property, Halifax County, Nova Scotia, Exploration License 06219], by Duncan, D R, D R Duncan and Associates Limited; Nycon Resources Incorporated, Assessment Report ME 2009-092, 2009, 65 page(s), 10 map(s). ISN: 22518

Graves, M. 1991 Gold, Salmon River [Dufferin Mines], Halifax County, Nova Scotia. Reports on Tailings Deposits, a Laboratory Leach Recovery Test on a Tailings Sample, and a Preliminary Environmental Assessment [Assessment Report, Special License for Tailings and Waste Rock: # 9/90 .... Dufferin Mines ....], by Graves, M C; Cuesta Research Limited; McBeath, G A; Earthtech Engineering Limited; MacDougall, F H; Gold Bank

Resources Incorporated, Assessment Report ME 1991-091, 1991, 46 page(s), 9 map(s). ISN: 12239

Groves, D.I., Goldfarb, R.J., Gebre-Marian, M., Hagemann, S.G., and Robert, F., 1998, Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types: Ore Geology Reviews, v. 13, p. 7-27.

Hamilton C., and B. Degay Jr. (2011) Resue Mining with eDev™ Electronic Detonators at Stawell Gold Mines Proceedings of the AusIMM 11TH UNDERGROUND OPERATORS' CONFERENCE / CANBERRA, ACT, 21 - 23 MARCH 2011, pp 73-79

Hannon, P., (2003): Flotation Test Work, Report on Work Completed on the Dufferin Tailings Material, Dufferin Mine, by MineTech International Limited.

Horne, Richard and Culshaw, Nicholas, (2001): Flexural-slip folding in the Meguma Group, Nova Scotia, Canada: in Journal of Structural Geology, vol. 23, pp. 1631-1652, 2001.

Horne, R.J. and Jodrey, M., (2002): Geology of the Dufferin Gold Deposit, in Report of Activities 2001, Nova Scotia Department of Natural Resources, Minerals and Energy Branch, pp.51-67.

Hughes, T.R. and N.J. Grigg, (2008) "The Python – An Underground Processing Plant for Narrow Vein Mining", Narrow Vein Mining Conference, Ballarat, Vic, 14 - 15 October 2008, pp 105 - 107

Hupe, A, Geominex; May 2011 "Report on 2010 exploration work For RESSOURCES APPAL ACHES And DUFFERIN RESOURCES Exploration L icences 08619 County of Halifax NTS 11DI6C" Dufferin Resources Incorporated; Ressources Appalaches Incorporated, Assessment Report ME 2011-053, 2011, 23 page(s), 3 map(s). ISN: 22842

Johansen, G.F., (2001): An Exploration Strategy for the Redevelopment of the Bendigo Goldfield, in a portion of Annual Report of Bendigo Mining NL, Company web site, 2001.

Kennedy, C. and Yule, S., 2010, Assessment Report on the 2010 Work Program for Exploration License 06274 held by Nycon Resources Inc., Halifax County, Nova Scotia, Canada: Assessment Report, Nycon Resources, 27 p.

KPMG (1999) Consolidated Financial Statements of NEWFOUNDLAND GOLDBAR RESOURCES INC. Year ended December 31, 1998

Keppie, J.D., (2000): Geological Map of the Province of Nova Scotia, Nova Scotia Department of Natural Resources, Minerals and Energy Branch, Map ME 2000-1.

MacDougall J.I and J.L Nowland, 1972, Canada Department of Agricultural, Ottawa, 1:450,000 scale map.

Malcolm, W., (1929): Gold Fields of Nova Scotia, Geol. Surv. Can., Mem. 156, reprinted 1976 Geol. Surv. Can. Mem. 385, pp. 130-132.

Mitchell, B. (1987) "Assessment Report On 1986-87 Exploration Program General Exploration Licenses 12093, 12094, 12134, 12135, Halifax County, Nova Scotia NTS: 11D/16" by Seabright Explorations Incorporated, Lower Sackville, Nova Scotia" Bruce Mitchell Project Geologist

Mitchell, B. (1989) “Assessment Report, 1988-89 Exploration Program, General Exploration Licence 14945 Dufferin Halifax County, Nova Scotia NTS: 11D16C by Seabright Explorations Incorporated, Lower Sackville, Nova Scotia” Bruce Mitchell Project Geologist

Newfoundland Goldbar Resources Inc. (2003): Correspondence with Copies of permits.

Nova Scotia, (1928): Minister of Mines Report, 1928, Part II, pp. 124-125.

Orr, S.A. and M. Karmis, Section 21 Underground Mining: Comparison of Methods, Mining Engineering Handbook, pp 1837 – 1866, SME, AIME

Pheeney, Peter, (1994): Environmental Assessment-Dufferin Gold Mine by Jacques, Whitford and Associates Limited (Incomplete) forming part of the Appendices to A Feasibility Study, The Dufferin Project by Peak Engineering Ltd

Poirier S., J-M Fecteau, M. Laflamme and D. Brisebois (2002), “Thermal Rock Fragmentation Applications in Narrow-Vein Extraction” a paper presented at the CIM Annual Meeting, Vancouver

RBC Wealth Management, Global Insight, February 2015

Ryan, R.J. and Smith, P.K., (1998): A review of the mesothermal gold deposits of the Meguma Group, Nova Scotia, Canada in”ore”Geology Reviews v. 13, (1988) pp 153-183.

Rawsthorne, G.R., (2004): Testwork Summary of Metallurgical Work on the Dufferin Property for Azure Resources Corp.

Sanguinetti, M.H. and Ainsworth, B., (2003): Summary Report on the Dufferin Property, Private Report for Azure Resources Corp.

Sanguinetti, M.H., (2004): Assessment Report on the Dufferin Property for Azure Resources Corp.

Scotiabank Global Economics, Global Forecast Update, February 3, 2015

Smith, P.K., (2002): Personal Communication.

Smith, P.K and Kontak, DJ., (1987): Meguma Gold Studies: Some Basic Observations in NSDME Report 87-5, Report of Activities 1987 Part A, pp. 69-74.

Smith, P.K. and Kontak, D.J, (1996): Gold Deposits in the Meguma Group of Nova Scotia, NSDME information Circular ME 51, 1996.

TD Economics, Dollars & Sense, January 26, 2015

Therien, J-Y, Personal Communication, October 2016, Jean-Yves Therien, VP Business Development, NIPPON DRAGON RESOURCES INC. 7055 Taschereau Blvd., Suite 500, Brossard, Quebec, Canada J4Z 1A7

Thornton, E.W. (1997) Letter to Tom Lamb, Dept. of Natural Resources .N.S. regarding Jemma Resources Annual Pertaining to Annual Report for 2006” This report applies to the operating lease No. 94 ·2

Thorpe, Lionel, (1989): Geology Report on the Mooseland Property, Nova Scotia for Hecla Mining Company of Canada Limited, July, 1989, 22 p.

George, Michael W., U.S. Geological Survey, Mineral Commodity Summaries, Gold January 2016

Thirumalai K. (1970) Potential of Internal Heating Method for Rock Fragmentation, The 12th U.S. Symposium on Rock Mechanics (USRMS) 16-18 November, Rolla, Missouri, American Rock Mechanics Association

Ueda, H.T and P.G. Talalay, 2007, "Fifty Years of Soviet and Russian Drilling Activity in Polar and Non-Polar Ice, A Chronological History" US Army Corps of Engineers, Engineer Research and Development Center, Cold Regions Research and Engineering Laboratory, ERDC/CRREL TR-07-20, 145 pages

Vasiliev, N.I. , P.G. Talalay, N.E. Bobin, V.K. Chistyakov, V.M. Zubkov, A.V. Krasilev, A.N. Dmitriev, S.V. Yankilevich, And V.Ya. Lipenkov "Deep drilling at Vostok station, Antarctica: History and Recent Events" Annals of Glaciology, 47, 2007, pp 10-23

Wightman, J., 1936, Synopsis – Dufferin Mines Property – 1935: report prepared for Cominco, November 27, 1937, 7 p.

Wightman, J., 1937, Synopsis of Dufferin Mines Operation – 1937: report prepared for Cominco, January 2, 1936, 8 p.

Woodman, KK, (1999): Draft of Report on the 1999 Drilling Program, Dufferin Property, Halifax County, Nova Scotia by D.R. Duncan & Associates Ltd. For Newfoundland Goldbar Resources Inc. (Incomplete).

Woodman, KK, (2000): Change to Mine Plan, Dufferin Resources Inc., Dufferin Gold Mine, D.R.Duncan & Associates Ltd., September 20, 2000, 28 p.

Woodman, K.K., 2002, Compilation Report on the Dufferin Property, Halifax County, Nova Scotia, Exploration License 04703, 25 p.

Woodman, K.K., 2004a, Assessment Report on the Salmon River Property, Halifax County, Nova Scotia, Exploration License 00133, 25 p.

Woodman, K.K., 2004b, Assessment Report on the Dufferin Property, Halifax County, Nova Scotia, Exploration License 04703, 25 p.

Yule, S., 2012, Assessment Report on the 2012 Work Program for Exploration License 06274 held by Nycon Resources Inc., Halifax County, Nova Scotia, Canada: Assessment Report, Nycon Resources, 27 p.

Yule, S., 2013, Assessment Report on the 2013 Work Program for Exploration License 06274 held by Nycon Resources Inc., Halifax County, Nova Scotia, Canada: Assessment Report, Nycon Resources, 43 p.

Yule, S., 2014, Assessment Report on the 2014 Work Program for Exploration License 06274 held by Nycon Resources Inc., Halifax County, Nova Scotia, Canada: Assessment Report, Nycon Resources, 50 p.

Yule, S., 2015, Assessment Report on the 2015 Work Program for Exploration License 06274 held by Nycon Resources Inc., Halifax County, Nova Scotia, Canada: Assessment Report, Nycon Resources, 31 p.

Zalnieriunas, R.V. (1997): Geological Review of the Mooseland Gold Project, Halifax County, Nova Scotia, Globex Mining Enterprises Inc., January 20, 1997, 34 p.

Web Resources

CALA, Canadian Association for Laboratory Accreditation, 2016, http://www.caladirectory.ca/index.php, accessed November 8, 2016.

NRCAN site. Accessed May 8, 2016: Natural Resources Canada, Magnetic declination calculator

http://www.geomag.nrcan.gc.ca/calc/mdcal-en.php

Accessed May 17, 2016: Environment Canada, Canadian Climate Normals 1971-2000

http://climate.weather.gc.ca/climate_normals

Accessed May 18, 2016: Virtual Museum of Canada, http://novascotiagold.ca/theme/exploitation_de_lor-mining/salmonriver-eng.php

Accessed May 18, 2016: Online Interactive Map for Viewing Nova Scotia Geoscience Atlas https://fletcher.novascotia.ca/DNRViewer/?viewer=Geoscience

http://atlas.gc.ca/toporama/en/index.html Accessed Sept 28, 2016

http://www.theweek.co.uk/gold-price/61682/gold-price-buying-fatigue-could-be-a-turning-point Accessed Sept 28, 2016

CanadianForex http://www.canadianforex.ca/forex-tools/historical-rate-tools/yearly-average-rates Accessed Sept 28, 2016

http://www.mining.com/rbc-adds-200-gold-price-forecast/ Frik Els posted Aug. 14, 2016, 12:28 PM

http://www.azomining.com/Article.aspx?ArticleID=489 Electric Vehicles For Mining: An Interview With Erik Svedlund Product Manager – Electric Vehicles, Atlas Copco

http://www.cim.org/en/Publications-and-Technical-Resources/Publications/CIM-Magazine/2016/August/upfront/The-electric-underground.aspx#sthash.tOBSeGop.dpuf

www.globalminingstandards.org

www.papabravoinnovations.com

http://www.nippondragon.com/index.php/en/, Nippon Dragon Resources Inc. web page

Many of the defined geological terms below are from Jennifer Bates's "Gold in Nova Scotia" (Bates, 1987) " and from "A Dictionary of Earth Sciences, third edition, 2008" published by the Oxford University Press, edited by Michael Allaby, 654 pages. The definitions have been edited slightly in that extraneous material that is not relevant to this assessment report, has been left out. Mineral Resource and Reserve terms are from "CIM DEFINITION STANDARDS - For Mineral Resources and Mineral Reserve's, Prepared by the CIM Standing Committee on Reserve Definitions Adopted by CIM Council on December 11, 2005.Mining terms are mainly from the Underground Mining Methods Handbook, W.A. Hustrulid, Ed., Society of Mining Engineers of the American Institute of Mining, Metallurgy, and Petroleum Engineers, Inc. New York, 1982.

Unit		Notes
Billion	b	One thousand million (10^9).
Gram	g	Metric system unit of mass. 1/1000 kg = 1 gram.
Grams per tonne	g/tonne	Grams per tonne. Equivalent to ppm (parts per million). Common metric system unit used to measure concentrations of gold in untreated rock. 1 g/tonne = 0.0292 oz/ton
Hectare	ha	Metric system unit of area. Equal to 10,000 square metres.
Kilograms	kg	Base unit of mass in the metric system. 1 kg = 1,000 grams = 2.2046 pounds.
Kilometre	km	One thousand metres
Metre	m	Base unit of length in the metric system. 1 m = 3.28084 feet
Ounce	oz	Troy ounce. An imperial system unit primarily used to describe the mass of precious metals. **Not equivalent to the 'ounce' unit in common usage**. 1 oz = 31.1034768 g
Ounce per ton	oz/ton	Troy ounces per short ton. Common imperial system unit used to measure concentrations of gold in untreated rock. 1 oz/ton = 34.2857 g/tonne
Pound	lb	Avoirdupois pound. An imperial system unit of mass, commonly referred to as the 'pound'. 1 pound = 0.45359237 kg.
Ton		Short ton. An imperial system unit of mass, commonly referred to as the 'ton'. 1 ton = 2,000 pounds = 907.18474 kg
Tonnes	t	Also called the 'metric tonne'. 1 tonne = 1,000 kilograms = 2,204.6 pounds.

Word	**Definition**
Adjacent Property	A property in which the issuer does not have an interest, which has a boundary reasonably proximate to the property being reported on, and which has geological characteristics similar to those of the property being reported on (NI 43-101 definition).
Adsorption	Physical adherence of chemicals to substrates without chemical reaction, in this case soluble gold complexes to activated carbon.
Advanced Property	A property that either has mineral reserves, or has mineral resources with potential economic viability as supported by a preliminary economic assessment, a pre-feasibility study, or a feasibility study.
Agglomerate (vb.)	The act of binding fine particles together to create coarse particles as part of a mineral processing activity.

Word	Definition
Anticline	A fold, generally convex upward, whose core contains the stratigraphically older rocks. Antonym of **Syncline.**
Arenite	A general name for sedimentary rocks composed of sand-sized fragments irrespective of composition; e.g., sandstone, graywacke, arkose, and calcarenite.
Assay laboratory	A facility in which the proportions of metal in rocks or concentrates are determined using analytical techniques.
Auriferous	Containing gold; gold-bearing.
Back	The roof or overhead rock surface of an underground opening.
Bed	The smallest distinctive division of a stratified series, marked by a more or less well-defined surface or plane from its neighbours above and below; a layer or stratum.
Bias	A measurement procedure or estimator is said to be biased if, on the average, it gives an answer that differs from the truth. The bias is the average (expected) difference between the measurement and the truth. For example, if you get on the scale with clothes on, that biases the measurement to be larger than your true weight (this would be a positive bias). The design of an experiment or of a survey can also lead to bias. Bias can be deliberate, but it is not necessarily so (Glossary of Statistical Terms).
Billion	One thousand million (10^9).
Birimian	A geologic age occurring from 1.8 to 2.15 billion years ago.
Breast	The vertical end or face of a horizontal cut. The breast is a mining face that is generally as wide as the”ore”body and as high as the cut height.
Bouma Sequence	Idealized sequence of sedimentary structures observed in *turbidity current deposits. It is named after the geologist, Arnold H. Bouma, who first emphasized its generality (Sedimentology of Some Flysch Deposits, Elsevier, Amsterdam, 1962). The lowest unit, A, a massive or graded sand, is overlain progressively by the B (lower division of parallel lamination), C (ripple or convolute laminations), D (upper division of plane parallel laminations), and E (pelagic shale) units. Examples showing the entire sequence are not common. The sequence can be interpreted in terms of deposition under waning current conditions.
Cambro-Ordovician	Geological timeframe; between the Cambrian and Ordovician periods (approximately 560 million and 440 million years ago).
Caps	Round or square timbers generally greater than 200mm in diameter, placed perpendicular to the vein for wall and back support. Caps are part of a timber set.
Cash cost	A measure of the average cost of producing an ounce of gold, calculated by dividing the total working costs in a period by the total gold production over the same period. Working costs represent total operating cost less depreciation and amortization and other non cash items. In determining the cash cost of different elements of the operations, it is necessary to allocated overheads.
Centrifugal Separation	The separation of different particles by centrifugal action as used in cyclone separators and centrifuges.

Word	Definition
Chute	The loading arrangement that utilizes gravity flow in moving broken rock from a higher elevation to a lower elevation. A gate is used to control the flow. Chute is often used for the wooden tube extending upward into the stope.
Concentrate	A powdery product containing the valuable"ore"mineral from which most of the waste material has been eliminated.
Concentrate	The clean product recovered in froth flotation.
Contained ounces	Represents ounces in the ground without the reduction of ounces not recovered by the applicable metallurgical process.
Cross section	A diagram or drawing that shows features transected by a vertical plane drawn at right angles to the longer of the axis of a geologic feature.
Crosscut	A nominally horizontal tunnel, generally driven at right angles to the strike of the vein.
Crown Land	Land held by the state; synonymous with Public Land. Formally, land held by the monarch acting as the head of state.
Cut	The volume of the orebody that is mined and filled in one cut and fill cycle.
Cut-off grade	The lowest grade of mineralized material considered economic to extract; used in the calculation of the"ore"reserves in a given deposit.
Decline	A sloping underground opening for machine access from level to level or from surface; also called a ramp.
Deposit	A natural occurrence of a useful mineral, or an ore, in sufficient extent and degree of concentration to invite exploitation.
Development	The initial stages of opening up a new mine.
Diamond Drilling	Drilling with a hollow bit with a diamond cutting rim to produce a cylindrical core that is used for geological study and assays. Used in mine exploration. Infill diamond drilling at shorter intervals between existing holes, used to provide greater geological detail and to help establish reserve estimates.
Dilution	The contamination of"ore"with waste rock during mining, decreasing the overall grade of the ore.
Dip	The angle that a structural surface, i.e. a bedding or a fault plane, makes with the horizontal measured perpendicular to the strike of a structure.
Disseminated	Said of a mineral deposit (esp. of metals) in which the desired minerals occur as scattered particles in the rock, but in sufficient quantity to make the deposit an ore. Some disseminated deposits are very large.
Doré	Unrefined gold and silver bullion bars usually consisting of approximately 90 percent precious metals which will be further refined to almost pure metal.
Down dip	Down from the point of dip as described above.
Drift	A nominally horizontal tunnel, generally driven parallel to or coincident with a vein.
Estimate	(verb) "to judge or approximate the value, worth, or significance of; to determine the size, extent, or nature of". (noun) "an approximate calculation; a numerical value obtained from a statistical sample and assigned to a population parameter".

Word	Definition
Exploration	Prospecting, sampling, mapping, diamond drilling and other work involved in searching for ore.
Faulting	The process of fracturing that produces a displacement in the rock strata.
Feasibility Study	A definitive engineering estimate of all costs, revenues, equipment requirements and production levels likely to be achieved if a mine is developed. The study is used to define the economic viability of a project and to support the search for project financing.
Feldspar	A group of common rock-forming minerals that includes microcline, orthoclase, plagioclase and others.
Flotation	A process by which some mineral particles are induced to become attached to bubbles and float, and other particles to sink, so that the valuable minerals are concentrated and separated from the worthless gangue or waste.
Flowsheet	A diagram showing the progress of material through a preparation or treatment plant. It shows the crushing, screening, cleaning, or refining processes to which the material is subjected from the run-of-mine state to the clean and sized products. The size range at the various stages may be shown.
Footwall	The underlying side of a fault, an orebody, or mine workings.
Free milling	Gold that is not intimately associated with a host mineral but rather is physically associated in a way which makes it easy to liberate from the host.
Froth Flotation	A flotation process in which the minerals floated gather in and on the surface of bubbles of air or gas driven into or generated in the liquid in some convenient manner.
G&A	General & Administrative Costs.
Gangue	Waste material that is mixed or associated with a desired mineral.
Geophysics	A branch of physics dealing with the Earth, including its atmosphere and hydrosphere. It includes the use of seismic, gravitational, electrical, thermal, radiometric, and magnetic phenomena to elucidate processes of dynamical geology and physical geography, and makes use of geodesy, geology, seismology, meteorology, oceanography, magnetism, and other Earth sciences in collecting and interpreting Earth data.
Gold	A chemical element with the symbol Au (from its Latin name aurum) and atomic number 79. It is a highly sought-after precious metal which has been used as money, a store of value and in jewelry since the beginning of recorded history. The metal occurs as nuggets or grains in rocks, underground "veins" and in alluvial deposits. It is one of the coinage metals. Gold is dense, soft, shiny and the most malleable and ductile substance known.
Grab sample	A sample taken from a particular area at a particular time without undue reference to any specific sampling program.
Grade	the amount of valuable mineral in each ton of ore, expressed as troy ounces per ton or grams per tonne for precious metals and as a percentage for other metals.
Gram	Metric system unit of mass. 1/1000 kg = 1 gram.

Word	Definition
Grams per tonne	Grams per tonne. Equivalent to ppm (parts per million). Common metric system unit used to measure concentrations of gold in untreated rock. 1 g/tonne = 0.0292 oz/ton.
Gravity separation	The use of differential specific gravities to separate denser material (e.g. gold) from lighter material (waste).
Graywacke (Greywacke)	An old rock name that has been variously defined but is now generally applied to a dark gray, firmly indurated, coarse-grained sandstone that consists of poorly sorted, angular to subangular grains of quartz and feldspar, with a variety of dark rock and mineral fragments embedded in a compact clayey matrix having the general composition of slate and containing an abundance of very fine-grained illite, sericite, and chloritic minerals.
Hanging wall	The overlying side of a fault, an orebody or mine workings.
Haul truck	A self-propelled vehicle used to transport material.
Heading	The working face of a drift, crosscut or ramp. In timber sets, a heading is a bundle of wooden boards placed between a cap and the wall rock.
Hectare	Metric system unit of area. Equal to 10,000 square metres.
Hinge	The locus of maximum curvature or bending in a folded surface, usually a line.
In situ	In place, i.e. within unbroken rock.
Indicated Mineral Resource	An "Indicated Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
Induced Polarization	The production of a double layer of charge at a mineral interface, or production of changes in double-layer density of charge, brought about by application of an electric or magnetic field (induced electrical or magnetic polarization). Induced electrical polarization is manifested either by a decay of voltage in the Earth following the cessation of an excitation current phase, or by a frequency dependence of the apparent resistivity of the Earth.
Indurated	Said of rock or soil hardened or consolidated by pressure, cementation, or heat

Word	Definition
Inferred Mineral Resource	An "Inferred Mineral Resource" is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.
Interbedded	Occurring between beds, or lying in a bed parallel to other beds of different material.
Kilograms	Base unit of mass in the metric system. 1 kg = 1,000 grams = 2.2046 pounds.
Kilometre	One thousand metres.
Level	The workings of a mine which are on the same horizontal place.
Ligneous	Rocks solidified from molten material often produced as result of volcanic activity.
Lode	A mineral deposit consisting of a zone of veins.
London price (or fixing)	The twice daily (a.m. or p.m.) gold price fixing determined by the London Gold Market.
Matrix	The rock material in which a fossil, crystal, or mineral is embedded.
Measured Mineral Resource	A "Measured Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
Meta-	A prefix that, when used with the name of sedimentary or igneous rock, indicates that the rock has been metamorphosed, e.g. metabasalt.
Metallurgical recovery	Proportion of metal in mill feed which is recovered by a metallurgical process or processes.
Metallurgy	The science and art of separating metals and metallic minerals from their ores by mechanical and chemical processes; the preparation of metalliferous materials from raw ore.
Metamorphism	The mineralogical, chemical, and structural adjustment of solid rocks to physical and chemical conditions that have generally been imposed at depth below the surface zones of weathering and cementation, and that differ from the conditions under which the rocks in question originated.
Metre	Base unit of length in the metric system. 1 m = 3.28084 feet.

Word	Definition
Mil	One thousandth of an inch.
Mill	A mineral treatment plant in which crushing, wet grinding, and further treatment of”ore”is conducted. Also, separate components, such as ball mill, hammer mill, and rod mill.
Mineable	That portion of a resource for which extraction is technically and economically feasible.
Mineralized	Rock which has undergone mineralization.
Mineral Claim	Title issued by the Government concerned to an individual or group, which grants that individual or group the right to explore for or exploit mineral wealth in a specified area by approved methods in accordance with the ruling laws and regulations.
Mineral Reserve	A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.
Mineral Resource	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
Mineralization	The process or processes by which a mineral or minerals are introduced into a rock, resulting, in a valuable or potentially valuable deposit.
Mineralized Zone	Any mass of host rock in which minerals of potential commercial value occur.
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects.
NSR	Net Smelter Royalty or Net Smelter Return, a royalty based on the price of metal (gold) realized after deducting the cost of refining.
Open pit/opencut	Surface mining in which the”ore”is extracted from a pit. The geometry of the pit may vary with the characteristics of the orebody.
Ore	Rock that contains one or more minerals or metals, at least one of which has commercial value and which can be recovered at a profit.
Orebody	A continuous well-defined mass of”ore”or sufficient volume to make extraction economically feasible.
Ounce	Troy ounce. An imperial system unit primarily used to describe the mass of precious metals. **Not equivalent to the 'ounce' unit in common usage**. 1 oz = 31.1034768 g.
Ounce per ton	Troy ounces per short ton. Common imperial system unit used to measure concentrations of gold in untreated rock. 1 oz/ton = 34.2857 g/tonne.
Percussion drilling	A drilling method which involves advancing the hole by means of a pneumatically operated hammer.

Word	Definition
Pound	Avoirdupois pound. An imperial system unit of mass, commonly referred to as the 'pound'. 1 pound = 0.45359237 kg.
Precambrian	All geological time and the corresponding rocks before the beginning of the Paleozoic Era (i.e. older than approximately 570 million years).
Preliminary Economic Assessment	A Preliminary Economic Assessment (PEA) is a study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources.
Preliminary Feasibility Study	A Preliminary Feasibility Study (PFS) is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.
Pre-stripping	Removal of overburden in advance of beginning operations to remove”ore”in an open pit operation.
Probable Mineral Reserve	A "Probable Mineral Reserve" is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
Proven Mineral Reserve	A "Proven Mineral Reserve" is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.
Qualified Person	An individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the technical report; and is a member or licensee in good standing of a professional association.
Reagent	A chemical substance used to induce a chemical reaction.
Reclamation	The process by which lands disturbed as a result of mining activity are reclaimed back to a beneficial land use. Reclamation activity includes the removal of buildings, equipment, machinery and other physical remnants of mining, closure of tailings impoundments, leach pads and other mine features, and contouring, covering and revegetation of waste rock piles and other disturbed areas.
Recovery rate	A term used in process metallurgy to indicate the proportion of valuable material obtained in the processing of an ore. It is generally stated as a percentage of the material recovered compared to the total material present.
Refining	The final stage of metal production in which final impurities are removed from the molten metal by introducing air and fluxes. The impurities are removed as gases or slag.

Word	Definition
Refractory ore	Any"ore"that does not respond to conventional mineral processing (cyanidation) to produce acceptable product recoveries without an intermediate step to address its refractory attributes (usually, but not always, some form or oxidation).
Reverse circulation drilling	A drilling method employing double walled drill rods. The drilling fluid (usually air or water) is pushed down the annulus between the rods. The cuttings are blown up in the middle.
Run-of-the mine (ROM)	Rock of various sizes resulting from blasting activities within the stone, before any further processing is undertaken on it.
Saddle Reef	A mineral deposit associated with the crest of an anticlinal fold and following the bedding planes, usually found in vertical succession, esp. the gold-bearing quartz veins of Australia.
Sampling	Samples of soils, stream sediments or rock chips taken to determine the quantities of trace and minor elements.
Sedimentary	Formed by the deposition of solid fragmental material that originates from weathering of rocks and is transported from its source to a site of deposition.
Settling Pond	A pond, natural or artificial, for recovering the solids from a washery effluent.
Shaft	A vertical or inclined excavation in rock for the purpose of providing access to an orebody. Usually equipped with a hoist at the top, which lowers and raises a conveyance for handling workers and materials.
Siltstone	An indurated silt having the texture and composition of shale but lacking its fine lamination or fissility; a massive mudstone in which the silt predominates over the lay; a nofissile silt shale.
Slate	A compact, fine-grained metamorphic rock that possesses slaty cleavage and hence can be split into slabs and thin plates. Most slate was formed from shale.
Slime	Extremely fine sediment (0 mesh), produced in the processing of"ore"or rock, especially phosphate rock, which remains suspended in water indefinitely. Consists chiefly of clay.
Slurry	A fluid comprising fine solids suspended in a solution (generally water containing additives).
Smelter	An establishment where ores are smelted to produce metal.
Smelting	Thermal process whereby molten metal is liberated from a concentrate, with impurities separating into a lighter slag.
Soil sampling	Samples of soils taken to explore for mineral deposits.
Stockpile	A store of unprocessed"ore"or marginal grade material.
Stratigraphic	Pertaining to the composition, sequence, and correlation of stratified rocks.
Strike length	Horizontal distance along the direction that a structural surface takes as it intersects the horizontal.
Sump	Reservoir to collect fluids for pumping.
Syncline	A fold in which the core contains the stratigraphically younger rocks; it is generally concave upward. Antonym of **Anticline.**

Word	Definition
Tailings	The gangue and other refuse material resulting from the washing, concentration, or treatment of ground ore.
Tailings Pond	Area closed at lower end by constraining wall or dam to which mill effluents are run. Clear water may be returned after settlement in a dam, via penstock(s) and piping.
Technical Report	A report prepared and filed in accordance with NI 43-101 and 43-101 F1; it includes, in summary form, all material scientific and technical information in respect of the subject property as of the effective date of the technical report.
Ton	Short ton. An imperial system unit of mass, commonly referred to as the 'ton'. 1 ton = 2,000 pounds = 907.18474 kg.
Tonne	Also called the 'metric tonne'. 1 tonne = 1,000 kilograms = 2,204.6 pounds.
Total Cash Cost per Ounce (TCC)	A measure of the average cost of producing an ounce of gold, calculated by dividing the total operating costs in a period by the total gold production over the same period.
Thermal Stress	Temperature changes cause the body to expand or contract. The amount of linear change depends upon the material and the constraints to movement.
Turbidite	A sediment or rock deposited from, or inferred to have been deposited from, a turbidity current. It is characterized by graded bedding, moderate sorting, and well-developed primary structures.
Turbidity Current	A density current in water, air, or other fluid, caused by different amounts of matter in suspension, such as dry-snow avalanche or a descending cloud of volcanic dust; specifically, a bottom-flowing current laden with suspended sediment, moving swiftly (under the influence of gravity) down a subaqueous slope and spreading horizontally on the floor of the body of water.
VLF-EM	Very Low Frequency Electromagnetics. A geophysical technique that relies on VLF broadcasts inducing secondary responses in rock.
Waste	Rock lacking sufficient grade and/or other characteristics of”ore”to be economic.

28 Authors' Certificates

CERTIFICATE of Author – William Douglas Roy

I, William Douglas Roy, M.A.Sc., P.Eng., do hereby certify that:

1. I am a Mining Engineer with MineTech International Limited, located on Hollis Street, Halifax, Nova Scotia, Canada, B3H2P6.
2. I graduated with a bachelor's degree in Mining Engineering from the Technical University of Nova Scotia, now Dalhousie University, in 1997. In addition, I graduated with a Master of Applied Science in Mining Engineering from the DalTech, now Dalhousie University, in 2000.
3. I am a member of the Association of Professional Engineers of Nova Scotia (APENS), Registration Number 7472.
4. I have worked as a Mining Engineer for 20 years since graduating from university. Most of the local work that I have carried out has involved mineral resource estimation, engineering design, economic feasibility, and permitting for narrow vein gold deposits including the Goldenville, Goldboro, Forest Hill, Dufferin, Tangier, Mooseland, Moose River, and Mill Village gold deposits.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1, 2, 3, 11, 12, 14.1, 14.3, 15, 20, and 23-27 of the technical report titled "Preliminary Economic Assessment of the Dufferin Gold Deposit," and with an effective date of December 30, 2016 (the "Technical Report") relating to the Dufferin Gold property in Nova Scotia, Canada. I visited the Dufferin Gold property many times since 2003, most recently on December 14, 2016 for one day.
7. I have had prior involvement with the property that is the subject of the Technical Report, all of an independent nature. My prior involvement is has mainly involved the preparation of technical reports for regulatory purposes.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 3rd day of April 2017.

"Signed and Sealed"

William Douglas Roy, M.A.Sc., P.Eng.

CERTIFICATE of Author

I, Patrick J.F. Hannon, M.A.Sc., P.Eng. do hereby certify that:

1. I am mining engineer employed with MineTech International Limited, 1161 Hollis Street, Suite 211, Halifax, NS, B3H 2P6.
2. I am a graduate of the Dalhousie University, Halifax Nova Scotia (M.A.Sc., Mining Engineering, 1987), Queen's University at Kingston (B.Sc. Geological Engineering 1972), and the Haileybury School of Mines (Senior Mining Technician, 1968).
3. I am registered as a Professional Engineer with Engineers Nova Scotia.
4. I have worked as a geologist for a total of 44 years since graduating from university. Much of my work has been exploration and development of narrow vein gold deposits. This work included resource estimates and engineering design. Since May 1989 until the present (March 2017), I been President and Vice President and Past President of MineTech International Limited. Assignments have included valuation of mining properties at various stages of development, management of an open pit mine, technical consultant on mine safety regulations and team leader for mine feasibility studies. I have spent over five years in and about underground gold mines.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of section all sections of technical report with the exception of Sections 6.2, 9.2, 14 and 15 of the technical report titled "Preliminary Economic Assessment, Dufferin Gold Deposit, Nova Scotia, located in Nova Scotia, Canada 45° 00' North, 62° 24'West, NTS 11D/16C" within an effective date of December 30, 2016 . I visited the Dufferin properties on several occasions since 1987. During the last five years I have visited the property over twenty times, the last time in late December 2016.
7. I have completed prior Technical Reports on the property as discussed in the Introduction.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 3rd Day of April 2017.

"Signed and Sealed"

____

Patrick J.F. Hannon, M.A.Sc., P.Eng

CERTIFICATE OF GREGORY Z. MOSHER, P.GEO.

I, Gregory Z. Mosher, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Geologist with Global Mineral Resource Services, with an office at 179 West Second Street, North Vancouver, British Columbia V7M 1C5;
2. This certificate applies to the technical report titled “Preliminary Economic Assessment Dufferin Gold Deposit, Nova Scotia” for Resource Capital Gold Corporation., with an effective date of 30 December 2016, (the “Technical Report”);
3. I am a graduate of Dalhousie University (B.Sc. Hons., 1970) and McGill University (M.Sc. Applied, 1973). I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Licence #19267. My relevant experience with respect to vein-type gold deposits extends over 40 years and includes exploration and mineral resource estimations. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I have not visited the Property;
5. I am responsible for Sections 6.2, 9.2, 14.2, and the related parts of Sections 1, 25.1 and 26 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: 30 December 2016
Signing Date: April 3, 2017

“Signed and Sealed”

Gregory Z. Mosher, P.Geo.
Principal Geologist
Global Mineral Resource Services

Appendix I 2014 Drill Collars

2014 Exploration Diamond Drill Hole Collars

HOLE-ID	East-UTM	North-UTM	RL	Surface/UG
F14-01	548034	4979952	990.99	Underground
F14-02	548034	4979952	990.99	Underground
F14-03	548034	4979951	991.23	Underground
F14-04	548035	4979951	986.63	Underground
F14-05	548036	4979952	986.68	Underground
F14-06	548068	4979989	987.91	Underground
F14-07	548067	4979990	987.83	Underground
F14-08	548067	4979990	987.87	Underground
F14-09	548068	4979988	987.91	Underground
F14-10	548132	4980034	1055.7	Surface
F14-11	548132	4980034	1055.7	Surface
F14-12	548132	4980034	1055.7	Surface
F14-13	548153	4980042	1052.9	Surface
F14-14	548154	4980037	1053.3	Surface
F14-15	548154	4980032	1052.9	Surface
F14-16	548187	4980050	1047.5	Surface
F14-17	548188	4980047	1048.1	Surface
F14-18	548189	4980042	1049.8	Surface
F14-19	548247	4980047	1041.7	Surface
F14-20	548247	4980044	1043.4	Surface
F14-21	548339	4980068	1042.6	Surface
F14-22	548338	4980071	1042.8	Surface
F14-23	548171	4980043	1051.7	Surface

Appendix II Possible Mining Sequence

Dufferin Gold Mine

MINING, YEARS 1-3

Saddle	Easting	Crest tonnes	g/t	Leg tonnes	g/t
S2	2200	290	8.87		
S2	2300	3,232	20.52		
S2	2400	2,038	5.74		
S2	2500	4,616	5.09		
S2	2600	3,875	18.66		
S2	2700	511	4.59		
S2	2800	2,603	2.48		
S2	2900	2,879	2.34		
S2	3000	1,781	3.71		
S2	3100	1,197	3.05		
S2	3200	1,038	2.84		
S2	2200			10,421	4.91
S2	2300			17,724	12.84
S2	2400			11,546	6.94
S2	2500			7,673	5.34
S2	2600			10,681	20.49
S2	2700			6,678	3.90
S2	2800			4,553	2.80
S2	2900			4,697	2.39
S2	3000			4,331	3.82
S2	3100			2,954	3.11
S2	3200			2,375	3.35
S3	2200	755	2.73		
S3	2300	1,040	2.61		
S3	2400	2,087	4.11		
S3	2500	2,921	2.31		
S3	2600	2,219	2.58		
S3	2700	566	16.9		
S3	2800	1,287	4.42		
S3	2900	36	3.41		
S3	3100	257	2.77		
S3	3200	643	4.51		
S3	2200			3,049	2.49
S3	2300			4,269	2.32
S3	2400			5,669	3.46
S3	2500			4,664	2.37
S3	2600			2,607	2.40
S3	2700			2,773	15.9
S3	2800			6,419	5.55
S3	2900			201	3.82
S3	3100			831	2.93
S3	3200			2,256	4.58
S4	2200	892	11.34		
S4	2300	3,257	17.34		
S4	2400	2,995	13.32		
S4	2500	3,850	12.40		
S4	2600	2,718	8.72		
S4	2700	606	13.42		
S4	2800	6,105	3.27		
S4	2900	3,595	5.31		
S4	3000	4,189	2.54		
S4	3100	136	2.09		
S4	2200			867	11.23
S4	2300			4,651	17.42
S4	2400			4,070	13.62
S4	2500			9,879	18.16
S4	2600			4,680	9.40
S4	2700			1,328	15.35
S4	2800			4,892	3.67
S4	2900			4,781	5.99
S4	3000			2,966	2.56
S4	3100			74	2.09
S5	2200	1,959	17.1		
S5	2300	3,033	10.3		
S5	2400	1,466	5.85		
S5	2500	172	2.02		
S5	2600	1,879	2.85		
S5	2700	1,010	4.16		
S5	2800	3,865	8.10		
S5	2900	45	6.35		
S5	2200			2,584	12.0
S5	2300			7,067	8.24
S5	2400			2,270	5.20
S5	2500			811	2.17
S5	2600			3,365	2.80
S5	2700			2,619	4.28
S5	2800			6,098	9.64
S5	2900			174	6.85
S6	2200	2,665	4.91		
S6	2300	12,854	13.21		
S6	2400	2,701	4.96		
S6	2500	1,589	31.29		
S6	2600	6,750	32.35		
S6	2700	1,861	3.12		
S6	2800	6,962	10.95		
S6	2900	1,101	2.31		
S6	2200			3,776	5.24
S6	2300			8,332	11.25
S6	2400			5,680	4.58
S6	2500			785	27.48
		114,100		198,100	
			312,200		

MINING, YEARS 4-6.5

saddle	Easting	Crest tonnes	g/t	Leg tonnes	g/t
S6	2600			4,378	27.88
S6	2700			3,695	4.08
S6	2800			10,363	12.69
S6	2900			3,836	2.32
SM2	3200	3,204	2.90		
SM2	3300	6,782	3.01		
SM2	3200			3,063	2.75
SM2	3300			7,041	2.77
SM1	2300	-	-		
SM1	2400	-	-		
SM1	2500	1,088	2.47		
SM1	2600	364	2.34		
SM1	2800	312	2.56		
SM1	2900	1,454	2.21		
SM1	2300	-	-	802	5.12
SM1	2400	-	-	775	5.07
SM1	2500			1,223	3.42
SM1	2600			433	3.05
SM1	2800			2,489	4.22
SM1	2900			3,843	2.47
SM1	3100			248	2.40
SM1	3200			809	2.45
S0	2200	-	-		
S0	2300	-	-		
S0	2400	122	3.80		
S0	2200			362	6.53
S0	2300			1,208	6.06
S0	2400			2,932	5.73
S1	2200	55	15.5		
S1	2300	4,613	13.0		
S1	2400	7,114	18.4		
S1	2500	6,307	3.54		
S1	2600	5,079	11.6		
S1	2700	3,228	7.51		
S1	2800	1,521	8.04		
S1	2900	1,486	3.66		
S1	3000	3,644	4.73		
S1	3100	357	3.31		
S1	3200	4,079	9.14		
S1	3300	6,833	9.51		
S1	3400	2,057	3.25		
S1	2200			1,863	11.5
S1	2300			15,120	11.2
S1	2400			17,001	15.5
S1	2500			11,434	4.09
S1	2600			12,404	12.4
S1	2700			10,560	8.16
S1	2800			6,374	5.43
S1	2900			3,987	3.49
S1	3000			7,101	4.35
S1	3100			343	3.10
S1	3200			6,520	7.80
S1	3300			12,600	7.37
S1	3400			4,598	3.08
S7	2200	108	2.09		
S7	2300	2,566	2.62		
S7	2500	311	2.02		
S7	2600	3,021	4.33		
S7	2700	4,383	3.65		
S7	2800	501	2.91		
S7	2900	1,343	2.97		
S7	3000	6,011	3.93		
S7	3100	190	5.55		
S7	2200			34	2.16
S7	2300			4,327	2.60
S7	2500			275	2.02
S7	2600			3,250	4.18
S7	2700			5,142	3.92
S7	2800			854	2.76
S7	2900			2,098	2.74
S7	3000			6,697	3.88
S7	3100			179	4.50
S8	2500	957	2.20		
S8	2700	4,281	3.96		
S8	2800	511	3.04		
S8	2900	34	2.13		
S8	2500			1,316	2.20
S8	2700			3,206	3.97
S8	2800			858	2.99
S8	2900			-	-
S11	2700	836	3.08		
S11	2700			1,552	3.08
S13	2700	307	3.57		
S13	2700			1,115	3.57
		85,100		188,300	
			273,400		