Armistice
RESOURCES CORP.

Technical Report and Mineral Resource Estimate
McGarry Project
McGarry Township (Virginiatown), Ontario

Prepared by:

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1. Summary

Introduction and Infrastructure

Armistice Resources Corp. ("Armistice") completed an underground exploration campaign during 2007-2008 at its wholly-controlled McGarry gold project located at Virginiatown, Ontario. The work was carried out entirely on the 2250 Level. It consisted of 44,500 feet of underground diamond drilling; 2,408 feet of drifting; 130 feet of raising; bulk sampling from 34 drift rounds; and trial mining from two test stopes. In addition, metallurgical testing and a preliminary assessment (scoping study) was completed. The study was carried out by Python Mining Consultants ("Python"), an independent firm with experience in narrow vein gold mining operations in Ontario.

The McGarry Property consists of 33 contiguous, patented mining claims and mining licences of occupation totalling 1,134.6 acres with surface rights on a majority of the claims totalling 975.56 acres. The mining rights and surface rights are all in good standing and are maintained by the payment of annual taxes since no work requirements exist. All proceeds of production from the Property are to Armistice, subject to a royalty interest held by Sheldon-Larder Mines Limited which provides for a Net Smelter royalty payable to Sheldon-Larder starting at 2% and increasing to 4% when the price of gold exceeds $US 800 per ounce or an advance royalty of $C 21,573 payable quarterly. The current status of royalty payments to Sheldon-Larder is in compliance with the agreement.

The McGarry Property is located in the heart of an established mining district that is well served by local labour skilled at narrow vein gold mining techniques and by equipment and service suppliers specializing in underground mining. The Property is traversed by a part of the Trans-Canada Highway system and by a Hydro One three-phase electric transmission line, both within a few hundred feet of the headframe.

The McGarry Property has established mining infrastructure installed or upgraded since the mid 1980's and all owned by Armistice. This infrastructure includes a 110 ft production-ready headframe, a 3 compartment shaft to 2290 feet below surface with two 6 ft by 6 ft hoisting compartments equipped with a service cage and a 5 ton skip respectively. In addition, there is a fully operational 10 ft double drum hoist capable of production hoisting at 1000 tons per day to a depth of 4400 ft. There is a mine air heater installed over a ventilation raise capable of heating all the fresh air required for production mode. Other infrastructure includes fresh water supply, compressed air, high speed communications, pumping systems, electrical substation, surface change-house and workshop, surface equipment fleet and 3 scooptrams, a full compliment of underground fans, drills, pumps and electrical substations. The shaft has stations cut at 200 ft level intervals and established levels at 550, 650, 1250, 1650, 2050 and 2250 feet below surface. The most extensive level is 2250 which extends 2700 ft west of the shaft and 400 ft east.

These factors will give a producing mine at McGarry definite cost advantages.

Geological Setting, Mineralization, Drilling and Sampling

The McGarry Property has been actively explored since at least the 1930's with major underground campaigns in the mid 1940's, 1980's, 1990's and in 2007-2008. Exploration work consists of mainly underground diamond drilling in 407 holes totalling over 302,000 ft; by drifting on 6 levels; and bulk sampling from 5 different locations. The deepest drill holes test to 5600 feet below surface.

Geologically, the Property sits astride the Larder Lake "Break" which is a major feature extending from Val d'Or in Quebec to Kirkland Lake in Ontario. Numerous past and present gold mines exist in geological environments associated with this "Break" including the Kerr Addison Gold Mine immediately to the east of the McGarry Property. The Kerr Addison Mine produced over 11 million ounces of gold from 1938 to its closure in 1996 making it one of the world's premier gold mines. The geological setting for both the Kerr Addison gold mineralization and that identified at McGarry have similarities. The setting includes a band of nearly vertically dipping and highly altered volcanics probably mixed with some sediments. The intense alteration has resulted in carbonate-rich units with various amounts of quartz and pyrite. Two types of gold-bearing environments within the alteration zone were mined at Kerr Addison: "green carbonate" and "flow ore" in a ratio of about 40:60 and at grades of 0.23 and 0.33 oz/t gold, respectively. Economically, pyritic "flow ore" was the most important type at Kerr Addison.

Although both types of gold mineralization are recognized at McGarry, "pyritic mudstone" ("flow ore" at Kerr Addison) appears to be the most important.

The most extensive drilling data is from the 2250 Level where fans of a nominal 7 holes each have been drilled over a strike extent of about 3000 feet at 100 ft intervals. Part of the 2007-2008 programme resulted in the completion of drill testing of a 600 ft gap in this pattern. During this campaign, holes were also drilled to continue the depth confirmation of gold mineralization to the 4000 ft elevation and to test an exploration gap between the 2250 and 1250 Levels west of the shaft.

In order to understand the nature and continuity of the gold-bearing zones, detailed sampling programmes were undertaken on the 2250 Level in 2007-2008. This work included cross cut drifting and panel sampling at 3 sections along the strike; bulk sampling of 34 drift rounds; bulk sampling of 2 test stopes; and in-fill drilling on a 50 ft spacing at one location.

An evaluation of the results of the recent detailed sampling programmes, considered together with previous bulk sampling programmes, leads to the conclusion that drill testing on 50 to 25 ft centres will be required to sufficiently define gold-bearing zones for production planning. It is also concluded that a stoping method will require guidance from in-stope face sampling and geological mapping on a daily basis in order for mining to follow local changes in strike, dip and plunge. Further, any mining method applied at McGarry will have to be of the narrow-vein type, for example, shrinkage stoping. Structures related to shearing and faulting sub-parallel to the identified gold zones are not fully understood yet. It has been concluded that these structures may cause off-sets at a stope scale and will require special attention to reduce hanging wall and footwall dilution. In general, ground conditions in the underground workings are very good except within well defined graphic shear zones.

Mineral Resources

A mineral resource for the Property to NI 43-101 standards was estimated in 2004 and reported by an independent qualified person as an Indicated Mineral Resource of 433,981 tons at a grade of 0.25 oz/t gold using a cut off grade of 0.10 oz/t gold above a depth of 2600 feet. This mineral resource has been updated to include all the assay and geological data available at 8 April 2009. The updated mineral resource is presented in the table below. The existing shaft provides full access to the 2250 Level and a ramp could easily extend this access to the 2300 elevation, therefore the Indicated and Inferred Resources above and below the 2300 elevation are tabulated separately.

Undiluted Mineral Resource Estimate – April 8, 2009

Mineral Resource Category	Tons (short tons)	Cut to 1.50 oz/t		Uncut	
		Grade (oz/t gold)	Gold (oz)	Grade (oz/t gold)	Gold (oz)
Indicated					
Above 2300 elevation (all zones)	374,000	0.22	82,000	0.25	93,000
Below 2300 elevation (all zones)	118,000	0.25	30,000	0.26	30,000
Total Indicated (all zones)	**492,000**	**0.23**	**112,000**	**0.25**	**123,000**
Inferred					
Above 2300 elevation (all zones)	59,000	0.17	10,000	0.19	11,000
Below 2300 elevation (all zones)	113,000	0.16	19,000	0.16	19,000
Total Inferred (all zones)	**172,000**	**0.17**	**29,000**	**0.17**	**30,000**

- Mineral resources estimated according to CIM definition standards (2005).
- A 0.10 oz/t gold cut-off grade was used with high-grade values uncapped and capped at 1.5 oz/t gold.
- A fixed specific gravity of 2.79 was used.
- Undiluted resources, all drill hole intercepts are calculated using a minimum horizontal width of 5 ft, using the grade of adjacent material, if assayed, or zero if not assayed.
- Gold grades determined using the polygonal method with polygons determined from interpretation on vertical cross sections and elevation plans. Maximum distance to the edge of a block from a drill hole or chip sample intercept of 50 ft has been applied. Maximum block size is 10,000 sq ft.
- A confidence level of ±10% is estimated for the Indicated Mineral Resource and ±25% for the Inferred Mineral Resource.
- Effective date of resource estimate is 8 April 2009.
- Qualified Person for the mineral resource estimate is Erik Andersen, P.Eng. (Armistice Resources Corp.)
- Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues although the Qualified Person is not aware of any such issues.

Metallurgy

A review of metallurgical test data as previously reported in an NI 43-101 report in 2004 and the data from test work completed in 2007-2008 shows that recoveries of 95% of the gold has been demonstrated using conventional carbon-in-leach processes. In addition, test work has shown that recoveries of 44 to 65% of the gold in 6 to 16%, respectively, of the weight of the mill feed has been demonstrated by gravity or gravity/flotation techniques alone. The gravity testing has been done using shaking tables only. No test work with centrifugal concentrators has yet been done. These results warrant continued testing to optimize non-leach gold recovery options.

Scoping Study

In order to gain insight into the requirements for a basis to convert the investment at McGarry into an economic venture, Python Mining Consultants ("Python") was engaged to conduct a preliminary assessment (scoping study) for potential production between the 1250 and 2250 Levels where underground access and other infrastructure already exists. The Scoping Study has a ±25% level of accuracy.

Above the 2250 Level, diamond drill testing has been restricted in large part to a corridor about 300 feet east and west of the shaft. This has resulted because no practical drill platforms exist beyond the confines of the limited level development from the shaft stations – except for the 2250 Level. As a consequence, there is a significant gap in drill testing of the prospective geology for gold-bearing zones between the 2250 and 1250 Levels both east and west of the tested corridor.

As noted above, in order to define stope scale gold-bearing zones, it has been concluded that drill spacings of 50 to 25 feet will be required. This is not considered practical or feasible to complete without the establishment of new mined openings for drill platforms.

Assumptions are made for the Scoping Study that zones as described above can be discovered each with minimum mining widths of 5 feet and diluted grades of at least 0.19 oz/t gold. Assumptions about productivity and unit costs based on the experience of over 12 months of actual development mining at McGarry during 2007-2008 were used by Python to set out a template for mining operations. The template estimates a 7 month period of development before enough stopes could reasonably be expected to be outlined and developed for production at an initial rate of about 350 tons per day. After 12 months at this rate, the template contemplates that the mining rate could be increased to 600 tons per day. It would take about 4.5 years to completely mine out between the 2250 and 1250 Levels based on the assumptions above. Using these assumptions plus very conservative metallurgical recoveries of 85% and a gold price of $C 800 per ounce, it could reasonably be expected that by the end of the first year, there would be revenues from custom milling to cover ongoing operating costs. Repaying the initial investment could take about 3 years from the start-up. It must be emphasized that the Python work does not meet the requirements of a feasibility study and has an estimated accuracy of ±25%.

The study was conducted to determine if the project can demonstrate a robustness to warrant advancement to the next stage of underground development. The conclusion is positive, supporting the next stage of work at McGarry following the template set out in the Python Study. The study was based on certain forward looking assumptions that have not yet been proven and actual experience may vary from these assumptions.

Recommendations

Based, therefore, on all available results and evaluations, it is recommended that the McGarry Project be advanced to the next stage of work to consist of:

- Complete the ventilation and escapeway raise system from the 2250 Level to the 1250 Level and outfit the existing ventilation raise as an escapeway from the 1250 Level to surface.
- Complete a main access drift over the full strike length of the 2250 Level using mobile equipment and establish short-hole diamond drill testing at initial 50 ft centres closing to 25 ft centres as warranted.
- Establish a main access drift over the full strike length of the 2050 Level using tracked equipment and test with diamond drilling as on the 2250 Level.
- For all potential stoping areas identified by the short-hole definition work on the 2250 and 2050 Levels, establish stope access drifts and sill drifts at the base of potential stopes. Establish access raises and draw points for shrinkage mining and proceed with stoping operations. Stockpile all material exceeding a threshold grade of about 0.10 oz/t gold on surface for gold recovery by batch processing at a custom milling operation.
- On a continuing basis, evaluate the results of the programme for economic implications and make adjustments as warranted.
- Continue with metallurgical test work towards optimizing non-leach gold recovery.
- Continue to build the computer based geological database and the petrologic, mineralogical and structural knowledge to improve understanding of the controls on gold mineralization.
- The estimated cost of the first 12 months of this programme is $C 12 million ±25%, say, $C 15 million at the top cost range.

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2. Introduction

This technical report is prepared for Armistice Resources Corp., referred to throughout this report as "Armistice".

The purpose of this report is:

- to update the geological knowledge of the McGarry Project area at Virginiatown, Ontario based on work carried out in 2007-2008. This work, all completed on the 2250 Level, consists of

  - 44,500 feet of underground diamond drilling, related logging and assaying;
  - 2,408 feet of drifting and related chip sampling;
  - 130 feet of raising;
  - bulk sampling from 34 drift rounds; and
  - trial mining from two test stopes;

- to present the results of background metallurgical work required prior to setting parameters for a comprehensive metallurgical study;

- to present an estimate of the mineral resources on the McGarry Project updated from an NI 43-101 compliant report in June 2004 (Carmichael, S.J., June 6, 2004); and

- to present the results of a scoping study considering a potential mining strategy for the development and production of gold zones between the 1250 and 2250 Levels at the McGarry Project.

A technical report meeting NI 43-101 standards was prepared by S.J. Carmichael of S.J. Carmichael Consultants dated June 6, 2004 and entitled "Report on the Armistice Resources Ltd. Virginiatown Gold Project, McGarry Township, Ontario" (Carmichael, S.J., June 6, 2004). The Carmichael report is available on the SEDAR database where it is filed under the category of "Other" with a filing date of 17 June 2004. (Note that Armistice Resources Corp. is a continuation of Armistice Resources Ltd.) The current report provides a comprehensive update on the Carmichael report.

All the information and data reported in this report was collected by Armistice Resources Corp. and is available in its Kirkland Lake office. The diamond drilling, assaying, geological drill core logging, mining openings, chip sampling, bulk sampling and survey control data has been incorporated into a computer based three dimensional database using the Gemcom GEMS version 6.2 software developed by Gemcom Software International Inc. and referred to throughout this report as the "Gemcom database". The Gemcom database includes the data collected in the current work programme and all relevant historical data as available in the Armistice files. The database is maintained on a computer located in the Kirkland Lake office of Armistice Resources Corp. Information on the GEMS software is available at www.gemcomsoftware.com.

The 2007-2008 work was planned and supervised on a continuing basis throughout the period by Erik Andersen, P.Eng, Vice President and Chief Operating Officer of Armistice and author of this report. Mr. Andersen, the author of this report, visited the Property on a daily basis during the period of underground work, June 2007 to October 2008, and on at least a weekly basis since then to the date of this report.

The author of Appendix I, *Scoping Study, McGarry Mine Project, Kirkland Lake, Ontario*, and Qualified Person, Martin Drennan, P.Eng., inspected the Property on April 17, 2008.

In this report, the Imperial measurement system is used throughout. The following abbreviations are used in the report:

Dollar (Canadian)	$C or CAD
Dollar (US)	$US or USD
Foot (square feet)	ft (sq ft)
Troy ounces per short ton (2000 lb)	oz/t
Short tons per day	tpd or TPD
Gold	Au
Cross-cut drift	X/C

Although metric units are not used in this report, it may be useful for some readers to have reference to the following conversion factor:

1 Troy ounce per short ton = 34.286 grams per metric ton
= 34.286 parts per million ("ppm")

Reference is made to the "shrinkage" stoping method in this report. This is mining method with which many readers may not be familiar, therefore a brief generic description:

> In shrinkage stoping, the ore is mined out in successive flat slices, working upward from the level. After each slice is blasted down, enough broken ore is drawn off from below to provide a working space between the top of the pile of broken ore ("muck") and the back of the stope. Usually about 40% of the broken ore will have been drawn off when the stope has been mined to the top. After the stope has been mined to the top, the remaining 60% of the ore in the stope is drawn down until the stope is empty. Shrinkage stoping is a method often applied to the mining of narrow veins and has been widely used in the Abitibi gold district. There are variations on the method of access to the top of the stope, to the method of drawing off the broken ore and to the drill/blast techniques for bringing down each successive slice of ore.

Through out this report naming conventions for workings within the Property are used. These conventions include:

- elevations are always referred to in terms of feet below surface, the elevation at the shaft collar being zero. For example, an elevation 5400 ft means 5400 feet below the shaft collar at surface.

- established mining levels are named with respect to distances below surface. For example, the 2250 Level refers to workings at a nominal 2250 feet below surface.

- the mine grid is oriented 25° west of north so that “grid north” is 335° true north. All references to the workings on the Property refer to the mine grid. The location of the grid with respect to surface features is shown on Figure 4, page 7.

- the shaft location establishes the zero (“00”) reference both east-west and north-south. That is, distances are measured east-west or north-south from the shaft location. Distances east and north are indicated as “+” and distances west and south are indicated as “-“ if not explicitly stated.

- mineralized zones are designated with reference to a distance north or south where they are first encountered. For example, the 325N Zone was first encountered at 325 north on the mine grid, although its trace may wander significantly from this northing as it is further identified.

3. Reliance on Other Experts – Disclaimer

Of relevance to this report is geological and engineering work in respect to the McGarry Project for which the author must rely on others. These include the geological core logging and underground geological mapping carried out for the 2007-2008 drilling and mining programmes by James Thompson, BSc, Chief Geologist for Armistice and Jean-François Leclerc-Cloutier, MSc, independent geological consultant to Armistice. The historical records and reports in the files of Armistice have also been used to form a background of understanding of the project and historical data has also been incorporated into the Gemcom database. Historical data critical to this report was collected by other Qualified Persons.

For overview of the metallurgical test work performed by SMC (Canada) Ltd., the author has relied on a site visit report by Ghislain Belzil, Mineral Processing Technician, of the independent consulting firm, Genivar.

With respect to the ownership and status of the mineral and surface rights of the McGarry Property in the Township of McGarry, reliance has been placed on a legal opinion by Fasken Martineau DuMoulin LLP of Toronto, Ontario dated February 26, 2009.

4. Property Description and Location

The McGarry Property is located in the south-western part of McGarry Township within the Larder Lake Mining Division of Ontario, Canada. The Property lies within the limits of the town of Virginiatown which is incorporated as The Corporation of the Township of McGarry and is approximately 40 km east of the mining centre of Kirkland Lake. The boundary between Ontario and Quebec lies about 7 km to the east of the Property. The location of the McGarry Property is shown in Figure 1.

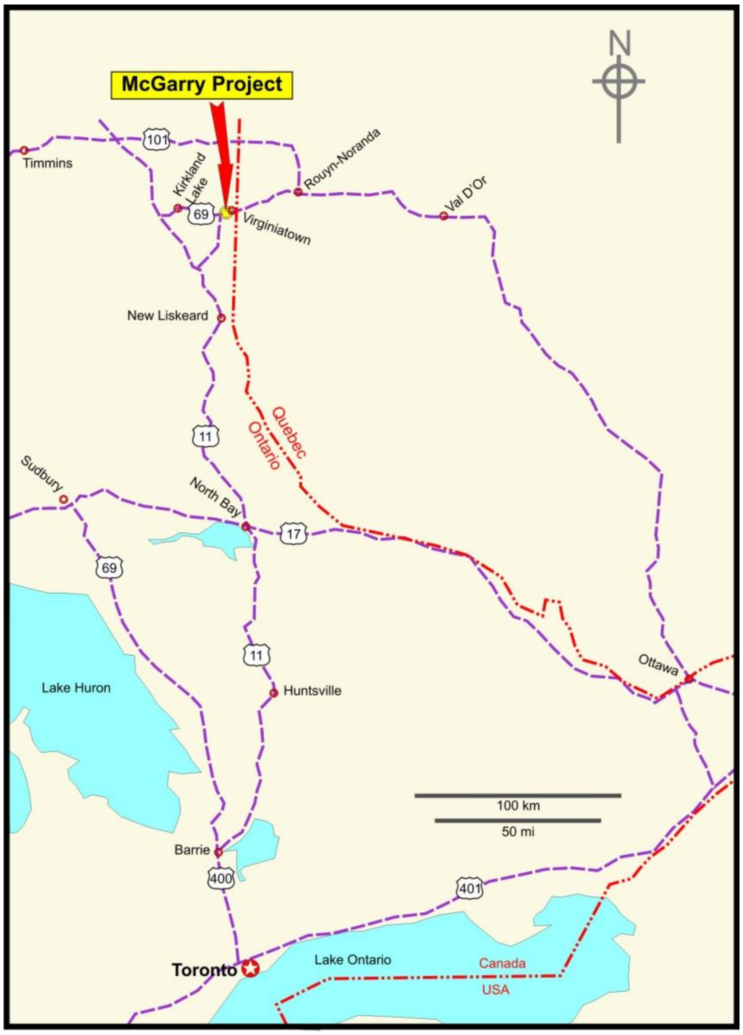


Figure 1 - General Location Map

The regional location of the Property is shown in Figure 2. The Property is centred at approximately 79°36'W and 48°07'N. The NTS reference is 32D/SW.

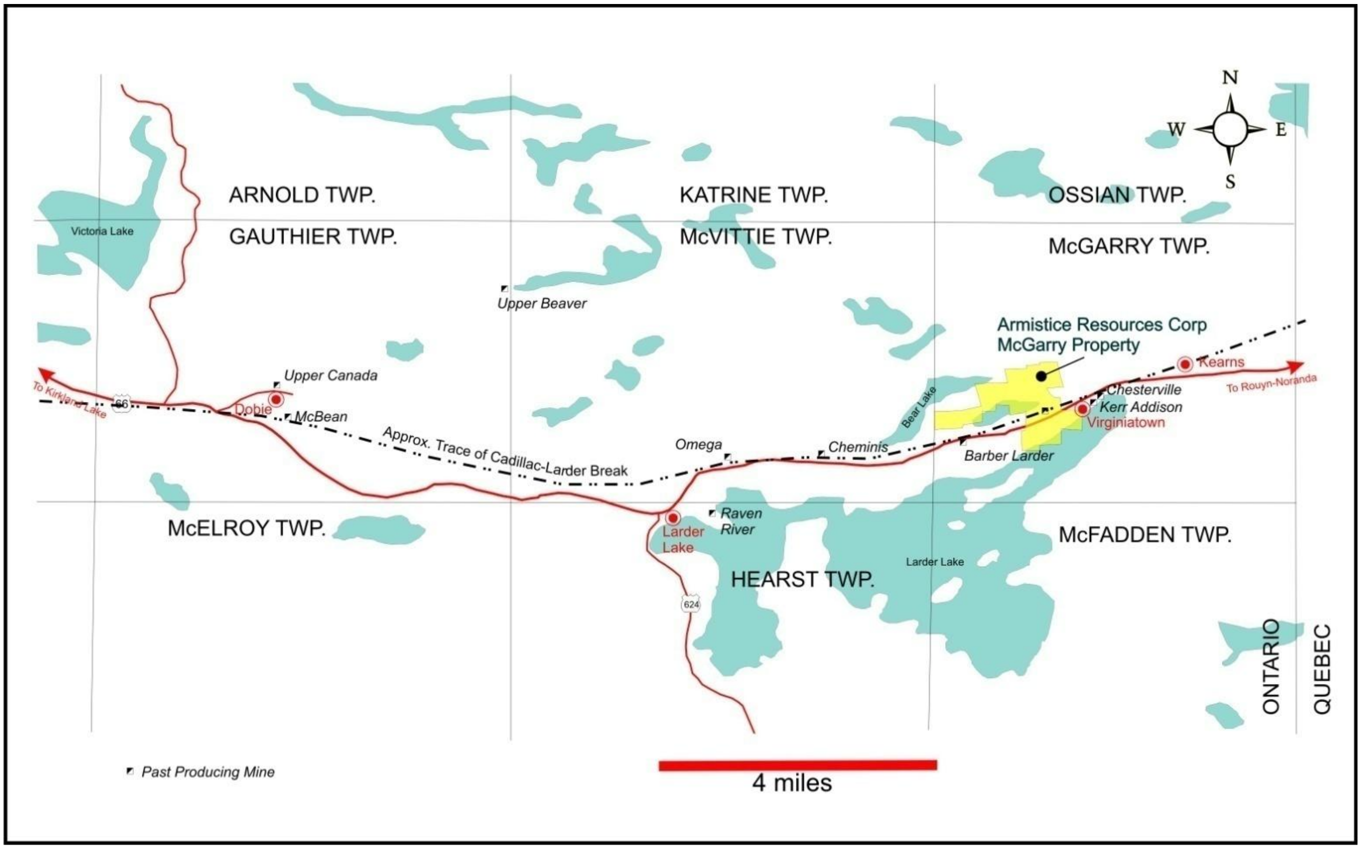


Figure 2 - Regional Location Map

The Property consists of 33 contiguous mining claims and licences of occupation (for water covered areas in Larder Lake). The rights include both mineral rights held by patent and surface rights. The individual patented claim boundaries have been surveyed by an Ontario Land Surveyor. The author has examined a selection of these plans and has discovered no issues. The surface rights exclude easements for highway and power line corridors. Mineral rights comprise a total area of 1,134.60 acres (459.17 hectares). Surface rights comprise a total area of 975.56 acres (394.80 hectares). All mineral and surface rights are registered to Armistice Resources Corp. There is no work assessment required to keep the mining rights in good standing. Annual payment of municipal (surface rights) and provincial (mineral rights) taxes are required. These taxes totalled $ 24,768 and $ 1,873 respectively in 2008. The author has confirmed from the tax statements that all tax liabilities are current and all the rights listed below are registered on the tax statements to Armistice Resources Corp. The mineral rights and surface rights do not expire provided annual taxes are paid.

The claims are shown in Figure 3 and are listed below in Table 1.

The mine grid and workings in respect to the property boundaries and geographic features is shown in Figure 4, page 7.

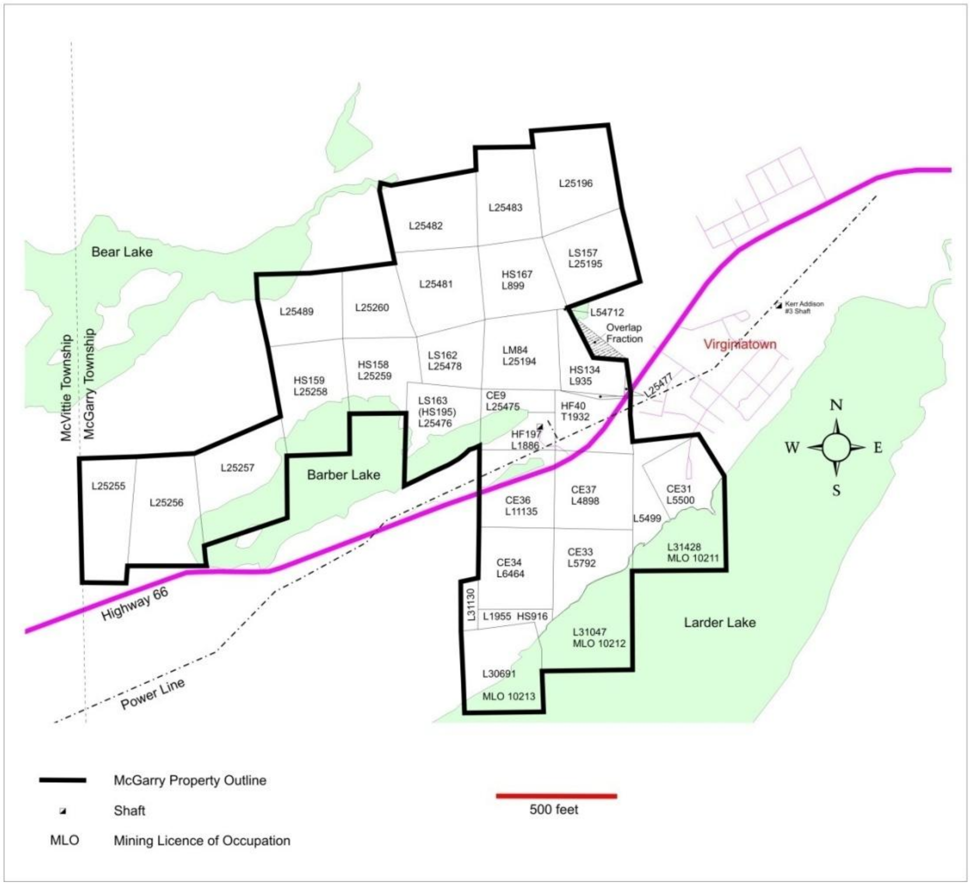


Figure 3 - Property Claim Map

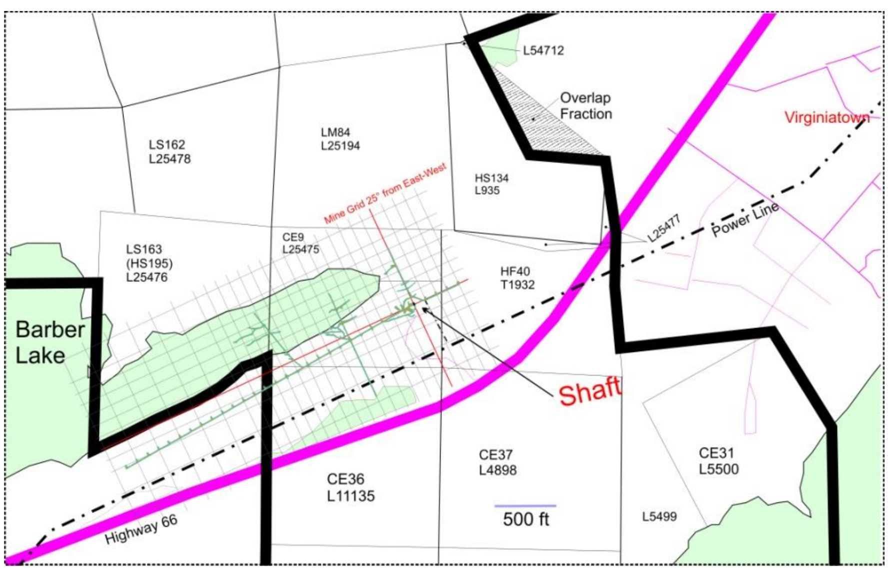


Figure 4 - Detail of Figure 3 Showing Workings on 2250 Level & Mine Grid

Claim Number	Original or Alternate Claim Number	Mineral Rights Acres	Surface Rights Parcel	Surface Rights Acres
L. 899	H.S. 167	37.00	1698 NND	37.00
L. 935	H.S. 134	33.70	6778 NND	33.70
L. 1886	H.F. 197	9.50	6665 CST	9.50
T. 1932	H.F. 40	29.50	7233 NND	25.80
L. 1955	H.S. 916	10.90	4667 CST Fifthly	10.90
L. 4898	C.E. 37	40.25	4667 CST Sixthly & RP 54R5188 Part 3	29.12
L. 5499		24.40	4667 CST Firstly	
L. 5500	C.E. 31	25.50	4667 CST Secondly	
L. 5792	C.E. 33	38.20	4667 CST Thirdly	38.20
L. 6464	C.E. 34	42.55	4667 CST Fourthly	42.55
L. 11135	C.E. 36	39.33	4667 CST Seventhly	34.49
L. 25194	L.M. 84	39.96	4864 CST	39.96
L. 25195	L.S. 157	42.56	7781 CST	42.56
L. 25196		51.66	4857 CST	51.66
L. 25255		44.21	4816 CST	44.21
L. 25256		53.40	4817 CST	53.40
L. 25257		60.31	4818 CST	60.31
L. 25258	H.S. 159	54.42	4807 CST	54.42
L. 25259	H.S. 158	35.45	4808 CST	35.45
L. 25260		31.78	4809 CST	31.78
L. 25475	C.E. 9	20.25	4810 CST	20.25
L. 25476	H.F. 195 or L.S. 163	45.60	4811 CST	45.60
L. 25477		3.85	4812 CST	3.85
L. 25478	L.S. 162	25.44	4813 CST	25.44
L. 25481		41.88	4814 CST	41.88
L. 25482		45.62	4819 CST	45.62
L. 25483		40.45	4820 CST	40.45
L. 25489		39.33	4815 CST	39.33
L. 30691	Land Only MLO 10213	32.17 15.00	4907 CST	32.17
L. 31047	MLO 10212	40.20		
L. 31130		5.86	4908 CST	5.86
L. 31428	MLO 10211	34.30		
L. 54712		0.10	7399 CST	0.10
Total	**33 Mining Claims including 3 Mining Licences of Occupation (“MLO’s”)**	**1,134.60 acres**		**975.56 acres**

Table 1 - List of Mining Claims

4.1. Royalty Interest

Armistice holds an undivided 75 % interest in the Property through an agreement with Sheldon-Larder Mines Limited ("Sheldon-Larder") dated 30 June 2004. The remaining 25% interest is a carried interest entitling Sheldon-Larder to a royalty. The carried interest means that all operating and other costs related to the Property or work carried out on the Property are borne 100% by Armistice. All work carried out on the Property is entirely the responsibility of Armistice and does not require the approval by or prior notice to Sheldon-Larder. This results in complete control by Armistice including 100% of proceeds of production subject only to the royalty interest. The royalty interest held by Sheldon-Larder is the greater of:

i) a Net Smelter Return royalty of a percentage of the price per troy ounce as follows:
   - 2% when less than $US 500
   - 3% when greater than $US 500 and less than $US 800
   - 4% when greater than $US 800;

(ii) $C 1.00 per short ton of ore derived from the properties; or

(iii) an advance royalty of $C 21,573 payable quarterly.

As of the date of this Report, Armistice was in compliance with all of its royalty payment obligations with respect to Sheldon-Larder under terms of the agreement.

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

As shown in Figure 2, page 6, the McGarry shaft and associated buildings are located near the eastern end of Barber Lake and are accessible via a short gravel road from Highway 66 which traverses the Property. Highway 66 is part of the Trans-Canada Highway system. Access to the rest of the Property is by bush roads and trails. The town of Virginiatown lies about one kilometre to the east of the shaft and associated buildings.

The Property is about 5% water covered, extending a short distance into Larder Lake and parts of Barber Lake. Topographic relief on the Property is subdued with maximum local relief of about 10 metres. Outcrop exposure varies from moderate to poor. Overburden consists of sand and glacial debris with many swampy areas. Daily mean temperatures range from about -20°C in winter to +20°C in summer. The areas receives ample precipitation. Common vegetation is second growth poplar, birch, pine, alder and scrub maple.

The south-central part of the Property is topographically low and forms part of the tailings disposal area for the past-producing Kerr Addison mine.

Electric power is available at the McGarry shaft site from a 44,000 volt transmission line owned by Hydro One which traverses the Property immediately south of the shaft site (see Figure 2 and Figure 3). Armistice owns a 3 MVA substation fed by the Hydro One transmission line.

The area offers well established and broad based services and suppliers for mining operations within a 50 km radius (Kirkland Lake to Rouyn-Noranda). There is an excellent labour pool within the same radius that can supply all the skills likely to be required by a mining operation.

The photos on the cover (which looks north from Highway 66) and in Figure 24, page 51, present excellent views of the local physiography and relationship of local topographic features, the Virginiatown town site and the former Kerr Addison Mine and Mill Complex.

Mining and other underground exploration operations can be carried out throughout the year with no restrictions.

In the opinion of the author, the surface rights owned by Armistice are sufficient for any mining operation on the Property including areas for waste rock and tailings disposal, as could reasonably be required.

6. History

Gold-bearing green carbonate rocks were discovered on Kerr Addison claims to the east of the McGarry Property in the early 1900's. The erratic distribution of the contained quartz veining and gold discouraged development until 1937. The Omega and Raven River mines in adjoining McVittie and Hearst Townships saw production from 1912 to 1928. The former from pyritic ores of the "flow ore" type, and the latter from veined, green carbonate rocks. The Omega Mine produced 214,000 ounces of gold from 1.6 million tons of ore before closure in 1947.

Production from the adjoining Kerr Addison property started in 1938 with initial production from veined, green carbonate ore. Later production came mainly from zones of pyritic "flow ore" which was found to increase in grade and continuity with depth. Operations at the Kerr Addison mine ceased in 1996, then under management by AJ Perron Gold Corporation. Over its 58 year operating life, the Kerr Addison mine recorded a production of about 16 million tons of green carbonate ore at a recovered grade of 0.233 oz gold per ton, and 25 million tons of pyritic "flow ore" at a recovered grade of 0.330 oz gold per ton (see Table 13, page 53).

Armistice Gold Mines Ltd. (not related to the present Armistice Resources Corp.) acquired the claims around Barber Lake and began sinking a vertical exploration shaft in 1945 which was completed in 1947. No substantial gold zones were encountered at shallower depths. On the 1250 foot level, a zone of "flow ore" material 170 feet in length and up to 20 feet in width was outlined grading a reported 0.20 oz gold per ton. However the work was terminated in 1947.

In 1974, Kerr Addison Mines Ltd. optioned the Property, then owned by Sheldon Larder Mines Limited, and drilled a deep exploration hole from surface collared just south of the shaft (DDH 74-1A and wedge cuts 1B, 1C and 1D). Wedging from this hole tested the target formations at three elevations reaching a maximum depth of 3,300 feet below surface. The units intersected were interpreted to be similar to those at the adjacent Kerr Addison mine to the east. The interpreted unit equivalent of the Kerr Addison No. 16 "flow ore" zone reported a core length intersection of 5.6 ft grading 0.11 oz gold per ton. Kerr Addison attempted to follow up underground exploration drilling from a drift heading on the 3850 ft level with inconclusive results. The option agreement was terminated in 1978.

Denison Mines Ltd. optioned the Property in 1980 and drilled a single exploration hole from the north collared about 1850 feet west and 1850 feet north of the McGarry shaft (DDH 80-1 and the two wedge cuts from this pilot hole, DDH's 80-1A and 80-1B). Difficulty was experienced penetrating a talc schist, but quartz-veined green carbonate rock was eventually intersected at a depth of about 3,300 feet. A zone of weak gold mineralization over a core length of 75 feet reported gold grades ranging from 0.050 to 0.005 oz per ton. Denison terminated its option soon after completion of this hole.

In 1986, Armistice Resources Ltd. (continued in 2005 as Armistice Resources Corp.) was formed and acquired the Property from Sheldon-Larder. During the period 1988 to 1990, the effective hoisting size of the shaft was enlarged to two standard 6 ft by 6 ft compartments and one manway by replacing the wooden sets with concrete set rings and the shaft was deepened to the 2250 ft level. A 9 ft by 8 ft drift was then driven on the 2250 Level to the west for 1200 ft. Underground diamond drilling was carried out from this drift. Several sub-parallel mineralized zones interpreted to be of the “flow ore” type were located within an alteration sequence about 300 ft in width with this drilling. Financial difficulties were experienced during 1990 and the operation was closed and the workings allowed to flood.

No further work was done on the Property until 1994 when the project was reactivated under new Armistice management. The hoisting plant was refurbished and the workings de-watered. The shaft was deepened by 40 ft to 2290 ft to accommodate a lip-type loading pocket and a sump. Bulk sampling was then carried out in four locations. Approximately 60,000 ft of diamond drilling was completed at and above the 2250 Level.

During 1997, the 2250 Level was extended an additional 1,500 ft to the west (to 2700W) and 400 to the east to provide a platform for drilling at and below the level. An “information for access” agreement was reached between Armistice and NFX Gold Inc. (now Bear Lake Gold Ltd.) which allowed the extension of the drift 1,400 feet west onto the NFX claims. Approximately 100,000 feet of drilling was completed in 1997, spread over a strike length of 3,200 ft and testing a maximum depth of 5,600 feet below surface.

In 1998, an additional 60,000 ft of drilling was completed. A prime objective of this programme was to reduce the hole intercept spacing to about 100 ft in the vicinity of the 2250 Level. Also, a 500 foot crosscut was driven south at 600W on the 2250 Level to facilitate future testing of the mineralized system to depth. At this point, the deepest hole to successfully traverse the entire system was DDH 22-107C which intersected seven “flow ore” type mineralized zones grading from 0.048 to 0.245 oz gold per ton over core lengths ranging from 2.5 to 15.5 feet at an approximate depth of 5,200 ft below surface.

During March 1998, Roscoe Postle Associates Inc. carried out a scoping study on the project, to assess the economic viability of production from the McGarry Property. It was concluded that production could be seriously considered by ramping down from the 2250 Level to a depth of 2,600 ft to provide sufficient tonnages and stoping areas to support sustained mining operations.

In January 1999, the project was placed on a care-and-maintenance basis pending further financing. The workings were kept dewatered until an accident with the skip caused damage to the headframe, shaft timbers, one hoisting rope and the skip itself. Hoisting operations in the shaft could not resume pending repair and re-certification. The workings were kept dewatered until finally allowed to flood in 2003.

Following the commencement of a major re-organization of Armistice in 2004, work began to de-water the workings, repair and re-certify the hoisting system and generally complete maintenance work in preparation of a resumption of underground activities.

In April 2005, Armistice Resources Ltd. was continued as Armistice Resources Corp.

In April 2007, Paul Whelan Mining Contractors ("Whelan") completed the dewatering of the workings. Whelan subsequently completed the rehabilitation of the underground infrastructure including the pumping, electrical, compressed air, ventilation and water systems.

The 2250 Level was ready to start an underground diamond drilling programme in June 2007 at which time Heath & Sherwood Drilling Inc. (now Cabo Drilling (Ontario) Corp.) was engaged to commence the programme reported on in this technical report.

Following acceptance and filing of The Closure Plan submitted to the Ontario Government, Whelan commenced mining activities to provide the basis for the sampling programmes as reported herein.

Figure 5 shows the mining development on the 2250 Level by work period. This figure also shows the location of diamond drilling stations occupied on the 2250 Level during the 2007-2008 work period.

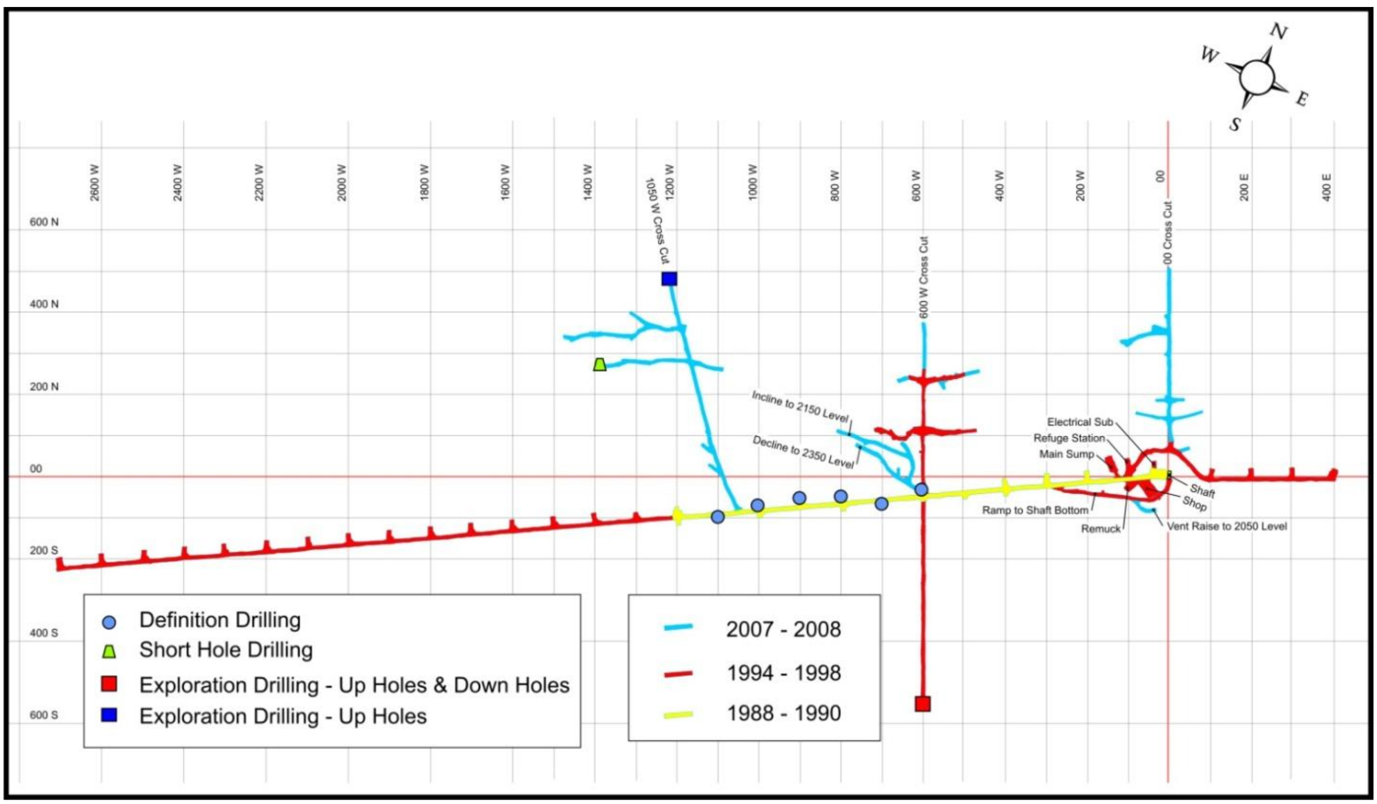


Figure 5 - 2250 Level Mining Development by Work Period and 2007-2008 Drill Stations

7. Geological Setting

7.1. Regional and Local Geology

The McGarry Property lies astride the Larder Lake “Break”, which is a relatively narrow, highly disturbed linear zone over 200 km in length that extends from about Kirkland Lake in Ontario to Val d’Or in Quebec. It constitutes a steeply-dipping and strongly faulted lithostructural unit, consisting mainly of a series of interlayered metasediments and mafic to ultramafic volcanics. Occasionally, the system is displaced by crossfaults. In the area of the McGarry Project, the formational strike is 070° and the dip is from 70° to 80° to the north. A regional geological map after Thompson (1941) is presented in Figure 6.

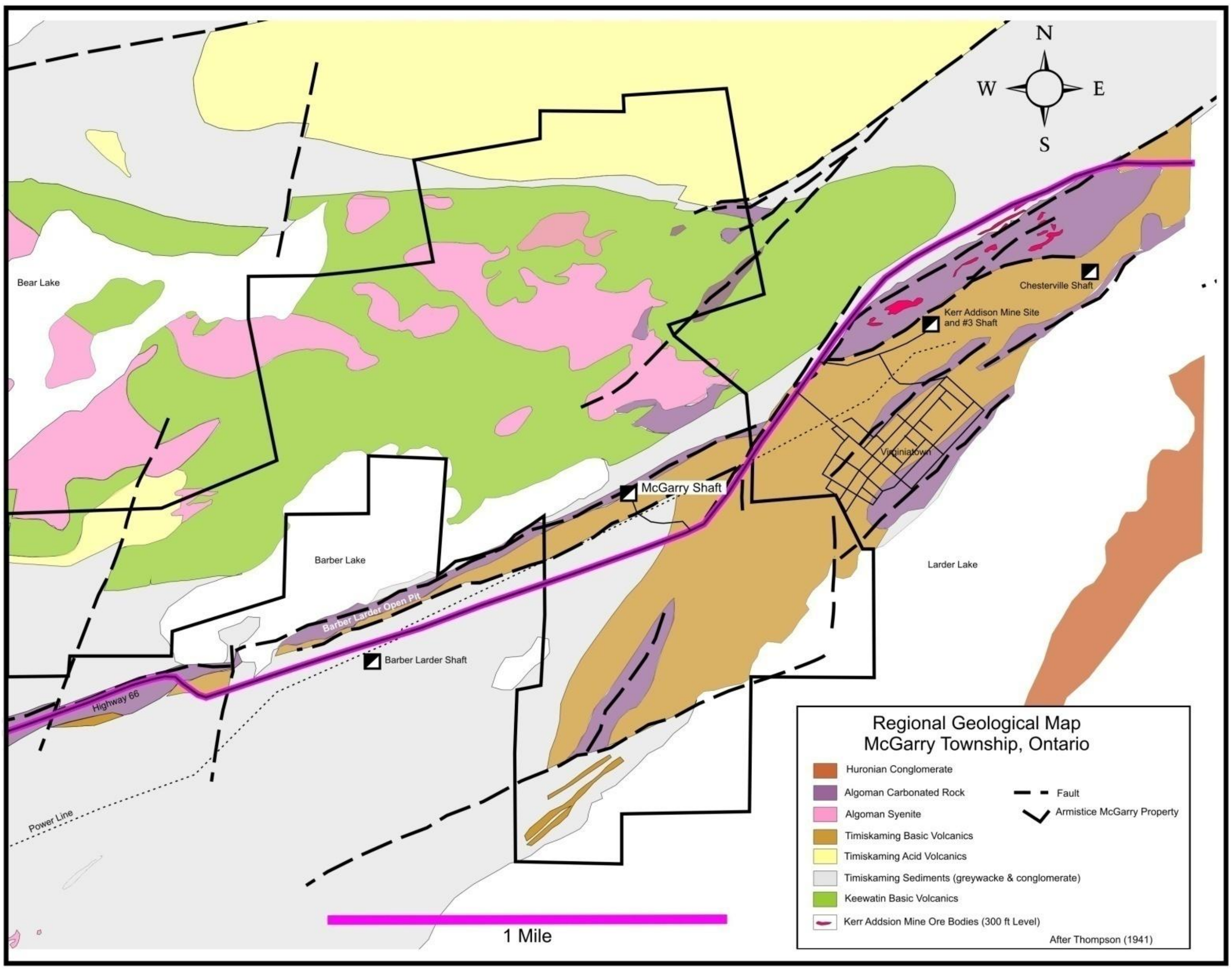


Figure 6 - Regional Geological Map

The Larder Lake “Break” forms the southern margin of the Abitibi geosyncline which was formed during a period of profound orogenic adjustment during the Neo-Archean Period. This involved the collapse of an extensive marine basin to the north accompanied by the extrusion of the Blake River volcanics, intrusive activity, and the intense deformation of marginal rocks. In reference to the regional Table of Formations shown in Table 2 (Mineralium Deposita, Vol. 21, No. 3, 1986) the units of the Kerr Group of the Timiskaming Supergroup represent these marginal remnants. They are exposed at intervals along the "Break" locus and host numerous gold deposits of the region.

Recent				Unconsolidated sands, gravels	Glacial, fluviatile deposits	
Unconformity						
Precambrian	Aphebian		Cobalt Group	Greywacke, Conglomerate	Glacial sediments	
	Unconformity					
	Kenoran			Diabase Granite, syenite, diorite		
	Archean		Blake River Group	Calcalkaline Volcanics, Minor intercalated sediments	Extensive volcanism, seafloor spreading in high energy basin. Full volcanic cycle	
		Timiskaming Supergroup	Crystal L. Group	Conglomerate, trachytic pebbles: sandstone	Restricted period of sedimentation	
			Disconformity			
				K-rich flows, Trachyte, Syenite	Strong subsidence, recrystallization, melting, volcanism, intrusion	Formations in McGarry Project Area
			Barber Lake Group	Argillite, carbonate/sericite-rich sandstone, stretched pebble conglomerate, conglomerate	High energy sedimentation on geosynclinal subsidence. Believed equivalent of Kewagama Group of Malartic area. May be equivalent of Kerr Group in certain depositional areas.	
			Kerr Group	Carbonate-chert-feldspar-pyrite mudstone, carbonate, conglomerate, sandstone, iron formation. Mafic to ultramafic volcanics sometimes present.	Low energy sedimentation, clastic to chemical. Intermittent evaporitic periods in marine shelf environment. Thickness and character quite variable.	
		Disconformity				
			Larder Lake Group	Basalt, massive, vesicular, pillowed. Ultramafic volcanics. Minor mudstone, carbonate, clastic sediments.	Tholeiitic to Komatiitic volcanism, extending into Kerr sedimentation period. Mior sediments. Note restricted late siliceous volcanic phase. Equivalent to Piche Group of Malartic area?	
			Kekeko Lake Group	Polymictic conglomerate, sandstone, minor carbonate	Shallow marine deposits, piedmont outwash, fluviatile sediments	
			Pontiac Group	Sandstone, minor conglomerates. Intercalated basalt and ultramafic volcanics.	Shallow marine deposits. Periods of volcanism.	

Table 2 - Regional Table of Formations

Formational units of the Kerr Group in the Larder Lake area include intercalated grey to green carbonate rock, cherty mudstone, variably graphitic shale, sandstone, conglomerate and mafic to ultramafic volcanics. These appear to have been deposited in a volcanically active, shallow marine environment prior to geosynclinal collapse, resting on the tholeiitic to komatiitic volcanics of the Larder Lake Group.

One interpretation is that gold, other metallic elements and silica, probably originating as weathering products from komatiites are believed by some to have become concentrated to varying degree in the carbonate rocks and cherty mudstones of the Kerr Group as part of the sedimentation process. During subsequent orogenic activity, the various formations making up the Kerr Group were pervasively faulted, variably metamorphosed and tilted to their present steeply dipping attitude. During this process, some limited redistribution of more mobile constituents took place.

An alternate interpretation is that the gold and associated veining and mineralization of the area is of hydrothermal origin, emanating as volatiles from deep seated intrusive bodies and volcanic fissures. Although it is likely that the origin of the gold mineralization in the Virginiatown area will continue to be debated well into the future, it is considered that the hydrothermal model has strong supporting evidence.

A hydrothermal model in which wide spread alteration processes including silicification and carbonization have been brought into a volcanic package that already contained gold-rich units is the one used as the foundation for all current geological interpretation at the McGarry Project.

7.2. Property Geology

The division of the geological package into “formations” has long been accepted and has been historically applied to describing the geological setting within the McGarry Project environment.

The practice of using “formations” in the working geological database for the McGarry Project is no longer in use. The intense secondary alteration within the gold-bearing zones has, to a large degree, completely obscured any primary formational characteristics. The simplified geological sequence as used on a practical basis within the project environment is illustrated in Figure 7.

Table 3 provides a detailed description of the rock types corresponding to the schematic cross section proceeding from north to south.

Figure 8 on page 19 below shows the interpreted geological plan based on the diamond drilling and mined openings on the 2250 Level. Figure 8 shows a simplified geological section through about 600 W.

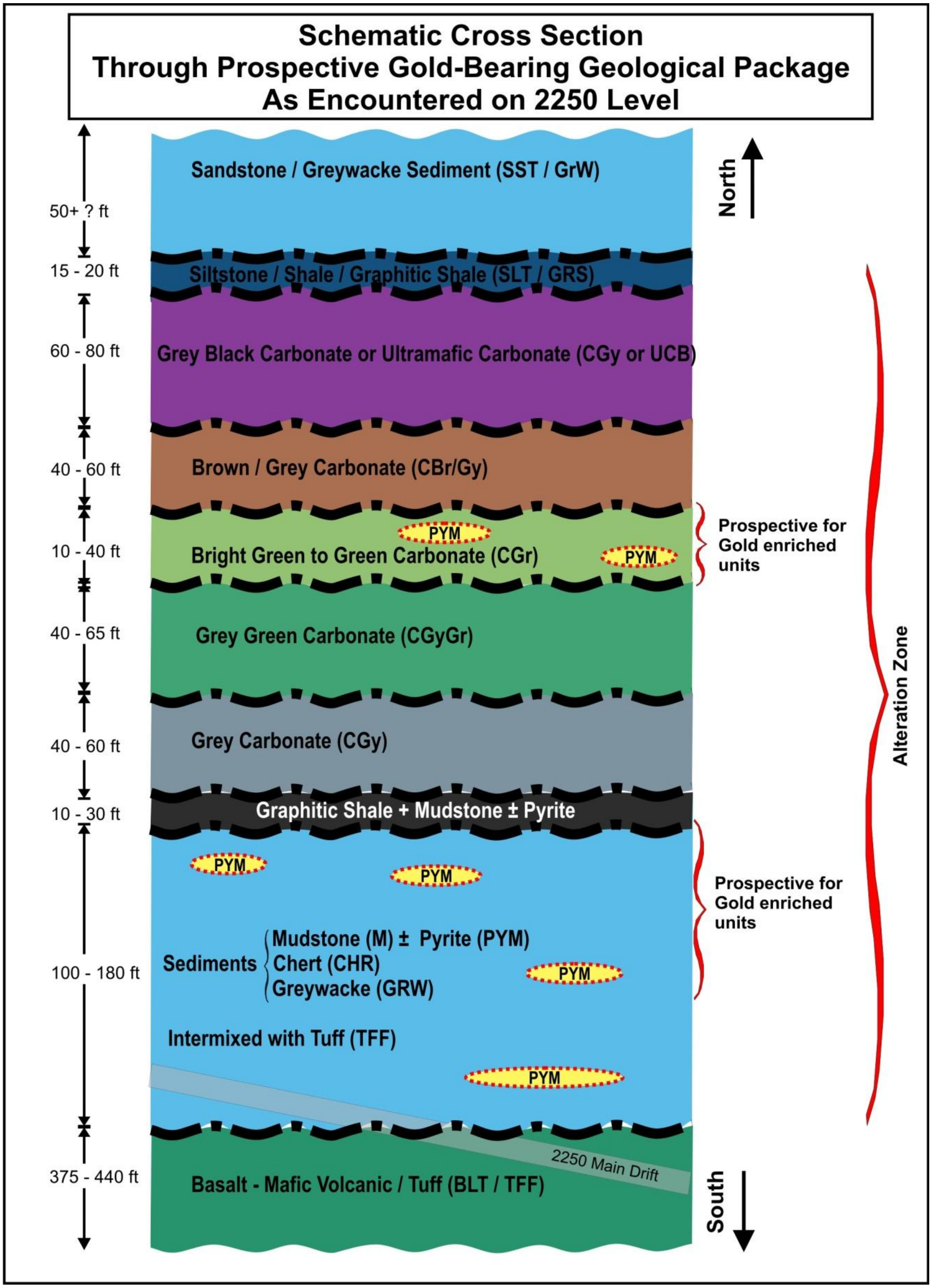


Figure 7 - Schematic Section Through the Prospective Geological Package as Encountered on 2250 Level

Rock Name	Gemcom Geocode	Lithological Description (North to South)
Conglomerate	CNG	stretched light grey/cream coloured subrounded conglomerate pebbles surrounded by a fine grained grey/light grey silicified cement, rare but usually in the Barber Larder Sediments
Sandstone/Greywacke	SST	light grey/grey fine grained/weakly to strongly silicified, massive to moderately foliated approaching schistose
	GRW	more metamorphosed/ foliated of above unit
Siltstone	SLT	light grey/light green tinge, very fine grained, silty appearance, can be interbedded with sandstones or shale units, moderate to strongly foliated, in core it looks disk like
Graphitic shale	GRS	grey,dark grey, black, shale with numerous graphitic slips/slickensides, sometimes has numerous small white calcite/quartz stringers, 1-2mm cubic pyrite growth, sometimes interbedded with mudstone, siltstone or sandstone laminae, evidence of faulting is observed by badly broken core, mud gouge
Dark Grey Black Carbonate	CGyBk	dark grey, black, altered carbonate, weakly to moderatly foliated, often interbedded with grey/light grey quartz flooding or grey fine grained chert-like mineral, localized small pebble like (1-2mm) augen growth
Ultramafic Carbonate Grey, Black, Brown, Green	UCGy, UCGyBk, UCGyBr, UCGyGr	similar to above, different colour tinges or tones, sometimes more localized augen growth (sometimes noticeably larger pebble size 1-5mm), grey/light grey quartz or grey fg chert like flooding, what differentiates this unit from above is the developed foliation always seems to be swirled, not matter what the orientation of the hole, it has been speculated that this unit has been extremely altered due to **external factors such as tectonic movement, folding or faulting etc.**
Brown (Buff) Grey Carbonate	CBr, CBrGy	light grey, grey brownish tinged carbonate, when foliation is evident it is weakly to moderate, there is similar amount of localized augen growth as above, it has been speculated that as this unit was being altered it was exposed to a hydrothermal fluid which resulted in the development of sericitic or biotitic mineral growth
Bright Green, Green Carbonate	CGr, CGrGy	lime green to dark green, grey lithological unit, fuchsite alteration is the greatest alteration product here, there are grey green flakey thin fuchsite laminae growth with creamy white quartz veinlets or flooding, or more solid looking green/dark green units, this zone usually encapsulates or surrounds an almost unaltered fine grained grey/light brownish mudstone or chert (PYM), these units are usually enriched with 5 to >20 % fg pyrite growth and/or grey quartz flooding, this zone is usually enriched in fg gold mineralization, historicaly called a "Carbonate Ore Zone"
Grey Green Carbonate	CGyGr, CGyGrBr	similar to the description to the brown carbonates, light grey, grey with a noticeable green tinge, this green tinge is produced by the growth of green fuchsitic laminae within the foliation,weak to moderately foliated, there is localized pebble like augen growth, grey light grey quartz flooding/veining, or grey fine grained chert like mineral
Grey Carbonate	CGy, CGyBr	light grey, grey carbonate, more sericite growth than fuchsite or biotite growth,weak to moderately foliated, grey, light grey quartz flooding/veining, or grey fine grained chert like mineral, localized pebble sized augen growth
Graphitic Shale + Mudstone +/- Pyrite	GRS, GRSM	similar to above description, usually a thicker sequence than above but often quite variable in width, moderately to strongly graphitic, it would not be a stretch to call this unit a fault zone, if mudstone is present it is usually a dark grey/black variety, often accompanied by 1/2 to 4 feet wide white bull quartz carbonate vein, which often has coarse cubic pyrite cluster growth associated with the original carbonaceous sediment ?? Some times gold enriched pyritic mudstones can be found within this lithological package but problems arise with the mining of these ore zones
Sediments (Flows) Mudstones, +/- Pyrite, +/- Gold = Pyritic Mudstone, Chert, Greywacke/Sandstone, Interbedded with Mafic Volcanic Tuff	M, MTFF, MCHR, CHRM, PYM	Mudstones; light grey, grey fg typical mudstone/argillite rock type, massive to moderatley foliated, often interbedded with mafic volcanic tuffaceous material, chert, greywacke. Pyritic Mudstone; similar to a mudstone, light grey, grey, light brownish tinge, in order to be defined as a PYM and not a Mudstone it has to have from 3 to >20 % fine grained pyrite, often this pyrite appears to be lined up in semi-parallel direction to foliation and disseminated within, if the PYM is enriched with a grey/black quartz flooding/silicification this rock will demonstrate very enriched gold values from .2 to > 1.0 ounce/ton, if the quartz/silicificiation is absent only subore gold values will be detected <.09 ounces/ton. Cherts; light grey, grey, light brown, brown colour, often associated with circular pyritic selvage rims, it has been speculated that this unit is an altered volcanic pillow structure, often strongly silicified therefore the chert description, if accompanied by 3 to > 20 % fine grained pyrite and grey/black quartz flooding veining often gold enriched. From logging observation it could be deduced that this Chert unit is exactly the same as the PYM unit. Throughout the history of the Armisitice McGarry Project there has never been a definitive definition of this unit. Greywacke/sandstone; fine grained light grey, grey, massive, weakly foliated unit
Mafic Volcanic Basalt/Tuff	BLT, TFF	grey, dark grey, sometimes dark green (chloritic) homogeneous massive to weakly foliated mafic volcanic unit. Tuff unit has thin small light grey or white phenocrysts spherules which appear to be lined up with foliation, sometimes associated with thin grey white quartz carbonate tension veinlets

Table 3 - McGarry Rock Type Descriptions

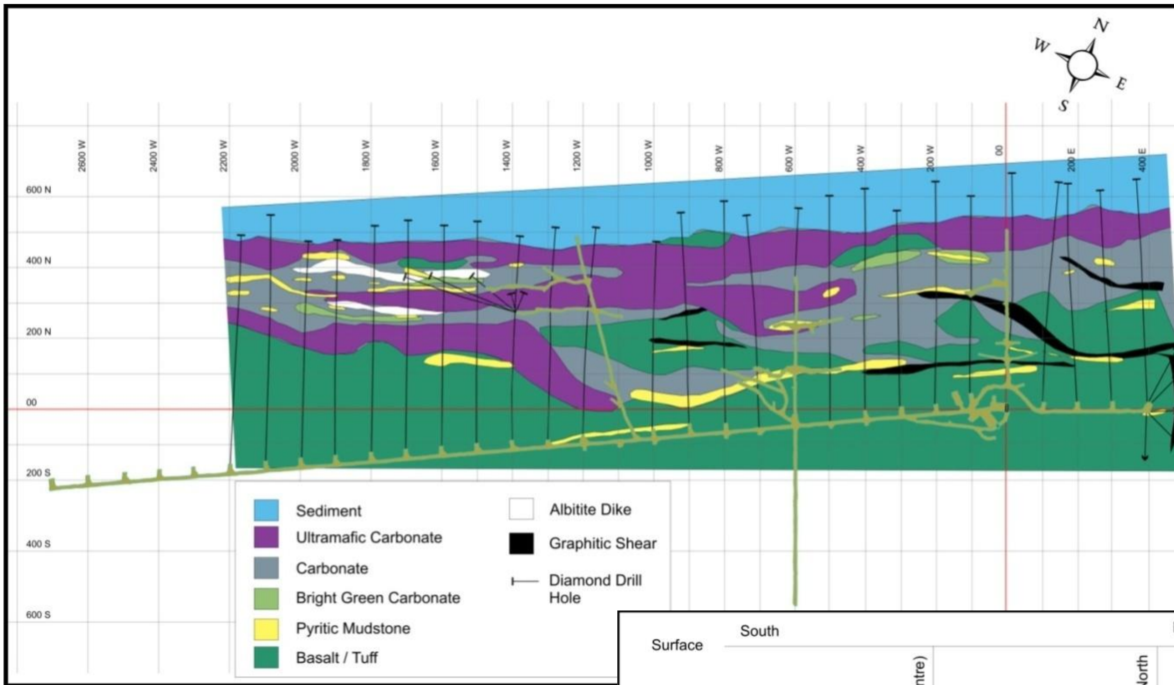


Figure 9 - Simplified Geological Plan – 2250 Level

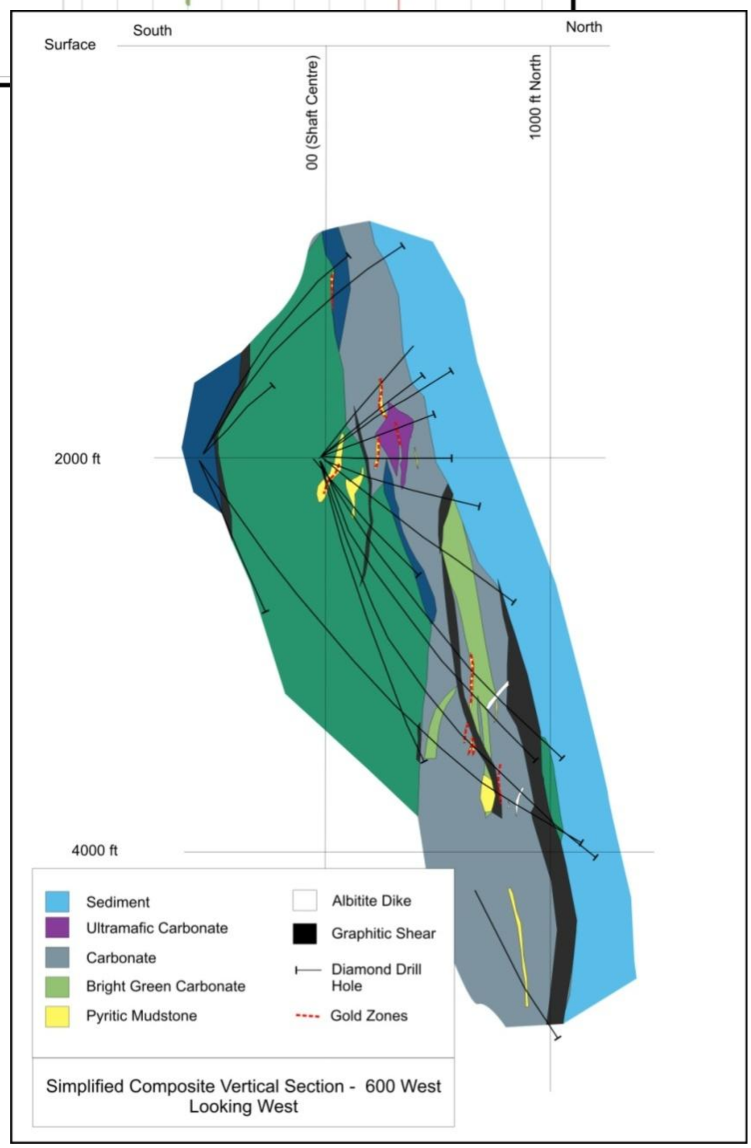


Figure 8 - Simplified Geological Vertical Section

8. Deposit Types

Gold-bearing zones in the Virginiatown camp area commonly occur as repetitive tabular lenses within veined green carbonate rock and pyritic cherty mudstones. In the carbonates, erratic and often coarse gold in native form occurs in contained quartz veining and is locally referred to as “green carbonate ore”. In the cherty mudstones, gold generally occurs in crystal intergrowth with disseminated pyrite and is locally referred to as “flow ore”. There are other deposit types such as graphite-rich and within altered albitite dikes but these are minor and local.

Brecciated stockwork zones of green carbonate rock also constitute an important deposit type in the area. In these, native gold occurs at quartz vein contacts and is particularly strongly associated with flat or low angle veining. Typically, such zones on the adjoining Kerr Addison mine property could be up to 100 ft in thickness and several hundred feet in length and vertical extent. The relative importance of the two major deposit types within the Kerr Addison mine are highlighted below in Table 13 on page 53.

Because of the erratic distribution of gold within them, the stockwork zones are very difficult to identify, and were often recognized in drilling at the Kerr Addison mine by the presence of 20 to 30 percent vein quartz carrying a few flecks of native gold. They were generally mined on a bulk shrinkage basis. The only well identified occurrence of this style of mineralization at the McGarry Project is on the 2250 Level in the gold zone encountered in the 600W cross-cut in the 260N Zone.

The predominant deposit type at Armistice is “pyritic mudstone” with varying amounts of quartz. It is difficult to determine the primary nature of these pyritic mudstone units. They may be inter-volcanic, shallow marine accumulations of sediments derived from the volcanics with the sulphides and gold originating from gases and fluids emanating from fissures related to the volcanic activity.

As discussed below, the current working model is that the green carbonate deposit type is a hydrothermal alteration variant of the “pyritic mudstone” type.

A resolution of the deposit type model is beyond the scope of this report and has been an ongoing discussion point among geologists working in this camp for many years. Work to date at McGarry has clearly shown, empirically, that the deposit type being referred to as “pyritic mudstone” is of the chief economic interest.

An excellent description of the deposit types at the Kerr Addison Mine is presented in a paper entitled The Kerr Addison-Chesterville Archean Gold-Quartz Vein System, Virginiatown: Time Sequence and Associated Mafic "Albite" Dike Swarm (Smith, 1990).

9. Mineralization

Pyritic mudstone units ("flow ore") constitute the most important gold-bearing zones at McGarry. Pyritic mudstone units appear to follow distinct formational horizons. Gold in such zones occurs mainly in intergrowth with pyrite and only sparingly in native form. The pyrite is medium to coarse grained and occurs in the cherty mudstones in disseminations varying from 1 to 25 percent of the rock volume. Minor quartz veining and silicification is commonly in evidence. Arsenopyrite and occasionally chalcopyrite may be present.

A second gold-bearing mineralization type has similarities to the "green carbonate ore" at Kerr Addison. This style of mineralization was of lower grade at the Kerr Addison than the "flow ore" type. The pyrite content is in the 1 to 25% range. Gold distribution is probably the same as in the pyritic mudstones but has been locally concentrated into larger grains so that visible gold is more common. The overall gold distribution appears to have been dispersed over a larger and more poorly defined volume which lowers the average grade over mineable widths and lengths and, as a corollary, increases the internal dilution within potential stoping areas.

Figure 10 shows the gold-bearing zones as identified on the 2250 Level.

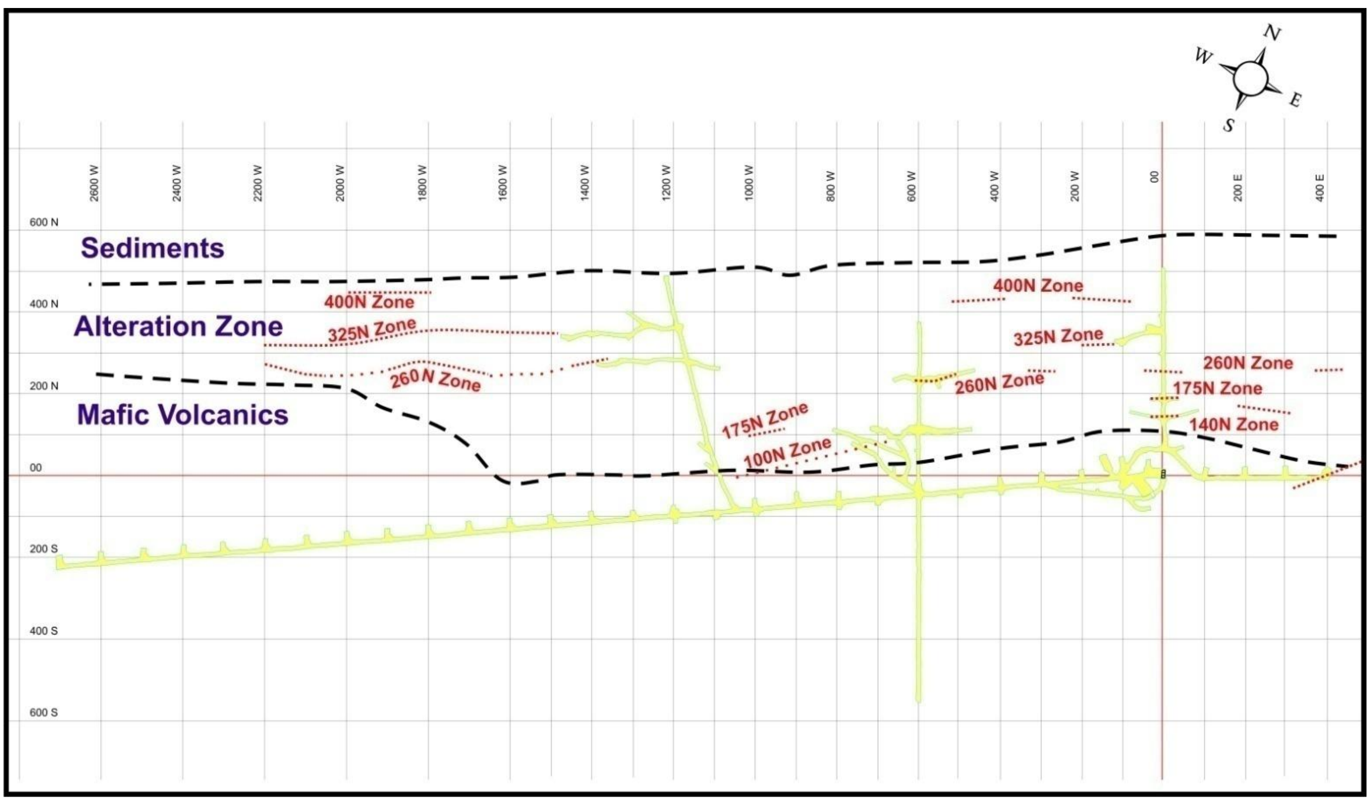


Figure 10 - Simplified Geology and Gold-Bearing Zones on the 2250 Level

10. Exploration

The McGarry Property has undergone various phases of exploration since the original claims were staked in the early 1900's. Most of this effort has been concentrated along the main axis of the Larder Lake "Break" which passes through the Property in the vicinity of Barber Lake. However, additional mineralized horizons in the southern part of the Property near Larder Lake have also been trenched, sampled and drilled, most recently by Armistice during the period 1986 to 1994. However, these gold prospects appear limited in size and grade as outlined by exploration completed to date. Historically, there are four named gold-bearing zones identified: the Dike Zone, the Mill Zone, the Western Zone and the Lamprophyre Zone. Although this area is outside the scope of this report, the area does present an ongoing exploration target of lower priority.

There has been virtually no historical exploration work in the northern part of the Property and none in the past 25 years.

No surface geological programmes were conducted during the 2007-2008 period. No geophysical (ground or airborne) or geochemical exploration programmes were conducted during the 2007-2008 period.

11. Drilling

Including the current programme, records have been located for 302,319 ft of diamond drilling in 407 holes plus 26 wedge cuts completed on the McGarry Property since the 1940's. Data from all these holes is included in the Gemcom database. There may be some additional holes from the 1940's for which records have not yet been located. Any such missing data would not be significant to the current resource evaluation or the interpretation of the economic geology of the Property in the opinion of the author. This work has been carried out by various contractors over the period. All post 1970's, drill core is BQ size or equivalent including all the drill core from the 2007-2008 programme.

Drill hole collars at the mine site have been surveyed and located on the mine grid. All drill holes completed between 1988 and 1998 have been downhole surveyed using a Sperry Sun unit. All holes drilled in the 2007-2008 programme were surveyed with a single-shot down hole Flex-It unit from which the dip, magnetic bearing, total magnetic field and temperature is read. Down hole surveys were taken every 100 feet in the 2007-2008 programme.

All drilling in the 2007-2008 programme was performed under contract by Cabo Drilling (Ontario) Corp. Two B-15 electric-hydraulic drilling rigs were used on the 2250 Level each with a rated capacity of about 2,000 feet, although Cabo regularly exceeded this capacity and managed to drill to 2,700 ft in the wedged hole DDH 22W60-7B, for example.

In addition, a short hole, hand portable, compressed air powered VEG drill was used for detailed testing ahead of drifting advance. The drill was set up in the western end of the 260W Drift west of the 1050 X/C to guide advance for the 325W Drift. The longest "short hole" was 330 feet.

All drilling, sampling, surveying (collar location and down hole deviation) and core logging data has been digitized and compiled into a Gemcom database. The complete database is available in Armistice's offices in Kirkland Lake.

A summary of the drilling performed on the Property and incorporated into the Gemcom database is presented in Table 4.

Period	Comment	Number of Holes	Number of Wedged Holes	Feet Drilled
Pre 1970		83		32,716
1974		1	5	8,036
1980		1	2	6,254
1988-89		56		42,423
1995-1998		223	14	168,439
Sub Total Historical		364	21	257,868
2007-2008	Definition	26		
	Down Holes	3	5	
	Up Holes	9		
	Short Holes	5		
Sub Total Current Programme		43	5	44,451
Grand Total to Date		**407**	**26**	**302,319**

Table 4 - Summary Drilling Statistics

To illustrate the overall density of the drilling included in the Gemcom database, Figure 11 below shows the traces of all holes in composite cross section and longitudinal section.

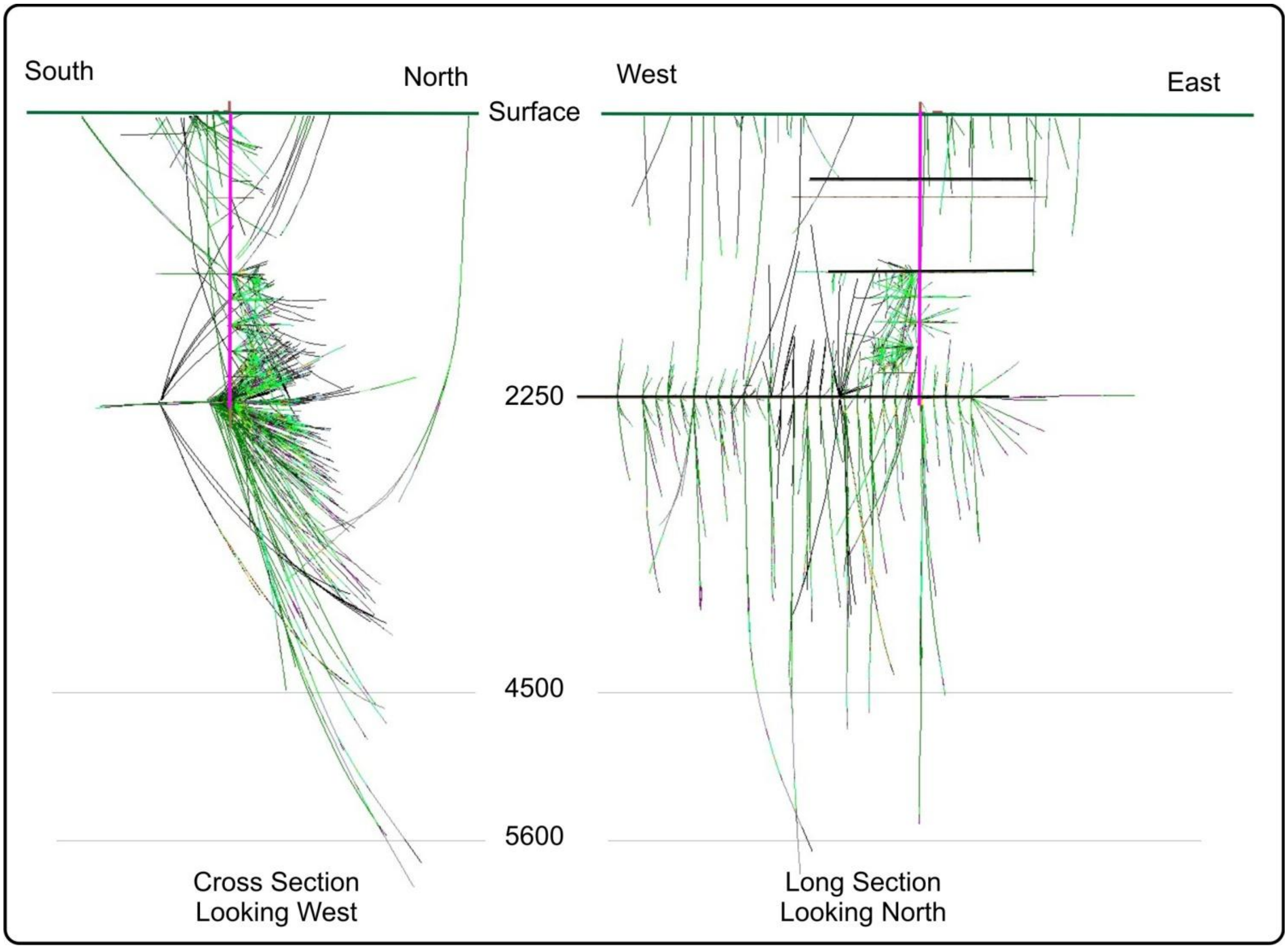


Figure 11 - Composite Cross Section and Long Section Showing All Drill Holes

11.1. Deep Drilling

Past geological concepts have focused on exploring the prospective geology below the 1250 Level with a strong emphasis below the 2250 Level as can be seen in Figure 11 above. Until the 2250 Level was established in the late 1980's and extended in the 1990's, underground drilling was limited to a narrow corridor near the shaft.

With the establishment of the 2250 Level footwall drift which extended over much of the prospective strike length of the Property, there was an active programme to define the potential gold zones to depth. At the end of 1999, a footwall cross-cut was driven 500 feet south of the main 2250 Level drift on the 600W section. A large drill station was established at the end of this cross-cut with the purpose of providing a platform for future deep drilling. This drill station was occupied during the 2007-2008 programme from which 3 deep holes were collared. A total of 5 wedge cuts were taken from the 3 pilot holes. These holes were targeted to explore the prospective geology at depths of about 3,500 to 4,000 feet below surface since the practical operating depth of the current hoisting plant is about 4,400 feet. The significant assay results from the 2007-2008 deep drilling programme are summarized in Table 5.

Drill Hole	Total Hole Length from Collar (ft)	From (ft)	To (ft)	Interval (ft)	True Width (ft)	Grade (oz/t)	Coordinates of Intersection Centre (on mine grid – ft)			Comments
							Easting	Northing	Depth	
22W60-7	1730									Not far enough - wedged
22W60-7B	2695									No significant assays
22W60-8	1220									Not far enough - wedged
22W60-8B	2230	1532.9	1534.2	1.3	0.9	0.10	605 W	415 N	-3435	Pyritic Mudstone
		1965.3	1967.3	2.0	1.5	0.11	620 W	745 N	-3715	Pyritic Mudstone
		2023.0	2025.6	2.6	2.1	0.43	620 W	790 N	-3750	Medium Grey Carbonate
22W60-8C	2400	1975.7	1984.2	8.5	7.1	0.09	630 W	750 N	-3710	
		1995.0	2002.3	7.3	6.1	0.12	635 W	775 N	-3730	
22W60-8D	2350	2011.8	2014.5	2.7	2.2	0.15	590 W	785 N	-3740	Graphitic Shale
22W60-9	2570	1658.8	1663.3	9.4	7.1	0.14	900 W	470 N	-3515	Pyritic Mudstone
22W60-9B	2200	1592.1	1598.2	6.1	5.1	0.11	890 W	425 N	-3475	Pyritic Mudstone

Table 5 - List of 2007-2008 Deep Drill Holes and Significant Intersections

Favourable gold-bearing environments were encountered in pre-2007-2008 drilling to depths of 5,600 feet. The downdip potential below 5,600 ft remains open. The depth potential below 4,000 ft has been tested by 8 drill holes all drilled prior to 2007-2008. Significant results from these 8 holes are highlighted below:

Drill Hole	Total Hole Length (ft)	From (ft)	To (ft)	Interval (ft)	True Width (ft)	Grade (oz/t)	Coordinates of Intersection Centre (on mine grid – ft)			Comments
							Easting	Northing	Depth	
22-96	2812	2147.5	2206.2	58.7	33.5	0.109	460 W	845 N	-4240	Pyritic Mudstone/Siltstone
	Including	2147.5	2157.1	9.6	5.5	0.304	460 W	835 N	-4225	Pyritic Mudstone
	and	2169.7	2175.1	5.4	3.1	0.316	460 W	845 N	-4235	Pyritic Mudstone
	and	2197.3	2206.2	8.9	5.1	0.182	460 W	860 N	-4260	Pyritic Mudstone
22-107C	4082	1721.0	1723.5	2.5	1.0	0.121	1050 W	400 N	-3905	Silicified Quartz Vein
		1787.5	1792.5	5.0	1.5	0.146	1050 W	425 N	-3970	Basalt with pyrite
		1816.0	1819.5	3.5	1.1	0.245	1050 W	435 N	-4000	Basalt with pyrite
		2687.0	2697.0	10.0	3.1	0.224	1070 W	770 N	-4800	Graphitic Shale with pyrite
22-66E	1934	3515.8	3519.0	3.2	1.8	0.103	85 W	1170 N	-5560	Pyritic Mudstone

Table 6 - List of Significant Intersections below 4,000 ft

Figure 12 below presents a schematic long section in the approximate plane of the mineralized zones showing the pierce points of all drill holes. A note of caution: it is difficult to show accurately the pierce points because: 1) the mineralized zones do not lie on a simple plane, and 2) the zones occur within a north-south corridor up to 600 feet wide so there is not a unique pierce point for each drill hole. The purpose of the figure is to give a visual impression of the density of drilling in schematic overview only. Major gaps in drill density are readily seen in this figure, of particular note are the areas west and east of the shaft above the 2250 Level.

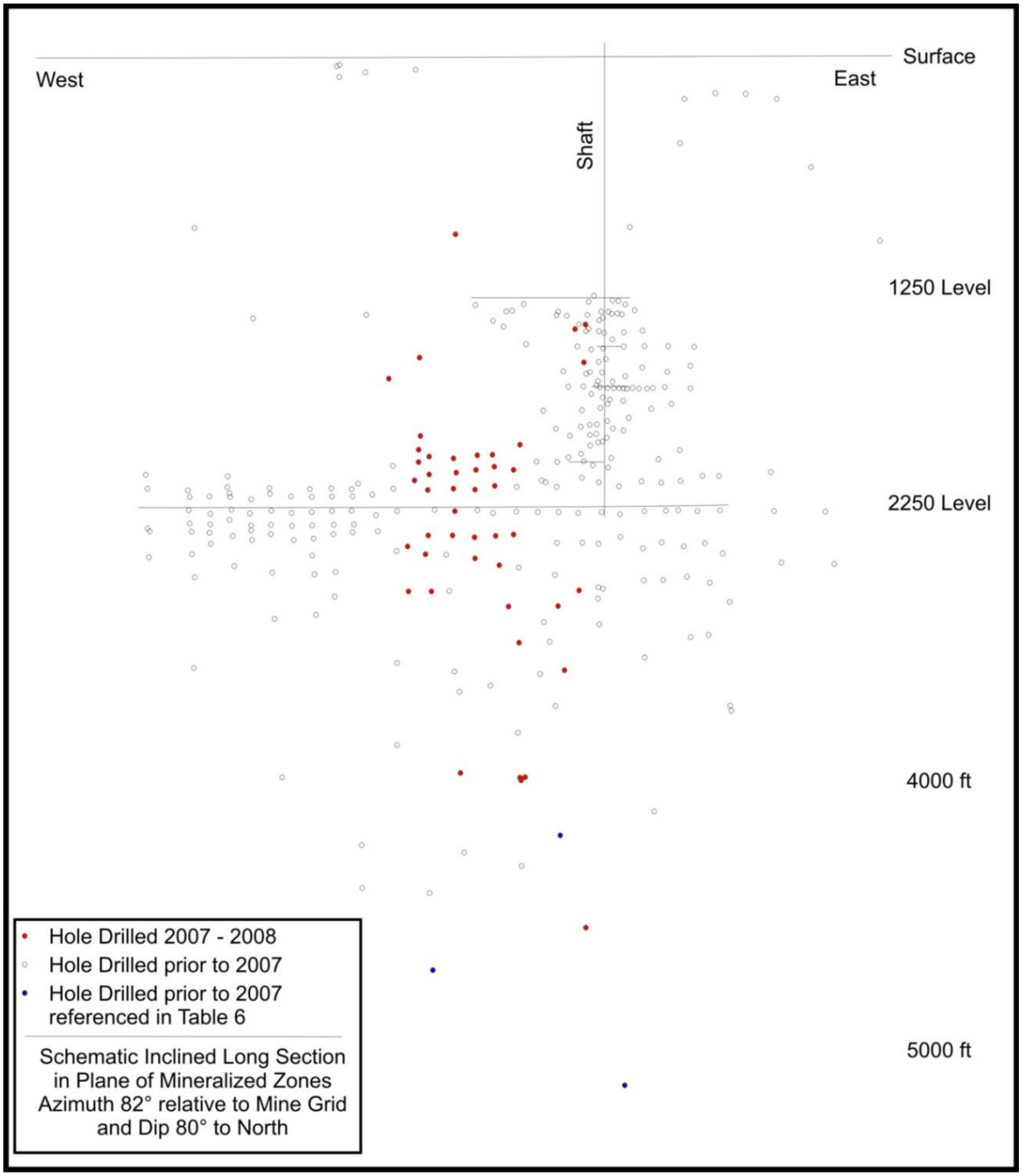


Figure 12 - Schematic Long Section in Plane of Mineralized Zones Showing Drill Holes

11.2. Definition Drilling

One of the prime objectives of the 2007-2008 work programme was to complete definition drilling on the 2250 Level as recommended in the previous NI 43-101 report (Carmichael, S.J., June 6, 2004). An ongoing programme of "definition" drilling from stations spaced 100 apart along the main 2250 E and W drifts had been undertaken during the periods 1988-89 and 1995-98. A nominal pattern of 7 holes drilled at dips of +52°, +40°, +22°, 0°, -22°, -40° and -52° with hole lengths of 600 to 700 feet were drilled north from each station. About 4400 feet of drilling from each station is required for the nominal pattern. This drill pattern tests the prospective geological package from about the 2450 elevation to the 2050 elevation; that is, respectively 200 feet below and above the 2250 Level.

Although this drill pattern is referred to as "definition drilling", and is considered sufficient by the author for the estimation of indicated mineral resources, the hole spacing provides only a "first pass" view of the mineral resource being explored. This issue is discussed further in Section 19.2.2. (page 77) under Interpretation and Conclusions.

Following the completion of work in 1998, there remained 6 partially or totally incomplete definition drill sections extending from 600W to 1100W. A nominal 20,500 feet of drilling was required to complete these sections. All these sections were completed in 2007-2008. See Figure 5, page 13, for the location of the drill stations.

Definition drilling on the 2250 Level is now largely complete over a strike distance of 3,000 feet from 400E to 2600W. Some sections still require a few holes to complete the full nominal pattern. On some sections, holes from pervious campaigns were also drilled at steeper dips (up to -70°) to test the prospective geological package at deeper elevations.

Table 7, below, lists all the holes (2007-2008 plus prior years) drilled on the definition drill sections completed in the current programme. All intersections over about 0.1 oz/t have been listed, together with the location of the intersection centre in 3D space and a brief note on the geological unit in which the intersection is located.

Significant Assay Results from 2007-2008 Definition Drilling Programme - 2250 Level - Sections 600W to 1100W
Intersections with 0.10 oz/t gold or greater (includes results from previously drilled holes on the subject section)

Drill Section	DDH #	Dip at Collar	From	To	Interval	True Width	Coordinates of Intersection Centre			Average Gold	Geology	Comments
			feet	feet	feet	feet	Easting	Northing	Depth	oz/t		
1100 W	22W110-1	+64°	--	--	--					--		No significant assay results
	22W110-4	+53°	--	--	--					--		No significant assay results
	22W110-2	+44°	57.8	60.0	2.2	1.6	1095 W	40 S	-2200	0.09	Pyritic Mudstone	
			70.0	72.3	2.3	1.7	1095 W	30 S	-2190	0.10	Pyritic Mudstone	
			360.0	365.0	5.0	3.9	1070 W	190 N	-2010	0.10	Grey/Brown Carbonate	
	22W110-3	+30°	270.0	271.9	1.9	1.7	1095 W	155 N	-2120	0.13	Pyritic Mudstone	
			377.1	383.0	5.9	5.3	1090 W	255 N	-2070	0.09	Grey/Green Carbonate	
	22W110-5	-25°	7.0	9.5	2.5	2.3	1100 W	75 S	-2250	0.26	Pyritic Mudstone	
			170.0	175.0	5.0	4.6	1110 W	70 N	-2320	0.50	Grey-Green Carbonate	cut to 1.0 oz/t (0.60 oz/t uncut)
			511.0	512.0	1.0	0.9	1105 W	390 N	-2430	0.12	Pyritic Mudstone	
	22W110-6	-35°	5.2	9.2	4.0	3.3	1100 W	75 S	-2250	0.15	Pyritic Mudstone	
			242.1	246.0	3.9	3.4	1120 W	125 N	-2380	0.17	Pyritic Mudstone	
			730.0	740.0	10.0	9.0	1175 W	565 N	-2600	0.10	Pyritic Mudstone	
	22W110-7	-52°	2.4	8.7	6.3	3.9	1100 W	75 S	-2250	0.13	Pyritic Mudstone	
			145.2	150.0	4.8	3.2	1110 W	15 N	-2360	1.00	Altered Pyritic Mudstone	cut to 1.0 oz/t (3.35 oz/t uncut)
1000 W	22W100-1	+51°	74.8	76.3	1.5	1.0	1005 W	5 S	-2180	0.17	Pyritic Mudstone	
	22W100-2	+38°	27.9	36.9	9.0	7.0	1005 W	30 S	-2220	0.12	Quartz Vein / Pyritic Mudstone	
			320.0	322.0	2.0	1.8	1025 W	210 N	-2060	0.12	Pyritic Mudstone	
	22W100-3	+22°	150.0	152.5	2.5	2.3	1015 W	80 N	-2190	0.16	Green Carbonate	
	22-23	0°	--	--	--					--		No significant assay results Drilled in 1996
	22W100-4	-23°	238.8	240.0	1.2	1.1	1020 W	165 N	-2335	0.10	Pyritic Mudstone	
	22W100-5	-36°	218.8	220.0	1.2	1.1	1025 W	125 N	-2370	0.14	Pyritic Mudstone	
	22W100-6	-49°	51.2	54.0	2.8	1.8	1005 W	20 S	-2290	0.10	Mudstone	
			134.6	136.0	1.4	0.9	1005 W	30 N	-2350	0.66	Pyritic Mudstone	
			794.7	804.0	9.3	7.0	1045 W	500 N	-2820	0.12	Pyritic Mudstone	
			840.9	842.1	1.2	0.9	1050 W	530 N	-2845	0.11	Pyritic Mudstone	
	22-107C	-80°	10.2	21.3	11.1	1.9	995 W	60 S	-2265	0.10	Pyritic Mudstone	First 800 ft only of 4082 ft hole Drilled in 1997
900W	22W90-6	+48°	--	--	--					--		No significant assay results
	22W90-7	+37°	--	--	--					--		No significant assay results
	22W90-8	+22°	93.1	96.9	3.8	3.5	900 W	35 N	-2185	0.12	Pyritic Mudstone	
			110.0	112.0	2.0	1.9	900 W	47 N	-2170	0.85	Pyritic Mudstone	
			420.0	423.0	3.0	2.9	905 W	355 N	-2100	0.10	Pyritic Mudstone	
	22W90-9	0°	69.1	74.6	5.5	5.5	900 W	30 N	-2245	0.16	Pyritic Mudstone	
	22W90-10	-23°	26.0	28.4	2.4	2.2	900 W	15 S	-2260	0.11	Pyritic Mudstone	
			54.3	56.2	1.9	1.8	900 W	10 N	-2270	0.16	Mudstone	
			127.5	139.1	11.6	10.8	900 W	80 N	-2300	0.18	Light Grey-Green Carbonate	
	22W90-1	-37°	--	--	--					--		No significant assay results Drilled in 1989
	22W90-2	-55°	--	--	--					--		No significant assay results Drilled in 1989
	22W90-4	-64°	44.2	47.2	3.0	1.1	900 W	20 S	-2290	0.14	Mudstone	First 800 ft only of 1898 ft hole Drilled in 1997
	22W90-5	-68°	47.5	50.0	2.5	0.8	900 W	20 S	-2295	0.10	Mudstone	Drilled in 1989
			226.6	230.0	3.4	1.3	895 W	50 N	-2260	0.11	Mudstone	
	22W90-3	-70°	47.4	50.0	2.6	0.8	900 W	30 S	-2285	0.15	Cherty Mudstone	First 800 ft only of 1898 ft hole Drilled in 1997
800W	22W80-1	+50°	323.1	329.0	5.9	4.2	790 W	185 N	-2005	0.13	GreyBrown Greywacke Carbonate	
	22W80-2	+38°	334.6	339.1	4.5	3.8	795 W	235 N	-2050	0.20	Pyritic Mudstone	
	22W80-3	+20°	339.2	344.1	4.9	4.7	800 W	280 N	-2130	0.23	Altered Greywacke Mudstone	
	22-22	0°	--	--	--					--		No significant assays in first 800 ft of 1537 ft hole Drilled in 1996
	22W80-4	-24°	250.0	255.3	5.3	5.0	805 W	195 N	-2340	0.10	Mudstone/Pyritic Mudstone+Chert	
	22W80-5	-36°	68.0	75.0	7.0	5.7	800 W	20 N	-2290	0.10	Pyritic Mudstone	
			268.0	269.4	1.4	1.2	805 W	180 N	-2400	0.17	Pyritic Mudstone	
	22-103	-65°	103.6	105.0	1.4	1.2	790 W	15 S	-2345	0.17	Pyritic Mudstone	First 800 ft only of 2052 ft hole Drilled in 1997
700W	22W70-3	+52°	185.0	191.0	6.0	4.5	705 W	75 N	-2100	0.26	Dark Grey Carbonate	
			217.5	221.3	3.8	2.8	705 W	95 N	-2080	0.28	Grey-Green Carbonate	
			238.9	241.3	2.4	1.8	710 W	170 N	-2015	0.11	Graphitic Pyritic Mudstone	
			314.7	319.4	4.7	3.6	710 W	170 N	-2015	0.19	Medium Grey Carbonate	Quartz Vein - Visible Gold
			355.0	361.4	6.4	4.9	710 W	205 N	-1990	0.17	Grey-Green Carbonate	
	22W70-4	+43°	167.8	171.3	3.5		700 W	75 N	-2140	0.20	Graphitic Pyritic Mudstone	Visible Gold
	22W70-5	+22°	145.6	150.0	4.4	4.1	700 W	85 N	-2190	0.10	Pyritic Mudstone	
			435.0	439.0	4.0	3.8	705 W	355 N	-2090	0.24	Medium Grey Carbonate	
	22W70-1	0°	160.0	168.5	8.5	7.7	705 W	110 N	-2250	0.51	Dark Grey Carbonate	Cut to 1.0 oz/t (4.26 oz/t uncut) Drilled in 1989
	22W70-2	-23°	--	--	--					--		No significant assay results Drilled in 1989
	22W70-6	-39°	185.5	195.0	9.5	7.8	700 W	100 N	-2365	0.11	Pyritic Mudstone	Hole not drilled far enough
	22W70-7	-52°	230.0	233.0	3.0	2.0	960 W	95 N	-2430	0.10	Pyritic Mudstone	Hole not drilled far enough
600W	22W60-4	+51°	414.0	418.6	4.6	3.0	575 W	250 N	-1915	0.29	Ultramafic Conglomerte	
	22W60-5	+40°	332.0	336.0	4.0	3.4	605 W	260 N	-2050	0.16	Grey/Green Carbonate	
			344.7	349.0	4.3	3.7	605 W	270 N	-2040	0.25	Grey/Green Carbonate	
	22W60-1	+22°	272.2	276.0	3.8	3.2	600 W	240 N	-2140	0.16	Ultramafic Carbonate	Drilled in 1989
			358.7	363.6	4.9	4.1	600 W	320 N	-2100	0.12	Chert	
	22W60-2	0°	245.1	252.5	7.4	6.8	600 W	235 N	-2245	0.34	Bright Green Carbonate	Drilled in 1989
			352.5	355.2	2.7	2.5	595 W	340 N	-2245	0.10	Medium Green Carbonate	
	22W60-6	-24°	--	--	--					--		No significant assay results
	22W60-3	-37°	206.5	208.2	1.7	1.2	600 W	150 N	-2375	0.11	Siltstone	Drilled in 1989
			620.0	623.0	3.0	2.2	605 W	480 N	-2620	0.20	Dark Grey Carbonate	
	22-97	-62°	--	--	--					--		No significant assays in first 800 ft of 1881 ft hole Drilled in 1997
	22-111	-69°	--	--	--					--		No significant assay results Drilled in 1997
	22-111A	-72°	--	--	--					--		No significant assays in first 800 ft of 2287 ft hole Drilled in 1997
	22-101	-75°	--	--	--					--		No significant assays in first 800 ft of 2861 ft hole Drilled in 1997

Table 7 - Tabulation of All Significant Assays from Definition Drill Section 2007-2008

11.3. Up-Hole Drilling

Following a review of the exploration and development strategy for the Property, it was recognized that a gap in the drilling data existed between the 1250 and 2050 elevations both west and east of the corridor near the shaft (see Figure 12, page 26). Issues with respect to the economic significance of this gap are further discussed in Sections 18.1. Scoping Study, page 65, and throughout Section 19. Interpretation and Conclusions.

With the completion of the Definition Drilling and Deep Drilling programmes, it was possible to use both available diamond drills for initial, wide spaced testing of the gap west of the shaft. This drilling programme is referred to as “Up-Hole Drilling”. The two available drill platforms, at the northern end of the 1050W X/C and the south end of the 600W X/C (see Figure 5, page 13), are not ideal for this testing but they were the best available.

Table 8, below, summarizes the results from the Up-Hole programme. As noted in the table, two of the holes were abandoned well before the target area was reached and two do not appear to have penetrated the target area. The steep drill dips make control of the natural drill hole deviation difficult. Only 5 of the 9 holes actually tested the target gap.

Significant Assay Results from 2007-2008 Up-Hole Drilling Programme - 2250 Level - Intersections with ~ 0.10 oz/t gold or greater

Drill Station	DDH #	Dip at Collar	Azimuth at Collar	From	To	Interval	True Width	Coordinates of Intersection Centre			Average Gold	Geology	Comments
				feet	feet	feet	feet	Easting	Northing	Depth	oz/t		
600W Drill Station at 500S	22W60-10	+50°	21°	--	--								No Significant Assays
	22W60-11	+64°	20°	1217.3	1221.4	4.1		375W	175N	-1275	0.26	Pyritic Mudstone	
	22W60-12	+65°	0°	998.2	1000.0	1.8		540W	105N	-1475	0.14	Pyritic Mudstone	
	22W60-13	+65°	350°	--	--						--		Too short to reach target
	22W60-14	+71°	320°	1172.5	1173.5	1.0		820W	125S	-1155	0.09	Pyritic Mudstone	Hole wandered too high and did not reach target zone
	22W60-15	+66°	320°	--	--						--		Hole aborted near start of hole
1050 X/C Drill Station at xx N	1050-1	+63°	177°	558.6	563.0	4.4		1205 W	195 N	-1740	0.11	Brown Carbonate	
				1044.9	1050.0	5.1		1190 W	115 S	-1370	0.09	Pyritic Mudstone	
	1050-2	+65°	152°	774.0	775.4	1.4		1065 W	135 N	-1550	0.12	Medium green Carbonate	
				846.3	851.5	5.2		1050 W	90 N	-1495	0.25	Medium green Carbonate	
	1050-3	+64°	200°	--	--						--		Hole aborted short of target

Table 8 - Tabulation of Results from Up-Hole Drill Programme

11.4. Short-Hole Drilling

It became clear that efficient drifting along the 260N and 325N Zones west of the 1050W X/C was difficult because there was no easily recognizable geological marker. Therefore, a short-hole air powered drill (VAG drill) was brought to the western end of the 260N drift west of the 1050N X/C (see Figure 5, page 13). This drill is light and is capable of drilling holes up to a practical maximum of 300 feet.

The prime objective of the short-hole programme was to provide detailed information to guide the westward advance of the 325N drift located about 75 feet north of the end of the 260N drift. All the holes drilled were flat or slightly elevated to avoid drilling into the pre-existing 325N drift. The azimuth angle to the 325N target was not ideal since the 325N drift was advanced almost 100 ft ahead of the 260N drift.

Figure 13, below, shows the results of the short-hole programme. Holes 260-325-4 and 260-325-3 clearly demonstrate the importance that holes spaced closer than 100 ft will have in locating mineralized zone continuity ahead of the 325N drift. The effective spacing of these two holes is only about 50 ft.

The good results just ahead and just to the north of the end of the 260N drift also confirm that if this information had been available before the drift was temporarily stopped that the 260N Zone mineralization could have been quickly exposed by mining. The 260N drift wanders a bit prior to this drill information because there were insufficient geological clues exposed in the drift to predict the exact location of the zone.

As discussed in Section 19.2.2. Interpretation and Conclusions – Diamond Drilling Strategy – Definition Drilling (page 77), a drill spacing density of between 25 and 50 ft is considered to be required for future programmes. The results below contribute significantly to this conclusion.

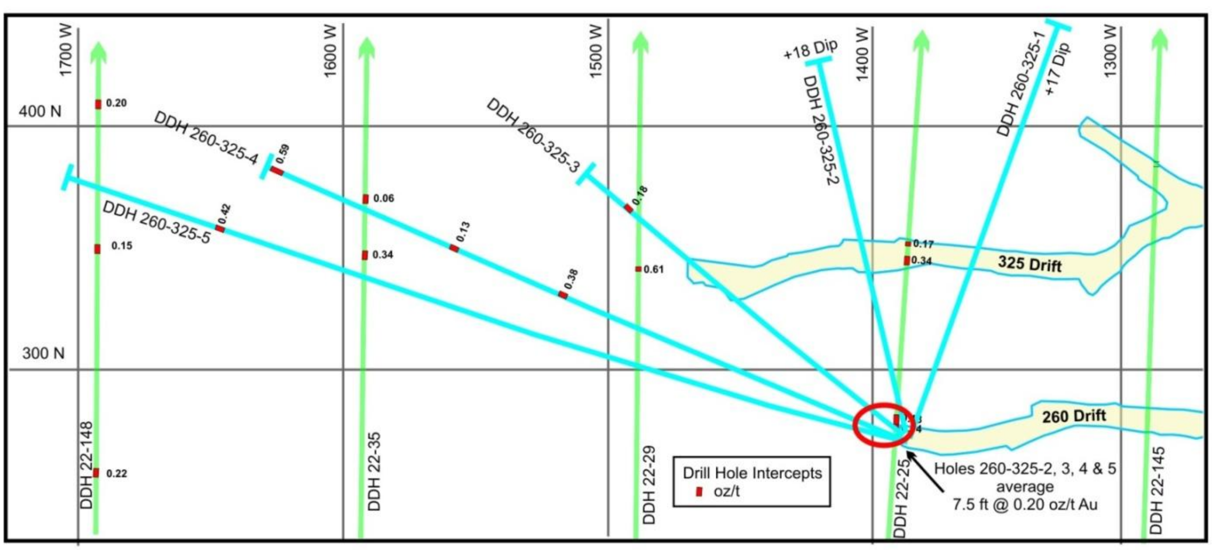


Figure 13 - Plan of Short Drilling at West End of 260N and 325N Zones

12. Sampling Method and Approach

12.1. Drill Core

In respect to BQ drill core, sample sections are selected during the logging process and include any mineralized sections noted. Generally, sample lengths are about 5 ft maximum and 1 ft minimum although most commonly 3 ft.. Attention is paid to ensure that selected samples are representative of any gold zones and of the potential waste rock on either side of the any gold zones.

The core is then halved using a diamond saw and one portion sent for assay and the other retained for future reference. Duplicate sample control tags are stapled into the core box at the start of each sample. Sample tag booklet control stubs are retained on file. All drill core is stored either in a locked building in Virginiatown owned by Armistice or in core racks at the McGarry project site.

For the short, closely spaced drill holes from the western end of the 260W drift, whole core was sent for assay which is standard industry practice for this type of very detailed drilling. Pulps and rejects plus core not sampled are retained as above.

For approximately one of every 20 core samples in the 2007-2008 drill programme, the sawn half of the core to be sent for assay was again sawn in half (producing two quarter cuts). One quarter cut was sent to each of two separate labs for assay. When quarter cuts were made, Armistice also inserted a blank (or other standard) sample into the sample batch. Sample control tags are placed into each sample bag. The results of drill core sampling and logging are incorporated into the Gemcom database.

12.2. Sludge Sampling

From time to time, holes are drilled into the walls of drifts (usually strike drive drifts) or stope walls with a jackleg percussion drill and the sludge produced is collected as samples. These holes are usually flat lying and extend 6 to 8 feet into the wall. Samples of the sludge are collected by the miner drilling the hole in plastic bags with each bag collecting sludge from successive 2-ft intervals. A control assay tag is placed in the bag.

The purpose of this type of sampling is to provide an indication if there is a gold-bearing zone just beyond the limits of current mining. Results of sludge sampling are not reliable in themselves and are never used for resource estimating although the results are recorded in the Gemcom database.

12.3. Chip Sampling

During mining operations, chip samples are taken as geologically warranted. Chip samples consist of a collection of pieces measuring about 1 inch per side chipped from the mining face at closely spaced intervals across the section to be sampled. The samples are chipped directly into the sample bag with the total sample weighing about two pounds. A sample control tag is also placed into each bag. Every attempt is made to collect the samples in an unbiased manner so that the sample is representative of the target sample section. The location of each chip sample is recorded on the face diagram that the geologist or sampler makes. The face sample sheets are retained and available in the Armistice files at the McGarry Project site. All chip sample locations and assays are incorporated into the Gemcom database.

12.4. Panel Sampling

In the case of the two 9 ft by 9 ft cross-cut drifts driven on the 2250 Level perpendicular to the trend of the mineralized zone, the 00 X/C and the 1050 X/C's, both the east and the west walls were sampled according to a comprehensive panel sampling protocol. One important objective of these two cross-cuts was to obtain a complete profile of the gold distribution through the prospective zone that would be significantly more representative than is possible with drill core. The entire length of each cross-cut was sampled without regard for the potential for gold mineralization as interpreted from the observed geology. This protocol was established to ensure that no potential gold zones would be missed and to obtain a full signature of gold values through the zone.

Panel sampling was done on a "mining round by mining round basis". Each mining round was either a nominal 6 or 8 ft in length (usually only about 5.5 and 7.5 ft respectively when the actual blasted break is considered). On each wall (west and east) a pattern was painted to divide the advance for each round into 6 or 9 equal squares or panels depending on the length of the round and as illustrated in Figure 14. Each panel was then chip sampled to provide a uniform distribution of chips from within the panel. The chip samples weighed approximately two pounds each. A panel has nominal dimensions of about 3 ft by 3 ft. In some cases, the top panels of the east wall could not be sampled because mining services (ventilation ducting and compressed air and water pipes) had already been installed. In some cases, non-standard panels were sampled to more accurately reflect the local geology. The extent of each mining round was surveyed for control.

The panel sample results are represented in the Gemcom database as three (or two) mock drill holes placed along the trace of each drift wall through the vertical centre of the panels (see Figure 14). That is, at about 1.5, 4.5 and 7.5 ft above the drift floor.

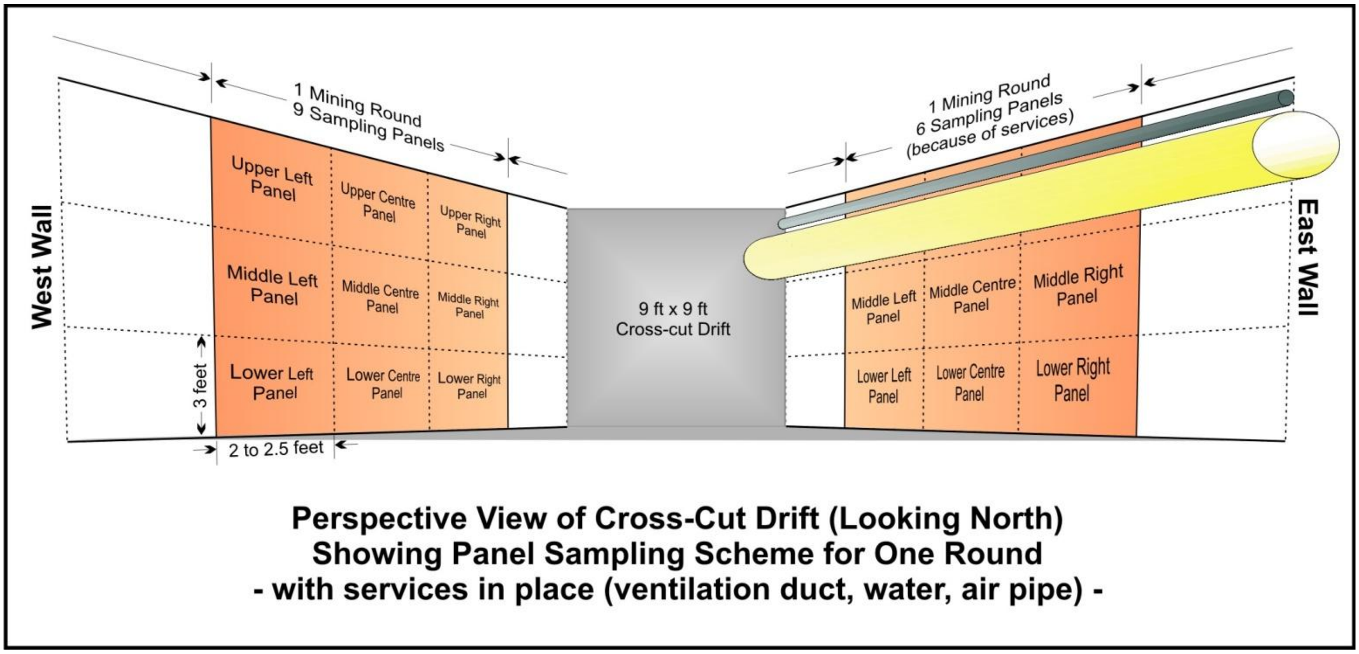


Figure 14 - Perspective View of Cross-Cut Drift Showing Panel Sample Scheme

12.5. Bulk Sampling

A programme of bulk sampling was carried out as a major part of the 2007-2008 work. In this report the term "bulk sampling" refers to the process of handling batches of mined rock from mucking to hoisting to crushing to sampling through to processing all in a manner that retains the integrity of the batch as a complete unit. A "batch" is nominally one 8 ft drift round of about 50 tons or, in the case of the test stopes, about 10 skips of rock. This process permits the identification of an eventual assay result with a specific mined round or stope.

The objectives of the bulk sampling programme were:

- to determine the relationship between drill hole data, chip samples and mined rock in the vicinity of the drill hole or mining face;
- to provide additional background to be used in developing an optimal diamond drilling strategy;
- to provide information on the horizontal and vertical continuity of gold mineralization to assist in mining method design;
- to provide a good estimate of the grade that would actually be delivered to the mill from a production mode stope or development muck; and
- to provide a sample of sufficient size for the determination of milling characteristics.

12.5.1. General Procedure for Bulk Sampling

Step 1 – choose which rounds to send for bulk sampling

Blasted rounds from along-strike drifting expected to return greater than 0.05 oz/t were targeted for bulk sampling. In most cases, the blasted rock from a single round could be stored in a re-muck area as a separate and complete sample without contamination from other rounds until the results of related face chip samples were available. The turnaround time for priority chip samples was normally 1 to 2 days, although in a few cases it took a week to get chip sample results back. There were sufficient storage spaces underground to permit this on-hold strategy. The most difficult decision was often whether the rock should go as “high grade” (0.10 oz/t or greater) or as “low grade” (0.05 to 0.10 oz/t).

In the author’s opinion, the ore-waste decision issues encountered during this programme mirror the decisions that are required during normal day-to-day mine operations. No special preparation of the re-muck areas between samples was made, again, in order to reflect how the muck would be handled in a normal operating mode. No doubt there may have been some dilution added and/or losses during the re-muck cycle. The author considers any such additions or losses normal for a mining operation where multiple handling of broken rock is required.

The rounds selected for bulk sampling are shown in the following plans of the 2250 Level (Figure 15 to Figure 20).

Step 2 – skip sample to surface

Before any sample was hoisted to surface, the grizzly and loading pocket on 2250 Level were cleaned out with an air lance to avoid contamination from the previous sample or waste. This cleaning procedure is in accord with normal mine operating procedures.

Each sample was hoisted in an uninterrupted batch, consisting of between 4 and 15 skip loads. Based on the skip volume of 90 cubic feet, a skip factor of 5.2 tons has been used which assumes a swell factor 1.5. This factor has not been check calibrated. The original plan was to calibrate the skip factor by comparing the truck weights for the rock as it was moved to a custom mill; however, this has not occurred at the time of writing. Nevertheless, in the opinion of the author, a skip factor of 5.2 tons is considered reasonable and even a 10% error one way or the other would not materially affect any of the conclusions in this report.

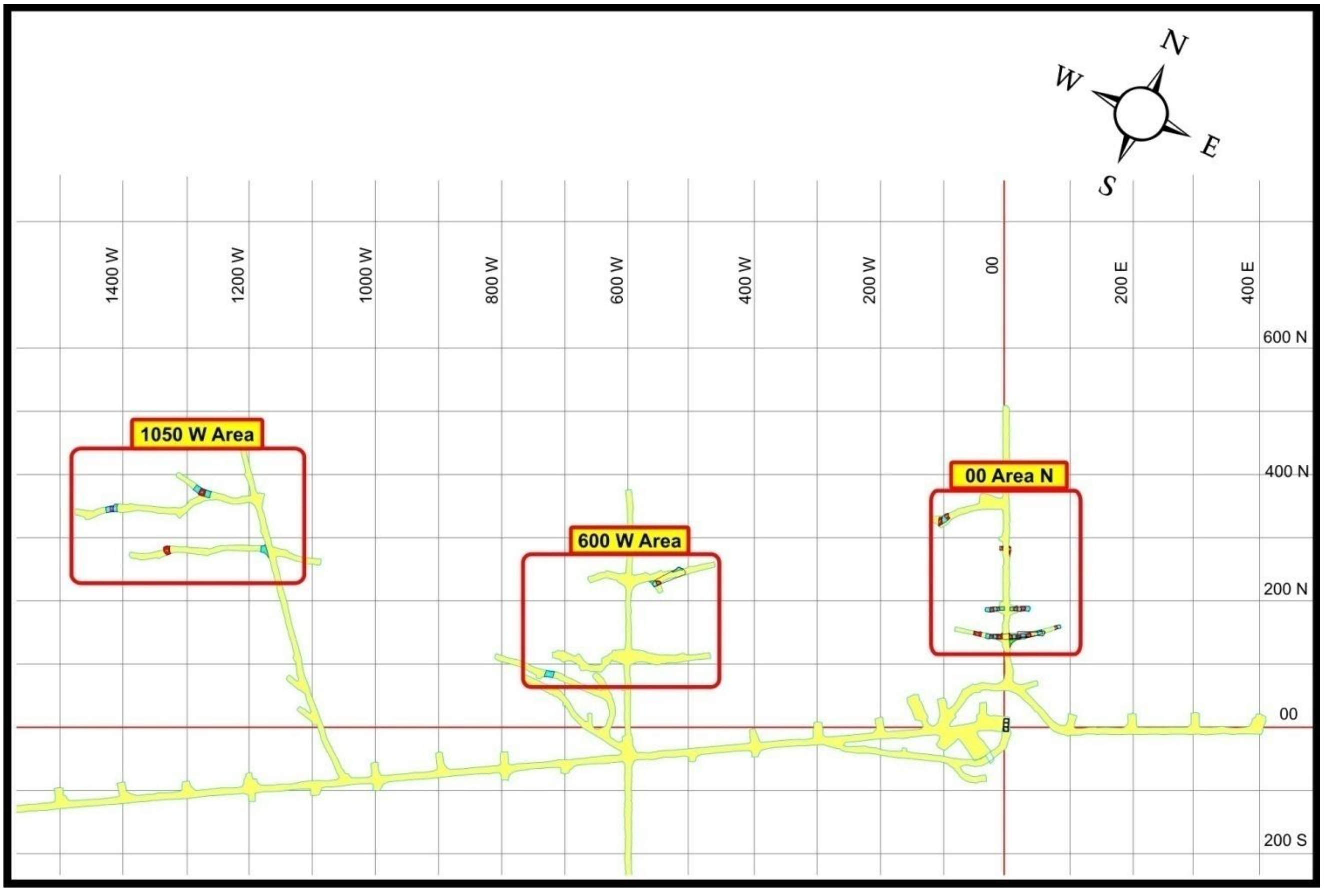


Figure 15 - Overview Location Map of Bulk Sampling and Test Stopes

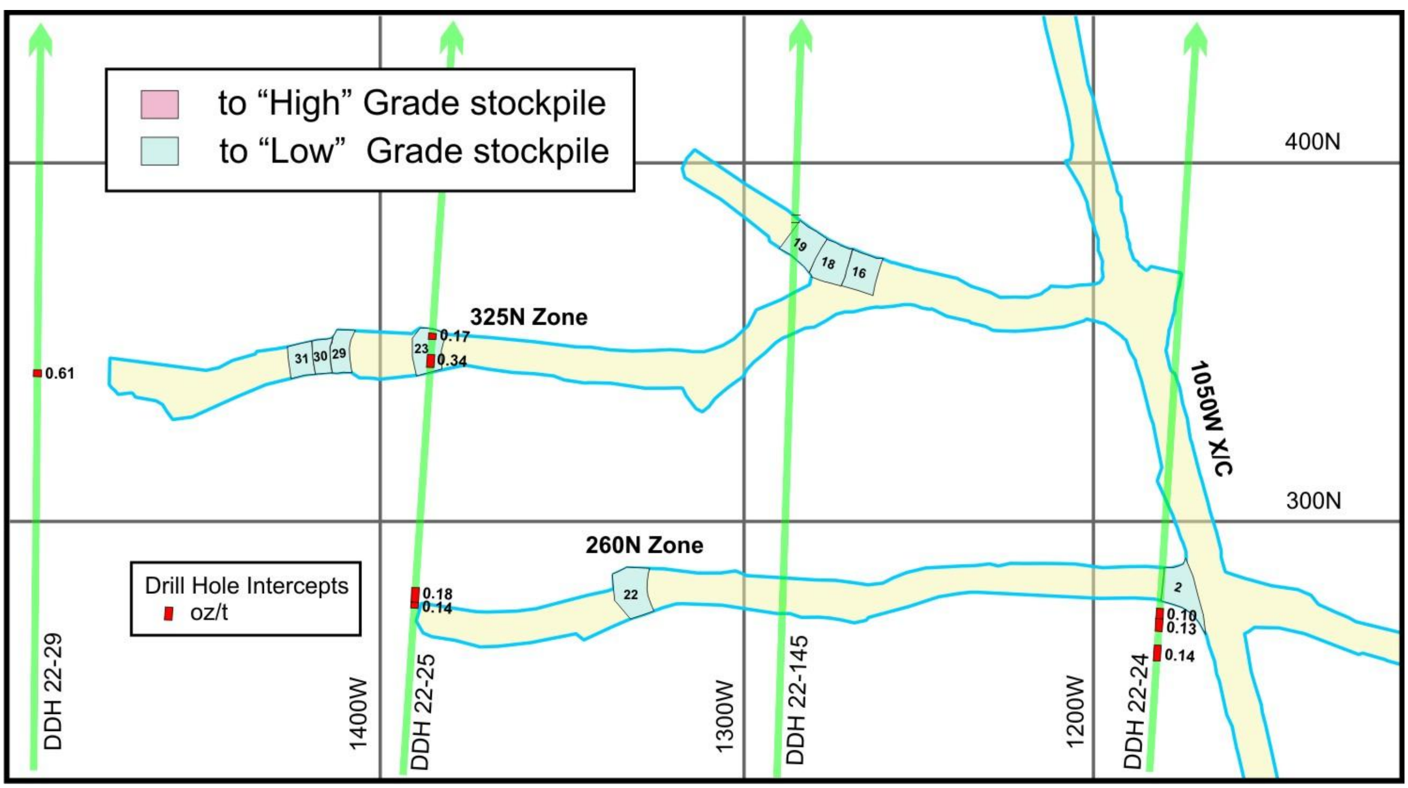
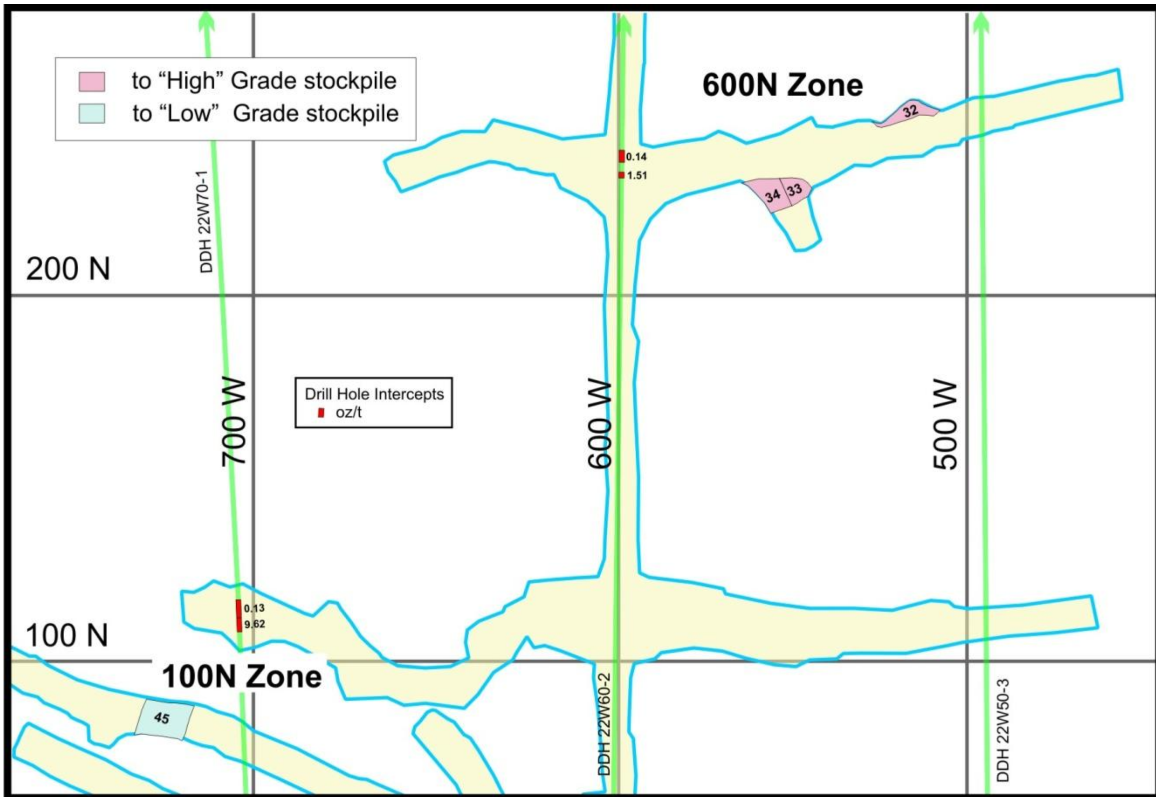


Figure 16 - 1050W X/C Area Bulk Sample Mining Rounds Detail Location

Figure 17 - 600W X/C Area Bulk Sample Mining Rounds Detail Location



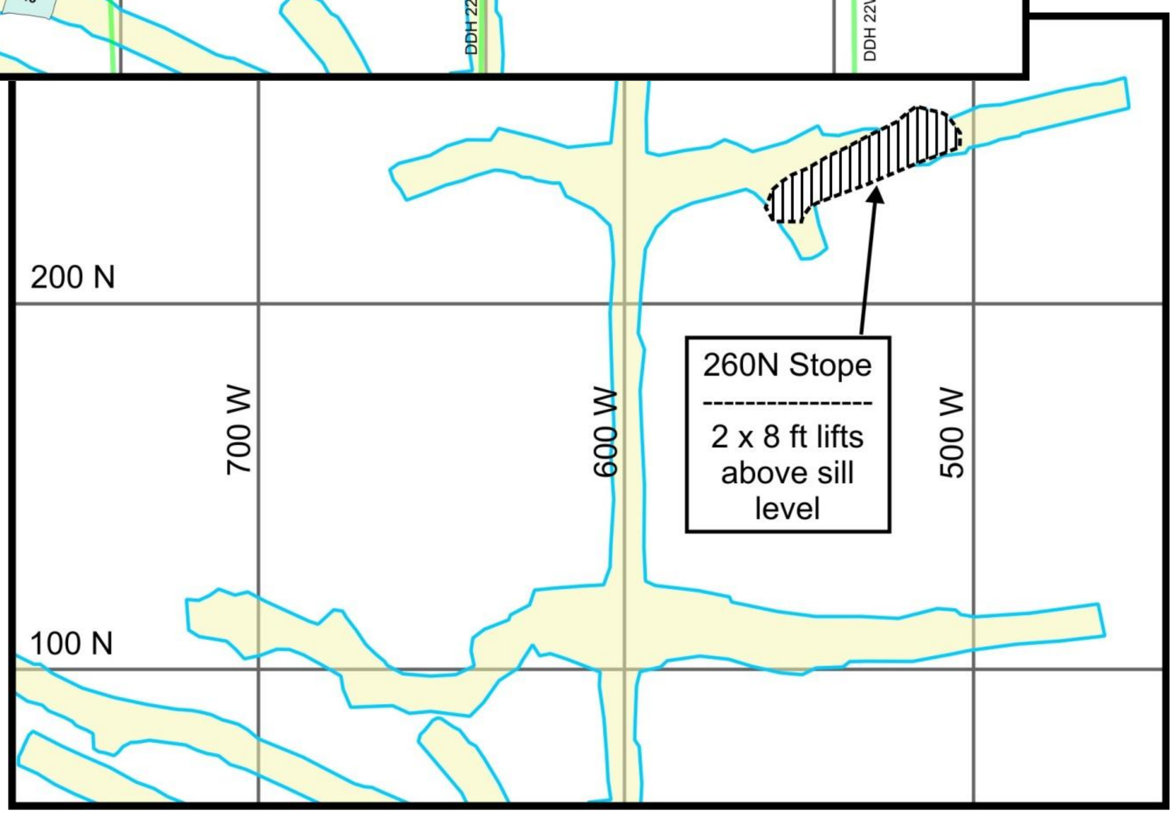
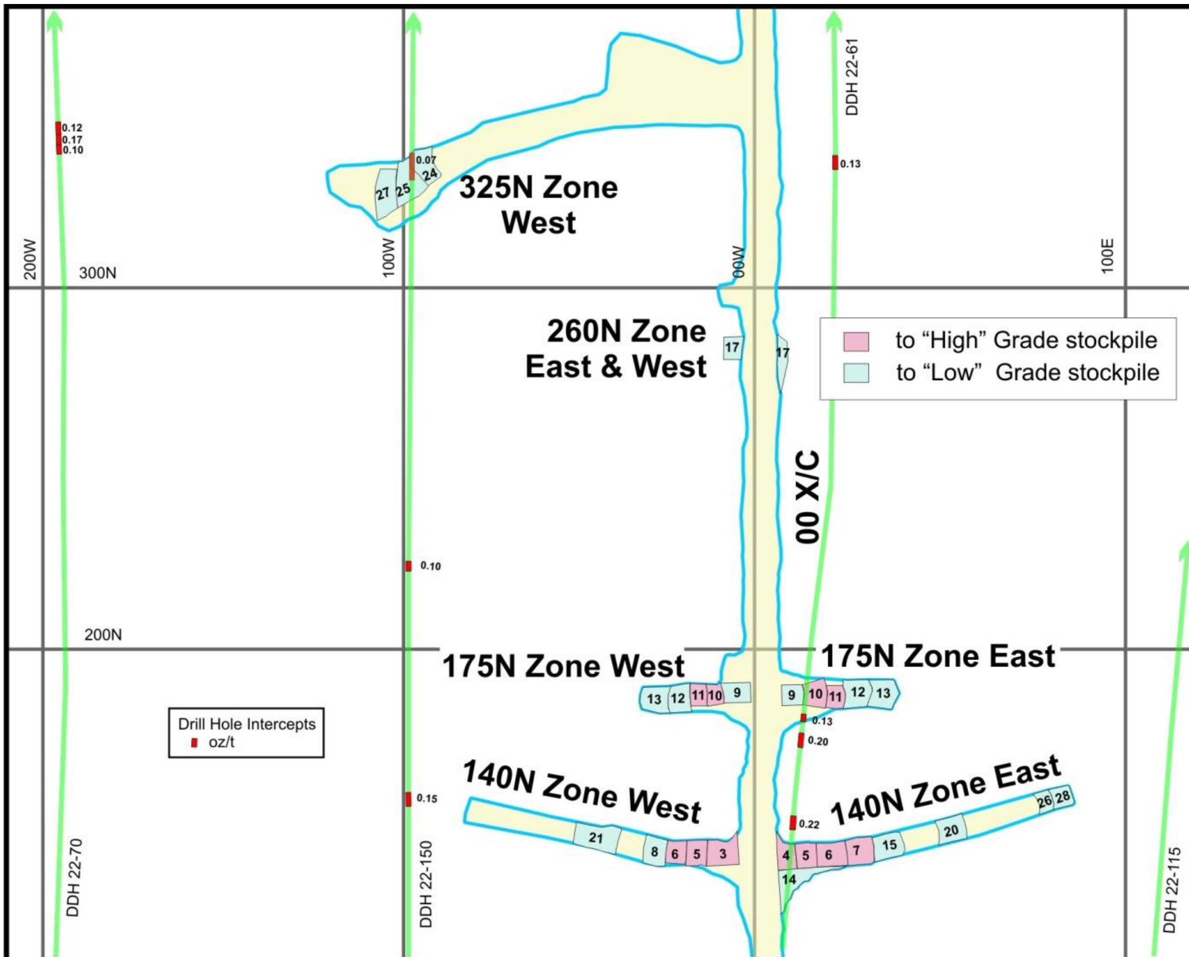


Figure 18 - 600W X/C Area Test Stope 260N Location

Figure 19 - 00 X/C Area Bulk Sample Mining Rounds Detail Location



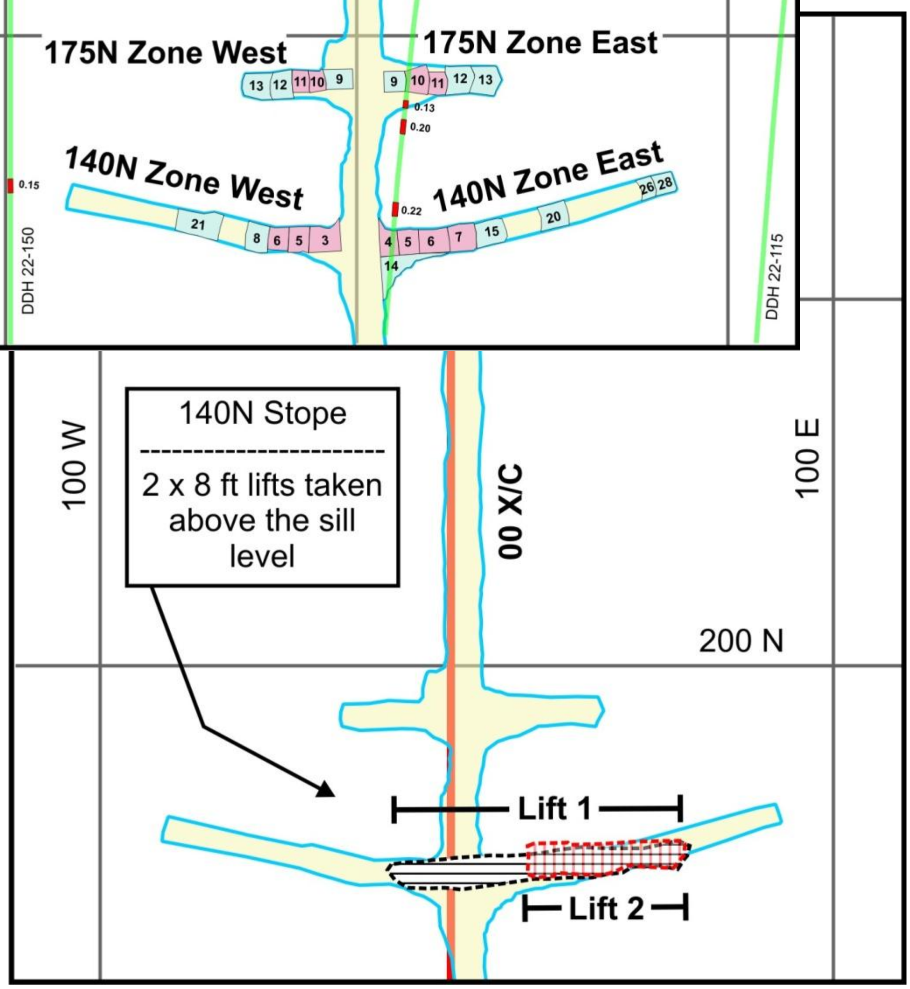


Figure 20 - 00 X/C Area Test Stope 140N Location

Step 3 – crush the rock to be sampled

The dump chute in the headframe was modified so that it would feed directly into a mobile two-stage gravel pit type crusher set up directly under the chute. The rock for sampling was crushed so it would pass through a 5/8 inch square mesh screen and into a holding hopper. The crusher and related conveyors were owned and operated by Teck Northern Roads Ltd., of Kirkland Lake.

Step 4 – sample the crushed rock

When the holding hopper was filled, a conveyor belt brought the material to a continuous linear sampler custom built by Gorf Contracting Ltd. of Timmins, Ontario (Model SBL-400-27). The sampler box opening measured 5 inches by 18 inches oriented with the long side parallel to the feeding conveyor belt. The sample stream was conveyed into a 45 gallon drum. The reject stream was conveyed into a pile directly on the ground. After the entire sample was run through the sampler, the reject pile was placed back into the holding hopper using a front-end loader and run through the sampler once again.

The collected sub-sample varied between 0.2% and 1.7% of the original batch sample by weight. The average collected sub-sample was 0.52% of the original sample weight (see Table 9, page 41).

Step 5 – process the collected sub-sample to determine average grade

Each collected sub-samples fit into a single 45 gallon drum which was sealed and placed on a pallet. The samples were shipped to PolyMet Labs in Cobalt, Ontario by Manitoulin Transport.

Each sub-sample was dry ground in a ball mill by PolyMet. The following general procedure is taken directly from the PolyMet Web Site (www.polymetinc.com).

> *"BULK SAMPLING CIRCUIT*
>
> *The production lot has been completely crushed and ground to the appropriate mesh size (minus screened off scales) and has become more homogeneous throughout the process. The material and all of the dust collected from the crushing and grinding circuit is loaded into a holding bin located above the bulk sampling circuit. The material is dropped at a constant rate into a cutter box that splits the production lot into four equal portions. The four portions are then dropped at a constant rate into a primary revolving drum that has equally spaced discharge slots cut into its periphery. The primary drum separates a uniform 10% portion of the material and deposits the other 90% into a holding bin.*
>
> *The 10% portion of the production lot is dropped into a secondary revolving drum that separates a further uniform 10% portion of this material and deposits the other 90% into the holding bin. The uniform material is finally separated into four samples that are each equivalent to 1/4% of the total production lot. The bulk sample circuit cuts the material 3,640,000 times per hour on a*

blended and fixed flow of approximately 5 pounds per minute. The circuit is extremely efficient and accurate in producing four similar homogeneous samples.

The four samples are provided to the laboratory for preparation and determination of content.

The dust produced by the bulk sample circuit is removed by a collector system and is reported as part of the production lot."

The procedure is further illustrated by the following diagram (Figure 21), modified from the PolyMet Web Site to reflect the actual circuit used for the Armistice samples:

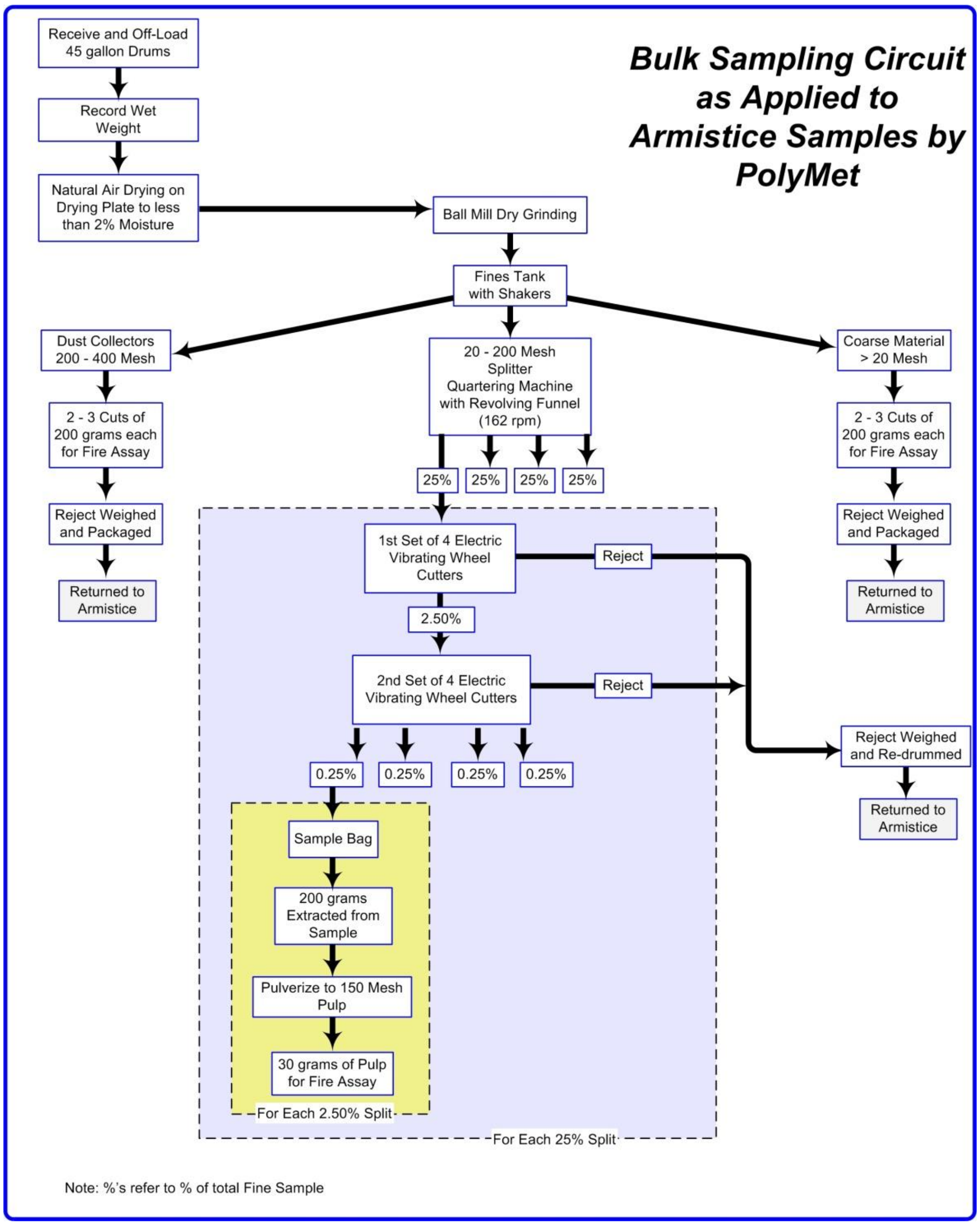


Figure 21 - PolyMet Bulk Sampling Circuit Diagram

There are three fractions of material produced by the sampling process:

- Oversize - +20 mesh
- Fines - -20 mesh to +200 mesh
- Dust - -200 mesh

There are four splits taken from the “fines”, which is the majority of the material produced for assay, representing about 87% of the total sub-sample by weight. Each of the four “fines” splits is assayed four times for a total of 16 assays.

There are either three or two splits taken from each of the “oversize” and “dust” fractions representing, on average, 6.5% and 6.7%, respectively, of the total sub-sample by weight. Each split is assayed twice for a total of either six or four assays for each of the “oversize” and “dust” fractions.

In total, there are either 28 or 22 assays taken from each sub-sample submitted to PolyMet. The representative grade for the sub-sample is then calculated by first averaging the assays for each fraction and then by weighting the averages by the weight of each of size fraction. The results are summarized in Table 9, below.

The crusher had a major mechanical breakdown before all the material from the 260N stope could be crushed and run through the Gorf sampler. All the uncrushed rock from the 260N stope was stored separately on surface. Two samples from the 260N stope were sampled according to protocol (samples ARM-46 and ARM-47) using the Gorf sampler. Three additional 45 gallon drums filled with samples of uncrushed rock were taken from different parts of the uncrushed surface pile. These samples were processed by PolyMet as samples ARM-48, ARM-49 and ARM-50. As shown in Table 9, one-third of the estimated tonnage of the uncrushed rock from the 260N stope surface stockpile was assigned for weighting purposes to each of these samples. The average grades from the PolyMet sampling procedure for all five samples appears consistent within a relatively narrow range. Therefore the author considers that the five samples from the 260N stope, taken together, fairly represent the bulk recovered grade from this stope.

The complete summary of results from the bulk sampling is presented below in Table 9.

Table 10 provides a summary of the bulk sampling results for the material stored in each of the “high grade” and “low grade” surface stockpiles. This table also provides a comparison of the bulk sample grade with the related estimates from chip sampling directly associated with the mined bulk sample and with the grade information from the nearest diamond drill hole.

Table 11 provides a summary of the bulk sampling results for the material from each of the two test stopes. Again related chip sampling and diamond drill hole information is presented for comparison.

Summary of Bulk Samples and PolyMet Processing Results of Extracted Sub-Sample															
Sample	Area	Heading and Source Type		PolyMet Processed Sub-Sample Weights				Average PolyMet Sub-Sample Grade by Fraction				*Standard Deviation Fines Fraction*	Total Sample Weight		Sub-Sample Weight as % of Total Skipped Sample Weight
				Fines (lb)	Oversize (lb)	Dust (lb)	Total (lb)	Fines (oz/t)	Oversize (oz/t)	Dust (oz/t)	Weighted Average All Fractions (oz/t)		High Grade (tons)	Low Grade (tons)	
ARM-01		Waste Blank		596	46	26	668	0.004	0.002	0.049	0.006	*0.004*			
ARM-02	1050W	260 E&W	Slash	417	42	40	499	0.036	0.026	0.035	0.035	*0.005*		26.0	0.96%
ARM-03	00 XC	140 W	Stope Sill	599	38	44	681	0.157	0.138	0.068	0.150	*0.012*	31.2		1.09%
ARM-04	00 XC	140 E	Stope Sill	603	42	64	709	0.115	0.087	0.073	0.110	*0.014*	67.6		0.52%
ARM-05	00 XC	140 E&W	Stope Sill	599	33	58	690	0.234	0.125	0.124	0.220	*0.040*	72.8		0.47%
ARM-06	00 XC	140 E&W	Stope Sill	510	46	22	578	0.131	0.132	0.110	0.130	*0.017*	72.8		0.40%
ARM-07	00 XC	140 E	Stope Sill	494	28	51	573	0.164	0.105	0.110	0.157	*0.009*	36.4		0.79%
ARM-08	00 XC	140 W	Drift	437	34	34	505	0.079	0.029	0.058	0.074	*0.012*		26.0	0.97%
ARM-09	00 XC	175 E & W	Drift	523	42	15	580	0.051	0.045	0.039	0.051	*0.015*		78.0	0.37%
ARM-10	00 XC	175 E & W	Drift	628	37	56	721	0.181	0.126	0.082	0.170	*0.020*	72.8		0.50%
ARM-11	00 XC	175 E & W	Drift	557	23	24	604	0.146	0.080	0.102	0.142	*0.024*	52.0		0.58%
ARM-12	00 XC	175 E & W	Drift	415	60	30	505	0.036	0.021	0.047	0.035	*0.007*		41.6	0.61%
ARM-13	00 XC	175 E & W	Drift	510	51	35	596	0.026	0.040	0.045	0.028	*0.007*		78.0	0.38%
ARM-14	00 XC	140 E	Slash	399	29	31	459	0.048	0.034	0.036	0.046	*0.011*		41.6	0.55%
ARM-15	00 XC	140 E	Drift	360	31	33	424	0.081	0.066	0.067	0.079	*0.008*		41.6	0.51%
ARM-16	1050 XC	325 W	Drift	429	19	29	477	0.018	0.018	0.025	0.018	*0.003*		52.0	0.46%
ARM-17	00 XC	260 E & W	Drift & Slash	369	20	36	425	0.033	0.014	0.033	0.032	*0.008*		78.0	0.27%
ARM-18	1050 XC	325 W	Drift	461	25	19	505	0.032	0.024	0.042	0.032	*0.004*		52.0	0.49%
ARM-19	1050 XC	325 W	Drift	477	47	50	574	0.081	0.043	0.025	0.073	*0.107*		36.4	0.79%
ARM-20	00 XC	140 E	Drift	366	39	37	442	0.176	0.160	0.072	0.166	*0.020*		41.6	0.53%
ARM-21	00 XC	140 W	Drift	370	68	38	476	0.051	0.042	0.064	0.050	*0.006*		26.0	0.92%
ARM-22	1050 XC	260 W	Drift	424	25	23	472	0.009	0.003	0.035	0.010	*0.005*		26.0	0.91%
ARM-23	1050 XC	325 W	Drift	389	25	13	427	0.033	0.014	0.040	0.032	*0.005*		26.0	0.82%
ARM-24	00 XC	325 W	Drift	352	29	46	427	0.090	0.054	0.051	0.083	*0.009*		52.0	0.41%
ARM-25	00 XC	325 W	Drift	149	22	16	187	0.111	0.011	0.058	0.095	*0.003*		52.0	0.18%
ARM-26	00 XC	140 E	Drift	560	27	40	627	0.024	0.067	0.022	0.026	*0.002*		41.6	0.75%
ARM-27	00 XC	325 W	Drift	386	35	41	462	0.037	0.024	0.034	0.035	*0.004*		41.6	0.56%
ARM-28	00 XC	140 E	Drift	301	40	20	361	0.043	0.015	0.044	0.040	*0.005*		57.2	0.32%
ARM-29	1050 XC	325 W	Drift	419	27	21	467	0.070	0.047	0.054	0.068	*0.005*		62.4	0.37%
ARM-30	1050 XC	325 W	Drift	606	40	48	694	0.047	0.026	0.032	0.045	*0.003*		67.6	0.51%
ARM-31	1050 XC	325 W	Slash	622	40	50	712	0.026	0.022	0.016	0.025	*0.004*		20.8	1.71%
ARM-32	1050 XC	260 E	Stope Sill Slash	376	32	34	442	0.029	0.013	0.017	0.027	*0.005*	36.4		0.61%
ARM-33	1050 XC	260 E	Stope Sill Slash	271	40	27	338	0.133	0.082	0.041	0.119	*0.016*	41.6		0.41%
ARM-34	1050 XC	260 E	Stope Sill Slash	353	35	24	412	0.158	0.163	0.082	0.154	*0.008*	46.8		0.44%
ARM-35	00 X/C	140N Stope	Stope	502	36	40	578	0.114	0.079	0.070	0.109	*0.008*		52.0	0.56%
ARM-36	00 X/C	140N Stope	Stope	549	24	29	602	0.149	0.099	0.070	0.143	*0.021*		62.4	0.48%
ARM-37	00 X/C	140N Stope	Stope	602	19	62	683	0.120	0.078	0.060	0.113	*0.012*	57.2		0.60%
ARM-38	00 X/C	140N Stope	Stope	605	29	46	680	0.130	0.102	0.058	0.124	*0.008*	57.2		0.59%
ARM-39	00 X/C	140N Stope	Stope	476	33	34	543	0.148	0.086	0.073	0.139	*0.012*	57.2		0.47%
ARM-40	00 X/C	140N Stope	Stope	356	34	31	421	0.152	0.170	0.050	0.146	*0.006*	57.2		0.37%
ARM-41	00 X/C	140N Stope	Stope	482	24	41	547	0.145	0.104	0.055	0.136	*0.012*	67.6		0.40%
ARM-42	00 X/C	140N Stope	Stope	573	48	43	664	0.106	0.116	0.059	0.103	*0.008*	57.2		0.58%
ARM-43	00 X/C	140N Stope	Stope	433	29	21	483	0.092	0.065	0.054	0.089	*0.006*	67.6		0.36%
ARM-44	00 X/C	140N Stope	Stope	408	18	32	458	0.116	0.098	0.050	0.111	*0.010*	31.2		0.73%
ARM-45	600N X/C	Incline	Drift	619	26	31	676	0.061	0.026	0.042	0.059	*0.005*		67.6	0.50%
ARM-46	600N X/C	260N Stope	Stope	503	20	27	550	0.061	0.041	0.047	0.060	*0.003*		67.6	0.41%
ARM-47	600N X/C	260N Stope	Stope	506	25	27	558	0.074	0.051	0.041	0.072	*0.007*		62.4	0.45%
ARM-48	600N X/C	260N Stope	Stope - Not Crushed	514	46	57	617	0.063	0.051	0.035	0.059	*0.006*		171.6	0.18%
ARM-49	600N X/C	260N Stope	Stope - Not Crushed	548	41	75	664	0.086	0.091	0.067	0.084	*0.004*		171.6	0.19%
ARM-50	600N X/C	260N Stope	Stope - Not Crushed	489	52	61	602	0.073	0.049	0.038	0.067	*0.007*		171.6	0.18%
Subtotal - High Grade Stockpile (incl. 140 Stope)				**8,925**	**588**	**714**	**10,227**	**0.139**	**0.104**	**0.073**	**0.133**	***0.044***	**982.8**		**0.52%**
Subtotal - Low Grade Stockpile (incl. 260 Stope)				**13,971**	**1,087**	**1,092**	**16,150**	**0.061**	**0.044**	**0.044**	**0.058**	***0.044***		**1,892.8**	**0.43%**
Subtotal - 140N Stope (ARM-03 to 07 & ARM-37 to 44)				**6,740**	**421**	**549**	**7,710**	**0.140**	**0.108**	**0.074**	**0.134**	***0.037***	**733.2**		**0.53%**
Subtotal - 260N Stope (ARM-32 to 34 & ARM-46 to 50)				**3,560**	**291**	**332**	**4,183**	**0.081**	**0.065**	**0.046**	**0.077**	***0.010***	**124.8**	**644.8**	**0.27%**
Total (excl. Blank)				**22,896**	**1,675**	**1,806**	**26,377**	**0.091**	**0.068**	**0.055**	**0.087**	***0.057***	**982.8**	**1,892.8**	**0.46%**

Table 9 - Summary of Bulk Samples and Processing Results of Extracted Sub-Samples

Comparison of Bulk Sample Grades with Grade Estimates Based on Chip Sampling and Diamond Drilling for Development Drift Rounds and Stope Sill Drifts

	Bulk Sample Reference	Weighted Average Grade from PolyMet Bulk Analysis (oz/t)	Related Chip Sampling					Nearest Drill Hole				Target Mineralized Zone
			Number of Chip Samples (#)	Aggregate Lineal Feet of Chip Sampling (ft)	Chip Sample Assay Range		Weighted Average Assay (by sample length) (oz/t)	DDH Number	Average Grade (oz/t)	True Width Interval (ft)	Distance from DDH to Bulk Sample (ft)	
					Lowest Assay Value (oz/t)	Highest Assay Value (oz/t)						
High Grade Stockpile	ARM-03	0.150	5	32	0.034	0.600	0.213	22-61	0.108	7.7	15 ft west of DDH	140N W of 00 X/C - Stope Sill Drift
	ARM-04	0.110	3	21	0.007	0.486	0.256	22-61	0.108	7.7	centred on DDH	140N E of 00 X/C - Stope Sill Drift
	ARM-05	0.220	7	45	0.116	0.606	0.361	22-61	0.108	7.7	2 ft E & 24 ft W of DDH	140N E & W of 00 X/C - Stope Sill Drift
	ARM-06	0.130	6	33	0.021	0.606	0.212	22-61	0.108	7.7	8 ft E & 30 ft W of DDH	140N E & W of 00 X/C - Stope Sill Drift
	ARM-07	0.157	2	13	0.035	0.086	0.059	22-61	0.108	7.7	16 ft east of DDH	140N E of 00 X/C - Stope Sill Drift
	ARM-10	0.170	4	26	0.001	0.040	0.015	22-61	0.052	8.0	adj. & 23 ft W of DDH	175N E & W of 00 X/C
	ARM-11	0.142	4	27	0.001	0.040	0.014	22-61	0.052	8.0	6 ft E & 29 ft W of DDH	175N E & W of 00 X/C
	ARM-32	0.027	2	16	0.020	0.366	0.193	22W50-3	--	--	15 ft W of DDH	260N E of 600N X/C
	ARM-33	0.119	2	13	0.039	0.185	0.138	22W60-2	0.349	7.5	44 ft east of DDH	260N E of 600N X/C
	ARM-34	0.154	2	18	0.185	0.480	0.341	22W60-2	0.349	7.5	38 ft east of DDH	260N E of 600N X/C
	Weighted Average	0.144	37	243			0.196					
Low Grade Stockpile	ARM-02	0.035	7	63	0.057	0.304	0.120	22-24	0.090	11.0	adjacent to DDH	260N W of 1050X/C
	ARM-08	0.074	2	11	0.021	0.021	0.021	22-61	0.108	7.7	36 ft west of DDH	140N W of 00 X/C
	ARM-09	0.051	7	40	0.002	0.338	0.057	22-61	0.052	8.0	11 ft east of DDH	175N E&W of 00 X/C
	ARM-12	0.035	4	26	0.002	0.007	0.005	22-61	0.052	8.0	11 ft E & 33 ft W of DDH	175N E&W of 00 X/C
	ARM-13	0.028	2	16	0.006	0.006	0.006	22-61	0.052	8.0	18 ft E and 38 ft W of DDH	175N E&W of 00 X/C
	ARM-14	0.046	3	11	0.001	0.157	0.077	22-61	0.108	7.7	centred on DDH	140N E of 00 X/C
	ARM-15	0.079	2	15	0.035	0.349	0.202	22-61	0.108	7.7	35 ft east of DDH	140N E of 00 X/C
	ARM-16	0.018	2	20	0.005	0.029	0.017	22-145	--	--	15 ft east of DDH	hanging wall of 325N W of 1050X/C
	ARM-17	0.032	2	13	0.017	0.053	0.038	22-61	--	--	20 ft west of DDH	260N at 00 X/C
	ARM-18	0.032	2	20	0.005	0.029	0.017	22-145	--	--	7 ft east of DDH	hanging wall of 325N W of 1050X/C
	ARM-19	0.073	2	21	0.029	0.039	0.034	22-145	--	--	centred on DDH	hanging wall of 325N W of 1050X/C
	ARM-20	0.166	5	15	0.000	0.321	0.143	22-61	0.108	7.7	42 ft east of DDH	140N E of 00 X/C
	ARM-21	0.050	2	14	0.009	0.034	0.023	22-61	0.108	7.7	49 ft west of DDH	140N W of 00 X/C
	ARM-22	0.010	2	25	0.013	0.031	0.024	22-25	0.163	5.9	25 ft west of DDH	260N W of 1050X/C
	ARM-23	0.032	4	37	0.010	0.128	0.052	22-25	0.164	9.6	centered on DDH	325N W of 1050X/C
	ARM-24	0.083	2	20	0.017	0.146	0.086	22-150	0.065	7.0	adjacent to DDH	325N W of 00 X/C
	ARM-25	0.095	2	21	0.055	0.146	0.102	22-150	0.065	7.0	centred on DDH	325N W of 00 X/C
	ARM-26	0.026	2	14	0.022	0.128	0.073	22-115	--	--	25 ft west of DDH	140N E of 00 X/C
	ARM-27	0.035	2	23	0.049	0.055	0.052	22-150	0.065	7.0	4 ft west of DDH	325N W of 00 X/C
	ARM-28	0.040	2	14	0.003	0.128	0.063	22-115	--	--	31 ft west of DDH	140N E of 00 X/C
	ARM-29	0.068	2	15	0.000	0.080	0.021	22-25	0.164	9.6	21 ft east of DDH	325N W of 1050X/C
	ARM-30	0.045	2	14	0.014	0.080	0.033	22-25	0.164	9.6	27 ft east of DDH	325N W of 1050X/C
	ARM-31	0.025	1	10	0.014	0.014	0.014	22-25	0.164	9.6	33 ft east of DDH	325N W of 1050X/C
	ARM-45	0.059	2	18	0.022	0.109	0.066	22W70-1	--	--	15 ft west of DDH	100N W of 600N X/C
	Weighted Average	0.059	65	490			0.061					

Table 10 - Drifting: Comparison of Bulk Sample Grades with Chip Sampling & Drilling Results

Comparison of Bulk Sample Grades with Grade Estimates
Based on Chip Sampling and Diamond Drilling for Bulk Stope Mining

	Bulk Sample Reference	Weighted Average Grade from PolyMet Bulk Analysis (oz/t)	Chip Samples: Aggregate Lineal Feet of Chip Sampling (ft)	Chip Samples: Weighted Average Assay (by sample length) (oz/t)	Muck Samples: # of Samples	Muck Samples: Average Assay (oz/t)	Nearest Drill Holes: DDH Number	Nearest Drill Holes: Average Grade (oz/t)	Nearest Drill Holes: True Width Interval (ft)	Comments
140N Stope	ARM-03	0.150	32	0.213			22-61	0.108	7.7	Sill Drift
	ARM-04	0.110	21	0.256						Sill Drift
	ARM-05	0.220	45	0.361						Sill Drift
	ARM-06	0.130	33	0.212						Sill Drift
	ARM-07	0.157	13	0.059						Sill Drift
	ARM-37	0.113								Lifts 1 & 2 Bulk
	ARM-38	0.124								Lifts 1 & 2 Bulk
	ARM-39	0.139								Lifts 1 & 2 Bulk
	ARM-40	0.146	**145**	**0.153**						Lifts 1 & 2 Bulk
	ARM-41	0.136								Lifts 1 & 2 Bulk
	ARM-42	0.103								Lifts 1 & 2 Bulk
	ARM-43	0.089								Lifts 1 & 2 Bulk
	ARM-44	0.111								Lifts 1 & 2 Bulk
	Weighted Average	**0.133**	**288**	**0.202**						
260N Stope	ARM-32	0.027	16	0.193			22W50-3	--	--	Sill Drift Slash
	ARM-33	0.119	13	0.138			22W60-2	0.349	7.5	Sill Drift Slash
	ARM-34	0.154	18	0.341						Sill Drift Slash
	ARM-46	0.060								Lifts 1 & 2 Bulk
	ARM-47	0.072								Lifts 1 & 2 Bulk
	ARM-48	0.059	157	0.113						Lifts 1 & 2 Bulk Uncrushed from Surface Stockpile
	ARM-49	0.084			36	0.047				Lifts 1 & 2 Bulk Uncrushed from Surface Stockpile
	ARM-50	0.067								Lifts 1 & 2 Bulk Uncrushed from Surface Stockpile
	Weighted Average	**0.075**	**204**	**0.141**	**36**	**0.047**				

Table 11 - Stoping: Comparison of Bulk Sample Grades with Chip Sampling & Drilling Results

13. Sample Preparation, Analysis and Security

Core and rock chip samples are bagged and shipped together with a sample control tag to one of four assay labs. In the case of Swastika Laboratories Ltd. in nearby Swastika, ON; Techni-lab in Ste-Germaine Boulé, QC; and Kirkland Lake Gold Inc.'s lab at the Macassa Mine in Kirkland Lake, ON, the samples were directly delivered to the lab by Armistice personnel. In the case of PolyMet Labs in Cobalt, ON, samples were either delivered by bonded carrier, delivered directly to the lab by Armistice personnel or picked up at the McGarry site by PolyMet personnel. In all cases, deliveries to the labs were documented by accompanying bills of lading.

All rejects and pulps have been returned to Armistice and are stored at the Armistice locked facility in Virginiatown.

At the laboratory, samples are crushed pulverized and analysed for gold content using standard fire assay methods with AA finish. Routine checks are carried out to ensure accuracy. Re-checks are requested by Armistice in circumstances as warranted, and metallic separation and analysis may be requested when samples are known to contain erratically distributed native gold.

It is the author's opinion that sample preparation, security and analytical procedures meet or exceed industry accepted standards and are fully adequate to provide a reasonable basis for the evaluation of the mineral resources of the Property.

14. Data Verification

As noted above, all analytical work on core and rock samples from the McGarry Property is performed by well established independent firms who perform similar services for a large number of other mining industry clients.

14.1. Internal Lab Controls

Heavy reliance is placed on the internal verification procedures established by the labs themselves. These procedures include:

- assaying of a reference standard supplied by RockLabs Limited of Auckland, New Zealand. RockLabs is the world wide major supplier of reference material to mining assay labs and is used by each of the labs used by Armistice. Detailed discussion and specifications for the reference materials can be found on their Web site at www.rocklabs.com. A reference standard is assayed for each assay certificate which vary from 5 to 78 submitted assays, but averaging every 27 submitted assays.
- assaying of a blank standard at the same frequency as the reference standard as discussed above.
- check assaying (re-assaying of a second pulp cut) of about 7% of submitted samples including 90% of the samples returning over 0.10 oz/t on the first assay.

14.1.1. Comments on Reference Standard Evaluation

In order to evaluate the Process Performance by a lab using the RockLabs standards, RockLabs provides a recommended Microsoft Excel spreadsheet template. The following is extracted from RockLabs' instructions for using the template:

> *"RockLabs Template for Plotting Reference Material ("RM") Results*
>
> ***The chart is not fully useful until data from at least 30 analyses of the same RM have been entered – the more the better!***
>
> *Features of the charts produced by this template*
> - *The standard deviation (on which the control limits are based) is calculated in such a way that it is not severely inflated by trends or jumps in the data.*
> - *Any results that are grossly in error (eg the incorrect sample has been analysed) are excluded from these calculations so that the resultant statistics are realistic features representing a process under control.* [Anything under half the average or over 1.5 times the average.]
> - *Two charts are generated from the analytical data - a Process Performance chart and a Moving Range chart. Any point that is outside the control limits (shown in red in both charts) indicates that the process is out of control.*

Recommendations for using the template

- *Whenever a result for a RM is received it should be entered into the "Data" worksheet immediately and the Process Evaluation Charts inspected. If the data point falls outside the control limits of either chart, the laboratory should be alerted and the reason should be investigated.*
- *The average, as shown by the green line should then be compared with the assigned value as depicted at the left side of the chart. (The green whiskers on the vertical axis of the chart show the 95% confidence interval on this value, as specified on the RM certificate). It is not unusual for the chart average to show some small bias away from the assigned value (reflecting expected laboratory-to-laboratory variation), However, if the RM average does show more marked deviation, this indicates a significant bias exists, and the laboratory should be advised so that the cause can be investigated.*
- *A visual examination of the Process Performance chart will show whether there are any upward or downward trends. RockLabs Reference Materials are inherently stable. ROCKLABS sulphide matrix RMs are unlikely to increase in weight due to interaction with moisture in the air by more than 0.1% per year. Any trends will therefore be due to laboratory instability and not to the RM itself.*
- *Fourthly, a note should be taken of the Coefficient of Variation (CV). The Coefficient of Variation is the standard deviation expressed as a percentage of the mean. Many laboratories would obtain a CV of between 4 and 6 percent when analysing a quality RM, containing >1 ppm gold, over a period of time, involving different shifts, operators, AAS calibration standards etc. We would suggest that there is room for improvement if a CV of 7% or higher is being obtained. Obviously the lower the CV is, the better the methodology and laboratory performance. Some laboratories are able to achieve a CV that is lower than 4%. "*

Table 12 below summarize the statistics calculated using the RockLabs template in the format provided by RockLabs:

RockLab Standard Reference Samples - Statistics from Armistice Batches by Lab																
Reference Sample Number	**OxL51**				**OxJ64**				**OxL63**				**OxK48**			
		Control Limits				**Control Limits**				**Control Limits**				**Control Limits**		
	Value	**Lower**	**Upper**	**CV**	**Value**	**Lower**	**Upper**	**CV**	**Value**	**Lower**	**Upper**	**CV**	**Value**	**Lower**	**Upper**	**CV**
Average	0.169				2.357				5.963				3.574			
SD	0.001	0.166	0.173	0.7%	0.069	2.149	2.565	2.9%	0.088	5.699	6.226	1.5%	0.064	3.381	3.767	1.8%
SD(mssd)	0.001	0.167	0.171	0.3%	0.064	2.165	2.549	2.7%	0.065	5.767	6.158	1.1%	0.059	3.398	3.750	1.6%
SD(ave mr)	0.000	0.168	0.170	0.2%	0.061	2.175	2.540	2.6%	0.047	5.822	6.103	0.8%	0.053	3.414	3.734	1.5%
SD(med mr)	0.000	0.169	0.169	0.0%	0.036	2.248	2.466	1.5%	0.000	5.963	5.963	0.0%	0.036	3.466	3.682	1.0%
Average Moving Range	0.000				0.069				0.053				0.060			
Median Moving Range	0.000		0.000		0.035		0.199		0.000		0.000		0.034		0.133	
Lab	**PolyMet**				**Swastika**				**PolyMet**				**Swastika**			
Number of Samples	63				20				171				25			

SD	Traditional standard deviation includes trend and other effects - NOT RECOMMENDED for control limits
SD(mssd)	Mean square successive fifference method - adjusts for trends, etc.
SD(ave mr)	Based on average moving range - more robust
SD(med mr)	Based on median moving range - most robust estimate - used here
Median Moving Range	Moving Range (MR) chart based on this

Table 12 - RockLab Reference Sample Control Statistics by Lab

It is the opinion of the author that the process control of the assay labs used is best reflected in an analysis of the reference control statistics since these standards are prepared to a very high degree of uniformity and a standard method of evaluating the results is published.

The key parameter recommended by RockLabs for the evaluation of the reference control statistics is the standard deviation based on the median moving average range, “SD(mr)”. For the reference standard materials for which there is sufficient data for meaningful analysis (at least 30 analyses as recommended by RockLabs) the SD(mr) and as shown in Table 12, above, does not exceed 0.07 which is very low and indicates that the process controls at both PolyMet and Swastika are very good.

14.1.2. Blank Control Results

All the blanks inserted by the three labs used returned assay results of less than the detection limit (<0.001 oz/t gold).

14.1.3. Check Assay from Pulps

Figure 22 presents a graphical comparison of the assay results when an assay lab has taken two samples from the same pulp sample for separate gold content determination. The graphs show an envelop encompassing a range in which the two results are within 10% of the average of the two results. All the comparisons for Swastika Labs are within or very close to this range. For PolyMet Labs, there are 9 samples assaying more than 0.50 oz/t and outside this range, all of which have been subsequently submitted to Swastika Labs for independent check.

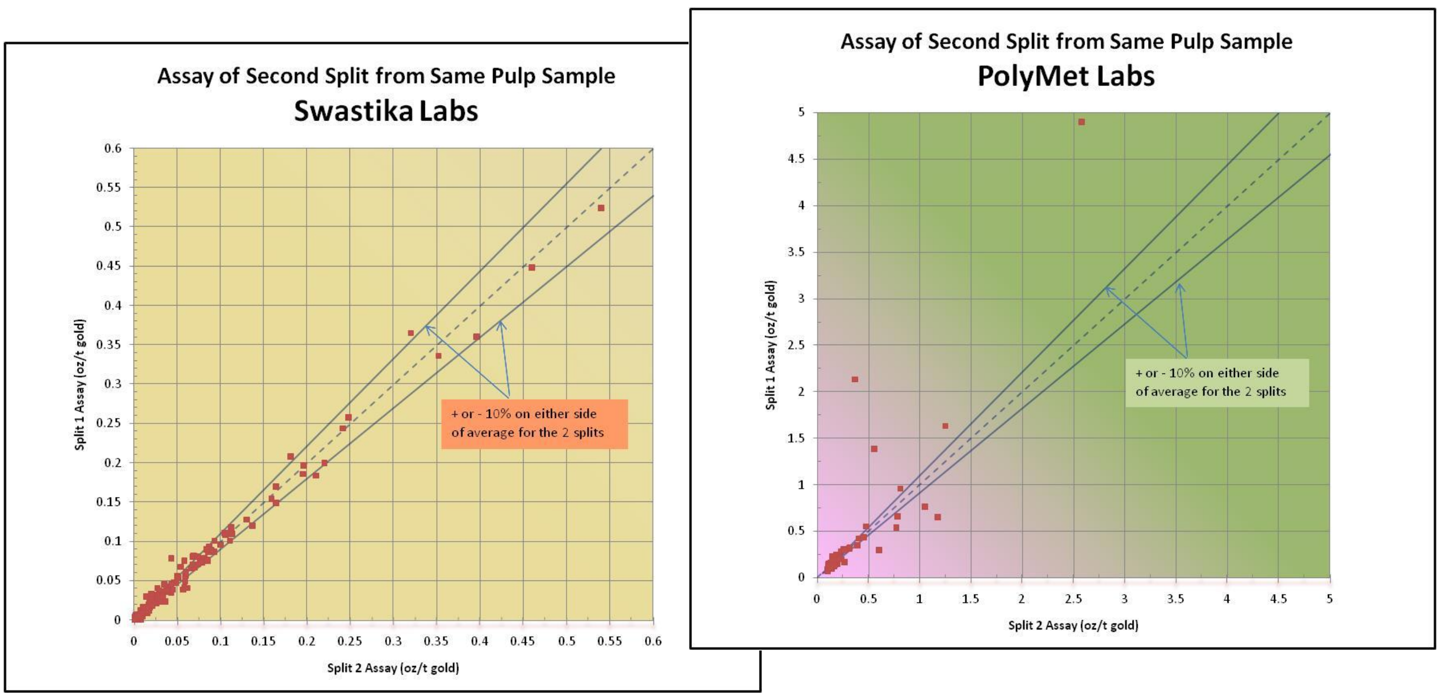


Figure 22 - Assay Comparison of Two Splits from Same Pulp for Swastika Labs and PolyMet Labs

14.2. Armistice Control Procedures

In addition to the internal controls at each lab, Armistice instituted its own verification procedures including:

- approximately one of every 20 submitted core samples was quarter cut. That is, the sawn half of the drill core to be submitted for assay was again sawn in half producing two "quarter cuts" from the same sample interval. The majority of the quarter cuts were submitted to each of two labs (Swastika and PolyMet).

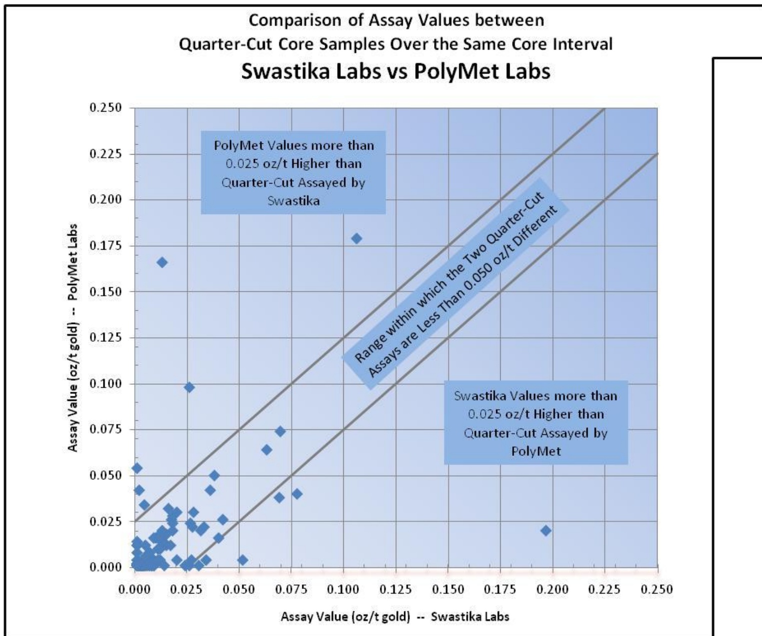
- Figure 23, below, presents the comparison of the results of the two assays for the two matching quarter cuts. An envelope is drawn on the graphs to show the range in which the results for the two quarter cuts is less than 0.05 oz/t different. All assays for samples in which the comparison falls outside this limit have had pulps submitted to opposing labs for re-assay.

- with each quarter cut core sample submitted, a control blank was also submitted. There were problems with the control of the Armistice submitted blanks until mid January 2009 and the author does not consider the results prior to this period valid. Blank submission protocol has been changed to ensure that submitted blanks are true blanks. The author is satisfied that the internal blank control procedures used by each lab is sufficient for this test prior to January 2009. Following the change in procedure for submitting blank material by Armistice, all Armistice control blanks have returned assays below the detection limit.

- in cases of discrepancies of more than 20% between two assays from the same submitted drill core sample, the pulp was re-submitted to a different lab for re-assay.

- in cases of large discrepancies between two assays from the same submitted drill core sample, the pulp was re-submitted for metallic assay procedure on the pulps plus reject material. There were 25 samples sent for metallic gold analysis. The results from the metallic analysis were in very close match with the average of the corresponding fire assays. For assays over 0.20 oz/t, all the metallic analyses returned equal or slightly higher results.

The author has verified the data to be relied upon and has discovered no unresolved discrepancies or issues of concern.



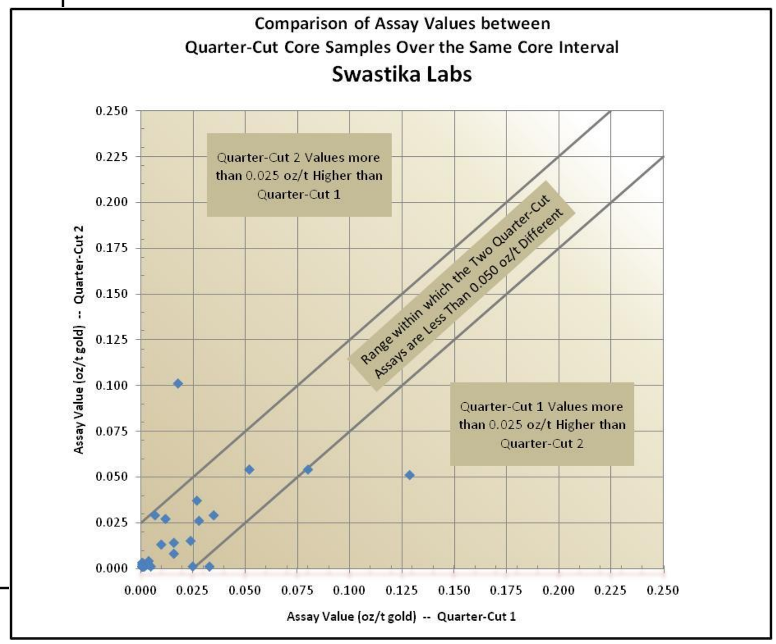


Figure 23 - Comparison of Assays from Two Quarter Cut Core Samples from Same Interval between Labs

15. Adjacent Properties

15.1. Kerr Addison

The Kerr Addison mine property adjoins the McGarry Property to the east, and has produced approximately 12 million ounces of gold over a 58 year operating life. Gold-bearing zones within its extensive mineralized system were mined from surface to a depth of 4,500 ft below surface, and over a strike length of about 3,200 ft. The past-producing Chesterville mine to the east also recorded notable gold production, and the property was absorbed into that of the Kerr Addison during the 1950's.

Information about the Kerr Addison and Chesterville Mines is taken from the following references: (Kishida & Kerrich, May 1987), (Buffam & Allen, 1948), (Gordon, Lovel, de Grijs, & Davie, 1979), (James, November 18, 1966), (Kerr Addison Mines Limited, January 1977), (Kerr Addison Mines Limited, 1962), and (Baker, 1957) (Smith, 1990). All the authors of these articles and papers acknowledge approval for release of information by Kerr Addison Mines Limited either explicitly or implicitly. There is a rich public record about the geology and production history of the Kerr Addison Mine since it was for a time the largest gold producer in Canada and subject to much industry and academic interest.

Figure 24, below, dramatically shows the physical relationship between the McGarry Project Site and the Kerr Addison #3 Shaft and Mill Complex as it existed until all the Kerr Addison surface facilities were completely raised in the late 1990's.

As has been described, the gold-bearing zones of the Kerr Addison mine occur within a belt of metasediments and mafic volcanics identified as the Kerr Formation. This belt is about 600 feet in thickness and 3,500 ft in length, strikes in an easterly direction and dips steeply north. Three major geologically distinct types of ore zones were mined at the Kerr Addison, including brecciated stockwork zones in green carbonate rock, tabular veined zones in green carbonate, and tabular pyritic zones in cherty mudstone which are termed "flow ore" zones.

Strong auriferous brecciated stockwork material was discovered at surface at the western part of the property area during the early 1900's, but the erratic distribution of gold therein discouraged production until 1937 under the aegis of Kerr Addison Mines Ltd. Bulk mining of such zones proved profitable, grading at about 0.25 oz gold per ton, soon augmented by production from higher grade material from tabular zones in green carbonate rock further to the west.

Accompanying "flow ore" zones at shallower depths proved to be of low grade and metallurgically complex and did not contribute significantly to early production. However, at depth the "flow ore" zones were found to steadily increase in grade and continuity, and by the late 1950's had become the major source of ore.

At depths of 3,000 ft, the No 21 "flow ore" zone was stoped over a length exceeding 1,000 ft and widths of up to 50 ft grading in the range of 0.50 to 1.00 oz gold per ton. This was an exceptional case, however, and the ultimate milling rate of about 4,200 tons per day ore was supplied from numerous mineralized zones, many restricted to a stoping length of 100 ft or so but exhibiting good vertical continuity.

“Flow ore”-type zones are present on the property in units of the Sheldon Formation, lying 200 ft south of the highly productive Kerr Formation. However, they proved to be of lower grade and were never mined.

At an approximate depth of 4,500 ft the mineralized system pinched out, and although extensive underground exploration was carried out to a depth of 6,000 ft on the property the auriferous Kerr Formation was not located and no mineralized zones of economic significance were encountered. Table 13, below, summarizes the gold production from the Kerr Addison and adjacent Chesterville Mines over their respective active mining periods.

Figure 24 - Arial View Looking South-easterly – Showing Kerr Addison and McGarry – circa 1990

15.2. Bear Lake Gold

The Barber Larder property of Bear Lake Gold Ltd., a merger of NFX Gold Ltd. and Maximus Ventures Ltd., ("Bear Lake Gold") adjoins the McGarry Property to the west, and it contains an auriferous system which is exposed at surface lying about 3,800 ft west of the McGarry shaft. A 410 ft vertical shaft was sunk on the property during 1939, and the system was explored by underground sampling and diamond drilling. No production was recorded at the time, but, from historical records in Armistice files, a total of 77,000 tons were mined and milled from a shallow open pit in 1987 from this location reporting at a recovered grade of 0.12 oz gold per ton (see Figure 2, page 6 and Figure 6, page 14). This production record cannot be relied upon for accuracy since it cannot be verified.

Over the past several years, Bear Lake Gold has conducted a surface diamond drilling programme on its properties immediately adjacent to the west of the McGarry Property. This drilling programme is still ongoing. Press releases in 2007 and 2008 by Bear Lake Gold indicate discovery of a zone of gold mineralization primarily of the "carbonate type" but also of the "flow type" located near the western end of Bear Lake (about 0.75 miles west of the McGarry Property – see Figure 2, page 6) at a depth to top of about 1300 feet and a strike extent of about 1950 feet. The following is taken from Bear Lake Gold's Web Site (www.bearlakegold.com) as posted on 22 January 2009:

> *"**Project Summary***
> *The discovery at Larder Lake, announced in November 2007, of new high grade mineralization confirms the presence of a multi-kilometer long geologic sequence with the potential to host a series of high grade gold deposits.*
>
> *On March 31, 2008 Maximus announced the highest gold grade intersected to date in the Bear Lake area of Larder Lake. Hole #35 intersects 18.3 g/t Au over 4.8 m, including 163.5g/t Au over 0.5m*
>
> *On June 4, 2008 Maximus announced that it had intersected new high gold grades in the Bear Lake area. Hole #44 intersects 13.6 g/t Au over 15.1 m, including 41.9 g/t Au over 4.4 m.*
>
> *Bear Lake Gold is currently completing a 43,000 m drilling program at Larder Lake to aggressively follow-up on the impressive intercepts. Their are currently three drills working on Bear Lake."*

15.3. Historical Production Statistics

Table 13, below, summarizes the gold production from the significant gold mines located along the Larder Lake Break within the 11 km stretch between Larder Lake and Virginiatown. The statistics on gold production is from a table compiled by the Ontario Ministry of Mines and Northern Affairs in the 2007 report prepared by the Resident Geologist's office in Kirkland Lake. The breakdown of ore from the Kerr Addison Mine into "Flow Ore" and "Carbonate Ore" is taken from a report in the Armistice files and is believed to be reasonably correct but cannot be verified and is included here because it shows the relative economic importance of the two types of mineralization.

There are no active producing mines operating in this area at the present.

Mine	Tons Milled	Production (oz Au)	Grade (oz/t)	Years of Production
Omega	1,615,000	214,000	0.133	1913, 26-28, 36-47
Barber Larder (Open Pit)		77,000	0.12	1987
Chesterville	3,260,000	359,000	0.110	
Kerr Addison	40,337,000	10,457,000	0.259	1938-1996
“Green Carbonate Ore”	16,000,000		0.233	
Pyritic “Flow Ore”	25,000,000		0.330	

Table 13 - Historical Gold Production between Larder Lake and Virginiatown

The production statistics are taken from the references about the Kerr Addison and Chesterville Mines cited above and from (Jenny, 1941) who acknowledges permission to publish from Omega Gold Mines, Limited. The production from the Barber Larder Open Pit is taken from information in Armistice’s files which is believed to be reliable but cannot be confirmed and should not be relied upon.

The author has been unable to verify the production information and the information is not necessarily indicative of the mineralization on the McGarry Property that is subject of the technical report.

16. Mineral Processing and Metallurgical Testing

Metallurgical testing of samples as summarized in the section are all taken from selected locations within the workings of the Property. The author cannot verify that any of the samples or the results of their analysis are representative of the metallurgical characteristics that could be expected from much larger batches produced from other locations within the Property.

16.1. Previous Metallurgical Testing

There were four bulk samples taken for processing in the Macassa Mill located just west of the town of Kirkland Lake operated by Kinross Gold Corporation on a custom basis in the period 1995 to 1997. (The Macassa Mill is now owned and operated by Kirkland Lake Gold Inc.) A detailed description of the sampling and processing is provided by Carmichael (June 6, 2004) and Hogg (December 17, 1996).

The results as presented by Carmichael are summarized below in Table 14:

Metallurgical Testing 1995-1997												
Sample Source					Processing							
Zone	Level	Mineralization Type	Related Chip Sampling Average (oz/t)	Tons Extracted (t)	Mill	Process Type	Tons Milled (dry t)	Calculated Feed Grade (oz/t)	Recovery	Ounces Produced for Sale	Tails Grade (oz/t)
185N	2050 Level at 1930 elevation	Pyritic Mudstone	~ 0.25	2,900	Macassa	Carbon-in-Leach	2,903	0.209	95.6%	580	0.0093
275N	1650 Level	Cherty Mudstone	0.25 oz/t (mixed with some waste rock)	~ 1,170	Macassa	Carbon-in-Leach	5,380	0.085	70.58%	321	Not Stated
100N	2250 Level	Graphitic Pyritic Mudstone	~ 0.05 oz/t drill indicated high grade not located	~ 3,240							
260N	2250	Green Carbonate	0.20 to 0.25 oz/t	~ 790							

Table 14 - Summary Results of Metallurgical Bulk Sample Processing – 1995 - 1997

There were reported problems with the unintended mixing of waste with the bulk samples. In particular, the surface stockpile from zones 275N, 260N and 100N became mixed and as a result these samples were processed as one batch. In addition, this batch was heavily weighted by the very low grade material from the 100N Zone. The author has back calculated an estimate of the combined grade from the 275N and 260N Zones by removing the influence of the 100N Zone material (3,240 tons at 0.05 oz/t gold). This back calculation gives an average estimated grade of about 0.15 oz/t gold, which appears to be a reasonable estimate following a review of the face and rib chip samples from the mined areas.

16.2. Gravity Based Processing

Work done by Lakefield Research as reported by Carmichael (June 6, 2004) and Hogg (December 17, 1996) included preliminary work on a Wilfley/Mozley Table combination to investigate the potential for upgrading of a mill feed. The results of this limited test work is presented in Figure 25. The implication of this test work is that the grade of the feed can be upgraded by a factor of at least four. These results provided encouragement for a larger scale test in 2008.

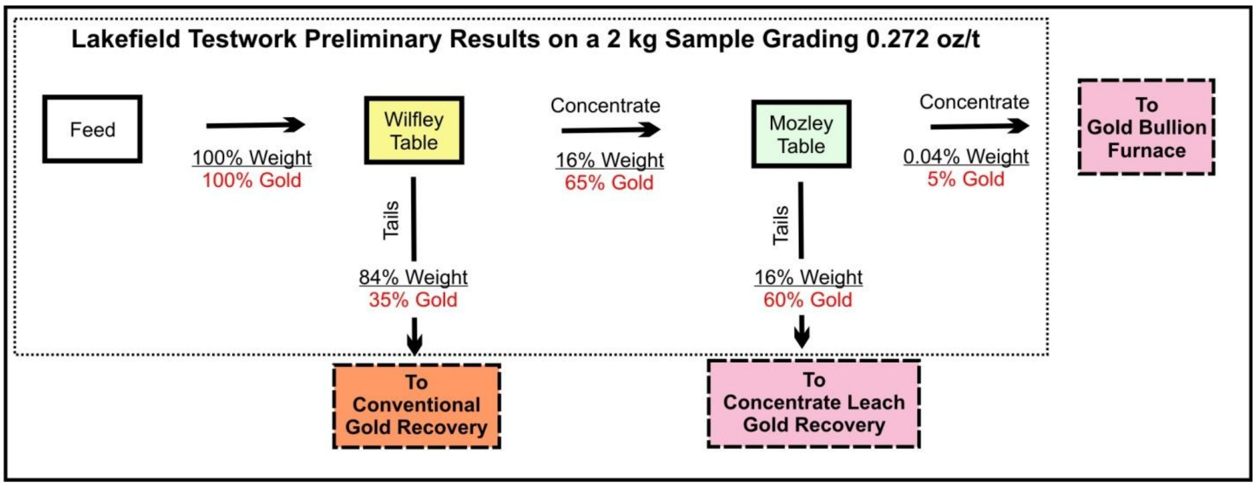


Figure 25 - 1995 Lakefield Preliminary Test Work Results on Gravity Recovery

SMC (Canada) Ltd., a subsidiary of Sabin Metal Corporation, (SMC) owns a mill located at Cobalt, Ontario that includes a circuit suited to processing small batches since the feed rate is about 1 ton per hour or less. The circuit includes crushing, grinding, a large Holman Table, a Maxwell cell and a bank of four flotation cells.

The circuit diagram for the SMC processing as applied to the Armistice sample is presented below in Figure 26.

A sample of crushed material taken at random from the “high grade” surface stockpile weighing 43.2 tons was delivered to SMC in November 2008 and processed on December 11 and 12, 2008. The processing was monitored on behalf of Armistice by an experienced metallurgical technician from Genivar in Val d’Or, Quebec. Genivar has reported to Armistice that all established procedures were followed and that no issues of concern were observed. The author also visited the SMC facility during the milling operation and noted no issues of concern.

The objective of the SMC test milling was to determine, if, on a larger scale than the preliminary Lakefield test work, an upgraded concentrate could be obtained that would be suitable either for:

(1) direct sale to a custom milling operation, or

(2) could be treated in an on site plant of significantly smaller size and capital cost than a plant built to treat the entire feed stream by one of the standard leach process alternatives.

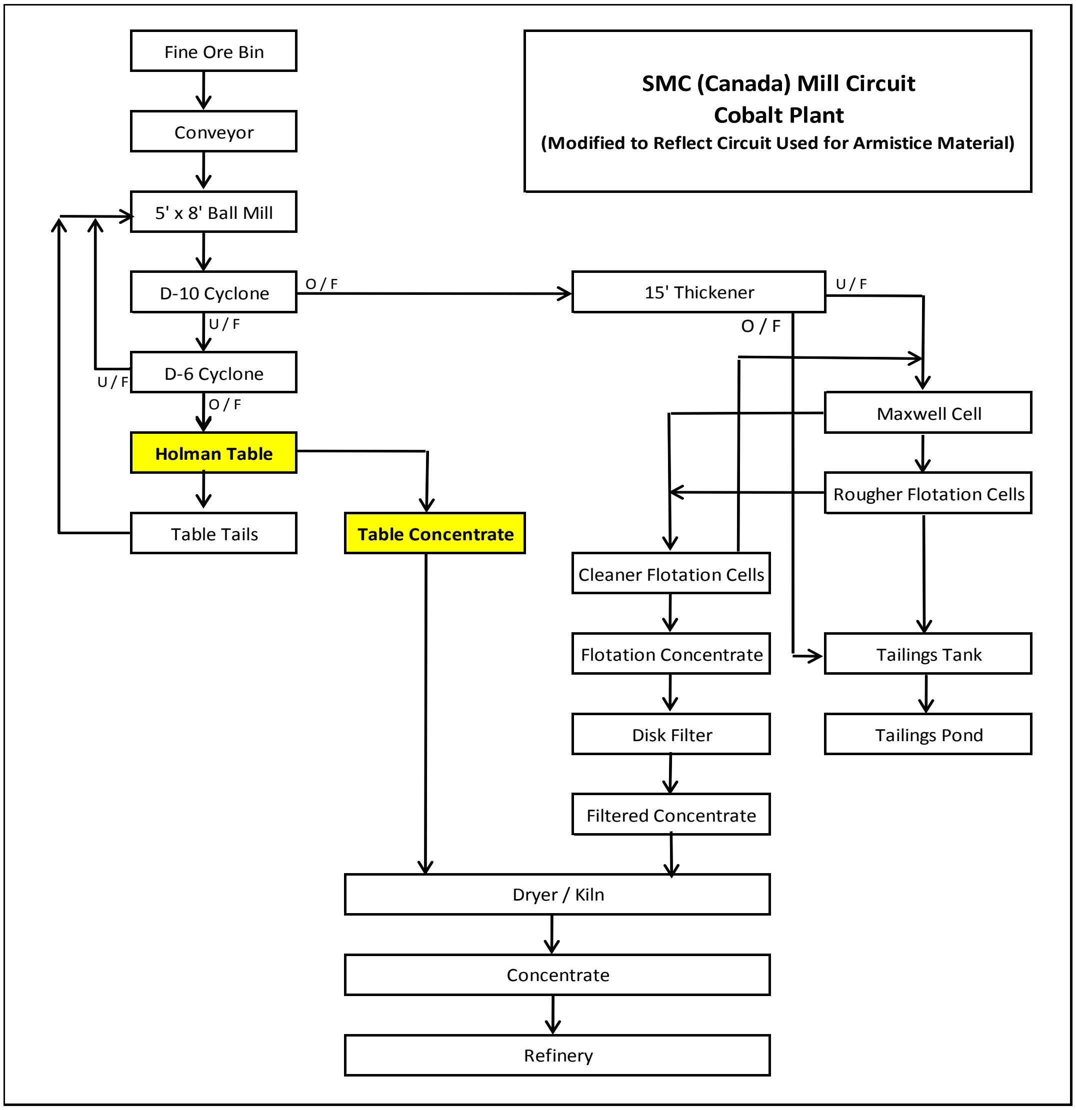


Figure 26 - SMC Processing Plant, Cobalt, Ontario – Circuit as Employed for Armistice Sample

The material reporting to the thickener was 80% -200 mesh and the material fed to the Holman Table was +60 mesh. Table 15, below, presents the results of the SMC processing.

There is a significant discrepancy between the estimate of recovered gold in the table + filter concentrates and the gold recovered in the melt of the same material at the SMC refinery. SMC has explained the discrepancy as due to the errors involved in obtaining a representative sample of the high grade concentrates for assay. SMC considers the melt recovered gold to be the reliable estimate of the contained gold in the concentrates. The author considers this to be a reasonable conclusion. The author also considers that the small quantities involved, especially in the refinery melt, may not be representative of the performance that can be expected from larger run processing batches.

Compared to the previous work by Lakefield as noted above, the gold recovery on the Holman/Flotation combination compared to the Wilfley Table decreased from 65% to 44% gold recovery, but the weights of the recovered concentrate was more efficient improving from 16% to 6%. The feed grinds to the Holman and Wilfley Tables were +60 mesh and 56% -200 mesh, respectively, that is, the Holman Table received a coarser grind than the Wilfley. The feed to the SMC flotation circuit was 80% -200 mesh.

As commented on in the Conclusions and Recommendations Sections of this report, the author finds these results warrant further research into improving non-leach recoveries by varying the grinds and testing in centrifugal concentrators such as the Falcon or Knelson.

Metallurgical Balance Results - SMC Canada Ltd.
Gravity / Flotation Circuit

Calculated Head Feed

Metallurgical Balance Based on In-Process Assay Results		Weight (Dry Tons)	Grade (oz/t gold)	Calculated contained ounces gold	Distribution	
					Tons	Gold
	Table Concentrate	0.324	2.09	0.68	0.8%	14.5%
	Filter Concentrate	2.363	0.87	2.06	5.5%	44.0%
	Total Concentrate	2.686	1.02	2.73	6.3%	58.5%
	Final Tails	40.050	0.05	1.94	93.7%	41.5%
	Calculated Head Feed	42.736	0.11	4.68	100.0%	100.0%

Belt Measured Feed	41.6	0.08

Metallurgical Balance Based on Concentate Melt Results		Weight (Dry Tons)	Calculated Grade (oz/t gold)	Contained Gold (oz)	Distribution	
					Tons	Gold
	Melt (Table + Filter Concentrate)	2.686	2.09	1.53	6.3%	44.1%
	Final Tails	40.050	0.05	1.94	93.7%	55.9%
	Calculated Head Feed	42.736	0.08	3.47	100.0%	100.0%

Table 15 - Metallurgical Balance Results from SMC Gravity/Flotation Test

16.3. Metallurgical and Environmental Characteristics of McGarry Mineralization

A number of operating gold mills within a 100 km radius of the McGarry Project have been identified by Armistice as having excess processing capacity that could be utilized for the custom processing of McGarry ore. Preliminary to discussions with these operators, it is important to determine the suitability of potential McGarry ore to the respective milling circuits and the environmental licences governing their tailings disposal impoundments. One concern with respect to tailings disposal issues today is acid generation, therefore pH is an important determination especially since the gold mineralized zones at McGarry do contain significant pyrite, minor arsenopyrite and very minor chalcopyrite and sphalerite.

Test work on samples from mineralized zones at McGarry were analysed by two labs, Multilab-Direct of Rouyn-Noranda, Quebec and Process Research Associates Ltd. (PRA) of Richmond, BC. The sample sent to Multilab-Direct was a composite from drill core assay reject material from the intersections of the 400N and 325N Zones on the 2250 Level in DDH's 22-44 and 22-126. The sample sent to PRA was reject fines material from bulk sample ARM-05 after return from PolyMet Labs. In addition, test work was done on waste rock from the 2250 Level by the Ontario Government's Geoscience Laboratories (Geo Labs) in Sudbury, Ontario.

No work has yet been done to determine long term arsenic leaching from tailings.

A summary of the main results is presented below in Table 16:

Summary of Metallurgical Characteristics Testing						
Lab	**Multilab-Direct**			**PRA**		**Geo Labs**
Sample Material	Drill core assay reject from 2250 Level Zones 325N and 400N			Reject material from PolyMet bulk sample ARM-05		5 grab samples taken from crushed waste pile
Sample Number	C-43880	C-43881	C-43882	Head 1	Head 2	
pH	9.24	9.27	9.25			8.24
Total sulphur	1.45%	0.73%	0.78%			0.34%
Sulphates	0.003%	0.00%	0.00%			
Acid generator	no	no	no			
Arsenic	390 ppm	533 ppm	735 ppm			
Copper	51 ppm	30 ppm	28 ppm			
Zinc	21 ppm	22 ppm	27 ppm			
Gold				8.51 g/t	7.42 g/t	
Al2O3				6%	6%	
CaO				13%	13%	
Fe2O3				9%	9%	
MgO				8%	8%	
SiO2				41%	41%	
Cabon-In-Leach						
Extraction (gold)				95.00%	95.10%	
Residue (gold)				0.38 g/t	0.38 g/t	
NaCN Consumption				0.31 kt/t	0.31 kg/t	
Lime Consumption				0.53 kg/t	0.56 kg/t	

Table 16 - Summary of Metallurgical Characteristics Testing Results

17. Mineral Resource and Mineral Reserve Estimates

The 2009 Mineral Resource and Mineral Reserve Estimates was carried out by the technical staff of Armistice Resources Corp. under the supervision of the author, a Qualified Person under the requirements of National Instrument 43-101. The estimate of mineral resources and reserves was made in compliance with the recommendations and regulations of NI 43-101. The mineral resources are grouped according to the classification established by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) and adopted by the CIM Council. The CIM Standards describe completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. A Preliminary Feasibility Study is a comprehensive study of the viability of a mineral project that is advanced to a stage where the mining method has been established, and where an effective method of mineral processing has been determined. A Preliminary Feasibility Study has not been carried out for the McGarry Project. Although the results of a Scoping Study are included in this report (see Section 18.1. below), the parameters for the Scoping Study establish it as a "preliminary assessment" and it does not meet the requirements of a Preliminary Feasibility Study. Only Mineral Resources are estimated for the McGarry Project in this report.

The source data and parameters used for the calculation of Mineral Resources correspond to the acquired knowledge, best estimation and the situation as at April 8, 2009. The technical staff of Armistice calculated mineral resources by zone according to local conventions used at McGarry.

The estimate is based on all the results available for the area included between Sections 700E and 2400W and from 300 feet below surface to 5400 feet below surface. Sufficient information to support an estimation of resources outside this area is not available on known mineralized zones. The resource estimate is not influenced by any penalizing factor.

Mineral resources are not mineral reserves since they have not demonstrated economic viability. Results from the resource estimates are presented with and without a cut off factor and in situ, undiluted.

17.1. Methodology and Estimation Parameters

The methodology used for the current resource estimate is by polygonal method on vertical cross sections and elevation (level) plans for each zone, using each intersection's true width. The author is of the opinion that this method is the best suited for narrow-vein type deposits such as at McGarry and that orienting plans and sections orthogonal to each zone's strike and dip produces the most accurate resource estimate possible. The calculation estimates undiluted resources.

The first step consists of geological interpretation based on all recognized geological features and not exclusively on grade. This interpretation is done on cross sections cut at grid north-south which is approximately at a right angle to each zone's strike. The features used to determine the continuity of the zone are continuity of lithological unit, often a "pyritic mudstone" or equivalent or close association to albitite bodies, and continuity of elevated gold values within the unit.

The second step is the generation of drill hole intercepts (composites) in horizontal and true thickness. The horizontal and true widths of each composite are calculated by trigonometry using the drill hole azimuth and dip and the zone strike and dip at the position of the intercept. A minimum horizontal width of 5 feet is used. The grade is calculated by weighted average, using surrounding grades, if needed, to achieve a 5 foot horizontal width. The composites are plotted on vertical longitudinal sections oriented grid east-west which is the closest practical common orientation of the mineralized zones.

The third step is the construction of the blocks of mineralized material. The blocks are constructed following mineralized trends observed in each zone based on the drilling information and on gold zones as observed in mine openings. Blocks are designed using mid-distances between drill hole intercepts or information from mined openings. A maximum distance to the edge of a block from a drill hole intercept of 50 feet has been applied. This maximum distance of influence has been determined based on the acquired experience at McGarry and is considered reasonable and appropriate by the author.

The last step is the tonnage calculation. Each block is measured, its lateral extension being measured directly on plan maps to account for zone strike, and its vertical extension being measured on the corresponding cross-section to account for zone dip. The true width of the composite is used as the third dimension.

Volume is then calculated for each block which leads to a tonnage calculation using a specific gravity of 2.90 for both waste and mineralized material as accepted in the previous Mineral Resource estimate (Carmichael, S.J., June 6, 2004). Carmichael's specific gravity factor was based on the average of 7 core samples for which the specific gravity was determined by Swastika Labs. (Note: there is a typographical error on page 20 of Carmichael's report where the specific gravity is incorrectly shown as 2.79 instead of 2.90; however the tonnage factor is correctly calculated as 11.0 cubic feet per short ton.) No additional determinations have been made for specific gravity. The tonnage factor for a density of 2.90 is 11.0 cubic feet per short ton in situ. Each block is given a unique reference designation.

The resource estimates are based on a total of 198 blocks (161 for the Indicated Resource and 37 for the Inferred Resource).

17.1.1. Minimum Cut-off Grades

A minimum cut-off grade of 0.10 oz/ton gold is used for the Mineral Resource estimate. There are no blocks included in the estimate of the mineral resources with an average grade below 0.10 oz/t. The bulk sampling and chip sampling reported on in Section 12 of this report clearly shows that very high grade intersections can be made within a few feet of very low grade intersections all being within the bounds of a defined gold-bearing zone.

17.1.2. Minimum Width

All drill hole intersections are calculated using a minimum horizontal mining width of 5 feet applying the grade of the adjacent material when assayed or zero when no assay data is available. The horizontal and true widths are calculated using trigonometry as described in Section 17.1., above. The minimum mining width is based on the horizontal width because this is the parameter that determines the stope width under normal mining practices.

17.1.3. High Grade Assay Cutting Values

The value used to cut the high grade assay values is 1.5 oz/ton. Cutting affects 8% of the blocks included in the calculation of the Indicated mineral resource estimate and 3% of the blocks included in the calculation of the Inferred mineral resource estimate. In the view of the author, cutting is a controversial issue and can only be resolved for a particular ore body when a cutting factor is required to fully reconcile mining grade estimates with actual mill head-grade calculations. Nevertheless, the author considers that cutting to 1.5 oz/ton at this stage provides an appropriate and conservative protocol for McGarry.

17.1.4. Assay Data Used in Resource Estimation

For the current estimate, drill hole intercepts, chip samples in drifts and test stopes including bulk sample openings are used to estimate grade and tonnage of the zones. Where more than one assay is available for a sample, then the arithmetic average of all the assays available for that interval has been assigned to the sample. Multiple assay values for the same interval arise for quarter-cut core samples, internal lab check assaying from the same pulp or re-check of pulps by a second lab. In the few cases in which metallic assays are available, then the metallic assay is used ignoring standard fire assay results. This is because the metallic technique analyses the total gold content of the sample including any larger gold grains that may or may not report to the standard amount of pulp normally used for fire assaying.

17.1.5. Resource Classification

The resource classification definitions used for this report are based on the Canadian Institute of Mining, Metallurgy and Petroleum as presented in the document “CIM Definition Standards – For Mineral Resources and Mineral Reserves” adopted by CIM Council on December 11, 2005.

> *“The term **Mineral Resource** covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase ‘reasonable prospects for economic extraction’ implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed*

and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

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.

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.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions."

The Indicated Mineral resource is estimated from blocks inside a regular drill pattern with drill hole intercepts having a maximum spacing of 100 feet. In some cases, a resource can be classified as Indicated on a wider drill pattern when it is part of a well defined gold-bearing trend. The shape and extension of the trend must be known laterally and some indication of good continuity must be present in the direction of the Indicated resource's trend. Otherwise, the resource is classified as Inferred. The maximum area assigned to any Indicated Mineral resource block is 100 ft by 100 ft (10,000 sq ft).

The Inferred Mineral resource is estimated from blocks intersected by a drill hole but located beyond 300 feet of an established mineralized trend or do not otherwise fall within the requirements for inclusion in the Indicated category. The maximum area assigned to any Inferred Mineral block is 10,000 sq ft.

No blocks in any category have been included that do not have a drill hole intercept.

17.2. Mineral Resource Estimate Results

A tabulation of estimated Mineral Resources is shown below in Table 17. The existing shaft and hoisting infrastructure fully services the Property to a depth of 2250 ft and a ramp to the 2300 ft elevation can easily be established from the 2250 Level. Work has already started to establish this ramp. Therefore, the tabulation of Mineral Resources is summarized as located above and below the 2300 ft elevation.

Undiluted Mineral Resource Estimate – April 8, 2009

Mineral Resource Category	Tons (short tons)	Cut to 1.50 oz/t		Uncut	
		Grade (oz/t gold)	Gold (oz)	Grade (oz/t gold)	Gold (oz)
Indicated					
Above 2300 elevation (all zones)	374,000	0.22	82,000	0.25	93,000
Below 2300 elevation (all zones)	118,000	0.25	30,000	0.26	30,000
Total Indicated (all zones)	**492,000**	**0.23**	**112,000**	**0.25**	**123,000**
Inferred					
Above 2300 elevation (all zones)	59,000	0.17	10,000	0.19	11,000
Below 2300 elevation (all zones)	113,000	0.16	19,000	0.16	19,000
Total Inferred (all zones)	**172,000**	**0.17**	**29,000**	**0.17**	**30,000**

- Mineral resources estimated according to CIM definition standards (2005).
- A 0.10 oz/t gold cut-off grade was used with high-grade values uncapped and capped at 1.5 oz/t gold.
- A fixed specific gravity of 2.79 was used.
- Undiluted resources, all drill hole intercepts are calculated using a minimum horizontal width of 5 ft, using the grade of adjacent material, if assayed, or zero if not assayed.
- Gold grades determined using the polygonal method with polygons determined from interpretation on vertical cross sections and elevation plans. Maximum distance to the edge of a block from a drill hole or chip sample intercept of 50 ft has been applied. Maximum block size is 10,000 sq ft.
- A confidence level of ±10% is estimated for the Indicated Mineral Resource and ±25% for the Inferred Mineral Resource.
- Effective date of resource estimate is 8 April 2009.
- Qualified Person for the mineral resource estimate is Erik Andersen, P.Eng. (Armistice Resources Corp.)
- Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues although the Qualified Person is not aware of any such issues.

Table 17 - Mineral Resource Estimate for the McGarry Project

The previous Mineral Resource estimate is dated 6 June 2004 (Carmichael, S.J., June 6, 2004) and is stated as an undiluted Indicated Mineral Resource of 433,981 tons at a grade of 0.250 oz/t gold (0.10 oz/t gold cut-off) without a capping cut-off factor and includes only blocks above the 2600 elevation. The current estimate has added approximately 14% to the tons and 14% to the contained ounces of gold. The previous estimate did not include an estimate of the Inferred Mineral resource.

18. Other Relevant Data and Information

18.1. Scoping Study

"Mineral resources" have been defined and classified at McGarry, but mining development has not reached the stage to permit the classification of any "ore reserves". Nevertheless, it is considered important at this stage to be able to set targets for the economic threshold required for a profitable gold mining operation at McGarry. Therefore, Armistice engaged an independent consultant, Python Mining Consultants Incorporated ("Python") to conduct a Scoping Study ("Python Report" or "Scoping Study") to estimate the costs, revenues and schedules for a potential mining operation between the 1250 and 2250 Levels given a set of parameters and assumptions supplied by Armistice. The level of confidence in the costs and revenues in the Scoping Study is ±25%.

> The Python Report is based on the knowledge of the McGarry Project as available at the end of December 2008. The Mineral Resource estimate included in this technical report was not available at that time. The author (Andersen) is of the opinion that an update of the Python Report to include all the data and information currently available, including the revised Mineral Resource estimate, would not materially change the conclusions reached by Python.

The Python Report is authored by Martin Drennan, P.Eng and Qualified Person and is attached hereto as Appendix A. The Qualified Person Certificate signed by Mr. Drennan is attached at the end of the Python Report following Appendix B of that report. Included with the Python Report is a letter from Armistice that sets out the scope of the requested study including parameters and assumptions (last two pages of Appendix B of the Python Report).

It is the opinion of the author (Andersen) that the parameters and assumptions set out do reasonably present a potential mining scheme for McGarry although, as new information is obtained about the mineral resources in the part of the mine under consideration by Python, the basis for the conclusions reached by Python may no longer remain valid.

The strategic approach on which the Python Report is based includes the following elements:

- the McGarry project's database of diamond drilling, bulk sampling and mining data indicates an east-west striking and steeply dipping geological package in which there is a reasonable likelihood that gold-bearing zones of sufficient extent to permit standard narrow-vein mining techniques exists between the 1250 and 2250 Levels;
- the actual distribution of gold-bearing zones on any one level cannot be determined at this time with sufficient confidence for the planning of mining stopes. However, it is reasonable to assume that a cumulative total of 1,000 feet of strike extent of gold-bearing zones exists on each level with each zone having a minimum strike extent of 100 feet. This assumption is

not proven and actual results may vary which could invalidate the results of the preliminary assessment.

- in order to provide definition drilling access and ore/waste haulage, an east-west main drift in waste near the centre of the prospective geological package could be driven as illustrated in the schematic, Figure 27, below; and in Figures 1 and 2 of the Python Report.
- a diamond drilling programme cannot be laid out that could adequately define production stopes prior to establishing significant mining infrastructure on each level at an acceptable cost. A schematic cross section of a practical short-hole drilling programme for gold zone definition and stope planning is shown in Figure 28 below.
- the plan presented is to establish the infrastructure and, as sufficient ore is discovered in conjunction with this work, to move quickly into a combined ore definition/development and production mode. Discovery of sufficient ore is not assured and actual results may require a re-evaluation of the work plan. Work should progress over 55 months in three phases:
    - 1) predevelopment lasting 7 months (see additional comment 3 pages below)
    - 2) produce ore at a nominal average rate of 350 tpd for the next 12 months; and finally
    - 3) increase ore development to produce at a nominal rate of 600 tpd for the final 36 months until all the ore available between the 2250 and 1250 levels is exhausted.

    [These production rates are "nominal" – the rates actually produced from the MineSched modelling performed by Python are 362 tpd and 602 tpd for use in Appendix B of the Python Report in which per day costs are calculated.]

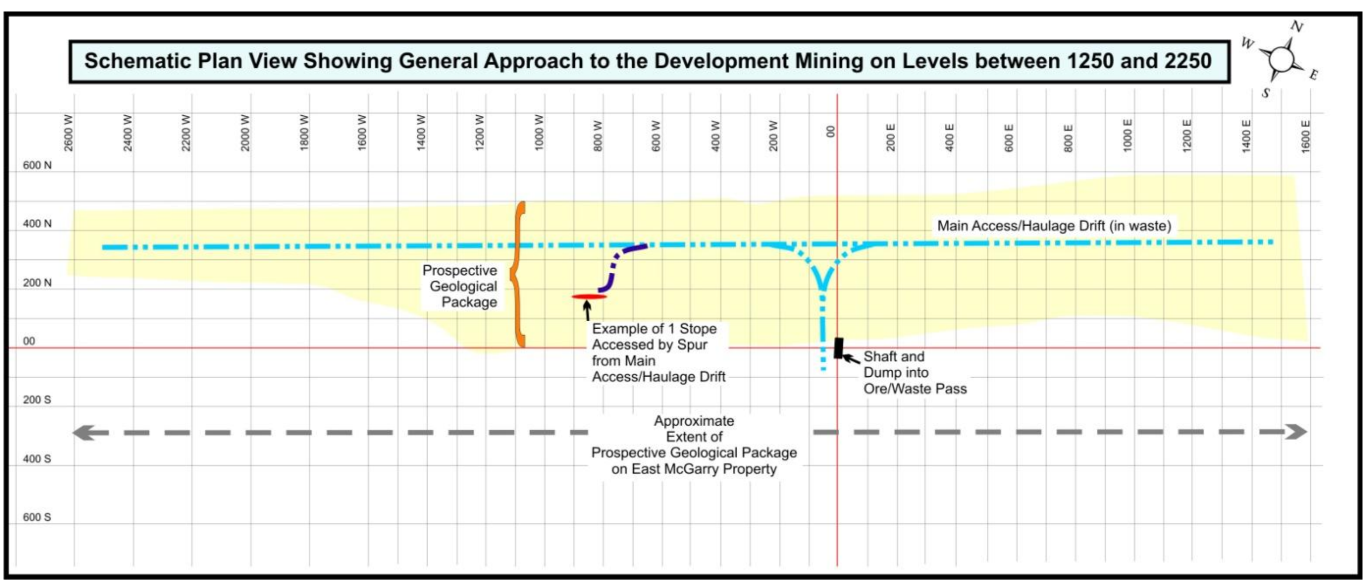


Figure 27 - Schematic Plan Showing Strategic Approach to Development Mining – Upper Levels

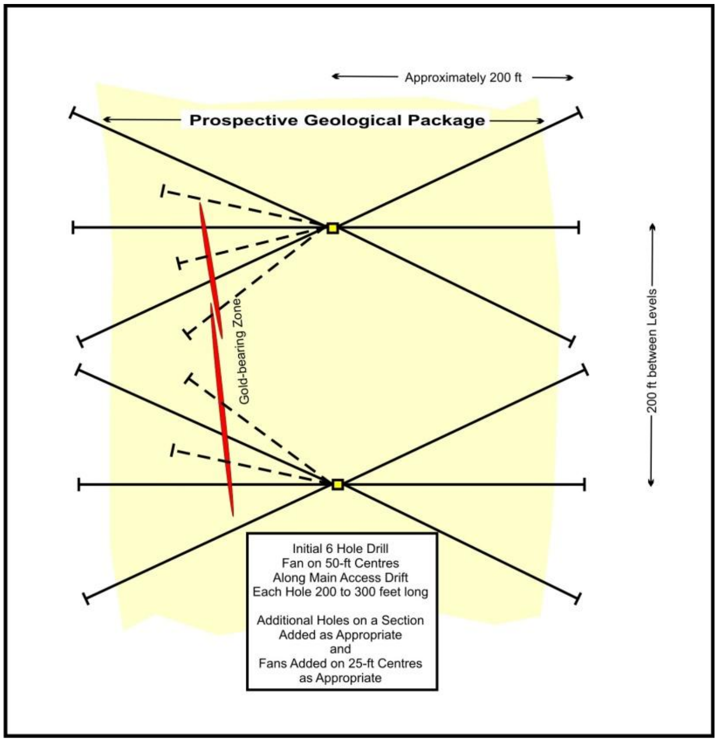


Figure 28 - Schematic Cross Section Showing Short Hole Diamond Drilling for Gold Zone Definition

The assumptions used by Python are summarized in the Python Report as:

- *"All technical source data has been supplied by Armistice and is correct. A site visit was completed, however, data auditing was not performed. This is assumed to be the responsibility of Armistice.*
- *In coordination with Armistice, this study has a confidence range of ±25%.*
- *Gold bearing areas are extrapolated on diamond drill holes and on sill information.*
- *Stope areas are based on gold bearing zone patterns identified.*
- *Gold bearing zones have been physically tested. Results of this have not been disclosed by Armistice to PMC.*
- *The development method of gold bearing zones is based on assumed structures.*
- *Economic analysis parameters are: mining recovery is 90% and milling recovery is 85%.*
- *Scheduling parameters are:*
    - *Dilution is assumed within the stope shells (i.e. dilution is accounted; not as a fixed percentage, but in the stope dimensions – minimum stoping width of 6ft and 1ft of dilution either side).*
    - *Development and production rates were discussed internally amongst Armistice, Paul Whelan Mining Contractors, and PMC.*
    - *The McGarry project is a narrow-veined gold deposit.*

- *Shrinkage and longhole mining, and a hybrid combination of these methods, are applicable to this project.*

- *Gold prices are assumed to range from $600 to $1100 at $50 intervals.* [$C per oz]
- *All values are Canadian Dollars (CAD).*
- *There are currently only inferred resources at McGarry."*

The economic modelling performed by Python is in cash flow form, is ±25%, and assumes the presence of gold zones suitable for stoping which have yet to be identified. There is no consideration for cost escalation, depreciation/amortization, or income taxes. No estimates of rates of return have been attempted, since the confidence level in the costs and mined grade is considered too low for meaningful estimates especially when combined with current volatility in gold prices.

A metallurgical recovery of 85% has been used throughout the Study. In part, this compensates for the 4% net smelter royalty payable to Sheldon-Larder Mines Limited when the gold price exceeds $US 800 per oz as described in Section 4.1. above and as it is at the date of this report. The test work to date, as described in Section 16. above, indicates that metallurgical recoveries in the order of 95% should be achievable in a leach circuit. The metallurgical test work has also shown that it may be possible to make an initial high grade concentrate using non-leach methods alone (gravity and flotation), but recoveries with these methods alone will probably be below 90% even after fine tuning. A final approach to metallurgical recovery of gold has yet to be determined. Therefore, a conservative assumption has been used for the Scoping Study.

Since March 2006, the gold price has only been below $C 685 per oz for two short periods (in late 2006). Since January 2008, the gold price has only been below $C 885 for three short periods and never below $C 785. For most of 2004 and 2005, the gold price was in the $C 500 range. The current gold price is $C 1090 per oz.

The cost estimates used in the Python Study are based on actual achieved costs and performance over the period September 2007 to September 2008 at McGarry.

The development and stoping plan assumed by Python includes three consecutive phases, summarized with duration periods and total costs from Table 3 in the Python Report as:

Phase	Duration (±25%)	Cost (±25%)
Preproduction	7 months	$4.1 million
350 tpd Development and Production	12 months	$7.4 & $8.4 million, respectively
600 tpd Development and Production	36 months	$4.2 & $41.5 million, respectively
Total	55 months	$65.7 million

Production potential during these phases, assuming a mined grade of 0.20 oz/t, 85% metallurgical recovery and $C 800 per oz gold and a range factor of ±25% is summarized as:

Period	Duration	Potential Production	Potential Mined oz Au	Potential Revenue @ 85% recovery
Preproduction	7 months			
350 tpd	12 months	126,000 tons	25,000 oz Au	$ 17.1 million revenue
600 tpd	36 months	520,000 tons	104,000 oz Au	$ 70.7 million revenue
Total	55 months	646,000 tons	129,000 oz Au	$ 87.8 million revenue

As noted in the Python Report under several points in the Recommendations and Conclusions sections, more work is required to update economic parameters.

The example above uses a gold price of $C 800. The reader can easily calculate other potential revenue totals, but $C 800 per oz, in the opinion of the author (Andersen), seems a reasonable value to use for purposes of illustrating the underlying economic robustness.

In order to provide a view of sensitivities, a simple comparison of cash flows over the 55 months modelled within a range of ±25% is summarized as:

Gold Price (per oz)	Cash Flow (millions) @ 0.20 oz/t	Cash Flow (millions) @ 0.18 oz/t
$C 500	$ 11.7 negative	$ 17.2 negative
$C 800	$ 21.2	$ 12.5
$C 1100	$ 54.2	$ 42.1

The Python Report includes the following comments on the economics of the model:

> *"The economic model, based on the assumptions in its development for both initial production (350 TPD) and the targeted final (600TPD) production, is linear.*
>
> *To be profitable, the initial production needs to have a diluted grade greater than 0.21 ounces per ton with a minimum assumed gold price of $650 per ounce* [$C]. *The initial production for the first year is assumed to be 350 TPD. The initial production period projected revenue is approximately $12,700,000, resulting in a relatively neutral profit – i.e. breakeven. If the first year of production can have an elevated production rate in excess of 350 TPD, the likelihood of being able to produce a profit increases.*
>
> *The economic cut-off diluted grade (breakeven) for 600 TPD is 0.19 oz/ton at $600 per oz gold. With improvements in grade and gold price, profitability increases. With a minimum 0.19 oz/ton diluted grade and a gold price of $600 per ounce, revenue would be approximately $54,000,000 over 3 years. This is the breakeven point and the minimum requirements for the project to move forward."*
>
> *"Assuming the market conditions continued for the four years it would take to mine the McGarry deposit, and the grade assumption of 0.19 oz/ton diluted grade, the potential revenue for the initial year of*

production would be approximately $17,800,000. The expenses for processing the initial production year is based on 350 TPD and a cost of $97.87 per ton. This would result in a positive cash flow circa $6,000,000. If the diluted grade was unable to be achieved and a diluted grade of 0.14 oz/ton was the realized grade (mill feed value @ $13.1M), then the initial year positive cash flow would be reduced to $1,500,000.

With the assumptions above ($1000 per ounce revenue and 0.19 oz/ton diluted grade), the revenue for the 600 TPD period would be approximately $91,000,000. The operating expense during this period would be around (3 years x 350 days/year x 600 TPD x $78.42) $53,500,000 resulting in a projected profit of $35,000,000. With the conditions varied to $1000 per ounce and a diluted grade of 0.14 oz/ton the revenue would fall to $67,000,000 giving a projected profit potentially of $13,000,000.

In any of the above instances there is a significant impact of diluted grade relative to profitability."

The Recommendations in the Python Report are:

- *"Continue logging and assembling the diamond drill database.*
- *Update the geological interpretation of the deposit with the new diamond drill data and development information.*
- *Complete an advanced geological model using the new interpretation – contours and block model.*
- *Create a resource meeting NI43-101 standards.*
- *Update the mine design to match the new geological model.*
- *Update unit costs and economic parameters to reflect regional costs.*
- *Based on the new mine design, reschedule and complete a new economic evaluation at a pre-feasibility level.*
- *Complete primary mine development, which will enable diamond drilling and bulk sampling to be completed in known gold bearing areas.*
- *Utilize the positive gold market conditions for investment.*
- *Review regional as well as similar scaled operations – specifically Kerr Mine, Kirkland Lake Gold – Macassa Mine, and smaller producers such as Wesdome's River Gold Mine and Island Gold's Patricia Mine – relative to McGarry as a means to support both geological and mining strategies."*

The Conclusions in the Python Report are:

"The positive aspects of the McGarry Project are:

- *The area is a known gold producing region which has produced significantly over past decades.*
- *The McGarry project is adjacent to the Kerr Mine, which produced approximately 10 million ounces of gold between 1938 and 1996.*
- *An existing operating infrastructure, including headframe, hoist, surface shop and key components meeting the permit and legal standards for operation.*
- *A recent bulk sampling history and diamond drill program indicating potential for the project.*
- *An existing underground shaft and development available to advance mine development and move material to surface.*
- *Operating costs in the region are less than other gold producing areas in Ontario.*
- *The mine design, based on the assumptions in the geological model, can be completed in a timely manner and at a reasonable cost.*

- *The majority of the considered ranges of grade and dollars per ounce are profitable.*
- *The production schedule indicates that a 7 month preproduction period would be necessary to achieve an annual production rate of 350 TPD. After the one year of production, a 600 TPD target is achieved readily.*

The aspects of the McGarry project requiring further consideration are:

- *Geological interpretation and data management need to be advanced to a NI43-101 resource with classified resources.*
- *Measured and Indicated resources need to be identified and targeted for advanced diamond drilling and underground bulk sampling.*
- *Geological correlation of previous work such as the bulk samples, and diamond drilling.*
- *Advanced economic parameters to fully capture cost components for detailing unit costs, site specific and corporate costs.*
- *More systematic underground bulk sampling from numerous levels including and above 2250 level.*

Based on the work to date, the McGarry project has financial potential. Some additional work needs to be completed to ensure the success of the project. The project appears to have a better than average likelihood for profit."

18.2. Environmental Monitoring and Closure Plan

Armistice has been issued two Permits to Take Water issued by the Ontario Ministry of Environment ("MOE") with respect to the McGarry Project. One Permit is for the taking of water from Barber Lake and the other is the taking of water from the underground workings. Armistice must report annual water volumes under these permits.

The Ontario Ministry of Environment has also issued a Certificate of Approval ("CofA") to discharge water from a treatment works (settling pond) which collects discharge water from the underground workings. The CofA establishes monitoring guidelines and quality limits. The results of the monitoring must be reported to the MOE annually.

Monitoring of the requirements and reporting as required by the Permits to Take Water and CofA are supervised on behalf of Armistice by N.A.R. Environmental Consultants Inc. of Sudbury, Ontario ("N.A.R."). The latest annual report filed under the CofA is dated March 25, 2009 and covers the year 2008. There were no variances noted in the report. No comments on the report have been received from the MOE. There are no outstanding orders from the MOE.

Armistice has filed an Advanced Exploration Closure Plan covering the current project scope (N.A.R Environmental Consultants Inc., July 2007) that has been accepted by the Ontario Ministry of Northern Development and Mines. This plan sets out measures to be undertaken at an eventual close out of the project including capping of underground access openings, removal of surface infrastructure, revegetation, and monitoring of the physical and chemical stability of the site following closure. Armistice has posted a bank guarantee for financial assurance of $410,400.

Armistice has engaged N.A.R. to begin work on developing a closure plan for expanded mining activity as may be undertaken during the next stage of mining activity. This work is in progress.

19. Interpretation and Conclusions

The McGarry Property has been explored in four major campaigns: 1945-1947, 1988-1990, 1994-1998 and 2007-2008 plus other more limited exploration programmes in between. The original impetus for exploration at McGarry came with the development of the adjacent property into the Kerr Addison Mine which became a major gold producer yielding 11+ million ounces over a span of almost 60 years until it finally closed in 1996.

From the start, near surface exploration failed to find economic gold-bearing zones at McGarry, although a small open pit was eventually mined by others a few thousand feet to the west of the McGarry Shaft. Therefore, exploration efforts became dominated by underground drifting, diamond drilling and bulk sampling beginning in the 1940's. To date there has been no commercial gold production on the Property.

As a result of the work on the Property, there is a fully functional mining plant in place including a production-capable headframe and hoist and a three-compartment shaft to 2290 feet below surface. This plant has the potential to service a mine producing over 1,000 tons per day combined ore plus waste to a depth of at least 4400 feet.

The Property is located in the centre of the long established Abitibi mining district in northeastern Ontario (see Figure 1, page 5). There are ready local labour pools that can provide all the skills required for narrow vein conventional mining. This location is also in the centre of one of the world's major service areas for underground gold mining support.

19.1. Geological Setting and Mineralization

The McGarry Property straddles the Cadillac-Larder Lake Break which has been a major locus for gold deposits from Val d'Or in Quebec to Kirkland Lake in Ontario (see Figure 6, page 14). In the area of the McGarry Project and the adjacent former Kerr Addison Mine, the "Break" encompasses, a series of east-west trending mafic to ultramafic volcanics and sediments which have been highly altered to various phases of "carbonates" with varying degrees of silicification. There remains controversy about the genesis of the original rock units and the mechanisms involved in the alteration/tectonic events. The McGarry technical staff, including the author, support the use of a working hypothesis assuming the alteration to be hydrothermal in nature. The relationship between the gold emplacement and the alteration events requires continued research in order to build a good working model as a basis for efficient ongoing exploration at the Project.

Current exploration strategies at McGarry are not heavily influenced by an interpretation of the geological history. Strategies are guided by empirical observations of geological relationships. Nevertheless, the development of geological concepts governing the controls for the localization of gold-rich zones remains an ongoing research project. As at all mining operations, an ongoing open mind to new ideas will be essential to ensure no opportunities are missed.

Only the eastern 4500 ft of the Property straddling the interpreted locus of the “Break” has been well explored. The prospective package is near vertical with a slight average dip to the north. Locally, dips may be steep to the south or to the north.

It is recognized that gold-bearing zones of potential economic interest are located within a zone of alteration that has a north-south thickness of about 600 ft on the 2250 Level. There are two important styles of gold mineralization recognized to date:

1) “carbonate associated” in which the gold appears to be concentrated in small random regions within a general envelope that has poorly defined boundaries. The carbonate alteration may be rich in fuchsite giving the rock a bright green colour. Grades may be very high (over 1 oz/t) in random samples and highly variable over distances of even a few feet. This style of mineralization is exhibited in the 260N Stope off the 600W X/C. Because of the highly variable nature of the gold distribution and the ill defined zone boundaries, this style of mineralization may have limited economic potential at McGarry although each situation where this style of mineralization is recognized needs to be independently evaluated.

2) “pyritic mudstone” in which the gold appears more uniformly distributed within a unit of fine grained, dark siliceous, pyrite-rich rock. Although a clastic name has been given to the unit, this is a local term only and does not imply that the unit is, in fact, of clastic origin. At the Kerr Addison Mine this style of mineralization was called “flow ore”. This style of mineralization appears to be the most important in terms of economic potential. Pyrite content can vary from a few percent to over 25%. Even though gold content appears more evenly distributed than with the “carbonate style”, gold assays can vary from near zero to 1 oz/t over a few feet.

From the mining along strike in mineralization on the 1650, 2050 and 2250 Levels, horizontal continuity of gold zones appears to be at least 100 feet with horizontal thicknesses varying from a few feet to 20 feet. Vertical continuity has been proven by mining to about 40 feet in the 140N Stope off 00 X/C. The mining of this stope does not limit the vertical continuity since both the stope floor and back remain in the gold zone.

An interpretation used in setting the parameters for the mineral resource estimate and in the scoping study for the mine design completed by Python Mining Consultants is that vertical continuity of about 200 feet or more is common. The author believes this to be a reasonable conclusion based on the current geological knowledge base. Much closer spaced diamond drilling than is currently available combined with extended mining experience will be required to establish the ultimate reliability of this assumption.

In defining gold-bearing zones of potential economic interest, hole-to-hole correlation has always been difficult. This is partly because hole spacing is usually 100 to 200 ft or more. It is also because of the complex patterns superimposed on the rocks by major alteration and deformation events. As discussed above, the empirical evidence is that economic interest should initially focus on units described as "pyritic mudstone" although all gold bearing environments have the potential for economic production.

Drill intersections through the pyritic mudstones are often elevated in gold and assay values above 0.10 oz/t gold are common. Direct comparisons between drill hole assay results and larger samples (chip samples, panel samples and bulk samples) shows a correlation that the author considers too erratic and unreliable (see Table 10 and Table 11, pages 42 and 43) to base the continuity of mineralized zone simply on assay values. In the author's experience,this is a common and important observation in most hydrothermal vein type gold deposits and should not be considered surprising or unusual.

Because continuity of gold-bearing zones cannot be strictly interpreted from drill hole assay data, the observed relationship between gold values and pyritic mudstone units – in combination with assay data – becomes very important in the interpretation of potential zones for stope design. Although still uncertain, hole-to-hole continuity of gold-bearing zones is easier to interpret by relying on the continuity of pyritic mudstone units than by relying entirely on the interpreted continuity of higher grade gold intersections alone. A low gold value within a pyritic mudstone unit does not mean that the unit does not contain economic grades of gold within a few feet. This variability of gold values is often referred to as a "nugget effect". No attempt to quantify the nugget effect at McGarry has been made, although in the author's experience, the nugget effect at McGarry is probably typical of narrow vein hydrothermal gold deposits in the Canadian Shield.

This approach to gold-bearing zone definition has been central to the conclusion that in the order of ten 100 ft long zones occur on each level as assumed for the Python Scoping Study. This is an unproven assumption and there is no assurance that further work will confirm this assumption.

19.1.1. Geological Structures (Faults, Slips, Shears)

There appears to be a westerly plunge to the mineralized zones varying from vertical to 65°. This plunge needs to be confirmed either by mining or more closely spaced drilling. At this point it is important to be aware that there may be pronounced local plunge changes and design of any prospective stopes should contain flexibility to deal with both shallow and variable plunges.

Local and regional structures appear to play an important part in controlling and defining gold zones. Because the entire original volcanic package has been subject to major overprinting by alteration and deformation events, there is a complex pattern of major and minor faulting with, as yet, indeterminate displacements. Some major faults or shears contain significant amounts of graphite which can make ground control during mining very difficult.

Some gold-bearing zones occur adjacent to major shears. For example, two attempts at different locations were made to mine along a graphitic shear on the 260N/325N Zone from the 00 X/C. Both attempts had to be abandoned due to ground control issues. A similar problem occurred in the 1995 bulk sampling programme on the 100N Zone from the 600W X/C.

Although most shearing appears to be east-west, sub-parallel to the general strike, there are strong signs that there may also be minor north-south trending shears resulting in offsets of the gold zones. One such possible situation occurs at the eastern end of the 140N Stope off the 00 X/C. The degree to which there are important north-south offsets has yet to be determined, but there is sufficient evidence at the moment to raise an awareness.

The mining of the 140N Stope off 00 X/C encountered slips sub-parallel to the target mineralized zone which in turn determined the stope width limits based on ground control issues. In order to keep dilution to a minimum, the eventual choice of mining method must be able to deal with ground control around such slips or shears.

Although there is very limited geotechnical work in the historical record of the project, in general, ground conditions encountered for mining are considered very good. Only in specific areas of graphitic faults and at overhanging brows has screening been required for ground control.

Although drill core logging has noted many faults and shears, a comprehensive interpretation of these observations has yet to be completed. This issue is further discussed in the Recommendations section of this report.

19.2. Diamond Drilling Strategy

Diamond drilling, both from underground and from surface, has been an important tool for the delineation of prospective gold-zones and the estimation of mineral resources. Drilling has confirmed gold enriched zones to depths of at least 5560 feet and open in all directions.

Drill testing of the Property has been limited by the availability of suitable drill stations within the underground workings which is a function of the extent of the underground workings. As a consequence, except for the 2250 Level, the area immediately adjacent to the shaft has been tested at a much greater density than areas even a few hundred feet along strike from the shaft location. Many holes from surface and from underground stations have been in excess of 2000 feet long which gives rise to hole wandering even when using controlled drilling techniques.

19.2.1. Deep Drilling

A number of deep holes have been drilled as a series of wedged holes from an initial pilot hole. This provides multiple drill tests within a semi-controlled distance of the pilot. There is no other practical way to provide close spaced drill density of deep gold zones.

The objective of deep drilling is to provide insight into the global potential for gold zones on the Property. A long term strategy for the development of the Property can then be framed into which shorter term exploration and development can be placed. That is, wide spaced deep drilling provides the overall geologic and economic scope to support evaluating the risks involved in the detailed exploration and development in specific high priority areas.

In the opinion of the author, if a specific area within the defined prospective geology can be demonstrated to be economically mineable by future work, then the deep drilling results give excellent encouragement that economic mining can be extended to a much larger area. This is an important conclusion since it provides the support for concentrating future work on making specific areas ready for production with good confidence that, if successful, there is ample “room to grow”.

It is the author’s opinion that continued deep drilling below the 2250 Level is not needed at this time since continuity of mineralized potential similar to that defined above the 2250 Level has been adequately demonstrated.

19.2.2. Definition Drilling

The only location within the mine in which a systematic approach to definition drilling has been applied is on the 2250 Level. Drill stations were established at 100 foot spacings along the main east-west drift over a total strike extent of 3000 feet. The 2250 Level main drift is located south of the prospective gold-bearing geology.

Fans of 7 drill holes with nominal dips at the collar of +52°, +40°, +22°, 0°, -22°, -40° and -52° have been drilled from each station. Holes are typically 600 to 700 feet in length.

Gold-bearing zones have been intersected at distances north of the drift varying from a few feet to 600 feet.

This drill pattern provides a drill density varying from 100 x 50 ft centres at 150 ft from the hole collars to 100 x 200 ft centres at 500 ft from the collars.

Mining along gold-bearing zones has demonstrated from chip sampling and bulk sampling that the density of drilling provided by the approach used on the 2250 Level provides adequate detail for the estimation of Indicated mineral resources.

In examining the sampling data available from McGarry and based on experience at five other narrow vein gold mines, the author concludes that a drilling strategy that will provide a pattern in the order of 25 x 25 ft centres or better will be required for stope definition at McGarry. Drilling a pattern of this density from a footwall drift such as the 2250 main drift, is not considered practical or economically feasible.

A strategic approach to stope definition was proposed for the Scoping Study undertaken by Python Mining Consultants (“Python”) (see Appendix A). In this approach, a main haulage/access drift is established along strike and in the centre of prospective geology on each main level. From this drift, a pattern of short holes (200 ft) can be drilled to give an initial effective 50 x 50 ft pattern that can be closed down to 25 x 25 ft as warranted. (This strategic approach is illustrated in Figure 27 and Figure 28, pages 66 and 67.)

19.3. Test Mining Results

Bulk sampling programmes were completed on the 1650, 2050 and 2250 Levels in the period 1995 to 1997. Mining design for the extraction of these samples did not take into account the transition of the openings into production stopes. The sample locations on the 1650 and 2050 Levels have indicated potential for the establishment of production stopes. From the 1997 bulk sampling programme results, the 2250 Level locations appear to have zones of lower grade gold. There were logistical problems with control of sample material that resulted in mixing so that the eventual milling results are not conclusive on a location by location basis. The actual chip sampling results in the 260N zone were actually quite encouraging which was part of the basis for the follow up 2008 test stoping on the zone.

The bulk sampling programme in 2007-2008 included a component that was designed to test potential production techniques as might be applied to stopes at McGarry. Although it is unlikely that a single stoping technique will be applicable to all situations at McGarry, it is considered that shrinkage will be a dominant method. Shrinkage allows the best opportunity for lift-by-lift control of stope dimensions thereby providing the best opportunity for dilution control. Shrinkage is also well suited to narrow stopes as are likely to dominate at McGarry – that is, stopes with design horizontal widths of 5 feet.

The 140N stope on the 2250 Level was established with a sill drift on the level directly from the 00 X/C (see Figure 20, page 37). Two lifts above the sill drift were taken using a combination of uppers and breasting. A complication with this stope was that it crossed the 00 X/C which resulted in non-routine dilution at the intersection. The stope was emptied and readied for placement of a stull floor and chutes above the sill level. The design width for the stope was 5 ft, however, slip faces on both walls forced a wider final width. Table 11, page 43, shows a comparison between the grade as determined from the bulk sampling protocol and from in-stope chip sampling. This comparison indicates an unplanned dilution of 20 to 25% which is consistent with an extra 0.75 ft of material on either side of the stope; that is, a final as-mined width of 6.5 feet instead of the planned 5 feet. The 140N stope is located in a zone of pyritic mudstone type mineralization.

Similar results were obtained for the 260N stope on the 2250 Level even though the recovered grades are lower than for the 140N stope. The 260N stope started from a sill drift established in the 1990's which was wider than the interpreted outline of the target mineralization resulting in unplanned dilution at the brow. Two lifts above the sill drift were taken (see Figure 18, page 36). This stope is in "carbonate type" mineralization and experience with in-stope grade control suggests that this type of mineralization is very difficult to plan a stope pattern for, even from one lift to the next, since gold distribution is very erratic.

Experience in mining the two test stopes has demonstrated that grade control and stope design on a lift-by-lift basis will require very close geological supervision and quick turnarounds on in-stope sampling.

In-stope dilution control will be a very important factor in production mining design and practices. Dilution is always an important factor, but the minimum mining widths in the range of 5 ft anticipated for a majority of any stopes magnify the issue at McGarry. Of course, the final mine design will ensure a balance between all cost factors to give the lowest cost per ounce of gold sold.

The test stopes were designed with manways at the ends of the stopes in mineralization. The observed variations in mineralized zone outlines as discussed above in Section 19.1. strongly suggests that access raises for shrinkage stopes should be outside the stope with connecting dogholes and that they should be Alimak driven. This is the approach applied by Python in the Scoping Study.

19.3. Processing Options

There have been a number of tests on the recovery of gold from mineralized zones at McGarry. The test results have generally been very encouraging.

Conventional carbon leaching processes have shown gold recoveries approaching 95% with head grades in the order of 0.2 oz/t. Gold grains appear to be freed at grinds of -200 mesh which is very encouraging.

The gold mineralization is associated with pyrite and other minor sulphides including chalcopyrite and arsenopyrite but test work has shown that neither the tailings nor waste rock is acid generating. Additional work is required on long term leaching of heavy metals.

Additional test work has shown that gravity or gravity/flotation processes can recover 44 to 65% of the contained gold in 6 to 16% of the feed tonnage, respectively.

The preliminary process results lead to several important conclusions:

1) There are several existing processing plants within 100 km of the McGarry site whose owners have shown an interest in considering custom milling contracts. This means that there is no immediate requirement for a producing operation at McGarry to finance and build a mill at the mine site until it can be proven that there is an ore body that can sustain such an investment. There are two costs to consider when evaluating custom milling options: the tolling charges and the trucking costs.

2) Establishment of a small gravity based processing plant at the site can be considered at an early point in any production plan. This option could provide an efficient upgraded product for secondary processing elsewhere on a toll basis. The gravity plant tailings could be stored on site for final recovery through either a full leach plant to be built at a later date or in batches by heap leaching.

19.4. Mineral Resources

The current estimate of Mineral Resources on the Property is in very good agreement with the previous estimate dated 6 June 2004 (Carmichael, S.J., June 6, 2004). The 2004 estimate included only Indicated Mineral Resources down to an elevation of 2600 feet with no top-cut applied. The current estimate is based on a fresh interpretation of gold zones independent from the previous interpretation. The author (Andersen) tested the results from the two estimates over regions in which direct comparison is possible and found them to be well within a variance envelope of less than 5% for both tons and grade. This test adds confidence to both estimates.

The table Mineral Resource estimate is summarized in Table 17 on page 63.

The 14% increase in the tons and contained ounces in the current Indicated Mineral Resource estimate over the 2004 estimate is in line with the 17% increase in the drill footage available for the revised estimate.

The estimate of an Inferred Mineral Resource is new to the current technical report. Although the grade estimated for the Inferred Resources is lower than that for the Indicated Resources the author considers this grade estimate to be ±25% considering the density of drill testing. It is the author’s opinion that the estimate of the Inferred Mineral Resource provides confidence in the larger potential of the Property beyond the region in which more intense drill hole spacing and mined openings exist.

19.5. Scoping Study

The investment in mining plant infrastructure at McGarry warrants ongoing evaluation for the potential of establishing a commercially viable venture at the Project. To this end, Python Mining Consultants was engaged to conduct a scoping level plan and model for mining production between the 2250 and 1250 Levels where underground access and other infrastructure already exists. The Scoping Study has a ±25% level of accuracy and is considered a “preliminary assessment”.

The full Python Report, including a complete description of parameters and assumptions, is attached as Appendix A to this technical report.

The mine development strategy discussed in the Python Report outlines a staged approach that sequences the exploration/ore delineation and bulk sampling phases that follow logically from all the previous work on the Property. In addition there is a component that moves potential stoping areas into production quickly at low risk and low incremental cost. The strategy takes full advantage of the “near production ready” infrastructure that already exists above the 2250 Level.

It is fully expected that prior to the completion of the mining modelled in the Python Report that the Project will expand to explore and develop other mining areas on the Property. Other mining areas could include both the prospective geology from the 1250 Level to surface and that from the 2250 Level down to at least 5600 feet below surface. In the case of expanding production levels below the 2250 Level, a deepening of the shaft will be required. The scope of the Python Report did not include considerations for on-going mining beyond the 1250 to 2250 Level interval. Once mining has been successfully established on the 2250, 2050 and 1850 Levels, planning for the next phase will have to be undertaken. The successful establishment of commercial mining on the Property cannot be assured at this time.

The part of the mine targeted for exploitation in the Python Report has, in the author's (Andersen) opinion, been under-explored in context of the level of exploration below the 2250 Level where there is no ready access. As a consequence, a large area of the prospective geology has not been tested by diamond drilling. Effective drill testing of the gaps at the density required (see Section 19.2.2. Definition Drilling, above) is not practical from existing drill platforms.

The 1995-1997 bulk sampling results on the 1650 and 2050 Levels support the conclusion that the area is now the top priority location for continued development of the Property and that gold-bearing zones with grades averaging in the 0.2+ oz/t range can be located. In the opinion of the author, it is reasonable to assume that the untested areas between the 1250 and 2250 Levels host gold-bearing zones similar to the zones discovered in the tested areas, although this has not been proved and actual results of exploration in the area may not bear out this expectation. Further, only a narrow corridor near the shaft has been well tested by diamond drilling and both bulk samples were taken from within this corridor.

During the 2007-2008 programme, 9 diamond drill holes were drilled into the exploration gap west of the shaft and below the 1250 Level. Of these holes, 4 holes were either too short or did not curve enough for the hole to enter the target zone. Of the remaining 5 holes, 4 encountered gold-bearing zones and only 1 failed to intersect a gold-bearing zone (see Table 8, page 29).

The Python Report outlines a strategy that includes a 7 month, pre-production period costed at $4.1 million, followed by a 12 month production period at 350 tpd and a subsequent 36 month period at 600 tpd until ore in the area is exhausted.

Assuming 1,000 aggregate feet of mineable gold zones per 200 ft spaced level; average diluted grades of 0.19 oz/t; final payable recoveries of 85%; and gold prices of $C 800 per oz, a simple economic model indicates a potential positive cash flow of $6 million during the first year of production following the 7 month pre-production period and a positive cash flow of $35 million during the next 36 months. It must be emphasized that Python Scoping Study does not meet the test of a pre-feasibility level study. Nevertheless, in the author's opinion, the Scoping Study does indicate a level of robustness in the project that supports continued advancement of the project.

The model outlined above is based on expectations that may not be achieved since parts of the subject area have yet to be fully explored. As a result, actual results may vary substantially from expectations. It is therefore, important to evaluate results on an ongoing basis to limit risks.

It is also the author's conclusion, that following the strategy set out in the Python Report is the most effective logical progression of the advancement of the project towards profitability with the lowest exposure to risk. This strategy is the first stage towards development of the gold potential of the entire Property.

19.6. Adequacy of Work

It is the author's opinion that the work programme and its results as reported in this technical report met the objectives of the work as set out and as stated in the Introduction on page 1.

20. Recommendations

It is recommended that the McGarry Project be advanced to the next stage of mining development following the strategy as outlined in the Python Scoping Study (Appendix A) and as discussed above in Section 18.1. The strategy is to establish main haulage/access drifts on levels spaced vertically 200 ft apart from the 2250 Level to the 1250 Level beginning with the lowest two levels: 2250 and 2050. Choosing these levels for the next stage of work will maximize the cost benefit of already existing underground infrastructure.

Concurrent with mining advance, close spaced diamond drilling at 50 to 25 ft centres to establish the outlines of gold zones and parameters for stope definition is recommended from the main access drifts. The main drifts are recommended to extend east-west over the entire strike length of the active part of the Property, that is, from about 1500 E to 2500 W. The drifts should be established along the centre of the prospective geologic package to minimize diamond drill footages and connecting cross cuts to each stoping area. The 2250 Level is recommended to continue to be developed with mobile equipment, but all future levels should be established with rail based haulage to reduce ventilation requirements and to keep development headings to minimum size.

As gold-bearing zones of sufficient grade and tonnage are located by the detailed drilling, it is recommended to develop them for production by shrinkage or other suitable narrow vein methods. It is expected that sufficient stopes will have been developed to sustain production at 350 tpd after about 7 months of pre-production activity. There is considerable uncertainty in this assumption since the exact location, size, grade and distribution of potential stopes cannot be predicted at this time. The author has concluded on the balance of all available evidence that there is sufficient encouragement to warrant the strategy described above although there can be no assurance that the expected economic zones will be found.

In addition to the work outlined by Python, some necessary infrastructure will be required in preparation for increased mining activity. This work includes, completing the ventilation/escapeway raise system from 2250 Level to the 1250 Level, outfitting the ventilation borehole from surface to the 1250 Level with escapeway ladders and adding a second skip and dump mechanism in the headframe. Future considerations, once economic mining has been proved, will include a better system for coarse ore handing including a coarse ore bin either attached to the headframe or standalone. The control system for the hoist motor is of an old design and replacement by modern solid state controls requires further investigation in order to increase power consumption efficiency and future reliability. Construction of expanded office, drill core logging and warehouse facilities on surface is recommended since these are very limited at the present. Additional professional staff to that provided for in the Python Study are also recommended including a mining engineer and purchasing/warehouse agent. An updated Closure Plan will also have to be completed and it is recommended that the process for this update be advanced as quickly as possible.

Although the Gemcom database is now well established, there remains work to seek out and add all historical data from the Property including surface diamond drilling of zones to the south near Larder Lake. Substantial progress has been made to integrate the historical geological descriptions with current concepts; however, there remains work to fully interpret the data using the Gemcom platform. In particular, the interpretation of structural information remains to be done. A programme of petrographic and mineralogical investigation should also be undertaken to better understand the geological setting of the deposit and the nature and characteristics of the gold mineralization.

Establishing metallurgical characteristics of the mineralized rock requires further investigation focussing on gravity/flotation options and environmental characteristics. The option of batch heap leaching of potential gravity/flotation tails during summer months requires investigation.

A budget for the recommended work is outlined below in Table 18. This budget contemplates an incremental approach to the ongoing development of the McGarry Project. At the end of the first year, a review of performance compared to budgeted will be required with appropriate adjustments to a 3-year plan. In particular, the demonstration of the project to develop and produce from sufficient shrinkage-type stopes will have to be evaluated in terms of ongoing operations. The estimated timing of revenues expected from mining operations has been delayed from the schedule presented by Python, to more conservatively reflect realistic delays in realizing revenue from gold potentially produced by custom milling. The estimate of potential revenue cannot be relied on since it assumes that gold zones suitable for stope production will be developed within the timing and at the frequency as described in the Python Scoping Study. Nevertheless, the author, has concluded that the basis for the assumptions is realistic and not to indicate this potential, within a ±25% range would be misleading.

		Expenditure Category					
Year	Period	Development (000's)	Production (000's)	Projects (000's)	Contingency (000's)	Total Expenditures (000's)	Potential Revenue (000's) ±25%
1	Q1	$ 1,090	$	$ 100	$ 179	$ 1,369	$
1	Q2	2,227		100	349	2,676	
1	Q3	2,431	1,488	100	603	4,622	
1	Q4	2,229	2,245	100	686	5,260	2,557
1	Total	7,977	3,733	400	1,817	13,927	2,557
2	Total	4,576	10,485		2,259	17,320	18,872
3	Total	2,612	13,900		2,477	18,989	29,220
1 - 3	Total	$ 15,165	$ 28,118	$ 400	$ 6,552	$ 50,235	$ 50,649

All numbers in $C '000's

Assumptions:
- First 2 months, only 40% of broken muck available for hoisting, remainder tied up in shrinkage stopes until stopes completed
- First 2 months of increased production rate to 600 tpd, only 450 tpd available for hoisting, remainder tied up in shrinkage stopes until stopes completed
- Payment for gold recovered by custom mill 2 months after start of treatment
- Fully diluted grade of material delivered to custom mill = 0.19 oz/t
- Gold Price = $C 800 / oz
- Metallurgical recovery 89% less 4% NSR = Net 85%
- Projects includes capital items not in Python Scoping Report
- Contingency = 15%
- Production Rate 350 tpd months 8 to 19, inclusive
- Production Rate 600 tpd months 20 to 36, inclusive
- Costs based on Python Scoping Study (±25% range of accuracy)

Table 18 - Recommended Budget for First 3 Years of McGarry Mine Development

The expenditure estimates are based on the work presented by Python in the Scoping Study (Appendix A). The Scoping Study is not a feasibility study and is stated to have a confidence range of ±25%. There is no contingency percentage factor built into the Python estimates – each expenditure was estimated individually on a conservative basis instead. The budget presented above includes an additional 15% contingency factor as a conservative measure to cover the costs of unanticipated requirements.

This budget shows a breakeven for the Project on a cash basis after about three years. If mining activity progresses in line with the assumptions contained in the Python Scoping Study, then completion of mining activity between the 2250 and 1250 Levels will not occur until about month 55 and all potential revenues would not be in hand until month 57, or so. Considering the state of the geologic knowledge of the Property, it is reasonable to expect that actual mining and production activity, costs and revenues will vary from those presented in this report. Ongoing evaluation of development targets and costs will be required.

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22. Signature Page

Technical Report and Mineral Resource Estimate, McGarry Project

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Signed at Kirkland Lake, ON
8 April 2009

23. Certificate of Author

I, Erik Andersen, P.Eng., residing at B1750 Thorah 8th Line, RR 3, Beaverton, Ontario, Canada, do hereby certify that:

1. I am employed full time by Armistice Resources Ltd, 6 Al Wende Ave, Kirkland Lake, Ontario continuously since May 2007.
2. I am a member of the Board of Directors of Armistice Resources Corp. continuously since October 2005.
3. I hold a Batchelor of Engineering (Geology) degree from the University of Saskatchewan, Saskatoon, Saskatchewan granted in 1968.
4. I am a registered member of Professional Engineers Ontario in continuous good standing since August 1970 (Licence number 880013).
5. I have over 40 years experience in mineral exploration and mine operations management.
6. As Vice President of Armistice Resources Corp, I have directed and supervised all work at the McGarry Project since May 2007. I have worked full time on the McGarry Project during this period usually visiting the Property daily. I most recently visited the Property in the first week of April 2009.
7. I am the author of the National Instrument 43-101 compliant report “Technical Report and Mineral Resource Estimate, McGarry Project” dated 8 April 2009.
8. I have not had any involvement with the McGarry Project prior to my association with Armistice Resources Corp. in 2005.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report, the omission to disclose which makes the Technical Report and Mineral Resource Estimate misleading.
10. I am not independent of Armistice Resources Corp.
11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report and Mineral Resource Estimate has been prepared in compliance with that regulation and form.
12. I authorize Armistice Resources Corp. to use this technical report for any lawful purpose. In particular, the report may be filed and my name may be used in the fulfillment of relevant reporting, disclosure and publishing requirements of any stock exchange or regulatory authority that recognizes my professional qualification including electronic publication in the public company files on their websites accessible by the public.

Dated this 8th day of April, 2009 at Kirkland Lake, Onta

Erik Andersen, P.Eng.

REGISTERED PROFESSIONAL ENGINEER
E. O. ANDERSEN
PROVINCE OF ONTARIO

Appendix A

Scoping Study – McGarry Mine Project
Kirkland lake, Ontario

Martin Drennan, B.Eng. P.Eng
Python Mining Consultants
February 4, 2009

44 pages including appendices.

PYTHON MINING CONSULTANTS

SCOPING STUDY – McGARRY MINE PROJECT

KIRKLAND LAKE, ONTARIO

Martin Drennan B. Eng, P. Eng

Python Mining Consultants Inc.

February 4, 2009

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Introduction

Armistice Resources Corp. (Armistice) retained the services of Python Mining Consultants Inc. (PMC) to development a preliminary economic assessment (scoping level) to review the economic potential of the McGarry Project above the 2250 level. Armistice has conducted bulk samples on the 1650, 2050 and 2250 levels, and drill programs covering elevations from surface to 5500 below surface. This report outlines a development and stoping strategy with a prospective geological package as defined by Armistice.

Assumptions

- All technical source data has been supplied by Armistice and is correct. A site visit was completed, however, data auditing was not performed. This is assumed to be the responsibility of Armistice.
- In coordination with Armistice, this study has a confidence range of ±25%.
- Level names are based on elevation in feet.
- Gold bearing areas are extrapolated based on diamond drill holes and on sill information.
- Stope areas are based on gold bearing zone patterns identified.
- Gold bearing zones have been physically tested. Results of this have not been disclosed by Armistice to PMC.
- The development method of gold bearing zones is based on assumed structures.
- Economic analysis parameters are: mining recovery is 90% and milling recovery is 85%.
- Scheduling parameters are:
    - Dilution is assumed within the stope shells (i.e. dilution is accounted; not as a fixed percentage, but in the stope dimensions – minimum stoping width of 6ft and 1ft of dilution either side).
    - Development and production rates were discussed internally amongst Armistice, Paul Whelan Mining Contractors, and PMC.
    - The McGarry project is a narrow-veined gold deposit.
    - Shrinkage and longhole mining, and a hybrid combination of these methods, are applicable to this project.

- Gold prices are assumed to range from $600 to $1100 at $50 intervals.
- All values are Canadian Dollars (CAD).
- There are currently only inferred resources at McGarry.
- Attached is a directive letter from Armistice with details of parameters and assumptions (see – Appendix B: Scoping Study – McGarry Project; Oct 1, 2008)

Assumed Mining Resource

The gold bearing estimates are extrapolated from gold bearing resource information in areas that have been mined at the 2250 and the 2050 levels. Both these areas have estimated gold bearing resource information. Two bulk samples have been completed and graded diamond drill hole estimations have been tested, resulting in reasonable confirmation of the sample results and the bulk sample results. In this instance, there are no minable reserves. However, there is an assumed minable resource for presentation, which can be collaborated and brought to conclusion with the existing diamond drilling data currently being analyzed and logged by Armistice personnel.

The mineable resource was defined by selecting only gold bearing resources above the cut-off grade from the block model, and then by removing any outliers. A block model was created for testing purposes, and represented any significant resource that appeared consistent and continuous within the range of the grade.

Studies were conducted to determine the optimal resource tonnage and grade. Analysis involved sensitivity to cut-off grade, high cutting factor for diamond drill hole database inclusions and having two phases of mining with an initial production rate of 350 tons per day (TPD), as well as a second advanced phase at 600 TPD with different cut-off grades. The results of these studies have been summarized in the following tables.

A table of the total development is included in Appendix A – Total Development, that PMC completed using MineSched software.

Mine Design: Existing and Future

The ore body is currently accessed via the McGarry shaft. The shaft has three compartments, two hoisting compartments that are 6 ft by 6 ft and one main way/service compartment that is 2 ft 10 in by 6 ft from surface to the 1250 level, and 6 ft by 6 ft below the 1250 level. The shaft extends to the 2250 level and the bottom of the sump is 2290. The mine was developed in two phases.

Phase One:

- A shaft was sunk slightly below the 1250 level.
- The 1250 level was established.

Phase Two:

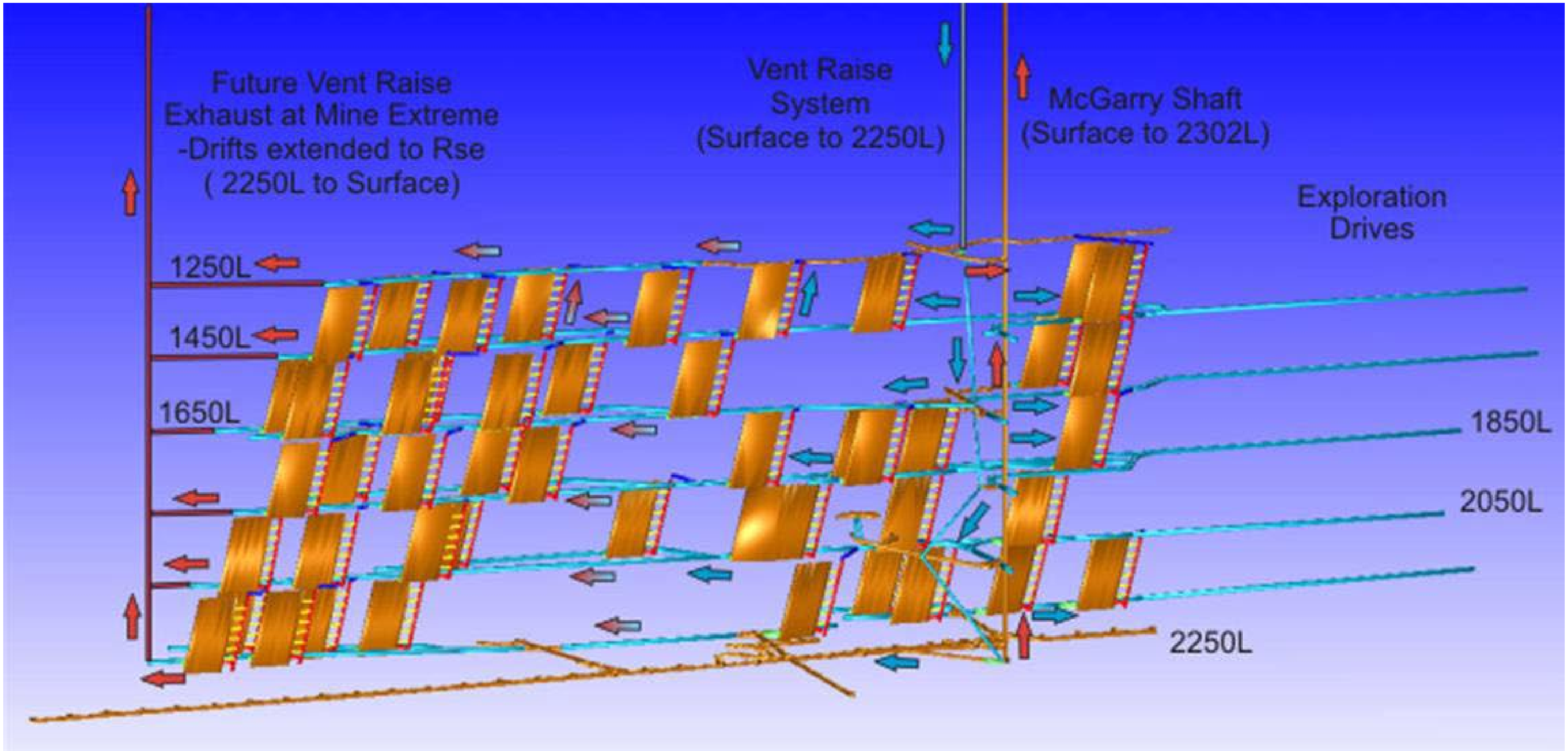
- Shaft sunk from 1250 to the 2290 elevation.
- Vent raise from the 1250 level to the surface (raise bored).
- Levels at 1250, 1650 (minimal level development – bulk sample taken in the 1990's), 1850 (station only), 2050, and 2250. The 2250 level is the most developed level within the mine and is entirely trackless.

The primary infrastructure for the operation has not yet been completed. Design updates exist for the completion of the 2250 to 1250 ventilation/escape way/rock pass system and the equipping of the bored ventilation raise for use as an emergency escapeway. Work has been started on the ventilation raise from 2250 although all underground work at the site is currently suspended until a review for the further development of the entire project is completed. The initial development design focused on a system to extend the ventilation, creating a ventilation loop throughout the mine in order to ensure that the mechanical ventilation assistance is reduced. The design incorporates a series of conventional raises driven at a maximum angel of 49°. The raises will connect all the levels that will initially allow the development to use the raise as an exhaust/escape system. As the infrastructure within the mine evolves, the ventilation system can be converted to a rock pass. Once waste development is completed, any economic material can be passed to a shaft pocket and taken to surface via the shaft. As the mine continues to evolve and developed, a permanent loading pocket can be established. See Figure 1.

Dogholes are 21 ft long and run until they contact the stoping area. The modeled stoping areas are relatively uniform. Therefore, the raises and the dogholes are relatively uniform. The dogholes are driven out of the footwall of the raise in most instances. However, in some circumstances, due to the configuration of the stoping area, as it occurs relative to the development, some dogholes do have to come out of the side of the raise. For more details, see Figure 2.

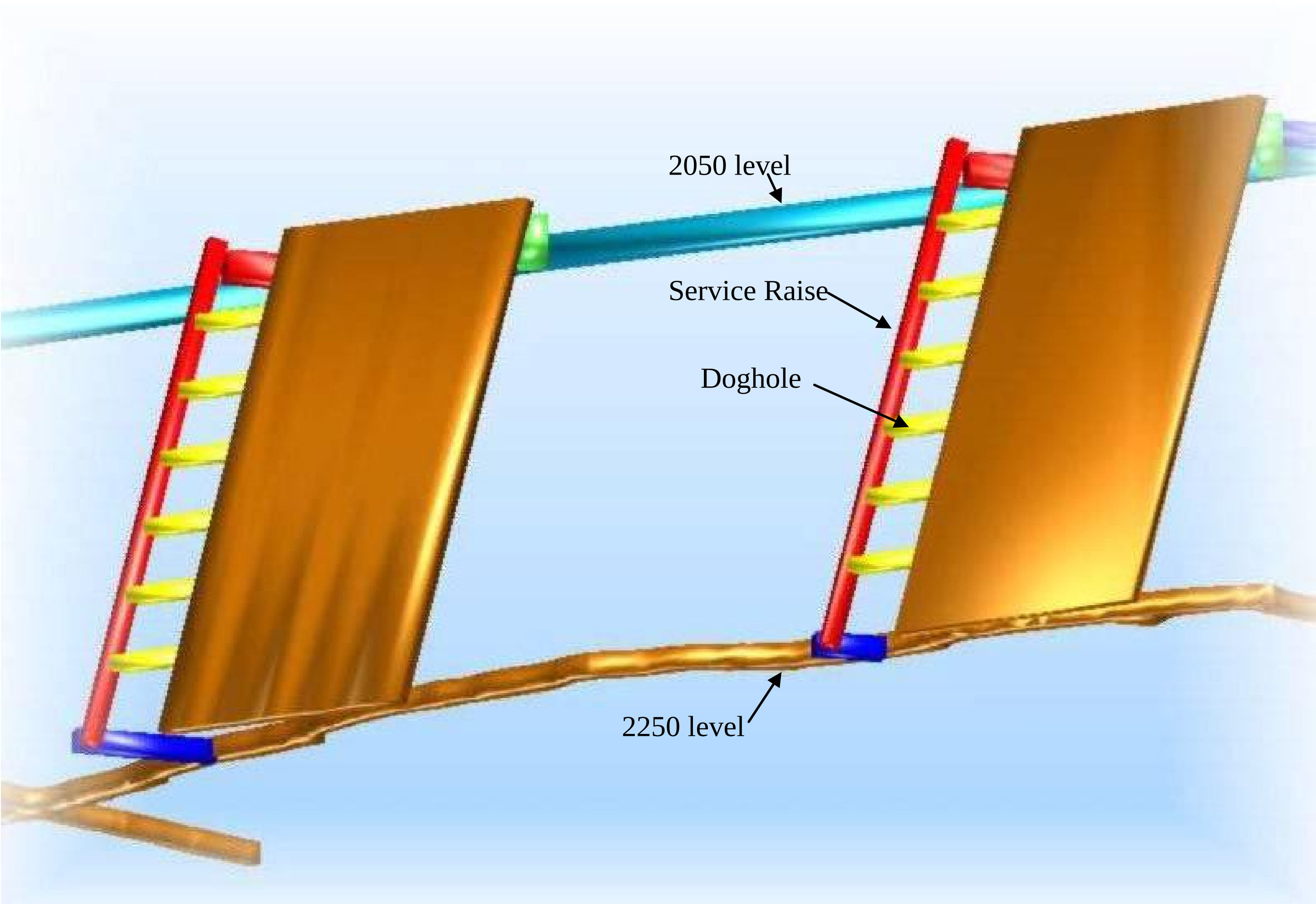
No geotechnical investigations have been conducted by PMC, and no geotechnical investigations are known at this time.

Figure 1: Ventilation Schematic



Red Arrow	**Exhaust**
Blue Arrow	**Fresh Air**
Blue to Red	**Transitional (Fresh Air to Exhaust)**

Figure 2: Example of Dogholes



Mining Method

The Mining methods considered for the McGarry project are shrinkage mining and longhole, as well as a combination of shrinkage and longhole.

Shrinkage mining is a common method for narrow-vein, high-grade ore deposits. McGarry Mine based on mining to date would not be considered a high-grade mine but does contain some pockets of high-grade ore. Shrinkage would be an excellent extraction method for the McGarry Project because the method provides the ability for inspection and evaluation of relatively small minable quantities allowing each production face to be sampled and "marked up" for the best recovery. Shrinkage allows dilution to be minimized and ground conditions to be controlled. Stope heights are limited to level heights and are projected to be 200 feet, which is a reasonably standard mining stope height for shrinkage.

The longhole method would be used in the wider, lower-grade areas. Longhole is less costly than shrinkage, but shrinkage gives better control on extraction and dilution. The McGarry Mine has some suitable areas for the longhole method and approach. Unfortunately, because of the 200 foot level frequency, longhole mining would require some modifications, such as the creation of sublevels in order to work between entire levels. Based on this, a sublevel frequency of 40 feet back to sill would be estimated as the maximum for the McGarry project.

The combination of shrinkage and longhole is proposed as a "hybrid mining method". This method has been used on other properties, such as Wesdome's Eagle River Mine when it was owned by River Gold Mines Limited. The method entails having 200 foot high stopes, using shrinkage mining to mine approximately the lower 100 feet of the stope, and then longholing the upper 100 feet. The upper 100ft longhole portion would have a 30 foot sill pillar, as well as sublevels (9 ft high). The sublevels would run the full length of the ore body, and be slightly wider than the ore structures to ensure that longhole drilling can be directly on the contact, as it is evaluated by the geologist. Vertically, from the sill of the subdrift to the back of the shrinkage stope, would be approximately 35 feet. The sill pillar would be 26 feet above the back of the subdrift. The sill pillar would be left in this method so that the shrinkage mining can occur on the level above while the development was underway. This approach would be used to recover any high-grade sill pillars, as they warranted recovery.

For costing purposes, shrinkage has been the only method used for costing analysis in the production schedule.

Equipment and Services

It is assumed that all equipment on site is owned by Armistice. Any additional equipment like drills and track rolling stock etc will be acquired on a lease to ownership basis. Services of a contractor will be labor based only. As a generality, equipment required for this mining will be trackless on 2250 level and track on all other remaining levels. Services will be utilized as currently established at the McGarry project with the exception of ventilation.

Scheduling

Development Schedule

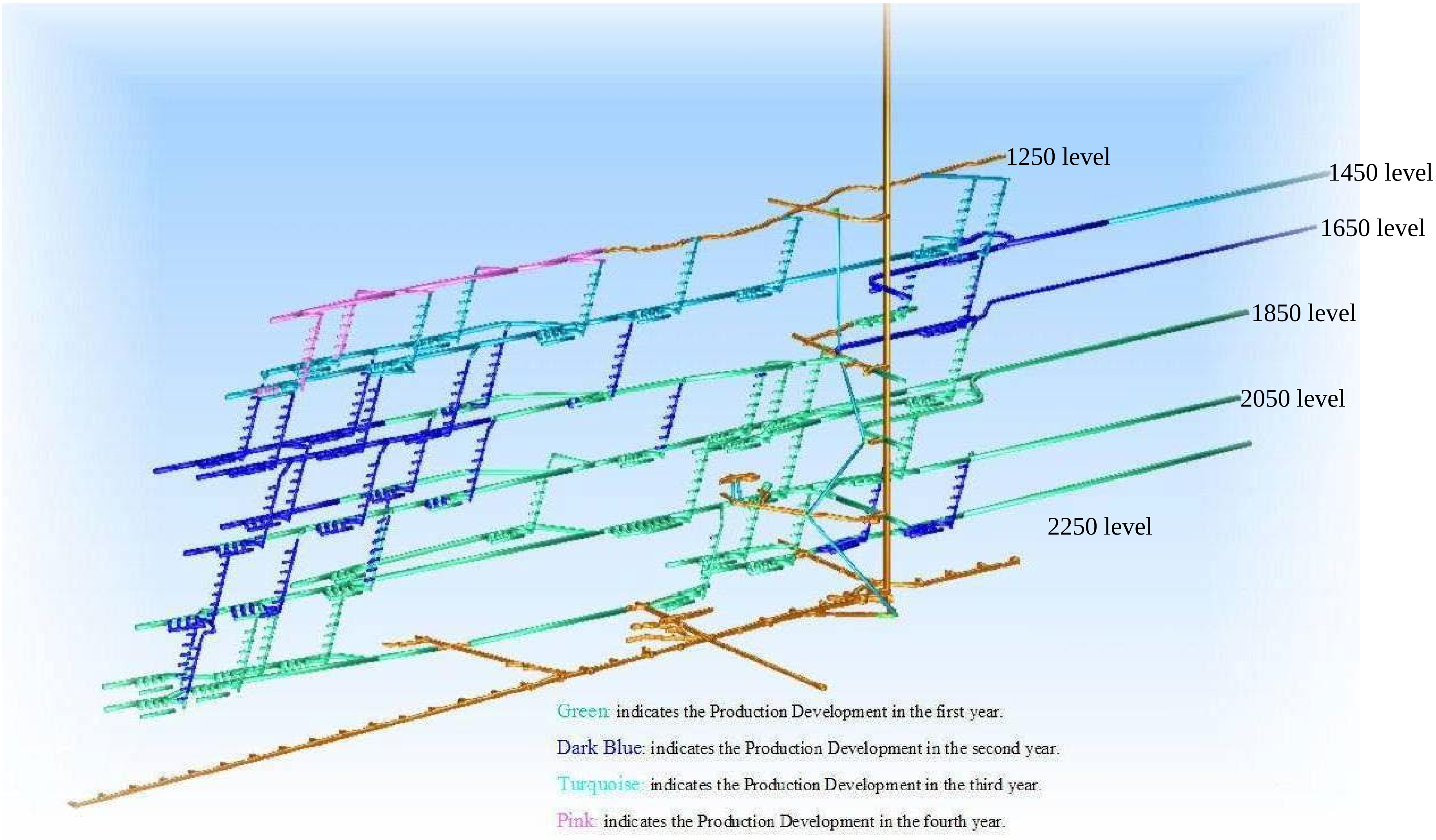
Development scheduling was completed with the help of Gemcoms Minesched software (for more information, see http://www.gemcomsoftware.com/products/MineSched). The assumed rates of cost were established by Armistice personnel. Scheduling was based on crew performance and costs based on 2007-2008 actual costs incurred. During this period, mine labor was supplied by Paul Whelan Mining Contractors based out of Kirkland Lake. The underground mine development schedule was initiated on the 2250 and the 2050 level. The 2250 level is already advanced in current development, and is the only trackless level in the mine. The development is completed in a logical manner starting from the bottom of the mine on the 2250 level and progressing upwards to the 1250 level.

The following are graphical representations of the development on a per annual basis, based on the legend provided.

For detailed and summary development schedules, see Appendix A.

Figure 3 outlines the progression of the development over the mine life.

Figure 3: Development Progression by Year



Production Schedule

There is no historical data specifically on continuous production of the McGarry Mine. Subsequently, all assumptions are based on previous experience with mines of similar size and type. The production rate during Year 1 has been assumed to be 350 TPD. All other years of production are assumed to be 600 TPD.

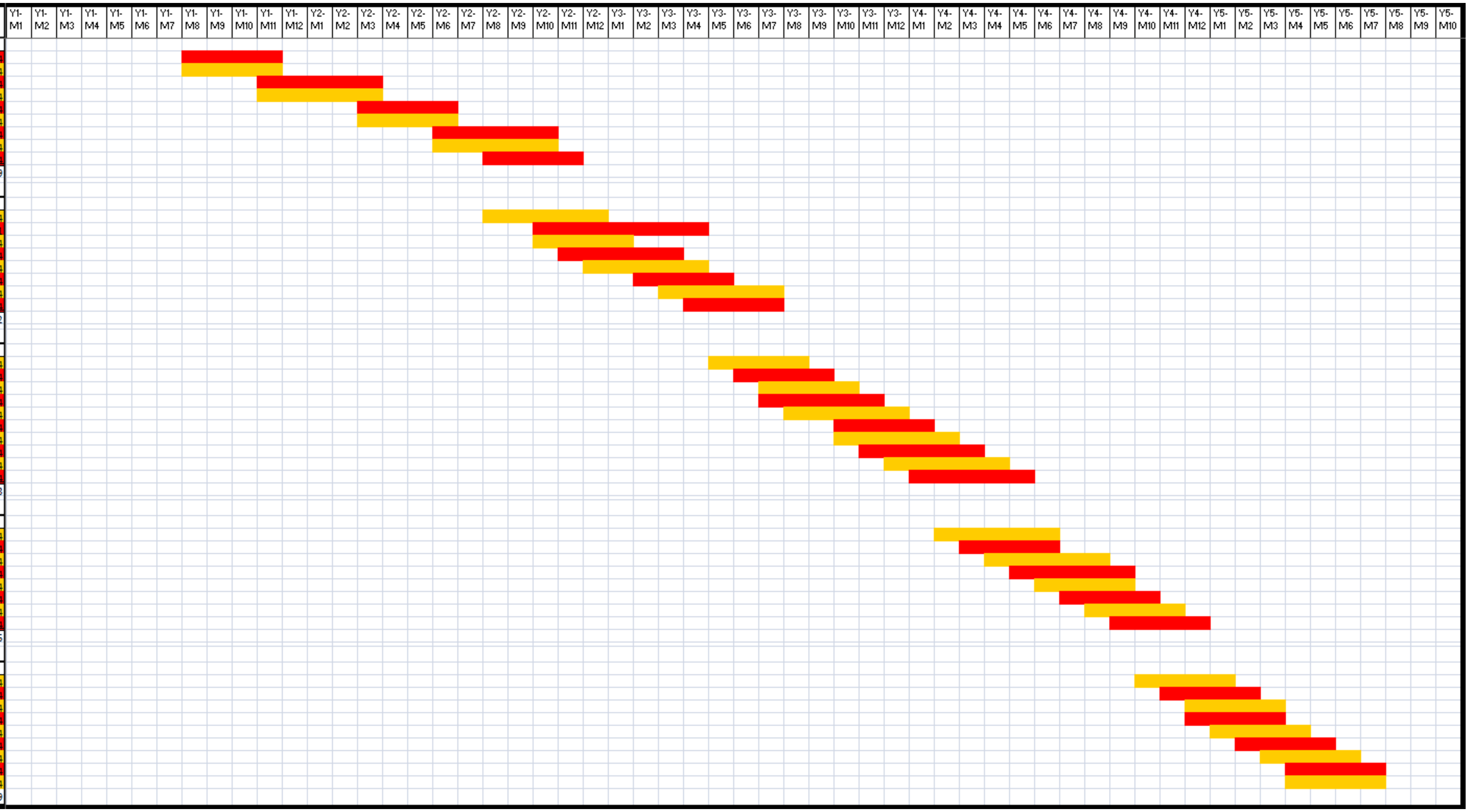
As noted earlier, stopes have been estimated due to the lack of diamond drill hole information, as well as the complexity of the gold bearing structure. Stopes are assumed to be reasonably regular in size and shape. Stope areas are based on diamond drilling holes and the 2250 level silling.

The development schedule is the precursor to the production schedule. The development schedule was used as a defining time component. Once all stope development was completed, it was assumed that the next stope would be available to begin the next production. Stope production was based on the availability of a stope crew. For the initial year of 350 TPD production, there are 4 crews available, and at 600 TPD, there are 8 crews available.

For the full production schedule, see Figure 4.

Figure 4: Production Schedule

Level 2250 (Trackless)		
Stope_Name	Volume (cu ft)	Tons
2250 st1	160.000.00	13.913.04
2250 st2	160.000.00	13.913.04
2250 st3	160.000.00	13.913.04
2250 st4	160.000.00	13.913.04
2250 st5	160.000.00	13.913.04
2250 st6	160.000.00	13.913.04
2250 st7	160.000.00	13.913.04
2250 st8	160.000.00	13.913.04
2250 st9	160.000.00	13.913.04
Total:	1.440.000.00	125.217.39
Level 2050 (Track)		
Stope_Name	Volume (cu ft)	Tons
2050 st1	160.000.00	13.913.04
2050 st2	303.877.32	26.424.11
2050 st3	160.000.00	13.913.04
2050 st4	160.000.00	13.913.04
2050 st5	160.000.00	13.913.04
2050 st6	160.000.00	13.913.04
2050 st7	160.000.00	13.913.04
2050 st8	160.000.00	13.913.04
Total:	1.423.877.32	123.815.42
Level 1850 (Track)		
Stope_Name	Volume (cu ft)	Tons
1850 st1	160.000.00	13.913.04
1850 st2	160.000.00	13.913.04
1850 st3	160.000.00	13.913.04
1850 st4	160.000.00	13.913.04
1850 st5	160.000.00	13.913.04
1850 st6	160.000.00	13.913.04
1850 st7	160.000.00	13.913.04
1850 st8	160.000.00	13.913.04
1850 st9	160.000.00	13.913.04
1850 st10	160.000.00	13.913.04
Total:	1.600.000.00	139.130.43
Level 1650 (Track)		
Stope_Name	Volume (cu ft)	Tons
1650 st1	160.000.00	13.913.04
1650 st2	160.000.00	13.913.04
1650 st3	160.000.00	13.913.04
1650 st4	160.000.00	13.913.04
1650 st5	160.000.00	13.913.04
1650 st6	160.000.00	13.913.04
1650 st7	160.000.00	13.913.04
1650 st8	160.000.00	13.913.04
Total:	1.280.000.00	111.304.35
Level 1450 (Track)		
Stope_Name	Volume (cu ft)	Tons
1450 st1	160.000.00	13.913.04
1450 st2	160.000.00	13.913.04
1450 st3	160.000.00	13.913.04
1450 st4	160.000.00	13.913.04
1450 st5	160.000.00	13.913.04
1450 st6	160.000.00	13.913.04
1450 st7	160.000.00	13.913.04
1450 st8	160.000.00	13.913.04
1450 st9	160.000.00	13.913.04
Total:	1,440,000.00	125,217.39



Economic Analysis

The preproduction phase of the McGarry project will occur during the first four months of the planned work. The total development cost of the preproduction period is $1.1 million, as outlined in the development schedule (see Appendix A). The Economic parameters and summaries for the McGarry Mine operation are found in Appendix B.

The duration of the development process in the McGarry Project is forecasted to be four years. An annual development cost summary is in Table 1. Initial development costs are the highest as preproduction and significant underground infrastructure needs to be completed up front. Development costs wane over time as infrastructure requirements diminish.

Table 1: Summary of Development Costs per Year of Operation

Year	Development Cost ($1000s)
1	7,977
2	4,576
3	2,612
4	589
Total: 15,754	

An annual production cost summary is in Table 2. Initial production costs are relatively low as development continues to create working areas. Once development progresses to allow continuous full production at 600 TPD, production costs level out to approximately $17 million annually. For more information on production and development costs over time, see Figure 6 and Table 3.

Table 2: Summary of Production Costs per Year of Operation

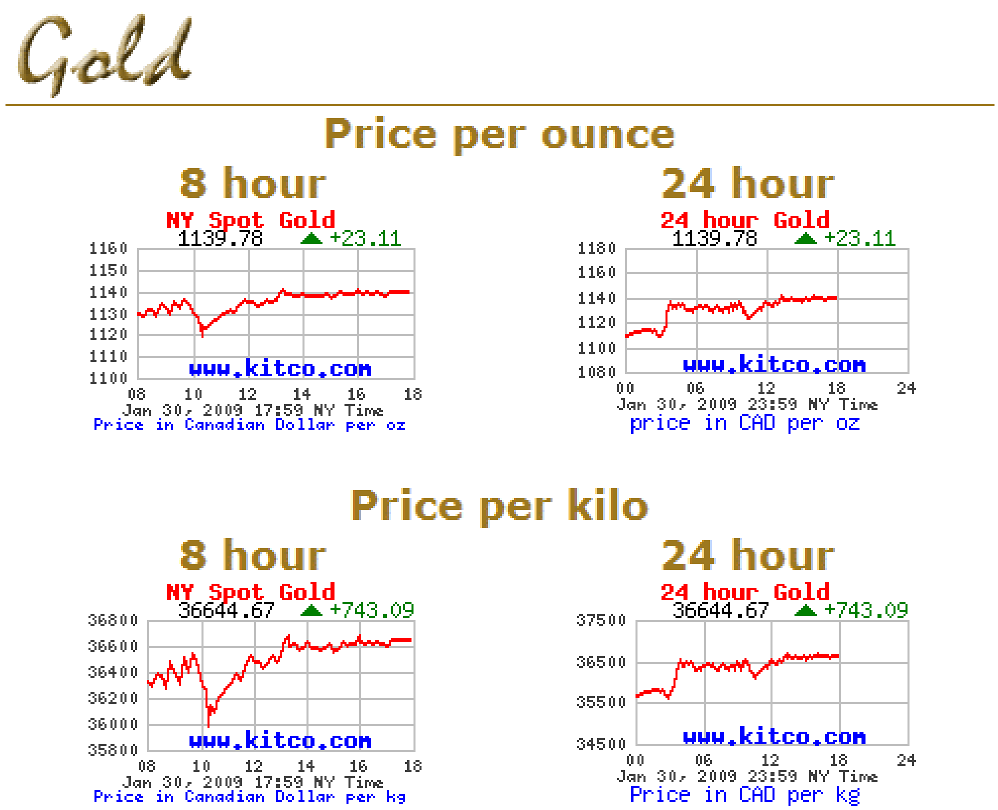
Year	Production Cost ($1000s)
1	3,733
2	10,485
3	13,900
4	14,090
5	7,784
Total: 49,992	

The economic model, based on the assumptions in its development for both initial production (350 TPD) and the targeted final (600 TPD) production, is linear. To be profitable, the initial production needs to have a diluted grade greater than 0.21 ounces per ton with a minimum assumed gold price of $650 per ounce. The initial production for the first year is assumed to be 350 TPD. The initial production period projected mill feed value is approximately $12,700,000, resulting in a relatively neutral profit – i.e. breakeven. If the first year of production can have an elevated production rate in excess of 350 TPD, the likelihood of being able to produce a profit increases.

The economic cut-off diluted grade (breakeven) for 600 TPD is 0.19 ounces per ton at $600 per ounce of gold. With improvements in grade and gold price, profitability increases. With a minimum 0.19 ounces per ton diluted grade and a gold price of $600 per ounce, mill feed value would be approximately $54,000,000 over 3 years. This is the breakeven point and the minimum requirements for the project to move forward. Gold prices are currently in the $1000 plus per ounce range.

For more information, see Figure 5 – or go to http://www.kitco.com/market/cad_charts.html. For this project, it is assumed that whatever diluted cut-off grade is selected, it is sustainable for the entire project operation.

Figure 5: Kitco Gold spot price – January 30, 2009



Assuming the market conditions continued for the four years it would take to mine the McGarry deposit, and the grade assumption of 0.19 ounces per ton diluted grade, the potential mill feed value for the initial production year would be approximately $17,800,000. The expenses for processing the initial production year is based on 350 days at 350 TPD and a cost of $93.87 per ton. This would result in a positive cash flow circa $6,000,000. If the diluted grade was unable to be achieved and a diluted grade of 0.14 ounces per ton was the realized grade (mill feed value @ $13.1M), then the initial year positive cash flow would be reduced to $1,500,000.

With the assumptions above ($1000 per ounce revenue and 0.19 ounces per ton diluted grade), the mill feed value for the 600 TPD period would be approximately $91,000,000. The operating expense during this period would be around (3 years x 350 days/year x 600 TPD x $78.42) $53,500,000 resulting in a projected profit of $35,000,000. With the conditions varied to $1000 per ounce and a diluted grade of 0.14 ounces per ton, the revenue would fall to $67,000,000 giving a projected profit potentially of $13,000,000. In any of the above instances there is a significant impact of diluted grade relative to profitability.

See Figure 7 and 8 for complete details.

Figure 6: Monthly Development and Production Costs

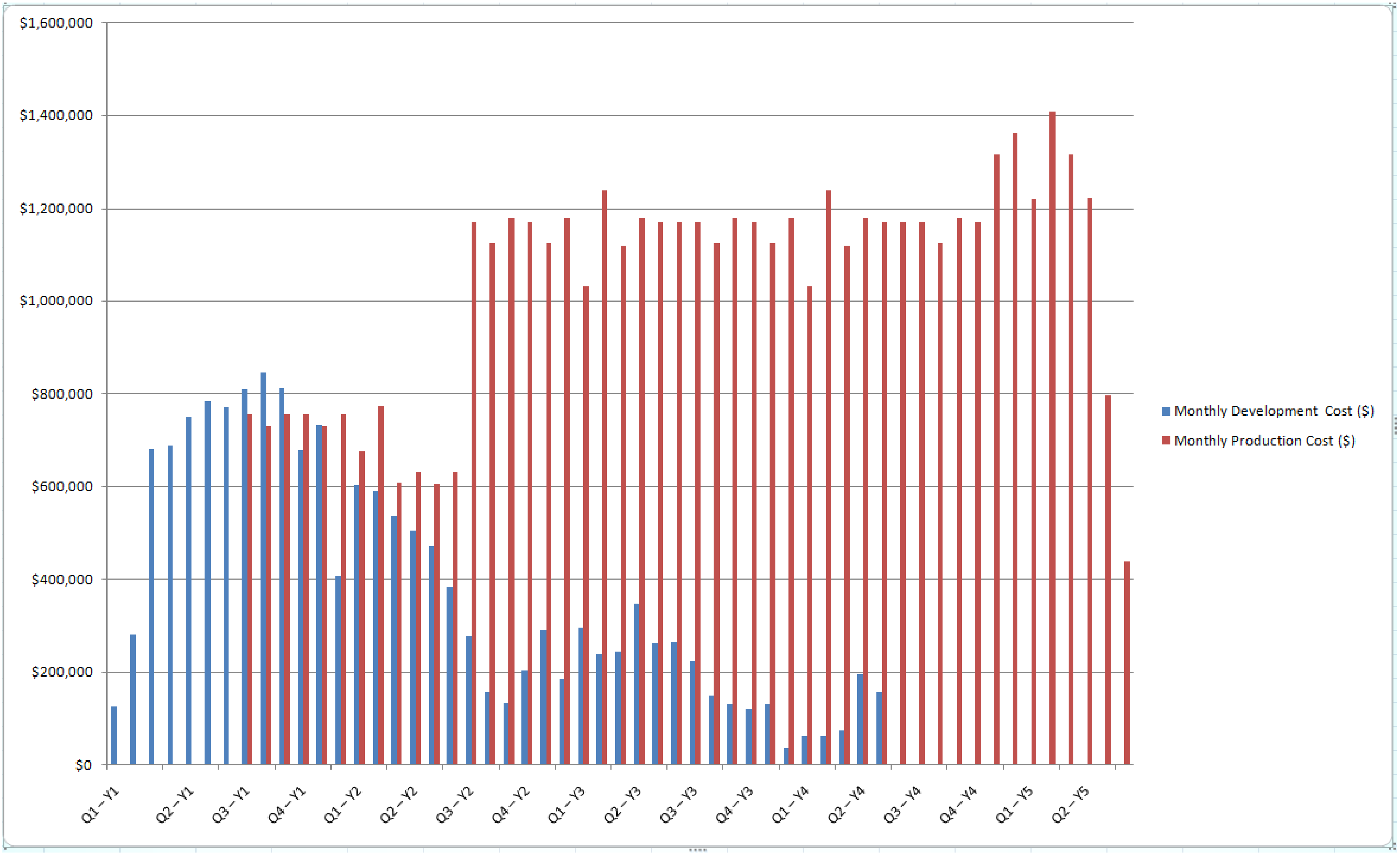


Table 3: Development and Production Cost Summary by Quarter

Preproduction

	Date	Development Cost ($)	Production Cost ($)
1		$ 126,000	$ -
2	Q1 – Y1	$ 282,000	$ -
3		$ 682,000	$ -
4		$ 690,000	$ -
5	Q2 – Y1	$ 752,000	$ -
6		$ 785,000	$ -
7		$ 772,000	$ -
8	Q3 – Y1	$ 811,000	$ 757,000
9		$ 848,000	$ 731,000
10		$ 815,000	$ 757,000
11	Q4 – Y1	$ 680,000	$ 757,000
12		$ 734,000	$ 731,000
13		$ 409,000	$ 757,000
14	Q1 – Y2	$ 604,000	$ 678, 000
15		$ 593,000	$ 777,000
16		$ 537,000	$ 611,000
17	Q2 – Y2	$ 507,000	$ 633,000
18		$ 473,000	$ 608,000
19		$ 386,000	$ 633,000
20	Q3 – Y2	$ 280,000	$ 1,175,000
21		$ 158,000	$ 1,128,000
22		$ 134,000	$ 1,182,000
23	Q4 – Y2	$ 203,000	$ 1,175,000
24		$ 292,000	$ 1,128,000
25		$ 185,000	$ 1,182,000
26	Q1 – Y3	$ 297,000	$ 1,034,000
27		$ 241,000	$ 1,242,000
28		$ 245,000	$ 1,122,000
29	Q2 – Y3	$ 350,000	$ 1,182,000
30		$ 264,000	$ 1,175,000
31		$ 267,000	$ 1,175,000
32	Q3 – Y3	$ 226,000	$ 1,175,000
33		$ 150,000	$ 1,128,000
34		$ 132,000	$ 1,182,000
35	Q4 – Y3	$ 122,000	$ 1,175,000
36		$ 133,000	$ 1,128,000
37		$ 36,000	$ 1,182,000
38	Q1 – Y4	$ 63,000	$ 1,034,000
39		$ 62,000	$ 1,242,000
40		$ 74,000	$ 1,122,000
41	Q2 – Y4	$ 196,000	$ 1,182,000
42		$ 158,000	$ 1,175,000
43		$ -	$ 1,175,000
44	Q3 - Y4	$ -	$ 1,175,000
45		$ -	$ 1,128,000
46		$ -	$ 1,182,000
47	Q4 – Y4	$ -	$ 1,175,000
48		$ -	$ 1,318,000
49		$ -	$ 1,365,000
50	Q1 – Y5	$ -	$ 1,224,000
51		$ -	$ 1,412,000
52		$ -	$ 1,318,000
53	Q2 – Y5	$ -	$ 1,226,000
54		$ -	$ 799,000
55	Q3 – Y5	$ -	$ 440,000
	Totals	**$ 15,754,000**	**$ 49,992,000**
	Total Development and Production	**$ 65,746,000**	

Figure 7: Revenue per Ounce Summary

	Cost limited
	Profitable

ton/day	Cost/ton
350	98.84
600	81.36

Total revenue per ounce for grade and price valuation.

Revenue per ounce for 350 tons per day

			Gold Price per Troy Ounce ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$55.08	$59.67	$64.26	$68.85	$73.44	$78.03	$82.62	$87.21	$91.80	$96.39	$100.98
0.13	0.9	0.85	$59.67	$64.64	$69.62	$74.59	$79.56	$84.53	$89.51	$94.48	$99.45	$104.42	$109.40
0.14	0.9	0.85	$64.26	$69.62	$74.97	$80.33	$85.68	$91.04	$96.39	$101.75	$107.10	$112.46	$117.81
0.15	0.9	0.85	$68.85	$74.59	$80.33	$86.06	$91.80	$97.54	$103.28	$109.01	$114.75	$120.49	$126.23
0.16	0.9	0.85	$73.44	$79.56	$85.68	$91.80	$97.92	$104.04	$110.16	$116.28	$122.40	$128.52	$134.64
0.17	0.9	0.85	$78.03	$84.53	$91.04	$97.54	$104.04	$110.54	$117.05	$123.55	$130.05	$136.55	$143.06
0.18	0.9	0.85	$82.62	$89.51	$96.39	$103.28	$110.16	$117.05	$123.93	$130.82	$137.70	$144.59	$151.47
0.19	0.9	0.85	$87.21	$94.48	$101.75	$109.01	$116.28	$123.55	$130.82	$138.08	$145.35	$152.62	$159.89
0.2	0.9	0.85	$91.80	$99.45	$107.10	$114.75	$122.40	$130.05	$137.70	$145.35	$153.00	$160.65	$168.30
0.21	0.9	0.85	$96.39	$104.42	$112.46	$120.49	$128.52	$136.55	$144.59	$152.62	$160.65	$168.68	$176.72

Revenue per ounce for 600 tons per day

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$55.08	$59.67	$64.26	$68.85	$73.44	$78.03	$82.62	$87.21	$91.80	$96.39	$100.98
0.13	0.9	0.85	$59.67	$64.64	$69.62	$74.59	$79.56	$84.53	$89.51	$94.48	$99.45	$104.42	$109.40
0.14	0.9	0.85	$64.26	$69.62	$74.97	$80.33	$85.68	$91.04	$96.39	$101.75	$107.10	$112.46	$117.81
0.15	0.9	0.85	$68.85	$74.59	$80.33	$86.06	$91.80	$97.54	$103.28	$109.01	$114.75	$120.49	$126.23
0.16	0.9	0.85	$73.44	$79.56	$85.68	$91.80	$97.92	$104.04	$110.16	$116.28	$122.40	$128.52	$134.64
0.17	0.9	0.85	$78.03	$84.53	$91.04	$97.54	$104.04	$110.54	$117.05	$123.55	$130.05	$136.55	$143.06
0.18	0.9	0.85	$82.62	$89.51	$96.39	$103.28	$110.16	$117.05	$123.93	$130.82	$137.70	$144.59	$151.47
0.19	0.9	0.85	$87.21	$94.48	$101.75	$109.01	$116.28	$123.55	$130.82	$138.08	$145.35	$152.62	$159.89
0.2	0.9	0.85	$91.80	$99.45	$107.10	$114.75	$122.40	$130.05	$137.70	$145.35	$153.00	$160.65	$168.30
0.21	0.9	0.85	$96.39	$104.42	$112.46	$120.49	$128.52	$136.55	$144.59	$152.62	$160.65	$168.68	$176.72

Figure 8: Yearly Revenue Summary

Revenue Over 1 Years at 350 tons per day ($1000s)

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$6,747	$7,310	$7,872	$8,434	$8,996	$9,559	$10,121	$10,683	$11,246	$11,808	$12,370
0.13	0.9	0.85	$7,310	$7,919	$8,528	$9,137	$9,746	$10,355	$10,964	$11,573	$12,183	$12,792	$13,401
0.14	0.9	0.85	$7,872	$8,528	$9,184	$9,840	$10,496	$11,152	$11,808	$12,464	$13,120	$13,776	$14,432
0.15	0.9	0.85	$8,434	$9,137	$9,840	$10,543	$11,246	$11,948	$12,651	$13,354	$14,057	$14,760	$15,463
0.16	0.9	0.85	$8,996	$9,746	$10,496	$11,246	$11,995	$12,745	$13,495	$14,244	$14,994	$15,744	$16,493
0.17	0.9	0.85	$9,559	$10,355	$11,152	$11,948	$12,745	$13,541	$14,338	$15,135	$15,931	$16,728	$17,524
0.18	0.9	0.85	$10,121	$10,964	$11,808	$12,651	$13,495	$14,338	$15,181	$16,025	$16,868	$17,712	$18,555
0.19	0.9	0.85	$10,683	$11,573	$12,464	$13,354	$14,244	$15,135	$16,025	$16,915	$17,805	$18,696	$19,586
0.2	0.9	0.85	$11,246	$12,183	$13,120	$14,057	$14,994	$15,931	$16,868	$17,805	$18,743	$19,680	$20,617
0.21	0.9	0.85	$11,808	$12,792	$13,776	$14,760	$15,744	$16,728	$17,712	$18,696	$19,680	$20,664	$21,648

Tons/day	Unit Cost	Annual Cost (1000s)	Total Cost (1000s)
350	$98.84	$12,627	$12,627

Cost Limited
Profitable

Total revenue for periods of consideration for tonnage, grade, and price per ounce valuation.

Revenue over 3 years at 600 tons per day ($1000s)

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$34,700	$37,592	$40,484	$43,376	$46,267	$49,159	$52,051	$54,942	$57,834	$60,726	$63,617
0.13	0.9	0.85	$37,592	$40,725	$43,857	$46,990	$50,123	$53,255	$56,388	$59,521	$62,654	$65,786	$68,919
0.14	0.9	0.85	$40,484	$43,857	$47,231	$50,605	$53,978	$57,352	$60,726	$64,099	$67,473	$70,847	$74,220
0.15	0.9	0.85	$43,376	$46,990	$50,605	$54,219	$57,834	$61,449	$65,063	$68,678	$72,293	$75,907	$79,522
0.16	0.9	0.85	$46,267	$50,123	$53,978	$57,834	$61,690	$65,545	$69,401	$73,256	$77,112	$80,968	$84,823
0.17	0.9	0.85	$49,159	$53,255	$57,352	$61,449	$65,545	$69,642	$73,738	$77,835	$81,932	$86,028	$90,125
0.18	0.9	0.85	$52,051	$56,388	$60,726	$65,063	$69,401	$73,738	$78,076	$82,413	$86,751	$91,089	$95,426
0.19	0.9	0.85	$54,942	$59,521	$64,099	$68,678	$73,256	$77,835	$82,413	$86,992	$91,571	$96,149	$100,728
0.2	0.9	0.85	$57,834	$62,654	$67,473	$72,293	$77,112	$81,932	$86,751	$91,571	$96,390	$101,210	$106,029
0.21	0.9	0.85	$60,726	$65,786	$70,847	$75,907	$80,968	$86,028	$91,089	$96,149	$101,210	$106,270	$111,330

Tons/day	Unit Cost	Annual Cost (1000s)	Total Cost (1000s)
600	$81.36	$17,819	$53,456

Cost Limited
Profitable

Total revenue for periods of consideration for tonnage, grade, and price per ounce valuation.

Recommendations

To advance the McGarry project and ensure profitability, Armistice should focus on the following key issues:

- Continue logging and assembling the diamond drill database.
- Update the geological interpretation of the deposit with the new diamond drill data and development information.
- Complete an advanced geological model using the new interpretation – contours and block model.
- Create a resource meeting NI43-101 standards.
- Update the mine design to match the new geological model.
- Update unit costs and economic parameters to reflect regional costs.
- Based on the new mine design, reschedule and complete a new economic evaluation at a pre-feasibility level.
- Complete primary mine development, which will enable diamond drilling and bulk sampling to be completed in known gold bearing areas.
- Utilize the positive gold market conditions for investment.
- Review regional as well as similar scaled operations – specifically Kerr Mine, Kirkland Lake Gold - Macassa Mine, and smaller producers such as Wesdome's River Gold Mine and Island Gold's Patricia Mine – relative to McGarry as a means to support both geological and mining strategies.

Conclusions

The positive aspects of the McGarry Project are:

- The area is a known gold producing region which has produced significantly over past decades.
- The McGarry project is adjacent to the Kerr Mine, which produced approximately 10 million ounces of gold between 1938 and 1996.

- An existing operating infrastructure, including headframe, hoist, surface shop and key components meeting the permit and legal standards for operation.

- A recent bulk sampling history and diamond drill program indicating potential for the project.

- An existing underground shaft and development available to advance mine development and move material to surface.

- Operating costs in the region are less than other gold producing areas in Ontario.

- The mine design, based on the assumptions in the geological model, can be completed in a timely manner and at a reasonable cost.

- The majority of the considered ranges of grade and dollars per ounce are profitable.

- The production schedule indicates that a seven month preproduction period would be necessary to achieve an annual production rate of 350 TPD. After the one year of production, a 600 TPD target is achieved readily.

The aspects of the McGarry project requiring further consideration are:

- Geological interpretation and data management needs to be advanced to a NI43-101 resource with classified resources.
- Measured and Indicated resources need to be identified and targeted for advanced diamond drilling and underground bulk sampling.
- Geological correlation of previous work such as the bulk samples, and diamond drilling.
- Advanced economic parameters to fully capture cost components for detailing unit costs, site specific and corporate costs.
- More systematic underground bulk sampling from numerous levels including and above 2250 level.

Based on the work to date, the McGarry project has financial potential. Some additional work needs to be completed to ensure the success of the project. The project appears to have a better than average likelihood for profit.

Appendix A: MineSched Period Detailed Report

MineSched Underground Demo Data

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Period Number	Period	Heading Name	Development Slope Distance (ft.)	Development Volume (cu.ft.)	Waste Development Mass (tons)	Ore Development Mass (tons)	Resources	Period Start Length	Period Start Width	Periods Start Height	Period Start SG	Period Start Priority
Grand Total:			63,178	5,754,319	474,098	25,904	2050_Level_crew/4660.540 2250_Level_crew/3534.554 2250_Prod_Dev_crew/7246.641 2050_Prod_Dev_crew/6418.744 1850_Level_crew/5486.218 Exploration_crew/5078.103 1850_Prod_Dev_crew/6914.950 1650_Level_crew/4786.942 1650_Prod_Dev_crew/6237.388 1450_Level_crew/4724.863 1450_Prod_Dev_crew/6305.238 1250_Level_crew/1783.313					
1	Q1/Y1	Total:	4,208	470,485	38,135	2,777	2050_Level_crew/1214.528 2250_Level_crew/1141.966 2250_Prod_Dev_crew/696.928 2050_Prod_Dev_crew/337.244 1850_Level_crew/529 Exploration_crew/288.758					
2	Q2/Y1	Total:	9,067	919,604	77,717	2,249	1850_Level_crew/1476.312 Exploration_crew/1501.5 2050_Level_crew/1493.962 2050_Prod_Dev_crew/1246.220 2250_Prod_Dev_crew/1447.628 2250_Level_crew/1476.523 1850_Prod_Dev_crew/424.438					
3	Q3/Y1	Total:	9,784	995,977	82,730	3,877	Exploration_crew/757.578 1850_Level_crew/1503.92 1850_Prod_Dev_crew/1383.821 2050_Level_crew/1515.312 2050_Prod_Dev_crew/1414.597 2250_Prod_Dev_crew/1518 2250_Level_crew/916.065 1650_Level_crew/763.745 1650_Prod_Dev_crew/11.000					
4	Q4/Y1	Total:	9,039	802,817	64,936	4,875	1650_Level_crew/1498.848 1650_Prod_Dev_crew/837.494 1850_Level_crew/1518.001 1850_Prod_Dev_crew/1517.467 2050_Level_crew/259.423 2050_Prod_Dev_crew/1517.997 2250_Prod_Dev_crew/1517.995 Exploration_crew/371.892					
5	Q1/Y2	Total:	6,481	523,324	41,553	3,953	1650_Level_crew/1233.385 1650_Prod_Dev_crew/1200.526 1850_Level_crew/458.985 1850_Prod_Dev_crew/1237.499 2050_Prod_Dev_crew/1077.22 2250_Prod_Dev_crew/1169.551 2050_Level_crew/103.389					
6	Q2/Y2	Total:	6,212	474,183	39,150	1,709	1650_Level_crew/1290.964 1650_Prod_Dev_crew/1501.5 1850_Prod_Dev_crew/1460.618 2050_Prod_Dev_crew/740.401 2050_Level_crew/73.926 2250_Prod_Dev_crew/798.539 Exploration_crew/346.500					
7	Q3/Y2	Total:	3,379	253,453	20,612	1,428	Exploration_crew/811.872 1650_Prod_Dev_crew/1517.078 1850_Prod_Dev_crew/866.606 2050_Prod_Dev_crew/85.065 2250_Prod_Dev_crew/98					
8	Q4/Y2	Total:	2,553	203,095	17,077	584	1650_Prod_Dev_crew/1085.290 1850_Prod_Dev_crew/24.501 1450_Level_crew/991.145 1450_Prod_Dev_crew/138.975 Exploration_crew/313.500					
9	Q1/Y3	Total:	2,815	292,213	24,297	1,113	Exploration_crew/686.503 1450_Level_crew/1195.504 1450_Prod_Dev_crew/848.628 1650_Prod_Dev_crew/84.5					
10	Q2/Y3	Total:	3,303	298,553	24,976	985	1450_Level_crew/1497.466 1450_Prod_Dev_crew/1492.644 1250_Level_crew/312.663					
11	Q3/Y3	Total:	2,515	223,930	18,231	1,241	1450_Level_crew/970.053 1450_Prod_Dev_crew/1518.003 1250_Level_crew/27.118					
12	Q4/Y3	Total:	1,532	101,253	7,692	1,113	1450_Prod_Dev_crew/1461.664 1450_Level_crew/70.695					
13	Q1/Y4	Total:	742	41,048	3,569	0	1450_Prod_Dev_crew/741.825					
14	Q2/Y4	Total:	1,547	154,385	13,425	0	1450_Prod_Dev_crew/103.499 1250_Level_crew/1443.532					

Appendix B: Unit Cost by Activity

Trackless Drift Cost/Shift	
Crew Size/Shift:	3
Cost Per Man:	$62.40
Hours per Shift:	8
Mobile Consumables	$390.25
Total Direct Cost/Shift:	*$1,887.85*
Total Cost/Shift:	*$1,887.85*
Trackless Drift Cost/Ft	
Performance:	*8.25 ft*
Total Cost/Unit:	*$228.83*
Trackless Sill Cost/Shift	
Crew Size/Shift:	3
Cost Per Man:	$62.40
Hours per Shift:	8
Mobile Consumables	$390.25
Total Direct Cost/Shift:	*$1,887.85*
Total Cost/Shift:	*$1,887.85*
Trackless Sill Cost/Ft	
Performance:	*5.5 ft*
Total Cost/Unit:	*$343.25*
Trackless Raise Cost/Shift	
Crew Size/Shift:	2
Cost Per Man:	$62.40
Hours per Shift:	8
Mobile Consumables	$390.25
Total Direct Cost/Shift:	*$1,388.65*
Total Cost/Shift:	*$1,388.65*
Trackless Raise Cost/Ft	
Performance:	*7.5 ft*
Total Cost/Unit:	*$185.15*
Track Drift Cost/Shift	
Crew Size/Shift:	3
Cost Per Man:	$62.40
Hours per Shift:	8
Track Consumables	$388.39
Total Direct Cost/Shift:	*$1,885.99*
Total Cost/Shift:	*$1,885.99*

Total Indirect Cost/Day:	**$21,790.00**
Number of Shifts per Day:	**2**
Total Indirect Cost/Shift:	**$10,895.00**

Appendix B: Unit Cost by Activity

Track Drift Cost/Ft	
Performance:	*7.5 ft*
Total Cost/Unit:	*$251.47*
Track Sill Cost/Shift	
Crew Size/Shift:	3
Cost Per Man:	$62.40
Hours per Shift:	8
Track Consumables	$388.39
Total Direct Cost/Shift:	*$1,885.99*
Total Cost/Shift:	*$1,885.99*
Track Sill Cost/Ft	
Performance:	*5.5 ft*
Total Cost/Unit:	*$342.91*
Track Raise Cost/Shift	
Crew Size/Shift:	2
Cost Per Man:	$62.40
Hours per Shift:	8
Track Consumables	$388.39
Total Direct Cost/Shift:	*$1,386.79*
Total Cost/Shift:	*$1,386.79*
Track Raise Cost/Ft	
Performance:	*7.5 ft*
Total Cost/Unit:	*$184.91*

Appendix B: Costs per Day Estimate

Cost Component	SubTopic	Detail	Total Cost per Day
Technical			
	Geology		
		Sampling (Lab)	$330
		DDH	$4,270
		Personnel	$1,320
		Geological Supplies	$330
	Engineering		
		Consultants	**$410**
	Survey (Assumed Contractor)		
		Personnel	$440
Technical Subtotal:	$7,100	Per Day	
Maintenance			
	Headframe		$50
	Shaft Crew		$1,850
	Level maintenance		$740
	Labour		$2,000
	Maintenance Materials		$990
Maintenance Subtotal:	$5,630		
Mining			
		Supervision	$1,150
		Bits and Steel	$1,070
		Explosives - All	$750
Mining Subtotal	$8,600	Per Day	
Overhead			
	Taxes		
		Municipal	$140
		Provincal	
		Federal	
	Environmental		$100
	Electrical		$1,510
	Fuel - Mine Air Heat		$1,010
	Fuel - General		$550
	Surface Maintenance		$70
	Safety and PPE		$50
	Advance Royalty		**$233**
	Vehicles - Surface		$50
	Corporate - Offsite		**$2,380**
Overhead Subtotal	$6,093	Per Day	
Total Cost per Day			**$21,793**

Cost Estimate (350 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
Cost Component						
Technical						
	Geology					
		Sampling (Lab)				$329
		Sampling (Materials)				$49
		DDH				$493
		Personnel				$493
		Computers				$5
		Software				$4
		Geological Consultants				$1,382
	Engineering					
		Personnel				$301
		Computers				$5
		Software				$4
		Engineering Consultants				$415
	Survey (Assumed Contractor)					
		Personnel				$440
		Computers				$5
		Software				$4
		Equipment				$55
Technical Subtotal:	$3,986	$11.02				
Maintenance						
	Headframe					$329
	Shaft					$329
	Level maintenance					$329
	UG Equipment					$658
	Pumps					$164
	UG Compressed Air					$658
	UG Elec Switch Gear					$658
Maintenance Subtotal	$3,123	$8.63				
Mining						
	Contractor (Labour)					
		Waste Dev.	139	2	$576	$1,152
		Silling (75% Ore)	83	2	$576	$1,152
		Stoping (100% Ore)	278	8	$576	$4,608
Tonnage Check:	362					
		Supervision				$1,152
		Bits		9	$50	$433
		Steel		3	$50	$150
		Oil (4 lire containers)		6	$4	$24
		Tools		0.25	$20	$5
		Powder (kgs)			$50	$600
		Caps (Nonel and 0)			$4	$1,043
		B-Line (ft)			$1	$130

Cost Estimate (350 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
		Blast Line (Red and Yellow)			$0	$150
		Timber				$150
	Services	Pipe				
		Chain				
		Hose				
		Ballast				
		Cageman				440
		Nipper				440
		Timbermen				880
Mining Subtotal	$12,509	$34.58				
Trucking/Milling						
	McGarry Surface Handling					
	Milling				$35	$12,661
Overhead						
	Taxes					
		Municipal				$137
		Provincal				$192
		Federal				$192
	Staff Salaries (offsite) and Benefits					$1,068
	Vehicles - UG and Surface					$902
	Insurance					$55
	Permits & Licences					$14
	Bank Charges					$10
	Professional Fees - Accounting					$4
	Professional Fees - Legal					$4
	Recruitment/Relocation					$27
	Security					$41
	Safety, Clothing and Training					$82
	First Aid					$49
	Dues & Subscriptions					$5
	Public Relations/Community Support					$27
	Environmental					$82
	Power					$197
	Professional Fees - General					$4
	Travel & Accommodation - Business					$75
	Freight					$137
	Communications, Office Supplies & Misc.					$33
	Miscellaneous					$137
Overhead Subtotal	$3,475	$9.61				
Total Cost per Day						$35,755

Cost Estimate (350 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
Total Cost per Ore Ton					$98.84	

Cost Component	Cost per Day	Cost per Ton
Technical	$4,000	$11
Maintenance	$4,000	$9
Mining	$13,000	$35
Milling/Trucking	$13,000	$35
Overhead	$4,000	$10
	Rounded	**Calculated**
Total Cost per Day	**$38,000**	$35,755
Total Cost per Ton Ore	**$98.84**	$98.84

Cost Estimate (600 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
Cost Component						
Technical						
	Geology					
		Sampling (Lab)				$329
		Sampling (Materials)				$49
		DDH				$493
		Personnel				$493
		Computers				$5
		Software				$4
		Geological Consultants				$1,370
	Engineering					
		Personnel				$301
		Computers				$5
		Software				$4
		Engineering Consultants				$411
	Survey (Assumed Contractor)					
		Personnel				$440
		Computers				$5
		Software				$4
		Equipment				$55
Technical Subtotal:	$3,970	$6.56				
Maintenance						
	Headframe					$329
	Shaft					$329
	Level maintenance					$329
	UG Equipment					$658
	Pumps					$164
	UG Compressed Air					$658
	UG Elec Switch Gear					$658
Maintenance Subtotal	$3,123	$5.16				
Mining						
	Contractor (Labour)					
		Waste Dev.	139	2	$576	$1,152
		Silling (75% Ore)	83	2	$576	$1,152
		Stoping (100% Ore)	522	15	$576	$8,640
Tonnage Check:	605					
		Supervision				$1,152
		Bits		12	$50	$608
		Steel		4	$50	$194
		Oil (4 lire containers)		8	$4	$31
		Tools		0.25	$20	$5
		Powder (kgs)			$50	$775
		Caps (Nonel and 0)			$4	$1,435
		B-Line (ft)			$1	$200
		Blast Line (Red and Yellow)			$0	$238

Cost Estimate (600 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
		Timber				$150
	Services	Pipe				
		Chain				
		Hose				
		Ballast				
		Cageman				440
		Nipper				440
		Timbermen				880
Mining Subtotal	$17,492	$28.90				
Trucking/Milling						
	McGarry Surface Handling					
	Milling				$35	$21,183
Overhead						
	Taxes					
		Municipal				$137
		Provincal				$192
		Federal				$192
	Staff Salaries (offsite) and Benefits					$1,068
	Vehicles - UG and Surface					$902
	Insurance					$55
	Permits & Licences					$14
	Bank Charges					$10
	Professional Fees - Accounting					$4
	Professional Fees - Legal					$4
	Recruitment/Relocation					$27
	Security					$41
	Safety, Clothing and Training					$82
	First Aid					$49
	Dues & Subscriptions					$5
	Public Relations/Community Support					$27
	Environmental					$82
	Power					$197
	Professional Fees - General					$4
	Travel & Accommodation - Business					$75
	Freight					$137
	Communications, Office Supplies & Misc.					$33
	Miscellaneous					$137
Overhead Subtotal	$3,475	$5.74				
Total Cost per Day						$49,243
Total Cost per					$81.36	

Cost Estimate (600 TDP)			Tonnage Calc	Rounds/ Breasts	Unit Cost	Total Cost per Day
Ore Ton						
		Cost Component	**Cost per Day**	**Cost per Ton**		
		Technical	$4,000	$7		
		Maintenance	$4,000	$5		
		Mining	$18,000	$29		
		Milling/Trucking	$22,000	$35		
		Overhead	$4,000	$6		
			Rounded	**Calculated**		
		Total Cost per Day	**$52,000**	$49,243		
		Total Cost per Ton Ore	**$81.36**	$81.36		

Appendix B: Revenue per Ounce Summary

	Cost limited
	Profitable

ton/day	Cost/ton
350	98.84
600	81.36

Total revenue per ounce for grade and price valuation.

Revenue per ounce for 350 tons per day

			Gold Price per Troy Ounce ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$55.08	$59.67	$64.26	$68.85	$73.44	$78.03	$82.62	$87.21	$91.80	$96.39	$100.98
0.13	0.9	0.85	$59.67	$64.64	$69.62	$74.59	$79.56	$84.53	$89.51	$94.48	$99.45	$104.42	$109.40
0.14	0.9	0.85	$64.26	$69.62	$74.97	$80.33	$85.68	$91.04	$96.39	$101.75	$107.10	$112.46	$117.81
0.15	0.9	0.85	$68.85	$74.59	$80.33	$86.06	$91.80	$97.54	$103.28	$109.01	$114.75	$120.49	$126.23
0.16	0.9	0.85	$73.44	$79.56	$85.68	$91.80	$97.92	$104.04	$110.16	$116.28	$122.40	$128.52	$134.64
0.17	0.9	0.85	$78.03	$84.53	$91.04	$97.54	$104.04	$110.54	$117.05	$123.55	$130.05	$136.55	$143.06
0.18	0.9	0.85	$82.62	$89.51	$96.39	$103.28	$110.16	$117.05	$123.93	$130.82	$137.70	$144.59	$151.47
0.19	0.9	0.85	$87.21	$94.48	$101.75	$109.01	$116.28	$123.55	$130.82	$138.08	$145.35	$152.62	$159.89
0.2	0.9	0.85	$91.80	$99.45	$107.10	$114.75	$122.40	$130.05	$137.70	$145.35	$153.00	$160.65	$168.30
0.21	0.9	0.85	$96.39	$104.42	$112.46	$120.49	$128.52	$136.55	$144.59	$152.62	$160.65	$168.68	$176.72

Revenue per ounce for 600 tons per day

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$55.08	$59.67	$64.26	$68.85	$73.44	$78.03	$82.62	$87.21	$91.80	$96.39	$100.98
0.13	0.9	0.85	$59.67	$64.64	$69.62	$74.59	$79.56	$84.53	$89.51	$94.48	$99.45	$104.42	$109.40
0.14	0.9	0.85	$64.26	$69.62	$74.97	$80.33	$85.68	$91.04	$96.39	$101.75	$107.10	$112.46	$117.81
0.15	0.9	0.85	$68.85	$74.59	$80.33	$86.06	$91.80	$97.54	$103.28	$109.01	$114.75	$120.49	$126.23
0.16	0.9	0.85	$73.44	$79.56	$85.68	$91.80	$97.92	$104.04	$110.16	$116.28	$122.40	$128.52	$134.64
0.17	0.9	0.85	$78.03	$84.53	$91.04	$97.54	$104.04	$110.54	$117.05	$123.55	$130.05	$136.55	$143.06
0.18	0.9	0.85	$82.62	$89.51	$96.39	$103.28	$110.16	$117.05	$123.93	$130.82	$137.70	$144.59	$151.47
0.19	0.9	0.85	$87.21	$94.48	$101.75	$109.01	$116.28	$123.55	$130.82	$138.08	$145.35	$152.62	$159.89
0.2	0.9	0.85	$91.80	$99.45	$107.10	$114.75	$122.40	$130.05	$137.70	$145.35	$153.00	$160.65	$168.30
0.21	0.9	0.85	$96.39	$104.42	$112.46	$120.49	$128.52	$136.55	$144.59	$152.62	$160.65	$168.68	$176.72

Appendix B: Yearly Revenue Summary

Revenue Over 1 Years at 350 tons per day ($1000s)

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$6,747	$7,310	$7,872	$8,434	$8,996	$9,559	$10,121	$10,683	$11,246	$11,808	$12,370
0.13	0.9	0.85	$7,310	$7,919	$8,528	$9,137	$9,746	$10,355	$10,964	$11,573	$12,183	$12,792	$13,401
0.14	0.9	0.85	$7,872	$8,528	$9,184	$9,840	$10,496	$11,152	$11,808	$12,464	$13,120	$13,776	$14,432
0.15	0.9	0.85	$8,434	$9,137	$9,840	$10,543	$11,246	$11,948	$12,651	$13,354	$14,057	$14,760	$15,463
0.16	0.9	0.85	$8,996	$9,746	$10,496	$11,246	$11,995	$12,745	$13,495	$14,244	$14,994	$15,744	$16,493
0.17	0.9	0.85	$9,559	$10,355	$11,152	$11,948	$12,745	$13,541	$14,338	$15,135	$15,931	$16,728	$17,524
0.18	0.9	0.85	$10,121	$10,964	$11,808	$12,651	$13,495	$14,338	$15,181	$16,025	$16,868	$17,712	$18,555
0.19	0.9	0.85	$10,683	$11,573	$12,464	$13,354	$14,244	$15,135	$16,025	$16,915	$17,805	$18,696	$19,586
0.2	0.9	0.85	$11,246	$12,183	$13,120	$14,057	$14,994	$15,931	$16,868	$17,805	$18,743	$19,680	$20,617
0.21	0.9	0.85	$11,808	$12,792	$13,776	$14,760	$15,744	$16,728	$17,712	$18,696	$19,680	$20,664	$21,648

Tons/day	Unit Cost	Annual Cost (1000s)	Total Cost (1000s)
350	$98.84	$12,627	$12,627

Cost Limited

Profitable

Total revenue for periods of consideration for tonnage, grade, and price per ounce valuation.

Revenue over 3 years at 600 tons per day ($1000s)

			Gold Price ($CAD)										
Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	0.9	0.85	$34,700	$37,592	$40,484	$43,376	$46,267	$49,159	$52,051	$54,942	$57,834	$60,726	$63,617
0.13	0.9	0.85	$37,592	$40,725	$43,857	$46,990	$50,123	$53,255	$56,388	$59,521	$62,654	$65,786	$68,919
0.14	0.9	0.85	$40,484	$43,857	$47,231	$50,605	$53,978	$57,352	$60,726	$64,099	$67,473	$70,847	$74,220
0.15	0.9	0.85	$43,376	$46,990	$50,605	$54,219	$57,834	$61,449	$65,063	$68,678	$72,293	$75,907	$79,522
0.16	0.9	0.85	$46,267	$50,123	$53,978	$57,834	$61,690	$65,545	$69,401	$73,256	$77,112	$80,968	$84,823
0.17	0.9	0.85	$49,159	$53,255	$57,352	$61,449	$65,545	$69,642	$73,738	$77,835	$81,932	$86,028	$90,125
0.18	0.9	0.85	$52,051	$56,388	$60,726	$65,063	$69,401	$73,738	$78,076	$82,413	$86,751	$91,089	$95,426
0.19	0.9	0.85	$54,942	$59,521	$64,099	$68,678	$73,256	$77,835	$82,413	$86,992	$91,571	$96,149	$100,728
0.2	0.9	0.85	$57,834	$62,654	$67,473	$72,293	$77,112	$81,932	$86,751	$91,571	$96,390	$101,210	$106,029
0.21	0.9	0.85	$60,726	$65,786	$70,847	$75,907	$80,968	$86,028	$91,089	$96,149	$101,210	$106,270	$111,330

Tons/day	Unit Cost	Annual Cost (1000s)	Total Cost (1000s)
600	$81.36	$17,819	$53,456

Cost Limited

Profitable

Total revenue for periods of consideration for tonnage, grade, and price per ounce valuation.

Appendix B: ConsTrack

Armistice Resources Corp.

Capital	**1891**
Consumables - Track	Account Code
Mine	(All costs in $C 000,'s)

Scope of Cost Centre and Explanatory Notes

To collect operating consumables for the underground tracked mining development work: 24" gauge 18 inch tie centres

Includes:

					Per ft Advance
Ties (18" centres)	3.50	per tie	use 0.67	per foot of advance	2.33
Rail (30#)	227	per 20 ft rail	use 0.10	per foot of advance	22.70
Fish plate & 4 track bolts	25	each	use 0.10	per foot of advance	2.50
Spikes	175	per 120	use 3	per foot of advance	3.89
Switch complete	2,000	each	use 1	per 500 ft of development	4.00

Pipe for air and water & couplings & valves

Ventilation ducting @ 4.70 / ft for 30" ducting

Ground support - rock bolts, screen, strapping

Timber

Mining Tools and Small Equipment

Miscellaneous

Lease charge for rolling stock and equipment:

4 ton loci with spare battery & charger	30,000
4 x 4 ton cars	5,000

Misc flat cars - car dump - rail bender	8,000		
2 x Mucking machine	32,000		
	75,000	10%	per month = 7,500

Details of Cost Estimate

	Standard Allowance	
Rail, Plates, Ties, Fish Plates, Spikes, Switches	35.42	$ per foot of drift advance
Pipe for air and water & couplings & valves	30.00	
Ventilation ducting / fans	5.00	
Ground support - rock bolts, screen, strapping	35.00	
	105.42	$ per foot of drift advance Total
Equipment Lease	7,500	
Timber/Lumber	1,000	$ per month
Mining Tools and Small Equipment	1,000	
Miscellaneous	1,000	
	10,500	$ per month Total

		Fixed + Variable Standard Cost		
		Month	Cumulative	
			Year	Project
2008 - 2009	Jul08			
	Aug08			
	Sep08			
	Oct08			
	Nov08	36.9	36.9	36.9
	Dec08	31.6	68.4	68.4
	Jan09	31.6	100.0	100.0
	Feb09	36.9	136.9	136.9
	Mar09	36.9	173.7	173.7
	Apr09	36.9	210.6	210.6
	May09	36.9	247.4	247.4
	Jun09	36.9	284.3	284.3
		284.3		
2009 - 2010	Jul09	36.9	36.9	321.2
	Aug09	36.9	73.7	358.0
	Sep09	36.9	110.6	394.9
	Oct09	10.5	121.1	405.4
	Nov09	10.5	131.6	415.9
	Dec09	10.5	142.1	426.4
	Jan10	10.5	152.6	436.9
	Feb10	10.5	163.1	447.4
	Mar10	10.5	173.6	457.9
	Apr10	10.5	184.1	468.4
	May10	10.5	194.6	478.9
	Jun10	10.5	205.1	489.4
		205.1		

Fixed Costs	Variable Costs		
		Standard Unit :>	Feet Drifting
Month	Month	Nº Standard Units	Per Standard Unit
			105.4
			105.4
			105.4
			105.4
10.5	26.4	250	105.4
10.5	21.1	200	105.4
10.5	21.1	200	105.4
10.5	26.4	250	105.4
10.5	26.4	250	105.4
10.5	26.4	250	105.4
10.5	26.4	250	105.4
10.5	26.4	250	105.4
84.0	200.3		
10.5	26.4	250	105.4
10.5	26.4	250	105.4
10.5	26.4	250	105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
10.5			105.4
126.0	79.1	750.00	

Class: **Capital**
Detail: **Consumables - Track**
Department: **Mine**
Code: 1891

Appendix B: Contractor Labour

Capital | **1876**

Contractor Labour | Account Code

Mine

(All costs in $C 000,'s)

Scope of Cost Centre and Explanatory Notes

Covers the cost of on site mining contractor labour.

See Contractor Supervision for scope of work and other details.

Other includes Carpenter
3 Watchmen included as 1 under Other

Details of Cost Estimate

Development Phase Rates								$
	Hoist/Cage/Other	Electrician	Mech/Weld	Miner		2008	Jul	220.3
	23.00	23.00	23.00	23.00	$/hr Base		Aug	200.3
Note	25.00	25.00	25.00	25.00	Rate $/hr		Sep	210.3
3 Watchmen	50	75	50	100	Standard % Standard		Oct	220.3
= 1 Other	30	30	30	30	Bonus % Burden		Nov	200.3
	46.15	54.28	46.15	62.40	$/hr Gross		Dec	160.2
						2009	Jan	205.3
	61.10	69.23	61.10	77.35	$/hr		Feb	195.0
					Overtime		Mar	225.8
	8.00	8.00	8.00	8.00	hr/day		Apr	215.5
	0.16	0.16	0.16	0.16	Regular hr/day		May	205.3
					Overtime		Jun	225.8
	378.98	445.28	378.98	511.58	$ Gross/day			
	56.85	66.79	56.85	76.74	15.00% overhead & profit		Jul	225.8
	435.83	512.07	435.83	588.32	$ Net/day		Aug	205.3
							Sep	215.5
2008	8.0	1	3	8	22 (incl. 3		Oct	
Jul Aug	8.0	1	3	8	watchmen) 22 (incl. 3		Nov	
Sep	8.0	1	3	8	watchmen) 22 (incl. 3		Dec	
Oct	8.0	1	3	8	watchmen) 22 (incl. 3	2010	Jan	
Nov	8.0	1	3	8	watchmen) 22 (incl. 3		Feb	
Dec	8.0	1	3	8	watchmen) 22 (incl. 3		Mar	
2009	8.0	1	3	8	watchmen) 22 (incl. 3		Apr	
Jan Feb	8.0	1	3	8	watchmen) 22 (incl. 3		May	
Mar	8.0	1	3	8	watchmen) 22 (incl. 3		Jun	
Apr	8.0	1	3	8	watchmen) 22 (incl. 3			
>>					watchmen)			

		Fixed + Variable Standard Cost		
		Month	Cumulative	
			Year	Project
2008 - 2009	Jul08	220.3	220.3	220.3
	Aug08	200.3	420.5	420.5
	Sep08	210.3	630.8	630.8
	Oct08	220.3	851.1	851.1
	Nov08	200.3	1,051.3	1,051.3
	Dec08	160.2	1,211.5	1,211.5
	Jan09	205.3	1,416.8	1,416.8
	Feb09	195.0	1,611.8	1,611.8
	Mar09	225.8	1,837.6	1,837.6
	Apr09	215.5	2,053.1	2,053.1
	May09	205.3	2,258.4	2,258.4
	Jun09	225.8	2,484.2	2,484.2
		2,484.2		
2009 - 2010	Jul09	225.8	225.8	2,709.9
	Aug09	205.3	431.0	2,915.2
	Sep09	215.5	646.6	3,130.7
	Oct09		646.6	3,130.7
	Nov09		646.6	3,130.7
	Dec09		646.6	3,130.7
	Jan10		646.6	3,130.7
	Feb10		646.6	3,130.7
	Mar10		646.6	3,130.7
	Apr10		646.6	3,130.7
	May10		646.6	3,130.7
	Jun10		646.6	3,130.7
		646.6		

Fixed Costs	Variable Costs Standard		
Month	Month	Nº Standard Units	Per Unit
220.3			
200.3			
210.3			
220.3			
200.3			
160.2			
205.3			
195.0			
225.8			
215.5			
205.3			
225.8			
2,484.2			
225.8			
205.3			
215.5			
646.6			

Class: **Capital**
Detail: **Contractor Labour**
Department: **Mine**
Code: 1876

Armistice
RESOURCES CORP.

Capital		**1879**
Consumables - Mobile		Account Code
Mine		(All costs in $C 000,'s)

Scope of Cost Centre and Explanatory Notes

To collect operating consumables for the underground mining development work:

Includes:

Tires for U/G mobile equipment	Variable by feet of drift advance	
Lubricants & Filters for U/G mobile equipment	Variable by feet of drift advance	
Pipe for air and water & couplings & valves	Variable by feet of drift advance	
Ventilation ducting	Variable by feet of drift advance	@ 4.70 / ft for 30" ducting
Ground support - rock bolts, screen, strapping	Variable by feet of drift advance	
Timber	Fixed Allowance per Month	
Mining Tools and Small Equipment	Fixed Allowance per Month	
Miscellaneous	Fixed Allowance per Month	

Details of Cost Estimate

	Standard Allowance	
Tires for U/G mobile equipment	2.00	$ per foot of drift advance
Lubricants for U/G mobile equipment	7.50	
Filters	2.00	
Pipe for air and water & couplings & valves	30.00	
Ventilation ducting / fans	7.00	
Ground support - rock bolts, screen, strapping	35.00	
	83.50	$ per foot of drift advance Total
Timber/Lumber	1,000	$ per month
Mining Tools and Small Equipment	1,000	
Miscellaneous	1,000	
	3,000	$ per month Total

		Fixed + Variable Standard Cost		
		Month	Cumulative	
			Year	Project
2008 - 2009	Jul08	31.4	31.4	31.4
	Aug08	17.2	48.6	48.6
	Sep08	31.4	80.0	80.0
	Oct08	31.4	111.4	111.4
	Nov08	23.0	134.4	134.4
	Dec08	18.0	152.4	152.4
	Jan09	18.0	170.5	170.5
	Feb09	23.0	193.5	193.5
	Mar09	23.0	216.5	216.5
	Apr09	23.0	239.6	239.6
	May09	23.0	262.6	262.6
	Jun09	23.0	285.7	285.7
		285.7		
2009 - 2010	Jul09	23.0	23.0	308.7
	Aug09	14.7	37.7	323.4
	Sep09	14.7	52.4	338.1
	Oct09	3.0	55.4	341.1
	Nov09	3.0	58.4	344.1
	Dec09	3.0	61.4	347.1
	Jan10	3.0	64.4	350.1
	Feb10	3.0	67.4	353.1
	Mar10	3.0	70.4	356.1
	Apr10	3.0	73.4	359.1
	May10	3.0	76.4	362.1
	Jun10	3.0	79.4	365.1
		79.4		

Fixed Costs	**Variable Costs**		
		Standard Unit :>	**Feet Drifting**
Month	Month	Nº Standard Units	Per Standard Unit
3.0	28.4	340	83.5
3.0	14.2	170	83.5
3.0	28.4	340	83.5
3.0	28.4	340	83.5
3.0	20.0	240	83.5
3.0	15.0	180	83.5
3.0	15.0	180	83.5
3.0	20.0	240	83.5
3.0	20.0	240	83.5
3.0	20.0	240	83.5
3.0	20.0	240	83.5
3.0	20.0	240	83.5
36.0	249.7		
3.0	20.0	240	83.5
3.0	11.7	140	83.5
3.0	11.7	140	83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
3.0			83.5
36.0	43.4	520.00	

Class: **Capital**
Detail: **Consumables - Mobile**
Department: **Mine**
Code: 1879

Armistice RESOURCES CORP.

Armistice Resources Corp.
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1 October 2008

Python Mining Consultants
173 Bold St
Hamilton ON L8P 1V4

Re: Scoping Study – McGarry Project

Armistice has a need for a Scoping Study that encompasses mining methods and sequences, including costs, revenues and work schedules for certain areas of the underground gold deposit at its McGarry Project located at Virginiatown, Ontario. This is to request that Python complete the Study on behalf of Armistice using the parameters and assumptions set out below:

Parameters:

Present a development and mining plan for the following levels: 2250, 2050, 1850, 1650, 1450 and 1250 starting with the infrastructure as it currently exists.

The plan should utilize trackless equipment on 2250 level and tracked equipment on all other levels.

The plan should incorporate the completion of a ventilation raise system from the 2250 to 1250 levels that will initially also serve as an emergency escapeway for men. The system should be able to convert to a waste/ore pass system at later stages as alternate ventilation and escapeways are established.

Mining should proceed from the lower levels and progress upward in an orderly and sustainable manner – that is, mining should start on the 2250 and 2050 Levels and progress upward adding the 1850 Level next, etc.

Production of ore should be planned at an initial rate of 350 tons per day with flexibility to increase progressively to 650 tons per day.

The Scoping is for mining only. Processing will be off site at a custom milling operation.

In 2002, an NI 43-101 Technical Report authored by S.J. Carmichael outlined an indicated mineral resource of 477,379 tons at a grade of 0.229 oz/t gold using a cut off grade of 0.10 oz/t and a dilution factor of 10% at a grade of 0.02 oz/t. This resource is located between 1600 and 2600 feet below surface. The resource is distributed between 4 main zones and 8 minor zones.

Armistice has conducted a drilling programme in 2007-2008 that has filled in the gap in the definition drilling on the 2250 Level as recommended by Carmichael. The 2250 Level now has definition drilling completed from 400E to 2700W at 100 foot spacings along strike. Each 100 foot section is composed of a fan of 7 holes drilled at angles from +52° to -52°. There remains a few missing drill holes in some of the sections but these are not considered material at this point. This definition drilling provides a drill hole density with about 100 foot centres through the expected mineralized geological package which lies from about 100 feet to 600 feet north of the 2250 Drift. This drill hole density extends from about 200 feet above the 2250 Level to 200 feet below the 2250 Level.

Armistice is also conducting a programme of drilling from the 2250 Level to test the prospective mineralized zone between the 1200 and 2000 foot elevations below surface in the area west of the shaft. This area is largely untested by previous diamond drilling, but is interpreted by Armistice to have similar potential for gold mineralization as the area below the 2000 foot elevation. This conclusion is also supported by the gold bearing zones identified near the shaft at these elevations from drilling and from bulk sampling on the 1650 Level. The planned limited drilling will be preliminary in nature only with the objective of confirming the existence of the prospective geology at these elevations only. Final results from this programme are not expected until February 2009. As indicated under Assumptions below, for purposes of the Study, it should be assumed that this drilling programme will confirm the presence of prospective geology at the target elevations.

Assumptions:

There are no existing ore reserves defined on the project and none are expected in the near future. A revised indicated mineral resource will be completed once all drill core assay results are available (expected in April 2009). The drilling completed to date is not at a close enough spacing to provide definition of potential stoping blocks. A review of all available data to date by Armistice's technical staff has led to the conclusion that there exists a favourable geological package extending from the 2250 Level to the 1250 Level of similar character to that indicated on the 2250 Level. This conclusion is not proven and subsequent exploration in this area may fail to confirm this conclusion.

The specific gravity used for the Carmichael estimate is 2.79 (which gives a tonnage factor of 11.5 cubic feet per ton in situ). This specific gravity should be used for both ore and waste in the Study.

Experience to date on the 2250 Level indicates that strike continuity of mineralized zones of about 100 feet is a reasonable assumption.

Vertical continuity of mineralized zones has yet to be well established by test mining, however, for purposes of this Study, vertical continuity of 200 feet should be assumed.

Undiluted stope widths of 6 feet should be assumed. Unplanned dilution of at a grade of 0.02 oz/t should be assumed based on 1.0 feet of waste from each stope wall. Considering the anticipated irregular stope envelopes, this is a reasonable assumption and conforms with the experience from test stoping carried out in 2008.

An undiluted grade in the range 0.16 to 0.25 oz/t gold should be modelled.

Mining recovery of mineralized gold zones should be assumed at 90%.

Because actual locations of prospective stopes cannot be established at this point, for purposes of the Study, it should be assumed that 10 gold zones with 100 foot strike continuity each will be located at intervals along the strike extent of the prospective geological package over the full east-west extent of the property. Where drilling results are available, this data should be used to guide the speculative placement of stoping areas for purposes of this Study. Work on the 2250 Level indicates that this is a reasonable assumption for a Scoping Study.

The strategic approach to each level's development and production plan should be based on the following outline:

An arrangement for dumping into the ore/waste pass system should be established.

A cross cut from the station should be driven north into about the centre of the prospective geological package.

From the access cross cut, strike drives both east and west should be driven in about the centre of the prospective geological package and trying to keep in waste.

From the strike drive, a short-hole air diamond drill should be used to drill 3 hole fans both to the north and south on 50 foot centres for stope definition. Where required for additional definition, this pattern may be closed to 25 foot centres.

Stope development is to be established from spurs off the main strike drive drift.

The mining method should be chosen for maximum in-stope flexibility to react to changing gold distribution patterns in all directions.

It should be assumed that dips will be close to vertical, strike will be close to mine grid east-west and plunges variable from vertical to 60°.

The mining method should accommodate strong shearing sub-parallel to the mineralization strike.

Unit costs for supplies and equipment will be provided separately by Armistice based on actual costs incurred during the 2007-2008 underground campaigns. For purposes of the Study, these costs should be assumed to be ±25%.

Manpower strengths and costs will be supplied separately by Armistice based on productivities and costs achieved during the 2007-2008 underground campaigns.

Overhead costs will be provided separately by Armistice.

It should be assumed that processing of the run-of-mine ore will be at an off-site custom milling operation. Combined trucking and milling costs of $C 35 per ton should be assumed with metallurgical recoveries of 85%.

Gold prices for revenues should be anticipated in Canadian dollars since all costs will also be in Canadian dollars. A range of gold prices from $C 600 to $C 1100 in $C 50 increments should be assumed for purposes of an economic evaluation.

Please do not hesitate to contact me for any clarification.

Yours truly,

Erik Andersen, P.Eng
Vice President & COO

CERTIFICATE OF AUTHOR

As a contributing author of this report on the mining section for McGarry Project, I, Martin Drennan do hereby certify that:

1. I am employed as an Associate and Senior Mining Engineer by, and carried out this assignment for:

   Python Mining Consultants Inc.
   37 Spruceside Ave
   Hamilton, Ontario
   L8P 3Y2
   Tel. (905) 540-1432
   Fax (905) 540-1136

2. This certificate applies to the technical report titled "Scoping Study - McGarry Mine Project", dated Feb. 4 2009, prepared for Armistice Resources Corp.

3. I hold the following academic qualifications:
   B.Eng. Mining Engineering Laurentian University 1991

4. I am a registered Professional Engineer with the Professional Engineers Ontario (membership number 90526286). As well, I am a member in good standing of the following technical associations and societies:
   The Canadian Institute of Mining, Metallurgy and Petroleum

5. I have worked as an engineer in the mining industry for 17 years.

6. I understand that my education, experience and professional registration, fulfill the requirements of a Qualified Person. My work experience includes mine design – development, mining method optimization, mine economics and analysis, winze design and underground headframe design.

7. I have visited the McGarry property.

8. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services.

9. I have had no prior involvement with the mineral properties in question.

10. As of the date of this certificate to the best of my knowledge, information and belief, the above listed technical report contains all the scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 19th day of March, 2009.

Martin Drennan, P.Eng.