Preliminary Economic Assessment and Mineral Resource Estimate

McGarry Project
McGarry Township (Virginiatown), Ontario

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1. Summary

Introduction and Infrastructure

Armistice Resources Corp. ("Armistice") completed an underground exploration campaign during 2007-2009 at its wholly-controlled McGarry gold project located at Virginiatown, Ontario. The work was carried out principally 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. There was also a programme of surface diamond drilling totalling 14,300 feet in the McGarry shaft area and in the McGarry Mill Zone, 3,000 feet to the south. A programme of metallurgical testing was also undertaken. In addition, a preliminary economic assessment was completed. The assessment was carried out by Python Mining Consultants ("Python"), an independent firm with experience in narrow vein gold mining operations in Ontario.

The author visited the project site in March 2008 and on September 16, 2011.

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. The McGarry property is owned 75% by Armistice Resources Corp. and 25% by Jubilee Gold Inc. All proceeds of production from the Property are to Armistice, subject to a royalty interest held by Jubilee Gold Inc. which provides for a Net Smelter royalty payable to Jubilee 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 (subject to cost of living adjustments). The current status of royalty payments to Jubilee 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 complement 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 would 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 2007-2009. Exploration work consists of underground diamond drilling in 421 holes totalling over 304,000 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. The geological 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. Of note economically, pyritic "flow ore" was the most important type at Kerr Addison. Both types of gold mineralization are recognized at McGarry.

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 16 September 2011. 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 (see Table 1 below).

Undiluted Mineral Resource Estimate – 16 September 2011

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 ±15% is estimated for the Indicated Mineral Resource and ±25% for the Inferred Mineral Resource.
- Effective date of resource estimate verification is 16 September 2011
- Qualified Person for the mineral resource estimate verification is Martin Drennan, P.Eng.
- 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 1 - Mineral Resource Estimate for the McGarry Project

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.

Preliminary Economic Assessment

The preproduction phase of the McGarry Project will occur during the first 7 months of the planned work. The total cost of the preproduction period is $4.7 million.

The duration of the development process in the McGarry Project is forecasted to 4 years. The initial annual cost of the development is approximately $9.2 million. In year 2 the development cost is forecasted to $5.3 million, in year 3 the development cost will be $3.0 million, and the final year of development will cost approximately $700,000 dollars.

Table 2: Summary of Development Costs per Year of Operation

Year	Development Cost ($1000s)
1	9,200
2	5,300
3	3,000
4	680
Total: 18,200	

The initial annual production year the cost is $9.7 million. In year 2 the cost of production is being forecast to $16 million, and for the third and fourth year of production, the cost will be approximately the same at $16 million dollars.

Table 3: Summary of Production Costs per Year of Operation

Year	Production Cost ($1000s)
1	9,700
2	16,000
3	16,000
4	16,000
Total: 57,700	

The economic model is linear based on the assumption in its development for both initial production and the targeted final 600 tpd production. With respect to the initial production with a minimum assumed gold price of $600 per ounce the required insitu grade to be profitable at 350 tpd is in around 0.24 oz/ton. The margin of profit for 600 tpd is in around 0.20 oz/ton or above. If the diluted production grade at the McGarry Mine is 0.18 oz/ton, then this would require a gold price of $785 per ounce at 350 tpd and $656 per ounce at 600 tpd to be profitable. See Figure 1 and 2 below for details.

Figure 1 - Daily Revenue Summary

Revenue Summary per ton for 350 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	90%	85%	$55	$60	$64	$69	$73	$78	$83	$87	$92	$96	$101
0.13	90%	85%	$60	$65	$70	$75	$80	$85	$90	$94	$99	$104	$109
0.14	90%	85%	$64	$70	$75	$80	$86	$91	$96	$102	$107	$112	$118
0.15	90%	85%	$69	$75	$80	$86	$92	$98	$103	$109	$115	$120	$126
0.16	90%	85%	$73	$80	$86	$92	$98	$104	$110	$116	$122	$129	$135
0.17	90%	85%	$78	$85	$91	$98	$104	$111	$117	$124	$130	$137	$143
0.18	90%	85%	$83	$90	$96	$103	$110	$117	$124	$131	$138	$145	$151
0.19	90%	85%	$87	$94	$102	$109	$116	$124	$131	$138	$145	$153	$160
0.2	90%	85%	$92	$99	$107	$115	$122	$130	$138	$145	$153	$161	$168
0.21	90%	85%	$96	$104	$112	$120	$129	$137	$145	$153	$161	$169	$177

Revenue Summary per ton 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	90%	85%	$55	$60	$64	$69	$73	$78	$83	$87	$92	$96	$101
0.13	90%	85%	$60	$65	$70	$75	$80	$85	$90	$94	$99	$104	$109
0.14	90%	85%	$64	$70	$75	$80	$86	$91	$96	$102	$107	$112	$118
0.15	90%	85%	$69	$75	$80	$86	$92	$98	$103	$109	$115	$120	$126
0.16	90%	85%	$73	$80	$86	$92	$98	$104	$110	$116	$122	$129	$135
0.17	90%	85%	$78	$85	$91	$98	$104	$111	$117	$124	$130	$137	$143
0.18	90%	85%	$83	$90	$96	$103	$110	$117	$124	$131	$138	$145	$151
0.19	90%	85%	$87	$94	$102	$109	$116	$124	$131	$138	$145	$153	$160
0.2	90%	85%	$92	$99	$107	$115	$122	$130	$138	$145	$153	$161	$168
0.21	90%	85%	$96	$104	$112	$120	$129	$137	$145	$153	$161	$169	$177

Cost Limited
Break Even
Profitable

Revenue Over 1 Years at 350 tons per day

Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	Gold Price ($CAD)										
			$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	90%	85%	$6,747,300.00	$7,309,575.00	$7,871,850.00	$8,434,125.00	$8,996,400.00	$9,558,675.00	$10,120,950.00	$10,683,225.00	$11,245,500.00	$11,807,775.00	$12,370,050.00
0.13	90%	85%	$7,309,575.00	$7,918,706.25	$8,527,837.50	$9,136,968.75	$9,746,100.00	$10,355,231.25	$10,964,362.50	$11,573,493.75	$12,182,625.00	$12,791,756.25	$13,400,887.50
0.14	90%	85%	$7,871,850.00	$8,527,837.50	$9,183,825.00	$9,839,812.50	$10,495,800.00	$11,151,787.50	$11,807,775.00	$12,463,762.50	$13,119,750.00	$13,775,737.50	$14,431,725.00
0.15	90%	85%	$8,434,125.00	$9,136,968.75	$9,839,812.50	$10,542,656.25	$11,245,500.00	$11,948,343.75	$12,651,187.50	$13,354,031.25	$14,056,875.00	$14,759,718.75	$15,462,562.50
0.16	90%	85%	$8,996,400.00	$9,746,100.00	$10,495,800.00	$11,245,500.00	$11,995,200.00	$12,744,900.00	$13,494,600.00	$14,244,300.00	$14,994,000.00	$15,743,700.00	$16,493,400.00
0.17	90%	85%	$9,558,675.00	$10,355,231.25	$11,151,787.50	$11,948,343.75	$12,744,900.00	$13,541,456.25	$14,338,012.50	$15,134,568.75	$15,931,125.00	$16,727,681.25	$17,524,237.50
0.18	90%	85%	$10,120,950.00	$10,964,362.50	$11,807,775.00	$12,651,187.50	$13,494,600.00	$14,338,012.50	$15,181,425.00	$16,024,837.50	$16,868,250.00	$17,711,662.50	$18,555,075.00
0.19	90%	85%	$10,683,225.00	$11,573,493.75	$12,463,762.50	$13,354,031.25	$14,244,300.00	$15,134,568.75	$16,024,837.50	$16,915,106.25	$17,805,375.00	$18,695,643.75	$19,585,912.50
0.2	90%	85%	$11,245,500.00	$12,182,625.00	$13,119,750.00	$14,056,875.00	$14,994,000.00	$15,931,125.00	$16,868,250.00	$17,805,375.00	$18,742,500.00	$19,679,625.00	$20,616,750.00
0.21	90%	85%	$11,807,775.00	$12,791,756.25	$13,775,737.50	$14,759,718.75	$15,743,700.00	$16,727,681.25	$17,711,662.50	$18,695,643.75	$19,679,625.00	$20,663,606.25	$21,647,587.50

Revenue over 3 years at 600 tons per day

Grade (Oz/ton)	Recovery (Mining)	Recovery (Milling)	Gold Price ($CAD)										
			$600	$650	$700	$750	$800	$850	$900	$950	$1,000	$1,050	$1,100
0.12	90%	85%	$34,700,400.00	$37,592,100.00	$40,483,800.00	$43,375,500.00	$46,267,200.00	$49,158,900.00	$52,050,600.00	$54,942,300.00	$57,834,000.00	$60,725,700.00	$63,617,400.00
0.13	90%	85%	$37,592,100.00	$40,724,775.00	$43,857,450.00	$46,990,125.00	$50,122,800.00	$53,255,475.00	$56,388,150.00	$59,520,825.00	$62,653,500.00	$65,786,175.00	$68,918,850.00
0.14	90%	85%	$40,483,800.00	$43,857,450.00	$47,231,100.00	$50,604,750.00	$53,978,400.00	$57,352,050.00	$60,725,700.00	$64,099,350.00	$67,473,000.00	$70,846,650.00	$74,220,300.00
0.15	90%	85%	$43,375,500.00	$46,990,125.00	$50,604,750.00	$54,219,375.00	$57,834,000.00	$61,448,625.00	$65,063,250.00	$68,677,875.00	$72,292,500.00	$75,907,125.00	$79,521,750.00
0.16	90%	85%	$46,267,200.00	$50,122,800.00	$53,978,400.00	$57,834,000.00	$61,689,600.00	$65,545,200.00	$69,400,800.00	$73,256,400.00	$77,112,000.00	$80,967,600.00	$84,823,200.00
0.17	90%	85%	$49,158,900.00	$53,255,475.00	$57,352,050.00	$61,448,625.00	$65,545,200.00	$69,641,775.00	$73,738,350.00	$77,834,925.00	$81,931,500.00	$86,028,075.00	$90,124,650.00
0.18	90%	85%	$52,050,600.00	$56,388,150.00	$60,725,700.00	$65,063,250.00	$69,400,800.00	$73,738,350.00	$78,075,900.00	$82,413,450.00	$86,751,000.00	$91,088,550.00	$95,426,100.00
0.19	90%	85%	$54,942,300.00	$59,520,825.00	$64,099,350.00	$68,677,875.00	$73,256,400.00	$77,834,925.00	$82,413,450.00	$86,991,975.00	$91,570,500.00	$96,149,025.00	$100,727,550.00
0.2	90%	85%	$57,834,000.00	$62,653,500.00	$67,473,000.00	$72,292,500.00	$77,112,000.00	$81,931,500.00	$86,751,000.00	$91,570,500.00	$96,390,000.00	$101,209,500.00	$106,029,000.00
0.21	90%	85%	$60,725,700.00	$65,786,175.00	$70,846,650.00	$75,907,125.00	$80,967,600.00	$86,028,075.00	$91,088,550.00	$96,149,025.00	$101,209,500.00	$106,269,975.00	$111,330,450.00

Figure 2 - Yearly Revenue Summary

Recommendations

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

- 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 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 13.4 million (development and production costs) ±25%, say, $C 15 million at the top cost range.

Table of Contents

Table of Tables

Table of Figures

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 2007-2011. This work, all completed on the 2250 Level and on surface, consists of
- 46,700 feet of underground diamond drilling, related logging and assaying;
- 10,200 feet of surface diamond drilling testing the near surface at McGarry;
- 4,100 feet of surface diamond drilling testing the Mill Zone, 3,000 ft south of the McGarry Shaft
- 2,408 feet of drifting and related chip sampling;
    - 130 feet of raising;
    - bulk sampling from 34 drift rounds;
    - trial mining from two test stopes; and
    - 1,000 feet of pre-development drifting on the 2250 Level

- to present the results of background metallurgical work;
- 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 preliminary economic assessment considering a potential mining strategy for the development and production of gold zones between the 1250 and 2250 Levels at the McGarry Project.
- To amend the April 8, 2009 technical report to meet the independence requirements of NI 43-101 and remove an inappropriate disclaimer

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 and/or at the McGarry Project Site at Virginiatown, Ontario. 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 6.4 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 and Virginiatown offices of Armistice Resources Corp. Information on the GEMS software is available at www.gemcomsoftware.com.

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 (oz/t) = 34.286 grams per metric ton (g/t)

= 34.286 parts per million ("ppm")

Reference is made to the "shrinkage" stoping method in this report. This is a 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.

Throughout 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 6.
- 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 and surface geological mapping carried out for the 2007-2011 drilling and mining programmes by James Thompson, BSc., Chief Geologist for Armistice and Jean-François Leclerc-Cloutier, MSc, independent geological consultant to Armistice. General supervision and planning was under the direction of Erik Andersen, P. Eng, Vice President and Chief Operating Officer of Armistice and non-independent Qualified Person. 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, specifically S.J. Carmichael in the NI43-101 compliant report, "Armistice Resources Ltd. Virginiatown Gold Project, McGarry Township, Ontario" (Carmichael, S.J., June 6, 2004) unless otherwise stated.

All maps and illustrations have been produced by Armistice Staff and are updated to the effective date of this report. The author is satisfied that these maps and illustrations present the data and information used in this report accurately within the scope of this report.

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 13 September 2011.

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 3.

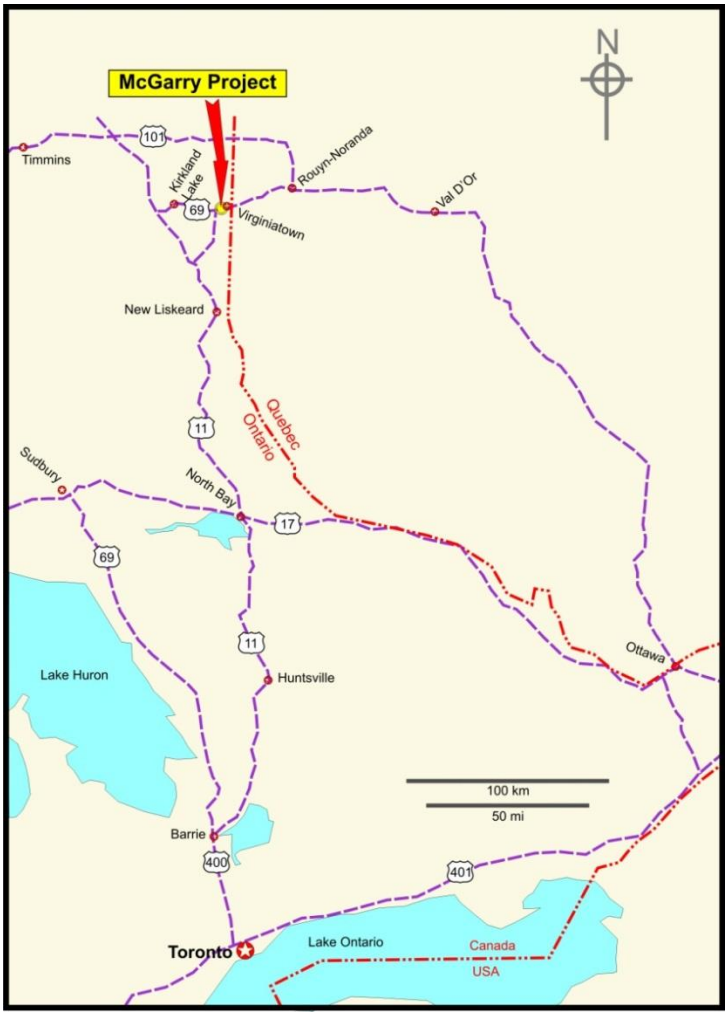


Figure 3 - General Location Map

The regional location of the Property is shown in Figure 4. The Property is centred at approximately 79°36'W and 48°07'N. The NTS reference is 32D/SW

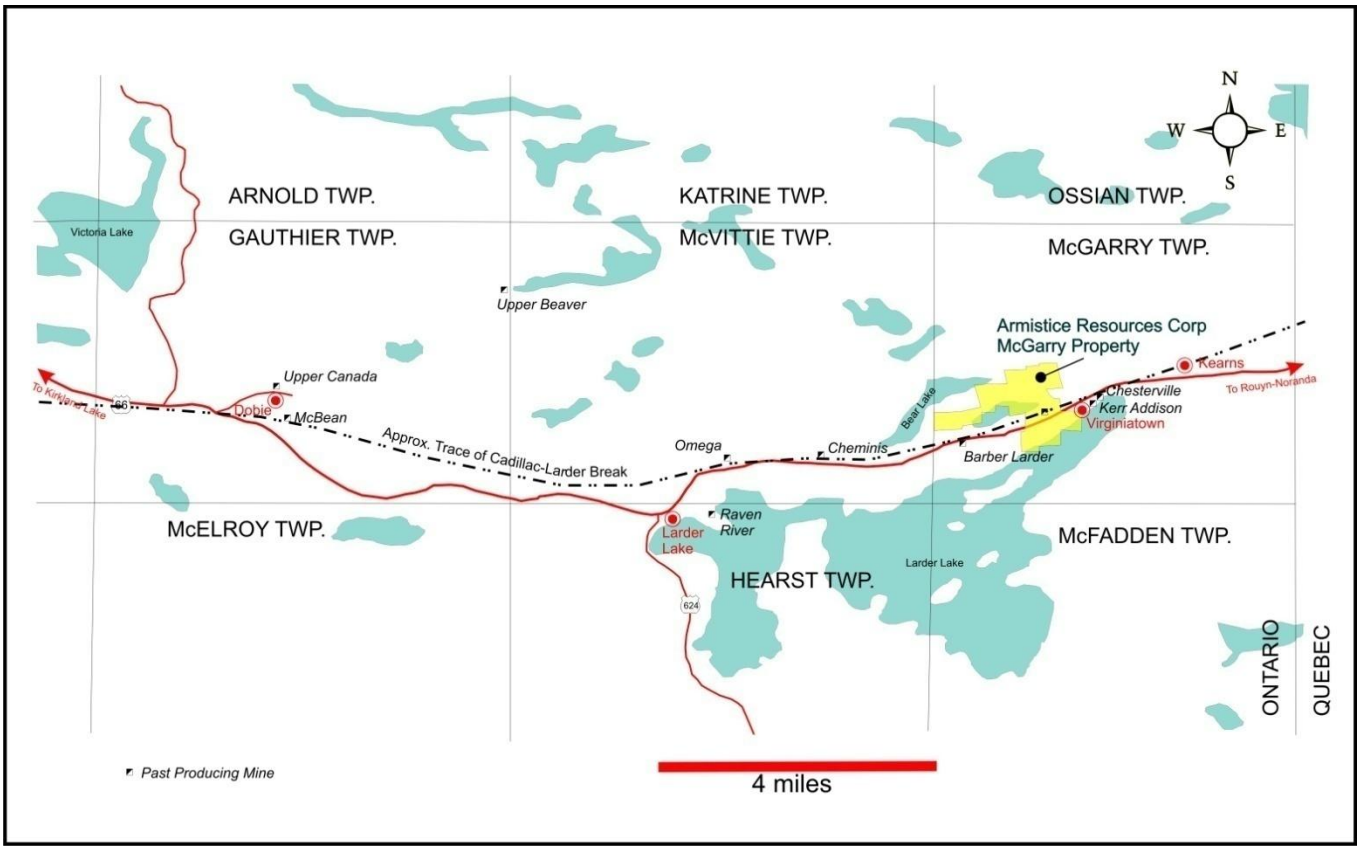


Figure 4 - Regional Location Map

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 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 $ 29,111 and $ 2,637 respectively in 2011. The author has relied on an opinion by Fasken Martineau DuMoulin LLP of Toronto, Ontario dated 13 September 2011 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 author is not aware of any significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

The claims are shown in Figure 5 and are listed below in Table 4. The mine grid and workings in respect to the property boundaries and geographic features is shown in Figure 6.

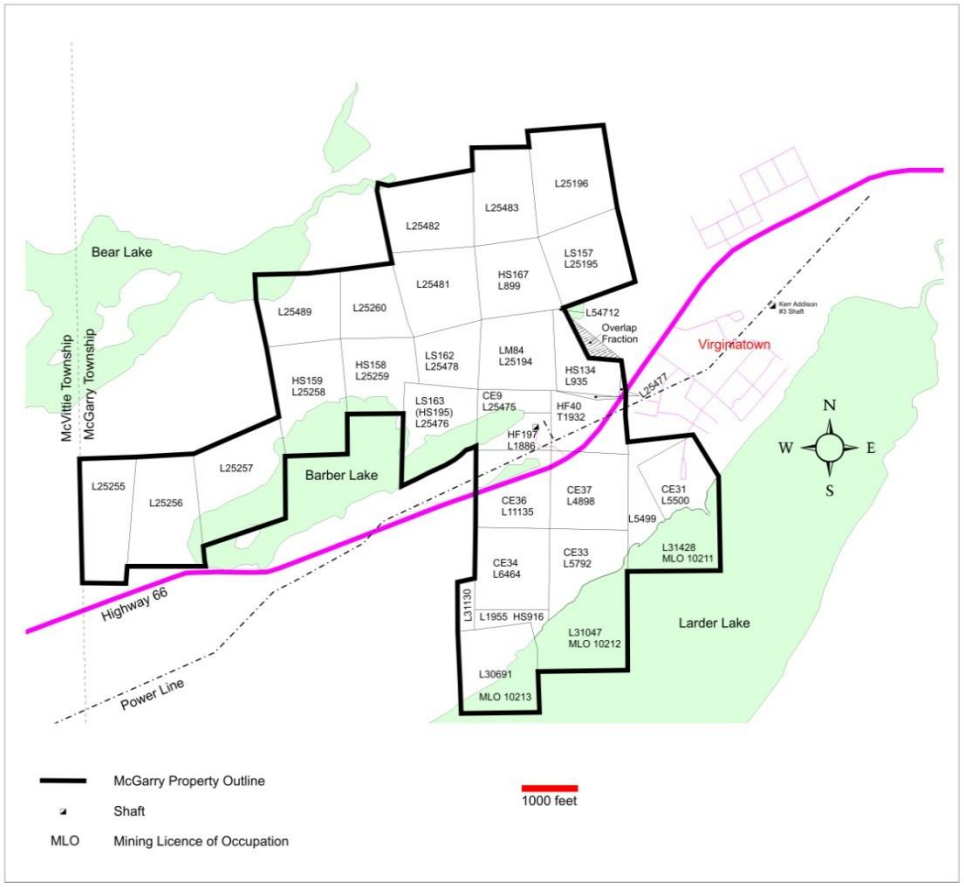


Figure 5 - Property Claim Map

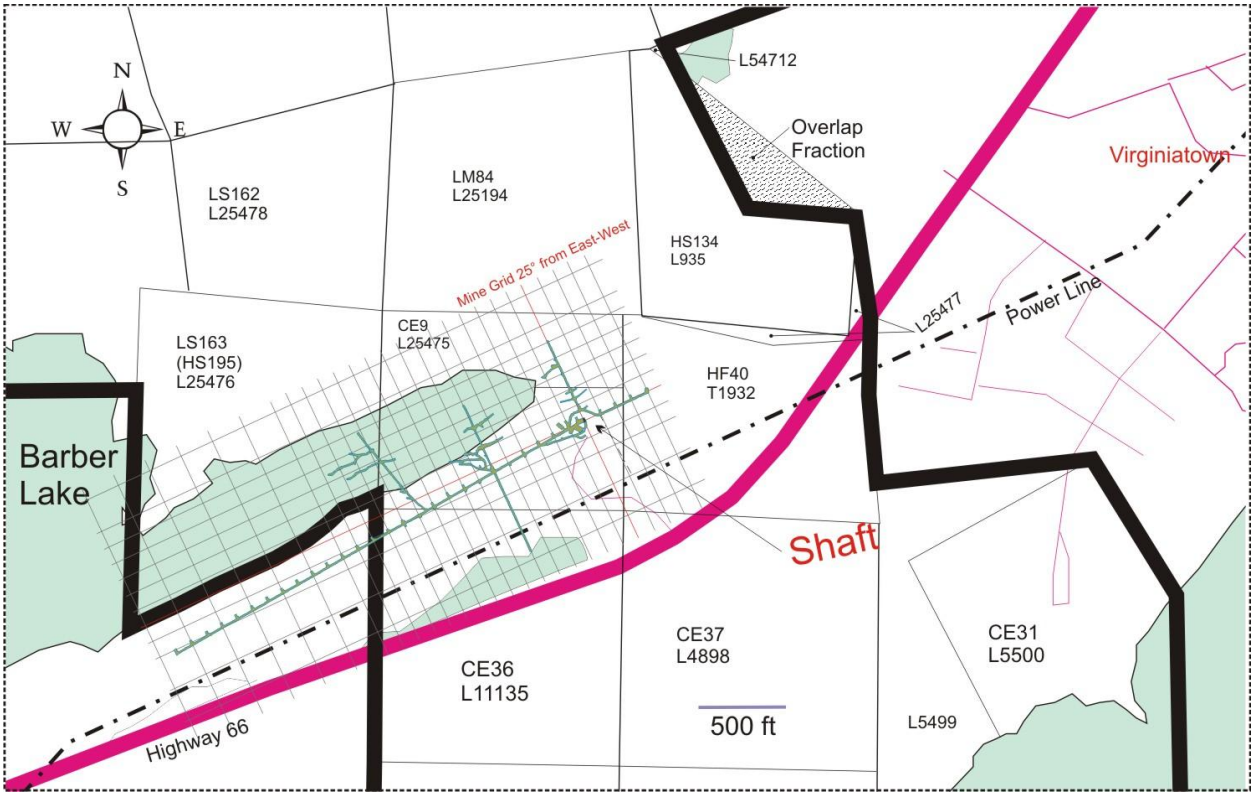


Figure 6 - Detail of figure showing workings on 2250 Level & Mine Grid

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 4 - List of Mining Claims

4.1 Royalty Interest

Armistice holds an undivided 75 % interest in the Property through an agreement with Jubilee Gold Inc. (formerly Sheldon-Larder Mines Limited) ("Jubilee") dated 30 June 2004. The remaining 25% interest is a carried interest entitling Jubilee 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 Jubilee. This results in complete control by Armistice including 100% of proceeds of production subject only to the royalty interest. The royalty interest held by Jubilee 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. The payment amount is subject to certain cost of living adjustments.

As of the date of this Report, Armistice was in compliance with all of its royalty payment obligations with respect to Jubilee under terms of the agreement. Armistice is entitled to not making certain advance royalty payments without being in default and such arrears will not be recoupable from royalty payments after commencement of commercial production.

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

As shown in Figure 6, 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 area receives ample precipitation. Common vegetation is second growth poplar, birch, pine, alder and scrub maple. The property is located at an elevation of about 1100 ft above sea level with locate topographic elevation relief up to 100 ft.

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 5 and Figure 6). 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.

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.

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 (now Jubilee Gold Inc.), 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 (now Jubilee). 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. This report is not NI 43-101 compliant and is not relied upon.

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 7 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 2007-2009 work period.

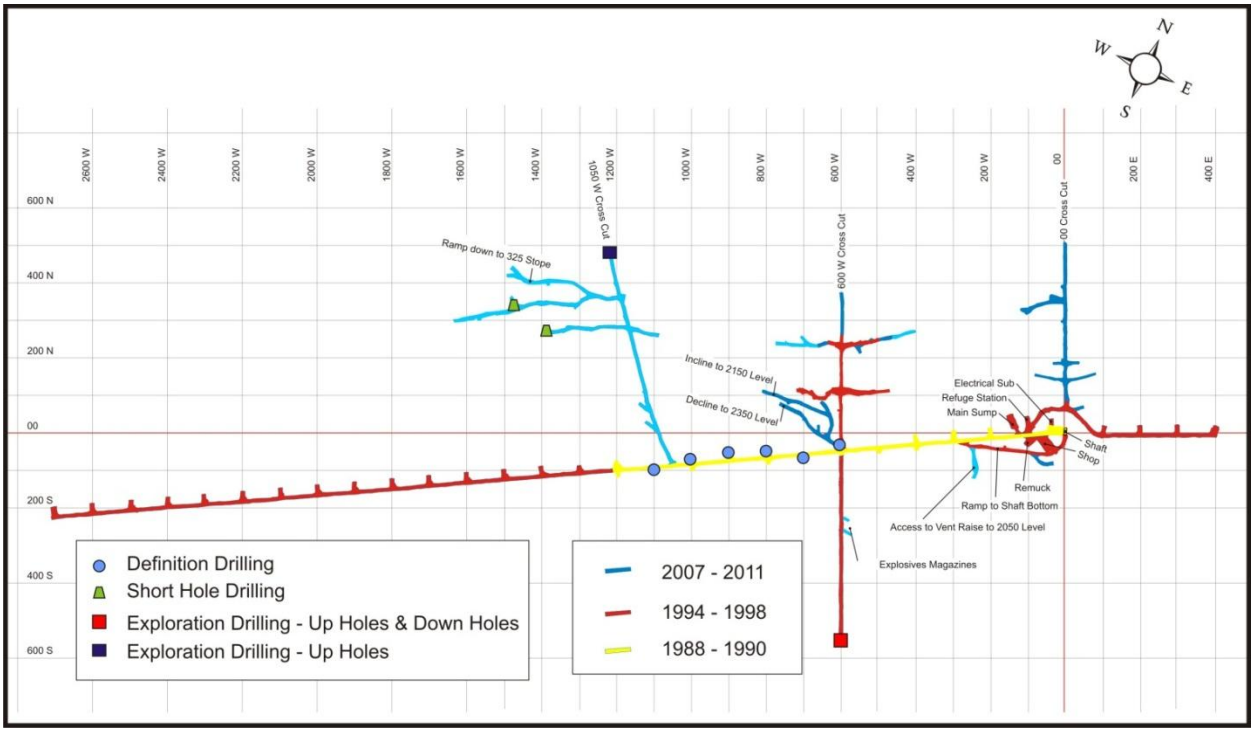


Figure 7 - 2250 Level Mining Development and Drill Stations by Work Period and 2007-2011

In 2009, Armistice carried a programme of diamond drilling in three parts of the property:

A series of 14 underground holes were drilled from the western end of the 325W drift off the 1050W X/C to test the mineralization in the 325W Zone initially test by the “short hole” programme in 2008.

Fifteen surface holes within 450 feet of the McGarry shaft to the west testing the McGarry gold mineralization potential above the 1250 level.

Fourteen surface holes testing the McGarry Mill Zone located approximately 3000 feet south of the McGarry shaft.

In April 2011, Armistice embarked on an underground mining programme to begin pre-production level development on the 2250 Level and to complete infrastructure projects required in advance of beginning production including 2250 Level loading pocket upgrade and access drifts for ore pass raises and ventilation/escapeway raises. New capital equipment purchases have also been made including adding one new skip/cage combination to double the skipping capacity, underground diamond drills, raise climber (“Alimak”) rebuild, long toms, and tracked equipment for the 2050 Level. This programme is ongoing. Approximately 1000 feet of level development in waste on the 2250 Level has been completed to date.

7. Geological Setting

7.0 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 cross faults. 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 8

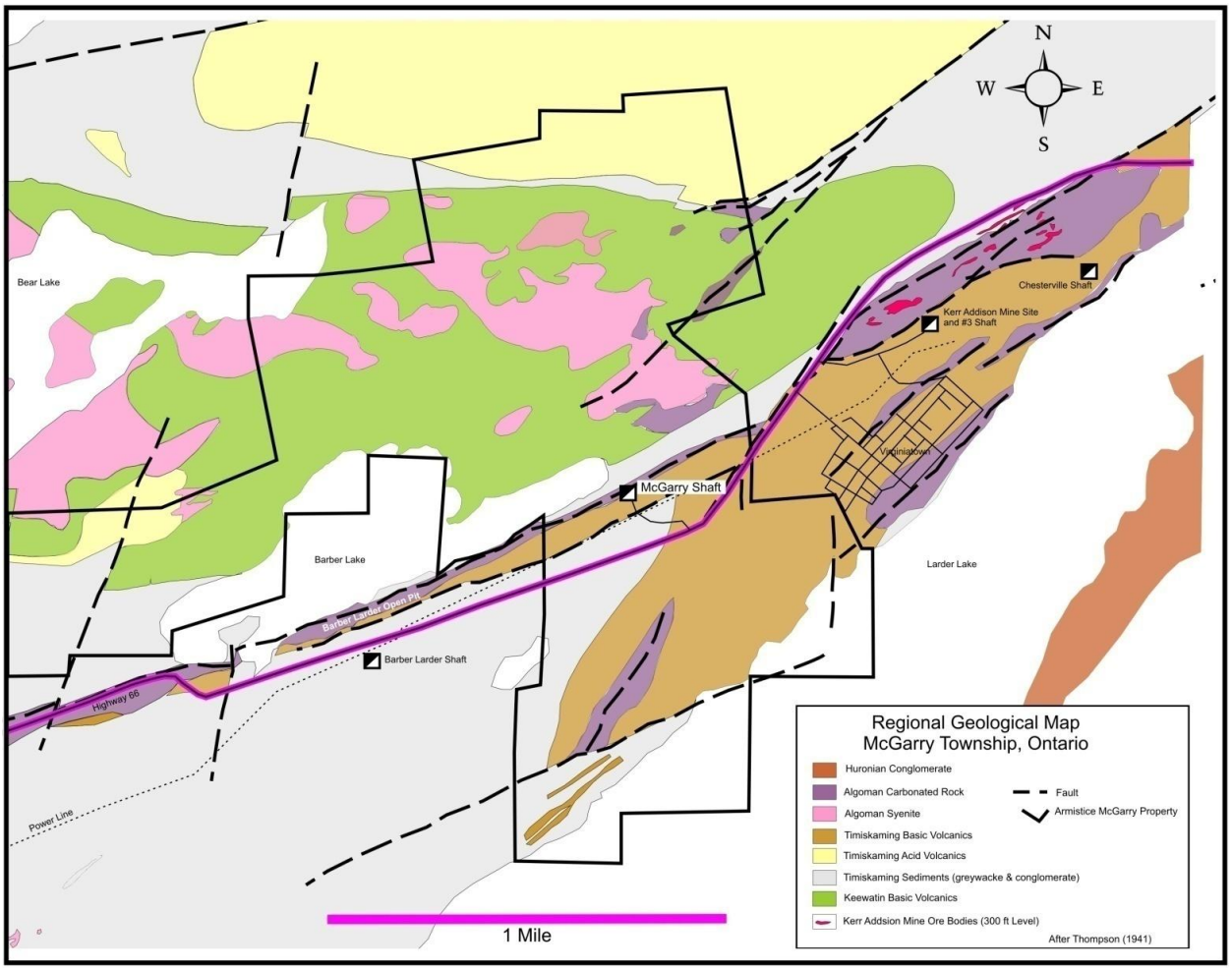


Figure 8 - Regional Geological Map

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 4 (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	Formations in McGarry Project Area
			Disconformity			
			K-rich flows, Trachyte, Syenite		Strong subsidence, recrystallization, melting, volcanism, intrusion	
			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 4 - 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.

7.1 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 9.

Table 5 provides a detailed description of the rock types corresponding to the schematic cross section proceeding from north to south.

Figure 10 shows the interpreted geological plan based on the diamond drilling and mined openings on the 2250 Level. Figure 11, shows a simplified geological section through about 600 W.

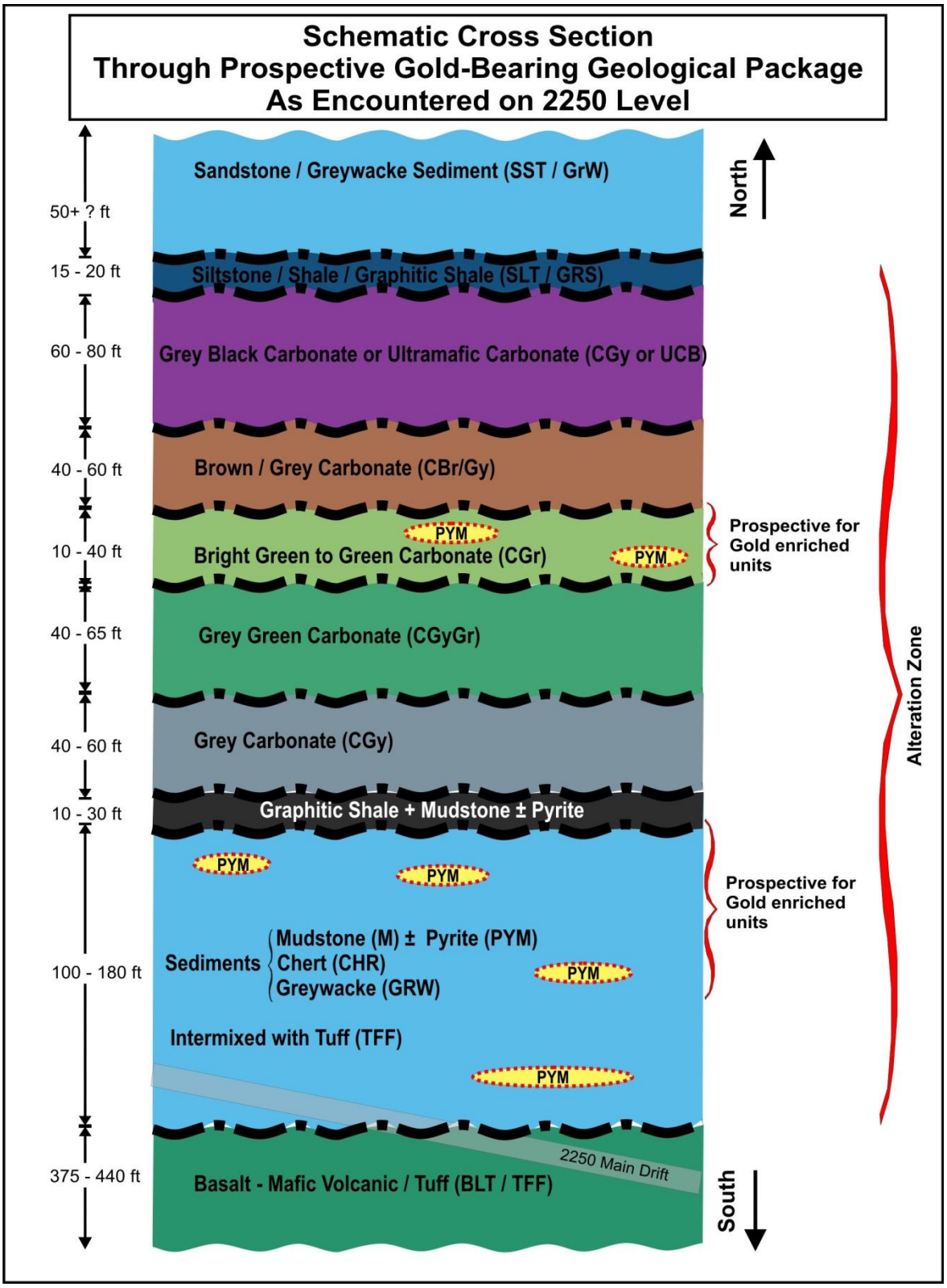


Figure 9 - Schematic Section through the Prospective Geological Package as Encountered on 2250 Level

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 5 - McGarry Rock Type Descriptions

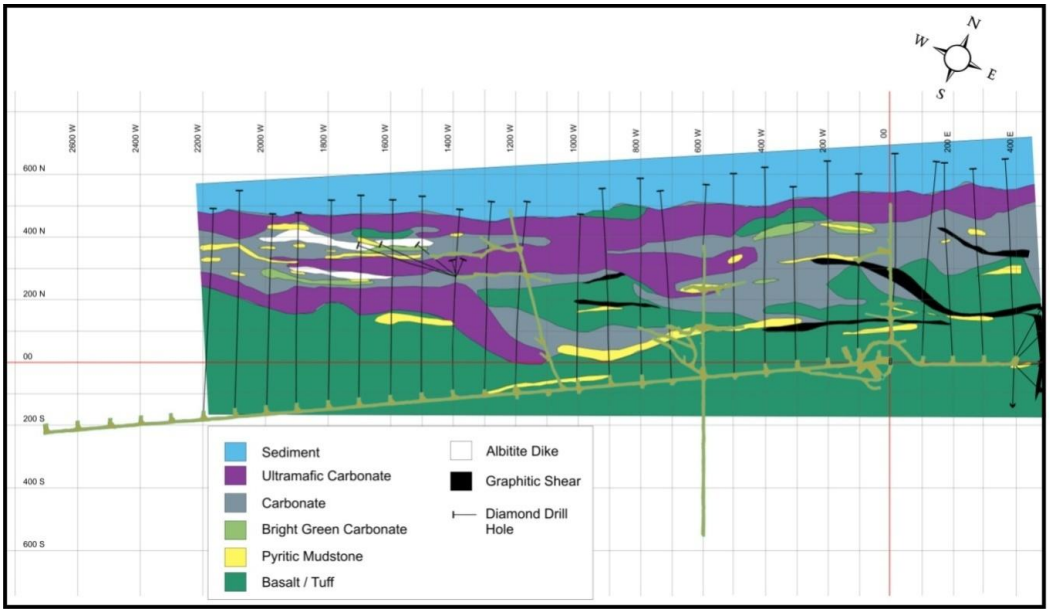


Figure 10 – Simplified Geological Plan – 2250 Level

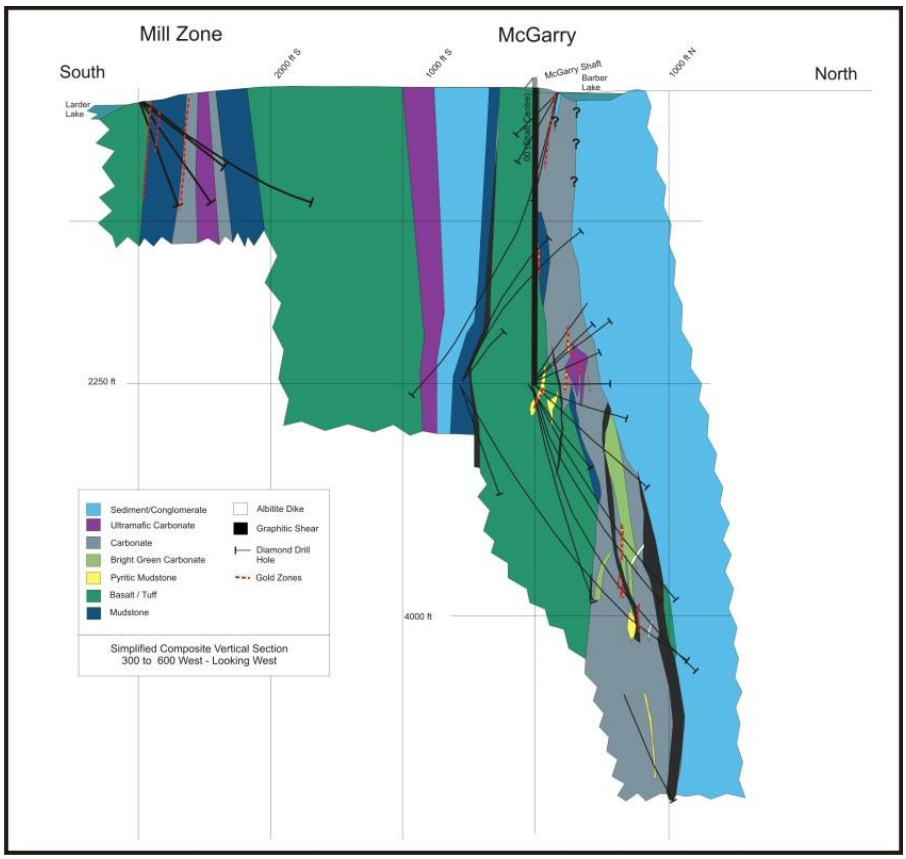


Figure 11 - Simplified Geological Vertical Section

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.

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.

An important 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. Adjacent Properties

9.0 Bear Lake Gold

The Barber Larder property of Bear Lake Gold Ltd. 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. 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. A press release dated 6 September 2011 by Bear Lake Gold indicates the definition of an NI 43-101 compliant Inferred Mineral Resource totalling 3.75 million metric tons grading 5.67 grams per metric ton in the Bear Lake Zone near the western end of Bear Lake (about 0.75 miles west of the McGarry Property.

10. Mineralization

Pyritic mudstone units ("flow ore") constitute an 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 12 shows the gold-bearing zones as identified on the 2250 Level.

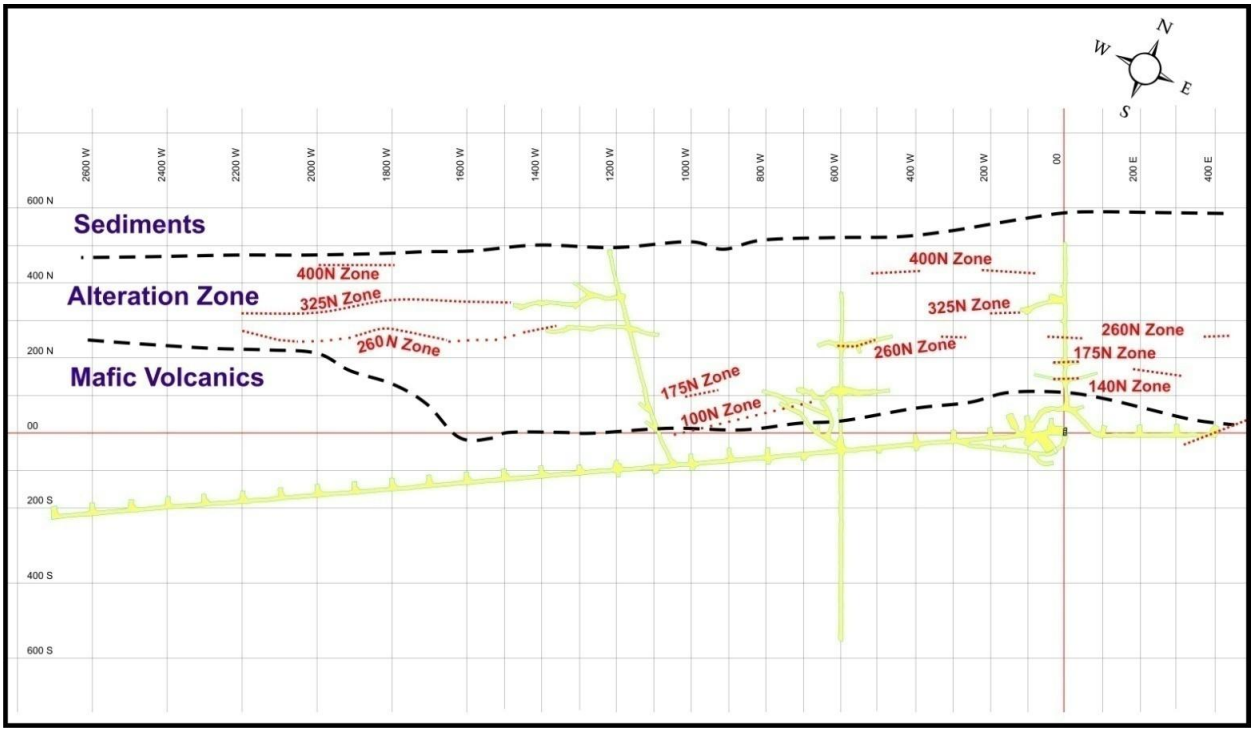


Figure 12 - Simplified Geology and Gold-Bearing Zones on the 2250 Level

11. 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 other than diamond drilling.

12. 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 2007-2009 programmes.

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 2007-2009 programmes were surveyed with a single-shot down hole Flex-It or PeeWee unit from which the dip, magnetic bearing, total magnetic field and temperature is read. Down hole surveys were taken every 100 feet in the 2007-2009 programmes.

All drilling in the 2007-2009 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 VAG drill was used for detailed testing ahead of drifting advance. The drill was set up in the western end of the 260W Drift and the 325W 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. The author is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

A summary of the drilling performed on the Property and incorporated into the Gemcom database is presented in Table 6.

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		44,451
	Down Holes	3	5	
	Up Holes	9		
	Short Holes	5		
2009	Short Holes	14		2,243
	Surface	15		10,242
	Mill Zone	7		4,099
Sub Total 2007-2009		79	5	61,035
Grand Total to Date		**443**	**26**	**318,903**

Table 6 - Summary Drilling Statistics

To illustrate the overall density of the drilling included in the Gemcom database, Figure 13, below shows the traces of all holes in composite cross section and longitudinal section.

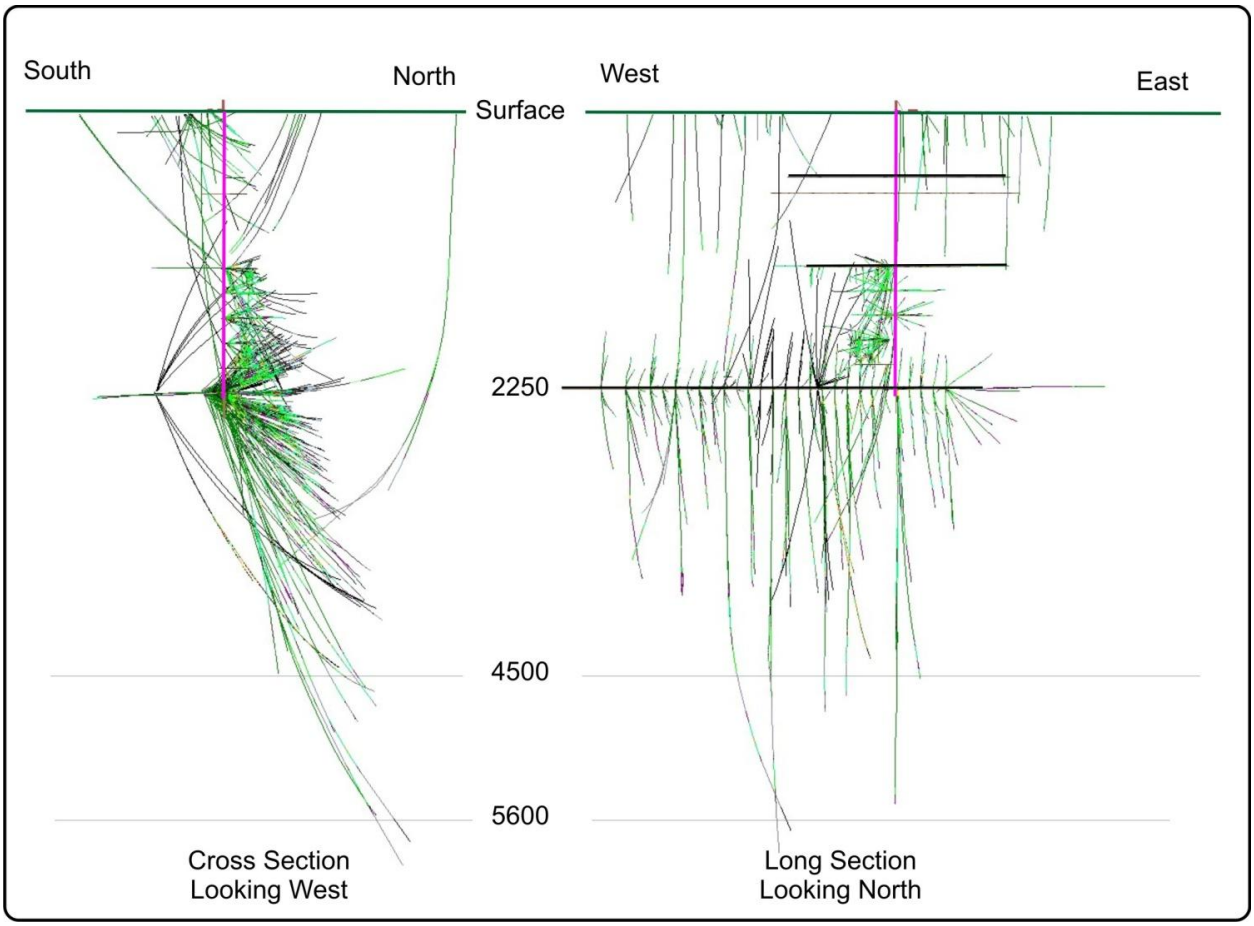


Figure 13 - Composite Cross Section and Long Section Showing All Drill Holes

12.0 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 13 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 7.

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 7 - 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 down dip 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 Table 8.

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 8 - List of Significant Intersections below 4,000ft

Figure 14 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.

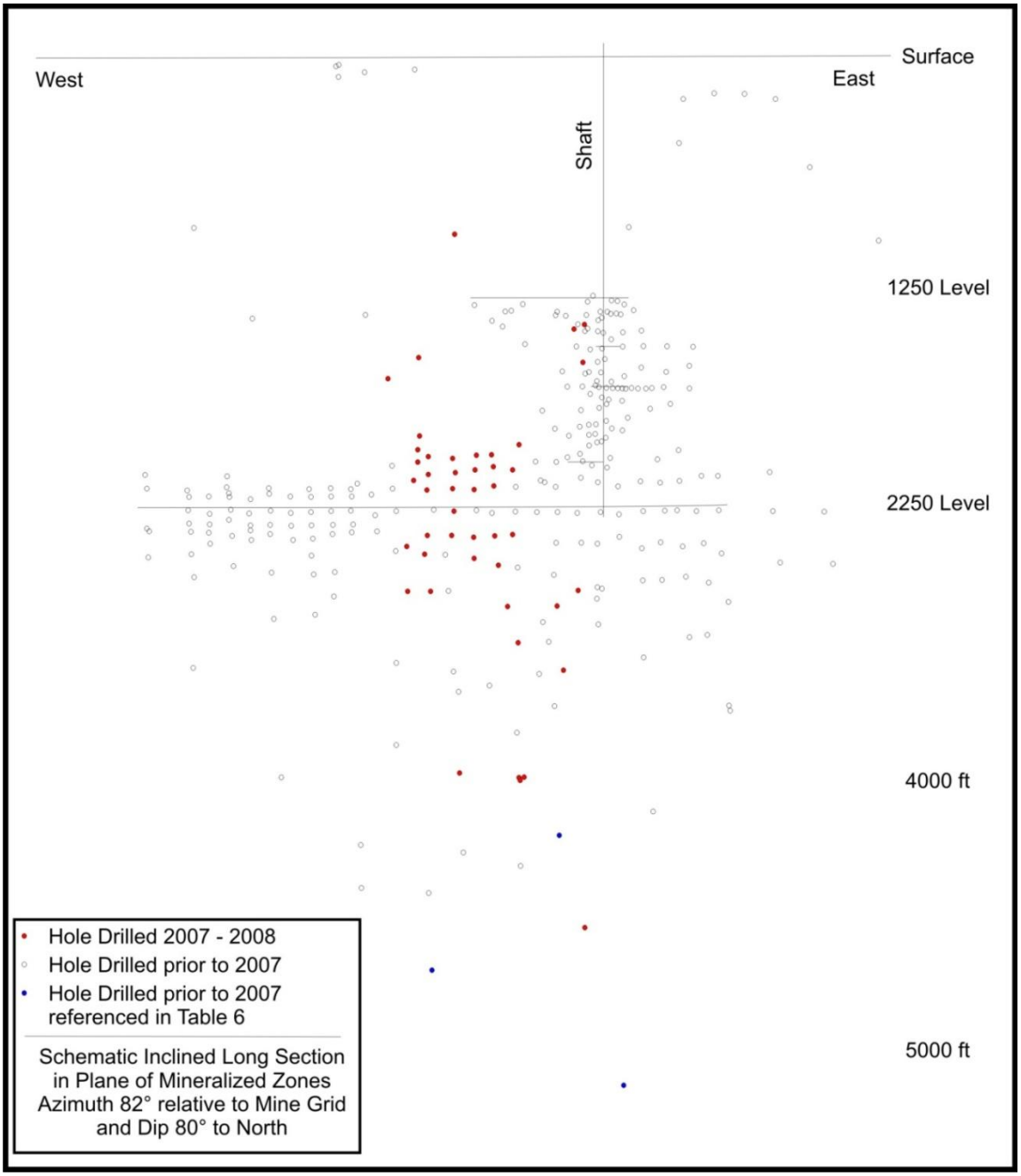


Figure 14 - Schematic Long Section in Plane of Mineralized Zones Showing Drill Holes

12.1 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.

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 7 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 previous campaigns were also drilled at steeper dips (up to -70°) to test the prospective geological package at deeper elevations.

Table 9 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 9 - Tabulation of All Significant Assays from Definition Drill Section 2007-2008

12.2 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 15.

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 7), are not ideal for this testing but they were the best available.

Table 10, 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 10 - Tabulation of Results from Up-Hole Drill Programme

12.3 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 7). 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 15, 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 – see Figure 15. 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.

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.

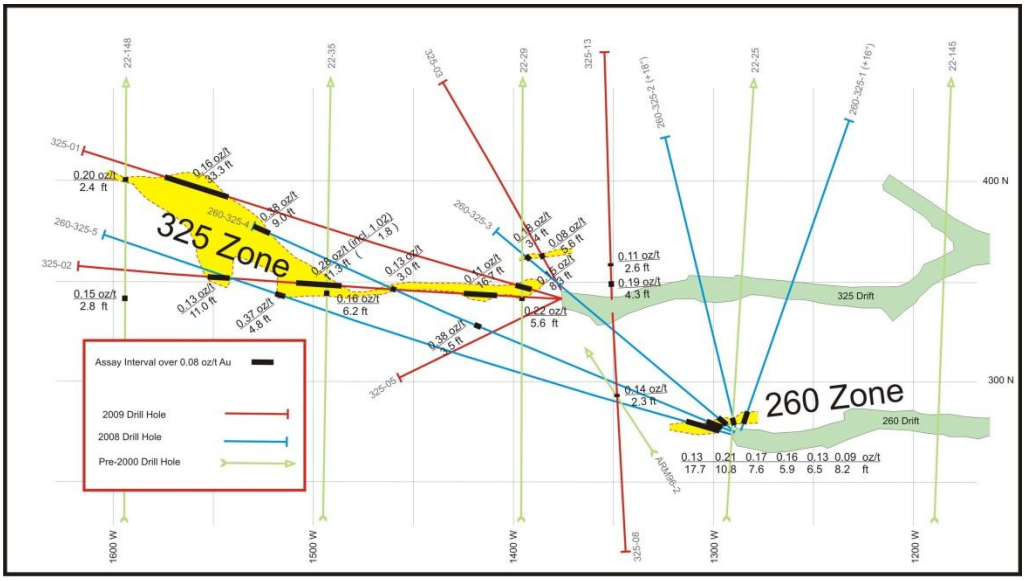


Figure 15 - Plan of Short Drilling at West End of 260N and 325N Zones

Significant Assay Results from 2008-2009 Short Hole Drilling Programme - 325N & 260N Zones - 2250 Level -								
Year	DDH #	Dip at Collar	Azimuth at Collar	From	To	Interval	Average Gold	Zone
				feet	feet	feet	oz/t	
2009	325-01	0.0°	282.9°	15.3	23.6	8.3	0.15	325N
			including	*17.6*	*18.9*	*1.3*	*0.46*	*325N*
				134.5	136.4	1.9	0.10	325N
				173.7	207.0	33.3	0.16	325N
			including	*182.0*	*204.5*	*22.5*	*0.20*	*325N*
	325-02	-0.1°	273.7°	32.0	48.7	16.7	0.11	325N
			including	*46.9*	*48.7*	*1.8*	*0.65*	*325N*
				110.0	132.8	22.8	0.17	325N
			including	*110.0*	*121.3*	*11.3*	*0.28*	*325N*
			and	*116.7*	*118.5*	*1.8*	*1.02*	*325N*
				166.0	177.0	11.0	0.13	325N
	325-03	+0.1°	344.7°	17.0	22.6	5.6	0.08	325N
	325-08	+1.3°	329.3	42.6	44.9	2.3	0.14	325N
	325-13	+0.5°	140.0°	7.0	11.3	4.3	0.19	325N
				16.1	18.7	2.6	0.11	325N
2008	260-325-1	+16.3°	+19.3°	0.6	8.8	8.2	0.09	260N
	260-325-2	+17.7°	+346.7°	0.0	6.5	6.5	0.13	260N
	260-325-3	-0.2°	310.2°	2.5	10.1	7.6	0.17	260N
	260-325-4	-0.5°	261.0°	3.6	14.4	10.8	0.21	325N
				136.6	140.1	3.5	0.38	?
				182.0	185.0	3.0	0.13	325N
				252.0	261.0	9.0	0.38	325N
	260-325-5	-0.4°	288.6°	6.5	24.2	17.7	0.13	260N
				235.0	239.8	4.8	0.37	325N
1996-1998	22-25	+3.0°	4.0°	362.0	367.9	5.9	0.16	260N
	22-29	+2.0°	0.0°	432.0	437.6	5.6	0.22	325N
	22-35	+2.0°	1.0°	446.8	453.0	6.2	0.16	325N
	22-148	-2.0°	+1.9°	452.2	455.0	2.8	0.15	?
				511.7	514.1	2.4	0.20	325N

Table 11 - Significant Assay Results from 2008-2009 Short Hole Drilling Programme - 325N & 260N Zones

12.4 Surface Drilling

In 2009, a diamond drilling programme consisting of four 3-hole fans at nominal -35°, -55° and -75° dips; and two partial fans of two and one hole each was completed to test the upward continuation of the gold mineralization identified below 1250 feet at McGarry. Drill fans are at nominal 50-ft intervals along strike and drilled from north to south. The shallow dipping holes did not encounter significant gold zones, but three of the four steep holes did. Fifteen holes were drilled in this programme totalling 10,242 feet. The holes are located between sections 180W and 450W.

Table 12, below, summarizes the significant assay results from this programme. Figure 16 presents a composite section at 350W that shows the relation of the surface drilling to the underground workings and drill results from previous underground programmes.

Drill Hole	Collar					Hole Length (feet)	From (feet)	To (feet)	Interval (feet)	True Width (feet)	Assay (oz/t)
	Easting (feet)	Northing (feet)	Elevation (feet)	Dip	Azimuth						
AR09-01	448W	166N	-42	-55°	180°	580	no significant assays				
AR09-02	448W	168N	-43	-76°	170°	800	no significant assays				
AR09-03	448W	167N	-42	-36°	179°	250	no significant assays				
AR09-04	402W	168N	-39	-75°	177°	3260	291.4	292.5	1.1	0.3	0.14
AR09-05	402W	165N	-39	-55°	178°	400	no significant assays				
AR09-06	402W	163N	-38	-40°	182°	300	no significant assays				
AR09-07	347W	168N	-35	-75°	185°	600	308.0	316.7	8.7	2.2	0.10
							347.2	348.3	1.1	0.3	0.10
							464.5	466.7	2.2	0.6	0.24
AR09-08	348W	165N	-35	-55°	180°	450	no significant assays				
AR09-09	348W	162N	-35	-40°	186°	350	14.0	15.8	1.8	1.5	0.14
							91.8	92.8	1.0	0.8	0.10
AR09-10	303W	163N	-32	-76°	168°	678	43.2	44.7	2.4	0.6	0.08
							63.7	67.0	3.3	0.8	0.10
							340.7	343.4	2.7	0.6	0.09
							569.3	573.5	4.2	1.0	0.11
AR09-11	303W	161N	-31	-54°	179°	400	no significant assays				
AR09-12	303W	157N	-32	-38°	179°	344	8.8	14.8	6.0	4.7	0.11
							26.4	27.4	1.0	0.8	0.31
AR09-13	250W	159N	-30	-74°	186°	212	no significant assays				
AR09-14	250W	159N	-30	-67°	177°	648	no significant assays				
AR09-15	180W	147N	-24	-74°	178°	630	no significant assays				

Table 12 - Tabulation of Significant Assays from Surface Drilling 2009

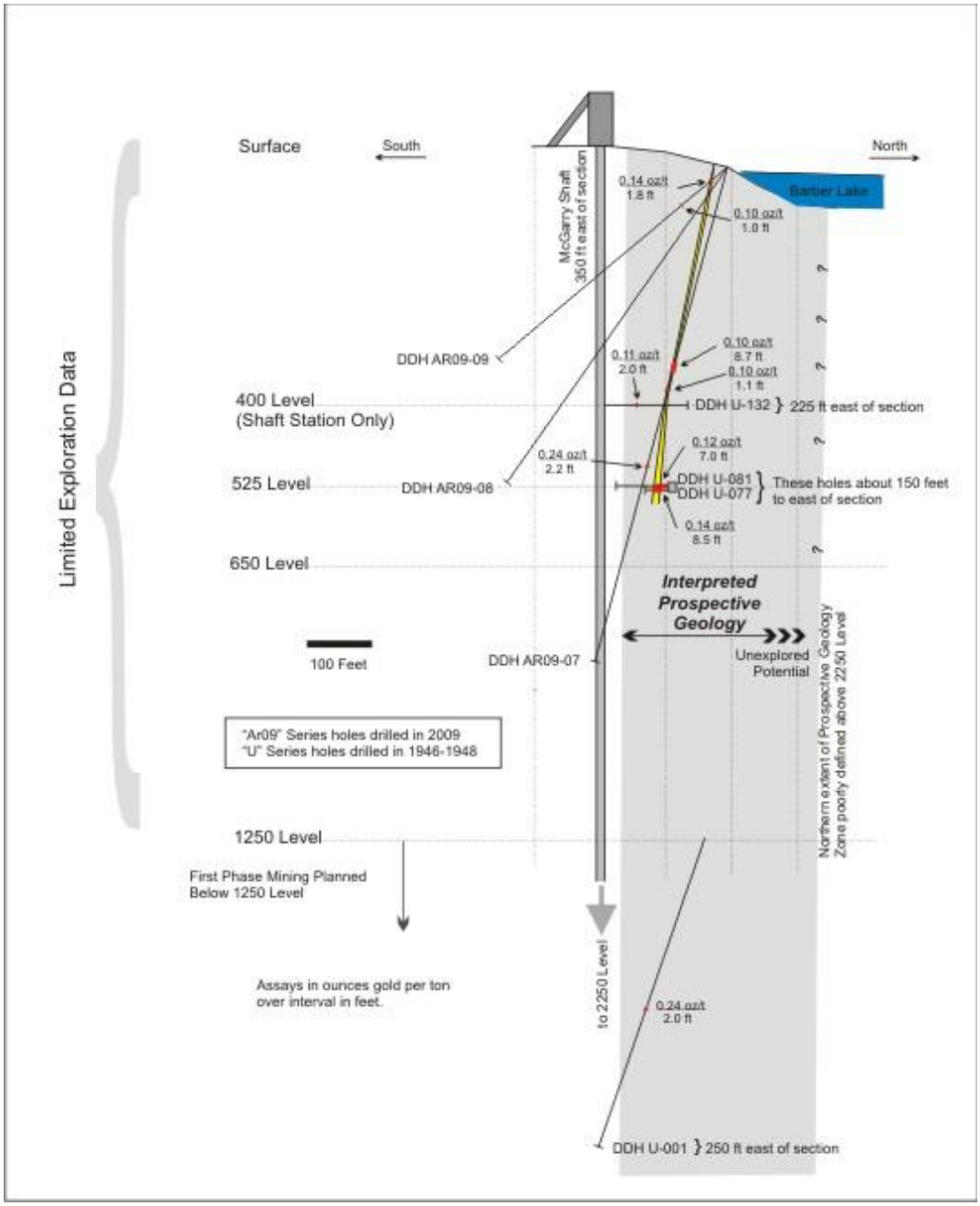


Figure 16 - Schematic Composite Section 350W Showing Surface Drilling and Mine Workings

12.5 Mill Zone Drilling

In 2009, seven surface holes totalling 4,099 feet were drilled on the McGarry Mill Zone.

There have been several phases of previous drilling along the McGarry Mill Zone. There are historical records in Armistice files showing this work dating back to the 1930s. In some cases only maps or drill sections are available. From these maps and sections, assay values and measured intercepts along the drill holes have been extracted. Although the extracted data is believed to be reliable, this cannot be verified and therefore the extracted results cannot be used in any future resource estimate. Historical drill holes were drilled with a variety of directions which may indicate the continuity of intersected gold zones was not easily determined at the time.

One of the objectives of the 2009 drill programme was to evaluate the reliability of the available historical data. Diamond drill hole (DDH) MZ09-05 was drilled to twin the 1940's DDH S38. The results of the two twinned holes are not identical but the gold zones located in DDH S38 are clearly identified in DDH MZ09-05. From this test, it is concludes that the results in the historical record can be used as a guide to exploration. The hole-to-hole comparison is tabulated below in Table 13 - Comparison of Assay Results between Historical Drill Hole S38 and 2009 Drill Hole AZ09-05.

Note that there is about a 10 ft difference in the down hole distances due to the relative positions of the hole collars. The final intersection in the 2009 hole does not appear to have a corresponding value in the 1940's hole – it is not possible to determine if this section of the 1940's hole was sampled or not since only significant assays are shown on the section plan in Armistice files.

DDH AZ09-05 (2009)					DDH S38 (1940's)			
From	**To**	**Interval**	**Assay**		**From**	**To**	**Interval**	**Assay**
72.0	**73.0**	**1.0 ft**	**0.12 oz/t Au**	**< -- >**	**65.0**	**67.5**	**2.5 ft**	**0.15 oz/t Au**
83.2	**85.4**	**2.2 ft**	**0.36 oz/t Au**	**< -- >**	**70.0**	**72.5**	**2.5 ft**	**0.68 oz/t Au**
177.1	**180.0**	**2.9 ft**	**0.13 oz/t Au**	**< -- >**	**155.5**	**160.5**	**5.0 ft**	**0.07 oz/t Au**
204.5	**210.0**	**5.5 ft**	**0.15 oz/t Au**	**< -- >**	**no corresponding assays shown on section map**			

Table 13 - Comparison of Assay Results between Historical Drill Hole S38 and 2009 Drill Hole AZ09-05

The second conclusion from the 2009 drilling is that there is support to interpret the gold zones to be nearly vertical and continuous over distances of at least 500 feet. Previous interpretation of the Mill Zone was complex. The results of the 2009 drilling provide a simpler interpretation opening the Mill Zone to systematic exploration drilling from surface along strike to the west and to the east.

The Table 14 below summarizes the results from the 2009 Mill Zone drilling on section 1100 W (see Figure 17)

DDH	From (feet)	To (feet)	Interval (feet)	Interpreted True Width (feet)	Assay (oz/ton gold)	Zone
MZ09-05	72.0	73.0	1.0	0.8	0.12	Mill Zone South
	83.2	85.4	2.2	1.7	0.36	Mill Zone South
	177.1	180.0	2.9	2.3	0.13	Mill Zone North
	204.5	210.0	5.5	4.3	0.15	Mill Zone North
MZ09-06	115.5	120.8	5.3	3.0	0.41	Mill Zone South
	267.6	272.9	5.3	3.0	0.12	Mill Zone North
MZ09-07	331.0	334.0	3.0	1.0	0.19	Mill Zone South

Table 14 - Summary of Assay Results from 2009 Drilling in McGarry Mill Zone

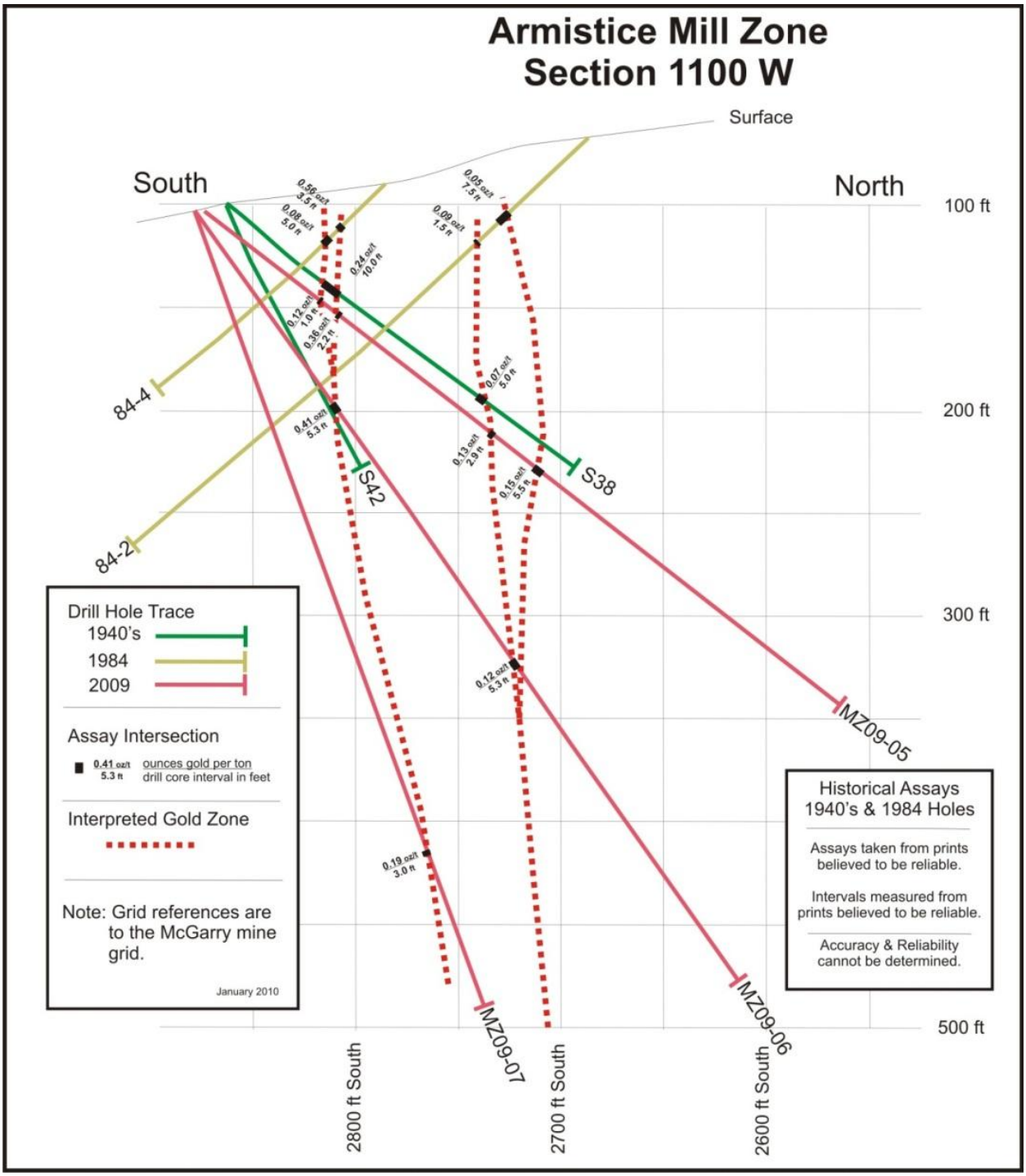


Figure 17 - Mill Zone Drill Cross Section 1100W

13. Geological 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.

The Property is located in the centre of the long established Abitibi mining district in northeastern Ontario (see Figure 4). 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.

14. Sampling Method and Approach

14.0 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 core racks or in bundles strapped onto pallets at the McGarry project site.

For the short, closely spaced drill holes from the western end of the 260W drift completed in 2008, 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 2007-2011 drill programmes, 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.

14.1 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.

14.2 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.

14.3 Panel Sampling

In the case of the two 9ft by 9ft 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 8ft in length (usually only about 5.5 and 7.5ft 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 18. 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 3ft 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 18). That is, at about 1.5, 4.5 and 7.5ft above the drift floor.

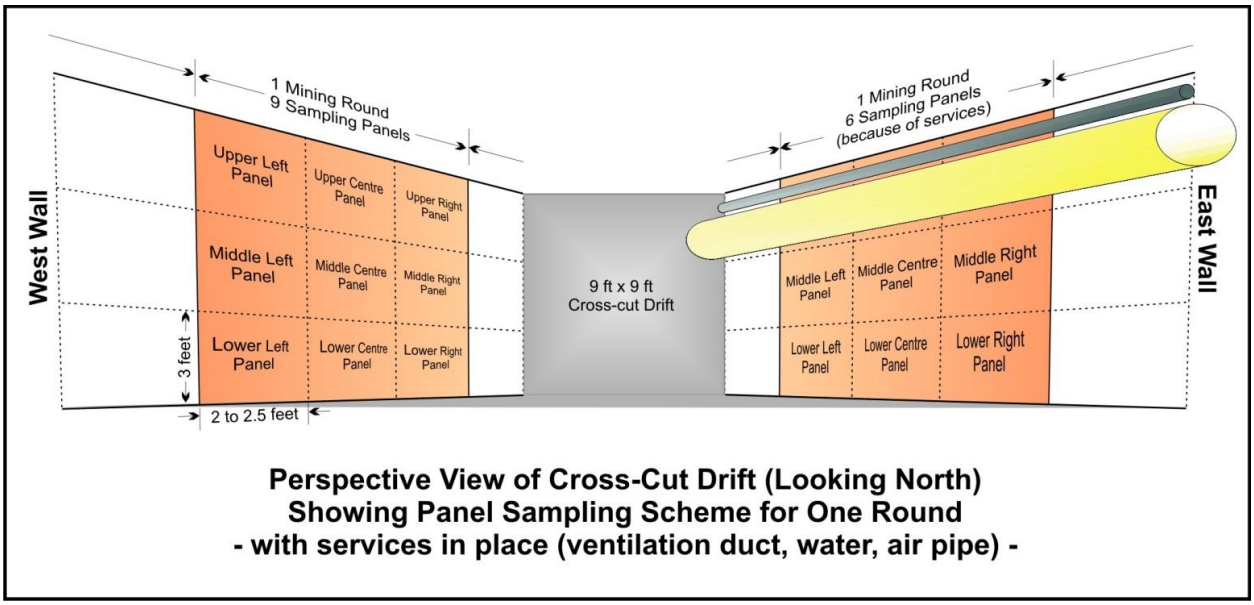


Figure 18 - Perspective View of Cross-Cut Drift Showing Panel Sample Scheme

14.4 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.

14.4.1 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 “grading” (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.

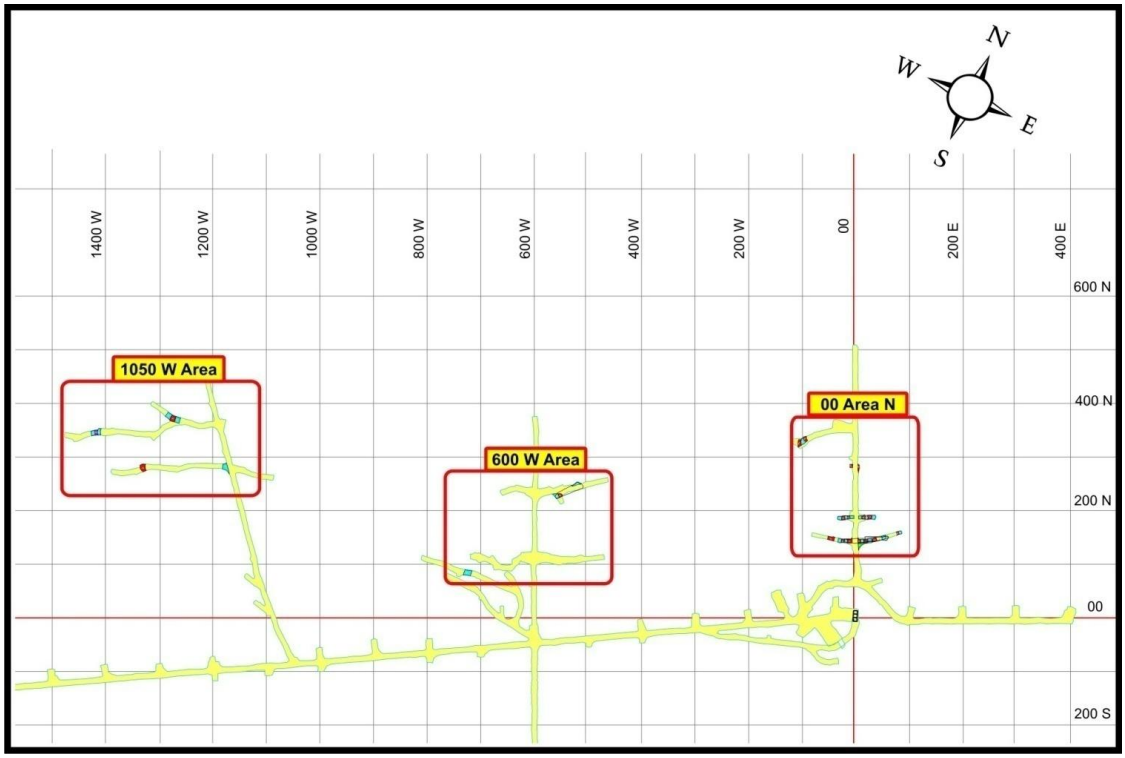
The rounds selected for bulk sampling are shown in the following plans of the 2250 Level (Figures 19 to 24).

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 confirmed via calibration. 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 19 - Overview Location Map of Bulk Sampling and Test Stopes



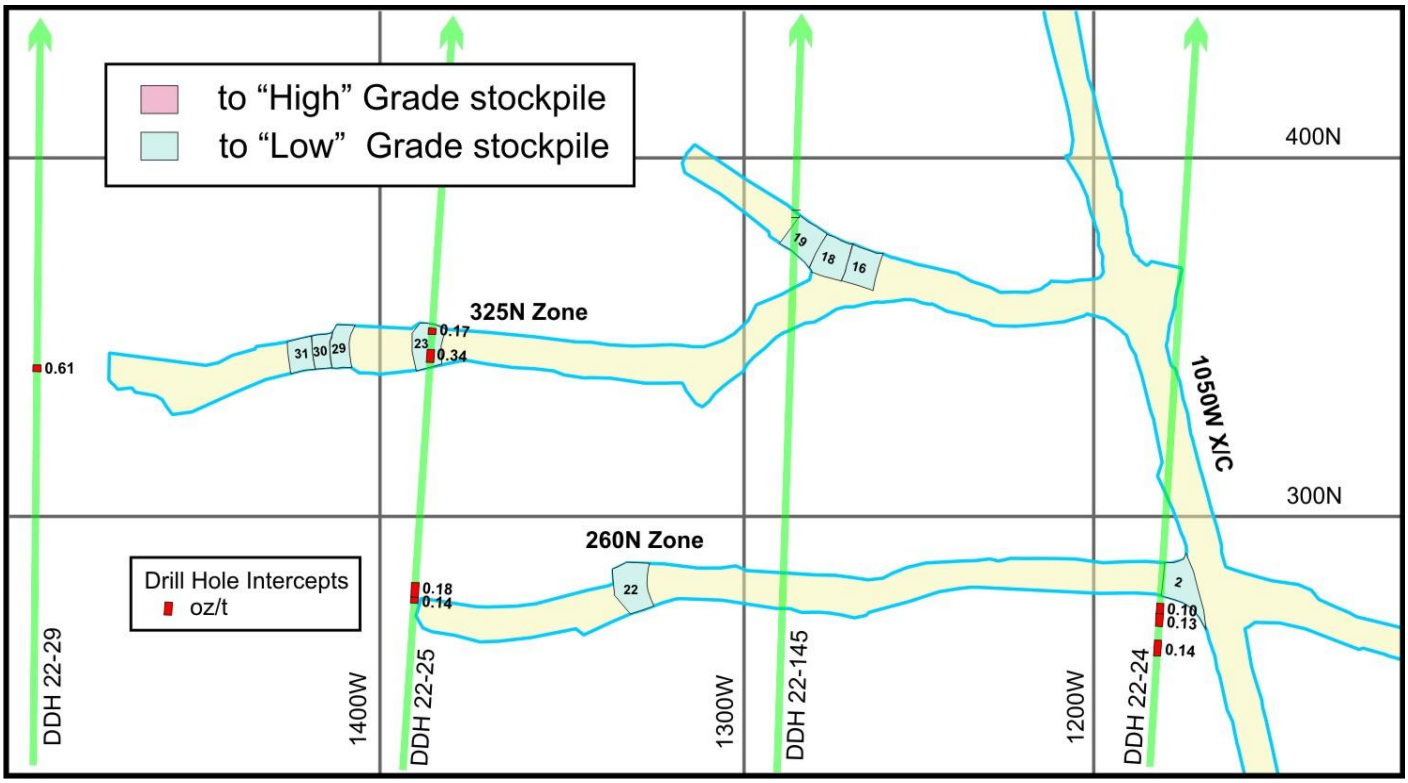
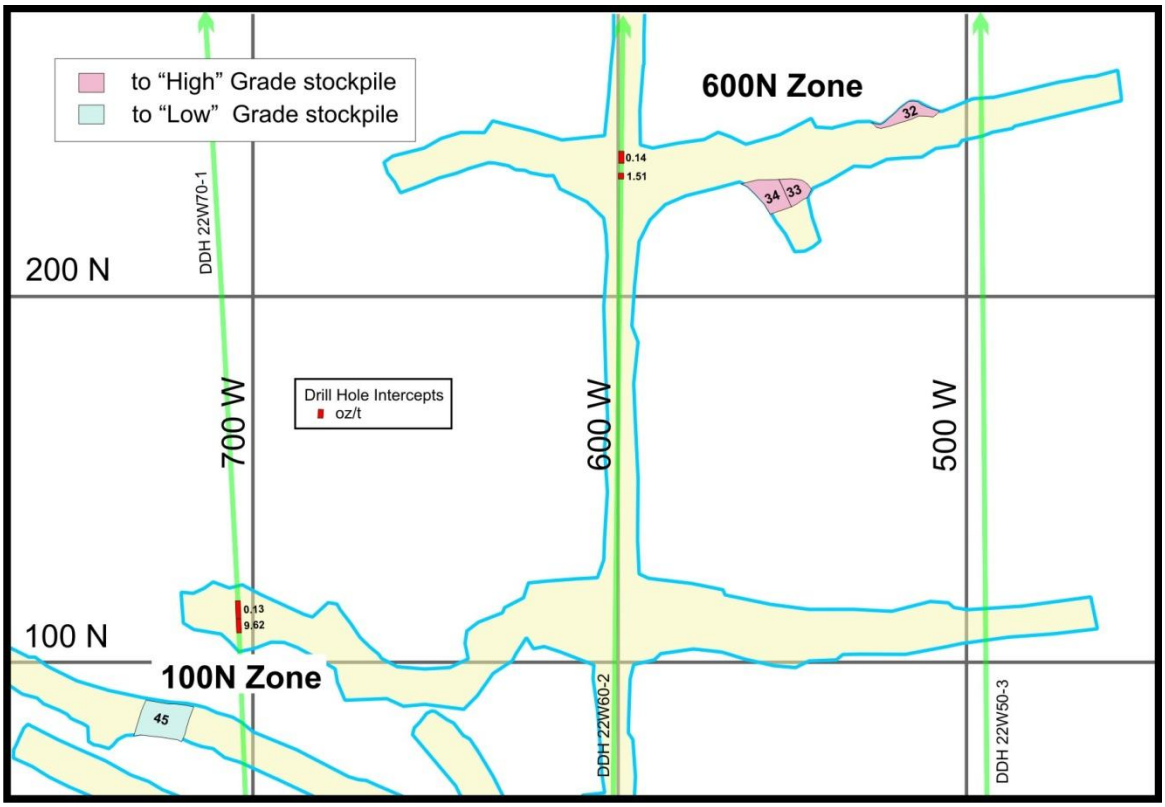


Figure 20 - 1050W X/C Area Bulk Sample Mining Rounds Detail Location

Figure 21 - 600W X/C Area Bulk Sample Mining Rounds Detail Location



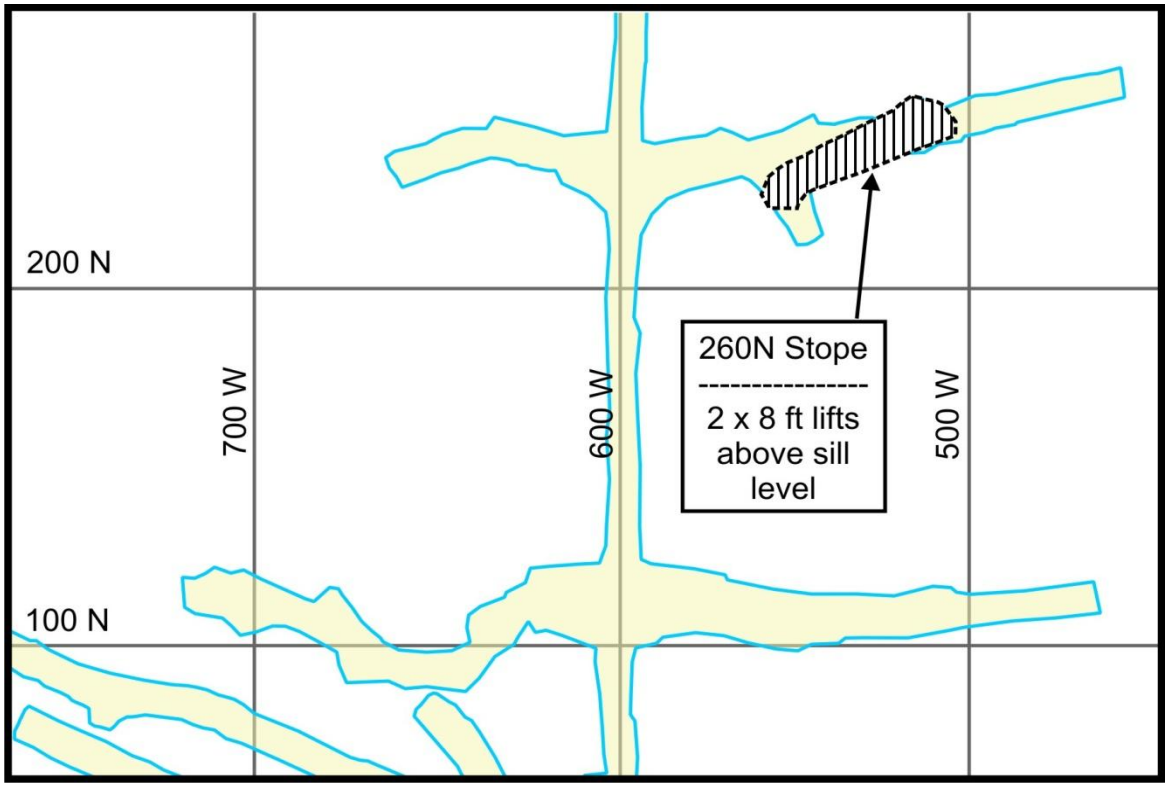


Figure 22 - 600W X/C Area Test Stope 260N Location

Figure 23 - 00 X/C Area Bulk Sample Mining Rounds Detail Location

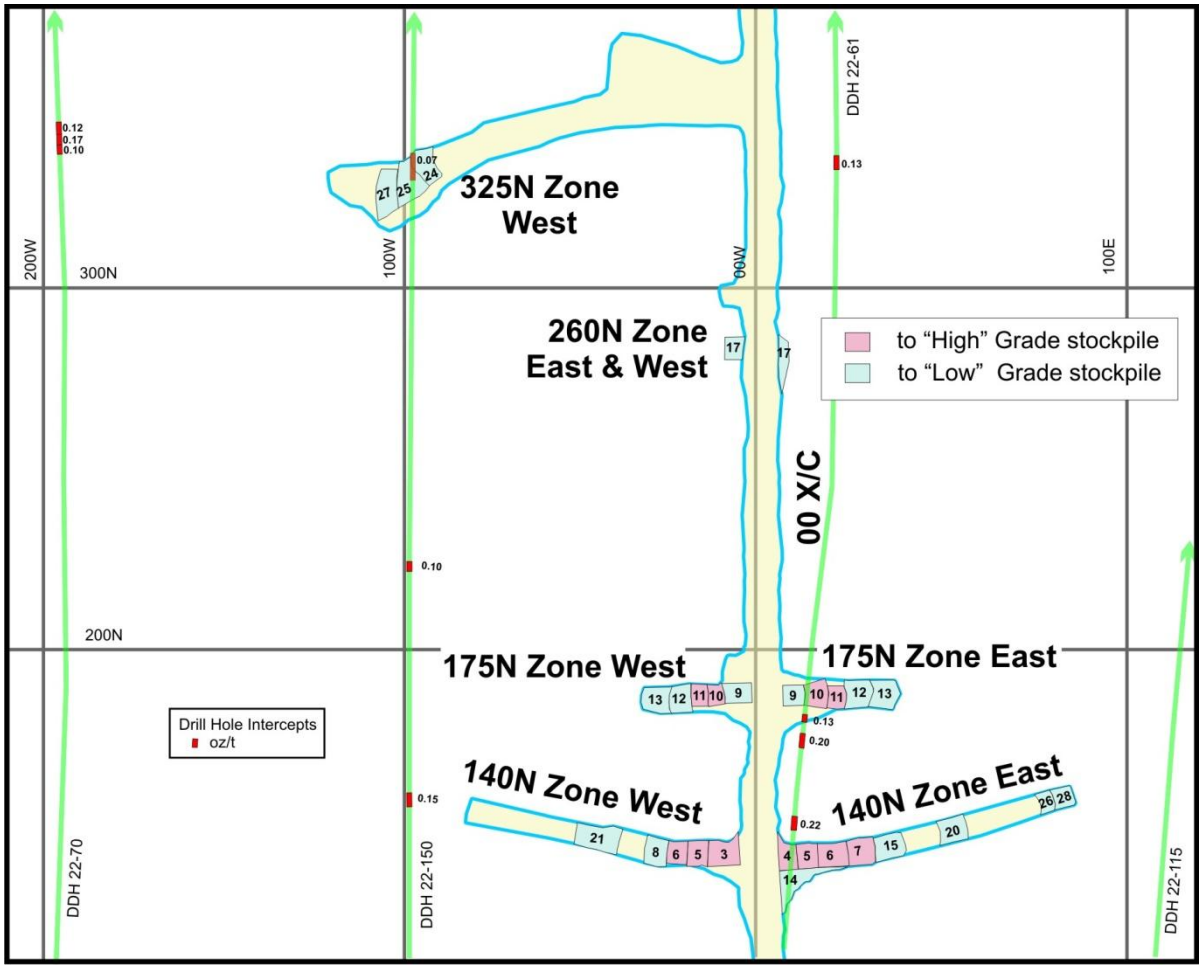
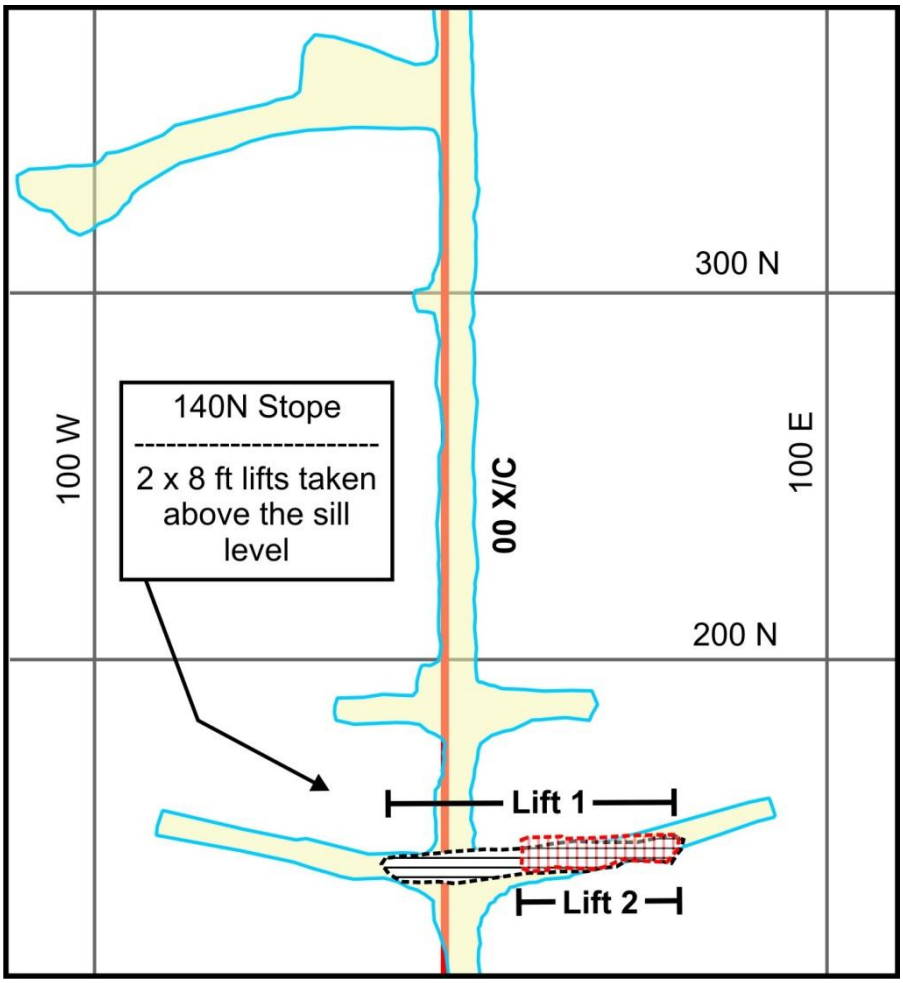


Figure 24 - 00 X/C Area Test Stope 140N Location



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 15).

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 25), modified from the PolyMet Web Site to reflect the actual circuit used for the Armistice samples:

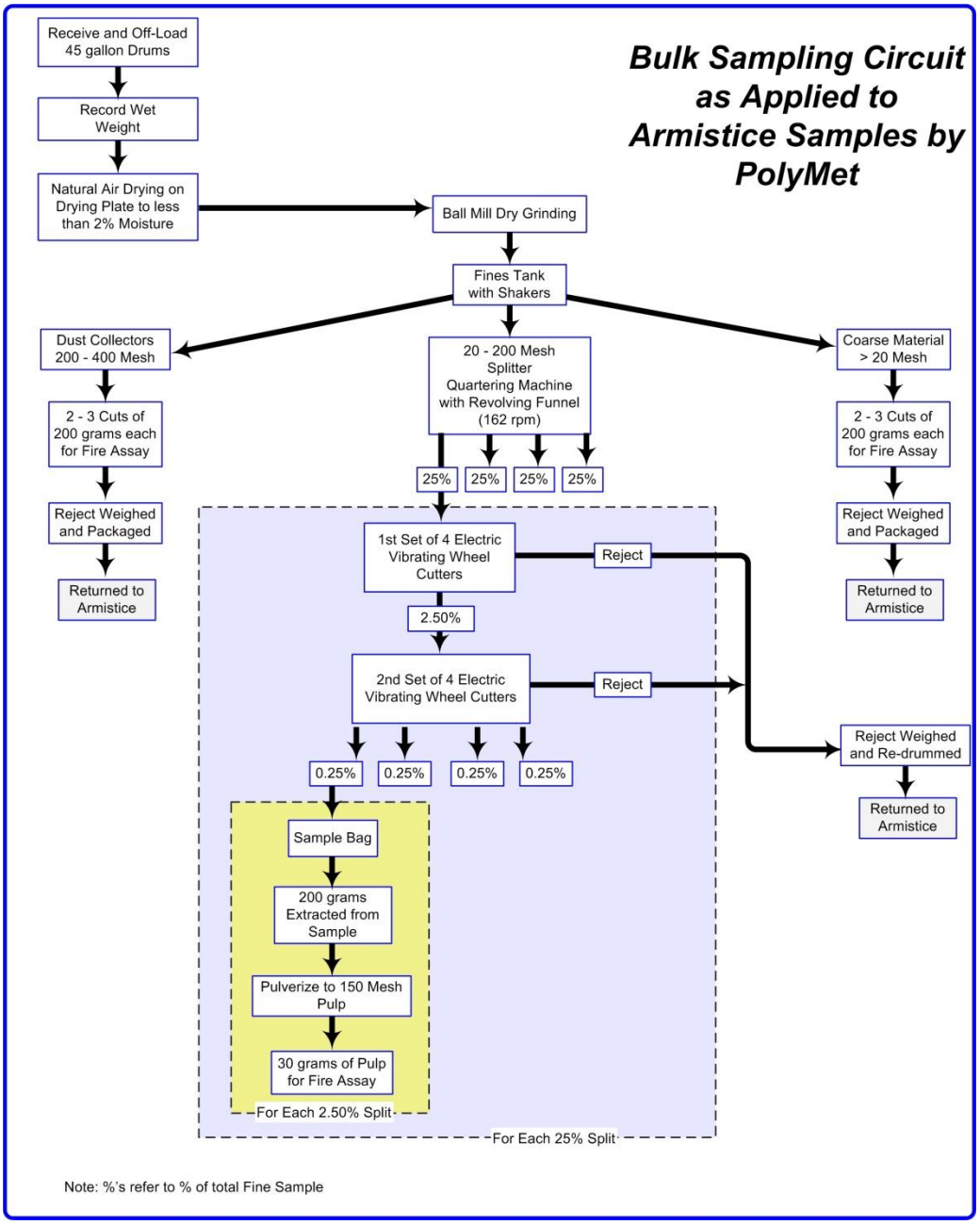


Figure 25 - PolyMet Bulk Sampling Circuit Diagram

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 15 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 15 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 grade 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 15.

Table 16 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 17 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	600 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	600 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	600 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 15 - 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: Lowest Assay Value (oz/t)	Chip Sample Assay Range: Highest Assay Value (oz/t)	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)	
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 16 - 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		Muck Samples		Nearest Drill Holes			Comments
			Aggregate Lineal Feet of Chip Sampling (ft)	Weighted Average Assay (by sample length) (oz/t)	# of Samples	Average Assay (oz/t)	DDH Number	Average Grade (oz/t)	True Width Interval (ft)	
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 17 - Stoping: Comparison of Bulk Sample Grades with Chip Sampling & Drilling Results

15. 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 Kirkland Lake Gold Mine (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 McGarry Project site.

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.

All the labs used are independent of Armistice. The author is not aware that any of the labs used is specifically other than PolyMet labs which is ISO 9001:2000 registered. Swastika Labs has been in business since 1928 and in the author's opinion has established an excellent reputation for independent reliability. The Kirkland Lake Gold Inc. Lab work is not used in the resource estimate.

16. 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.

16.0 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 worldwide 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.

16.1 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).

16.2 Check Assay from Pulps

Figure 26 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 envelope 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.

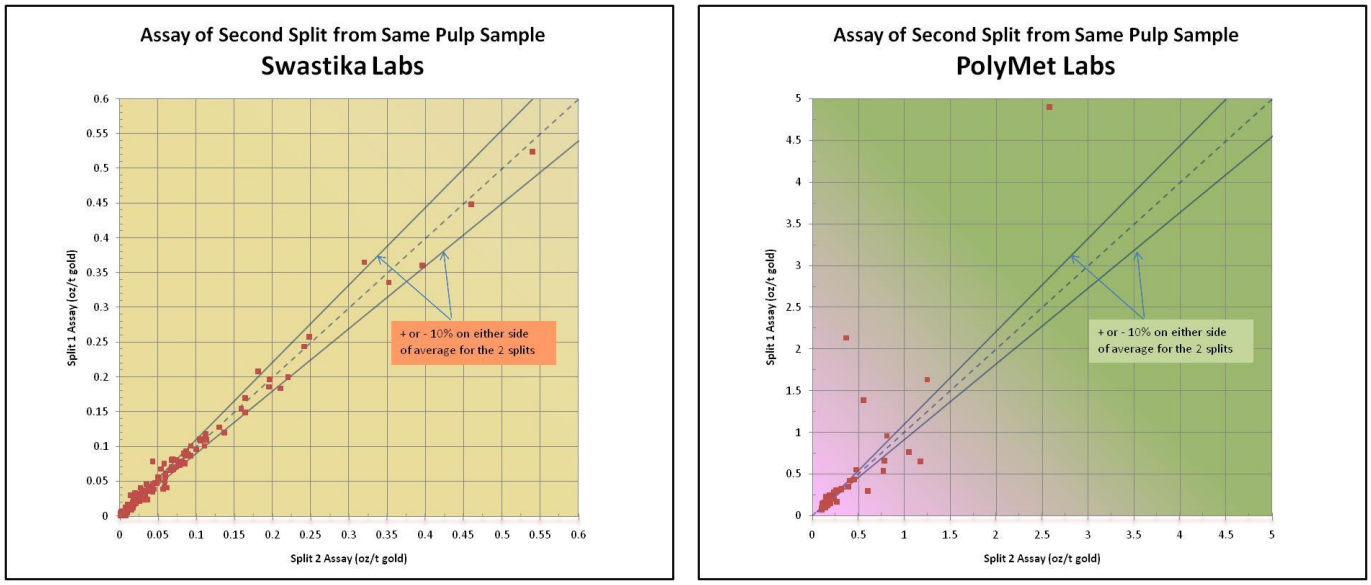


Figure 26 - Assay Comparison of Two Splits from Same Pulp for Swastika Labs and PolyMet Labs

16.3 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).

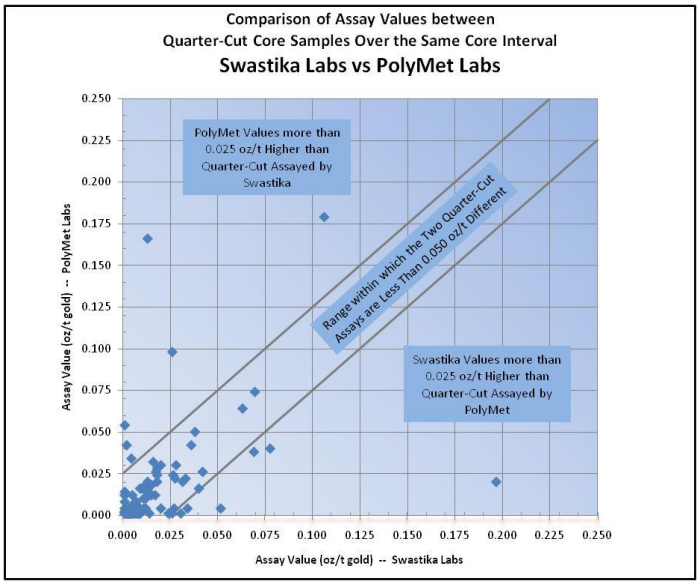
- Figure 27, below, presents the comparison of the results of the two assays for the two matching quarter cuts. An envelope is drawn on the graphs to show the range in which the results for the two quarter cuts is less than 0.05 oz/t different. All assays for samples in which the comparison falls outside 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.



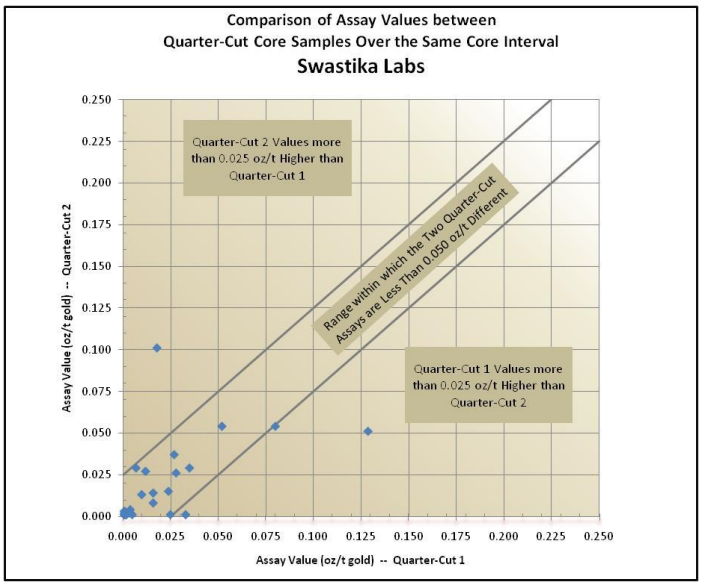


Figure 27 - Comparison of Assays from Two Quarter Cut Core Samples from Same Interval between Labs

17. 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.

17.0 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 18.

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 18 - 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 (in around 3,200 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.

17.1 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 28. The implication of this test work is that the grade of the feed can be upgraded significantly. These results provided encouragement for a larger scale test in 2008.

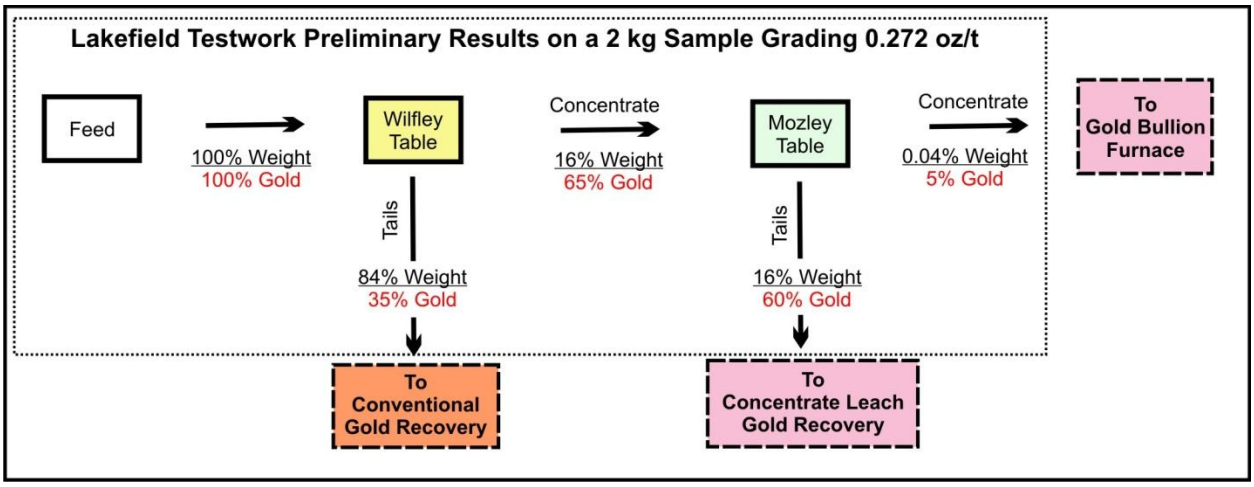


Figure 28 - 1995 Lakefield Preliminary Test Work Results on Gravity Recovery

17.2 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 19:

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 19 - Summary of Metallurgical Characteristics Testing Results

18. Mineral Resource Estimates

The current Mineral Resource Estimate was carried out by the technical staff of Armistice Resources Corp. under the supervision of Erik Andersen, P. Eng., a non-independent Qualified Person. The methodology, database, gold zone interpretation and calculations have been extensively tested and verified by the author. This verification and testing has included checking assay results in the Armistice database against original assay certificates obtained by the author directly from the assay labs. A selection of key mine sections and plans were also reviewed by the author. No material issues were identified and the author considers the resource estimate presented here to be verified. The author certifies the estimate of mineral resources was made in compliance with the recommendation 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 the Preliminary Economic Assessment are included in this report (see Section 18. below), the results do 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 16 September 2011.

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.

18.0 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 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 currently 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). The author is not aware of any environmental, permitting, legal title, taxation, socio-economic, marketing, political, or any other relevant factors which could materially affect the mineral resource estimates.

18.1 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.

18.2 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 16.1., 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.

18.3 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 protocol for McGarry.

18.5 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.

18.6 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.

18.7 Mineral Resource Estimate Results

A tabulation of estimated Mineral Resources is shown below in Table 20. 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 – 16 September 2011

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 ±15% is estimated for the Indicated Mineral Resource and ±25% for the Inferred Mineral Resource.
- Effective date of resource estimate is 16 September 2011.
- Qualified Person for the mineral resource estimate is Martin Drennan, P.Eng.
- 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 20 - 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.

19. Environmental Permitting 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 23, 2011 and covers the year 2010. 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.

20. Preliminary Economic Assessment

"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, we have been engaged by Armistice to conduct a Preliminary Economic Assessment ("Preliminary Economic Assessment") 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 Preliminary Economic Assessment is ±25%. It should be noted that this Preliminary Economic Assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, an there is no certainty that the Preliminary Economic Assessment will be realized.

The strategic approach on which the Preliminary Economic Assessment 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 29, below.
- 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 30 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

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.

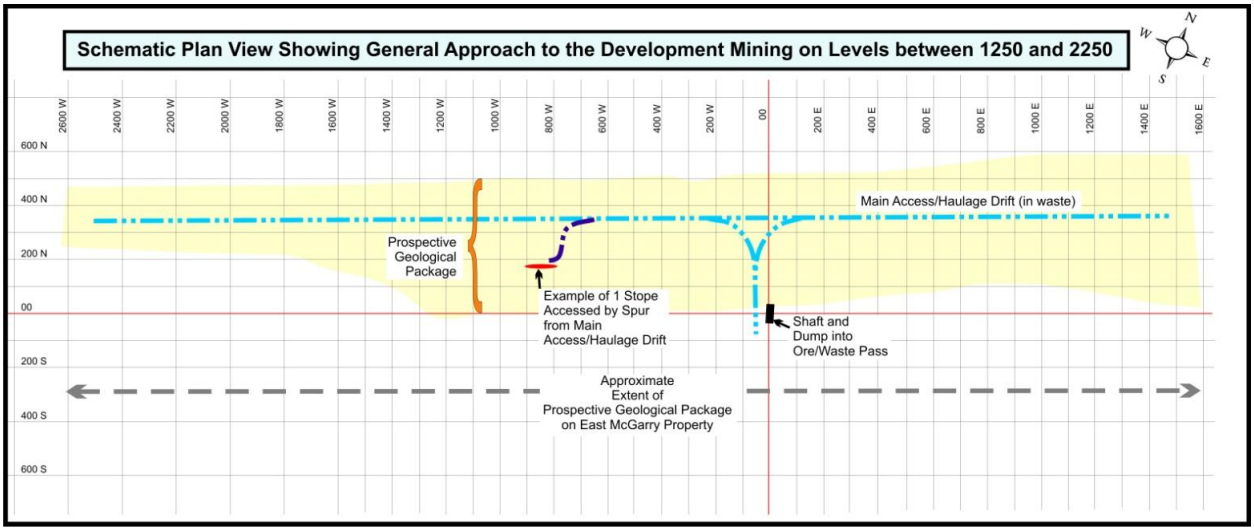


Figure 29 - Schematic Plan Showing Strategic Approach to Development Mining – Upper Levels

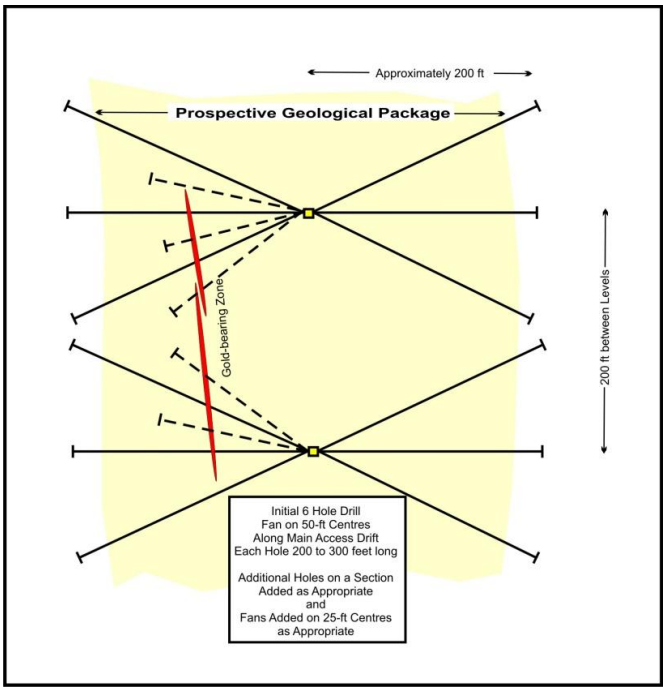


Figure 30 - Schematic Cross Section Showing Short Hole Diamond Drilling for Gold Zone Definition

The assumptions used by Python are summarized as:

- This study has a confidence range of ±25%.
- Gold bearing areas are extrapolated on diamond drill holes and on sill information.
- Mining areas are based on gold bearing zone patterns identified.
- Gold bearing zones have been physically tested.
- 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 mining shells (i.e. dilution is accounted; not as a fixed percentage, but in the mining dimensions – minimum mining width of 6ft and 1ft of dilution either side).
    - Development and production rates were discussed internally amongst Armistice, Paul Whelan Mining Contractors, and Python.
    - 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).
- Mining is on areas within the geological structure which includes both indicated and inferred resources at McGarry. The mining envelopes are not limited to existing mineral resources.

The economic modelling performed by Python is in cash flow form, within a ±25% estimation, and assumes the presence of gold zones suitable for mining 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. In part, this compensates for the 4% net smelter royalty payable to Jubilee Gold Inc. 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 Preliminary Economic Assessment.

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 currently fluctuating significantly, but is currently in the range of $C 1600 - 1700 per oz.

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

The development and mining plan assumed by Python includes three consecutive phases, summarized with duration periods and total costs as:

Phase	Duration (±25%)	Cost (±25%)
Preproduction	7 months	$4.7 million
350 tpd Development and Production	12 months	$8.5 & $9.7 million, respectively
600 tpd Development and Production	36 months	$4.9 & $47.8 million, respectively
Total	55 months	$75.7 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	122,500 tons	20,800 oz Au	$ 16.6 million revenue
600 tpd	36 months	630,000 tons	107,100 oz Au	$ 85.6 million revenue
Total	55 months	752,500 tons	127,900 oz Au	$ 102.2 million revenue

More work is required to update economic parameters as noted in the Recommendations and Conclusions sections.

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, seems a reasonable value to use for purposes of illustrating the underlying economics.

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	$ 30.6 negative	$ 36.4 negative
$C 800	$ 3.9	$ 5.3 negative
$C 1100	$ 38.4	$ 25.7

The following comments should be kept in mind:

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 slightly 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 $13,400,000,

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 slightly greater than 0.19 oz/ton at $800 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 $44,700,000 over 3 years including the pre-production period. This is the approximately breakeven point and the minimum requirements for the project to move forward.

Assuming a market price more in line with current market conditions (say $1200/oz Au) 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 $21,300,000 (350 tpd x 350 days/year x 0.19 oz/ton x 0.9 mine rec x 0.85 mill rec x $1200/oz). The expenses for processing the initial production year is based on 350 TPD and a cost of $108 per ton. This would result in a positive cash flow circa $8,100,000. If the diluted grade was unable to be achieved and a diluted grade of 0.14 oz/ton was the realized grade, then the initial year positive cash flow would be reduced to $2,500,000.

With the assumptions above ($1200 per ounce revenue and 0.19 oz/ton diluted grade), the revenue for the 600 TPD period (~3 years) would be approximately $109,000,000. The operating expense during this period would be around (3 years x 350 days/year x 600 TPD x $91) $57,300,000 resulting in a projected profit of $52,500,000. With the conditions varied to $1200 per ounce and a diluted grade of 0.14 oz/ton the revenue would fall to $78,700,000 giving a projected profit potentially of $23,000,000.

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

The Recommendations from Preliminary Economic Assessment include:

- 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.
- Update the mine design to match the new geological model as it develops.
- Update unit costs and economic parameters to reflect regional costs as they change from time to time.
- 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 form Preliminary Economic Assessment include:

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 over 11 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:

- 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 be able to achieve a positive cashflow in the current economic climate.

21. Other Relevant Data and Information

There is no other relevant data or information relied upon not otherwise disclosed in this report.

22. 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.

23. Recommendations

It is recommended that the McGarry Project be advanced to the next stage of mining development following the strategy as outlined in the Preliminary Economic Assessment (Section 18.). 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.

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 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 21. 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 in the Preliminary Economic Assessment, 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 Preliminary Economic Assessment. 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) @10%	Total Expenditures (000's)	Potential Revenue (±25%) (000's) @ $C 800
1	Q1	$ 1,300		$ 100	$ 140	$ 1,540	
1	Q2	$ 2,600		$ 100	$ 300	$ 3,000	
1	Q3	$ 2,800	$ 1,700	$ 100	$ 500	$ 5,100	$ 800
1	Q4	$ 2,600	$ 2,600	$ 100	$ 500	$ 5,800	$ 2,800
1	Total	$ 9,300	$ 4,300	$ 400	$ 1,440	$ 15,440	$ 3,600
2	Total	$ 5,300	$ 12,100		$ 1,700	$ 19,100	$ 16,700
3	Total	$ 3,000	$ 16,000		$ 1,900	$ 20,900	$ 24,400
4-Q1	Total		$ 1,500		$ 200	$ 1,700	$ 4,100
1 - 3	**Total**	**$ 17,600**	**$ 33,900**		**$ 5,240**	**$ 56,740**	**$ 48,800**

* All numbers are in Canadian Dollars ($C)

Assumptions:

1. Gold price is $C 800
2. Metallurgical recovery including NSR is 85%; Mining recovery is 90%; Avg Grade is 0.19
3. Contingency is 10%
4. Revenue is uniform – and offset by 3 months to account for shrinkage "free pull" based on 1/3 swell immediately available and 2/3 "freepull" available in 3 months
5. Gold revenue is instant of milling
6. Production rate is 350 tpd from month 8 to 19 inclusively
7. Production rate is 600tpd from month 20 to 36 inclusively
8. Costs are based on unit cost development and are at ±25% confidence

Table 21 - Recommended Budget for First 3 Years of McGarry Mine Development

The expenditure estimates are based on the work summarized in the Preliminary Economic Assessment. The Preliminary Economic Assessment is not a feasibility study and is stated to have a confidence range of ±25%. There is no contingency percentage factor built into the estimates – each expenditure was estimated individually on a conservative basis instead. The budget presented above includes an additional 10% contingency factor as a conservative measure to cover the costs of unanticipated requirements as well as Projects.

This budget shows a shortfall for the Project on a cash basis after about three years based on an $C 800/oz. If mining activity progresses in line with the assumptions contained in the Python Preliminary Economic Assessment, 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 58, 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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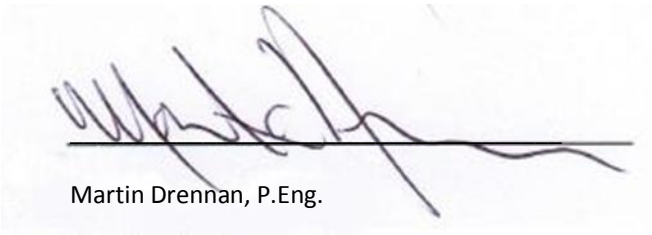
25. Signature Page

Preliminary Economic Assessment and Mineral Resource Estimate, McGarry Project



Martin Drennan, P.Eng.

Signed at Hamilton, Ontario

26. Certificate of Author

I, Martin Drennan, P.Eng, do certify that:

1. I am employed as an Associate and Senior Mining engineer by, and carried out this assignment for:
   Python Mining Consultants Inc.
   173 Bold Street, Hamilton, ON L8P 1V4
   Tel. (905) 540-1432 Fax (905) 540-1136

2. This certificate applies to the technical report titled "Preliminary Economic Assessment and Mineral Resource Estimate, McGarry Project, McGarry Township (Virginiatown), Ontario" dated 2009-Apr-08 and amended 2011-Sept-30 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). 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 19 years.

6. I understand that my education, experience and professional registration, fulfill the requirements of a Qualified Person. My work experiences 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 no prior involvement with the mineral properties in question other than those disclosed herein.

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 14th day of October, 2011

Martin Drennan, P.Eng.