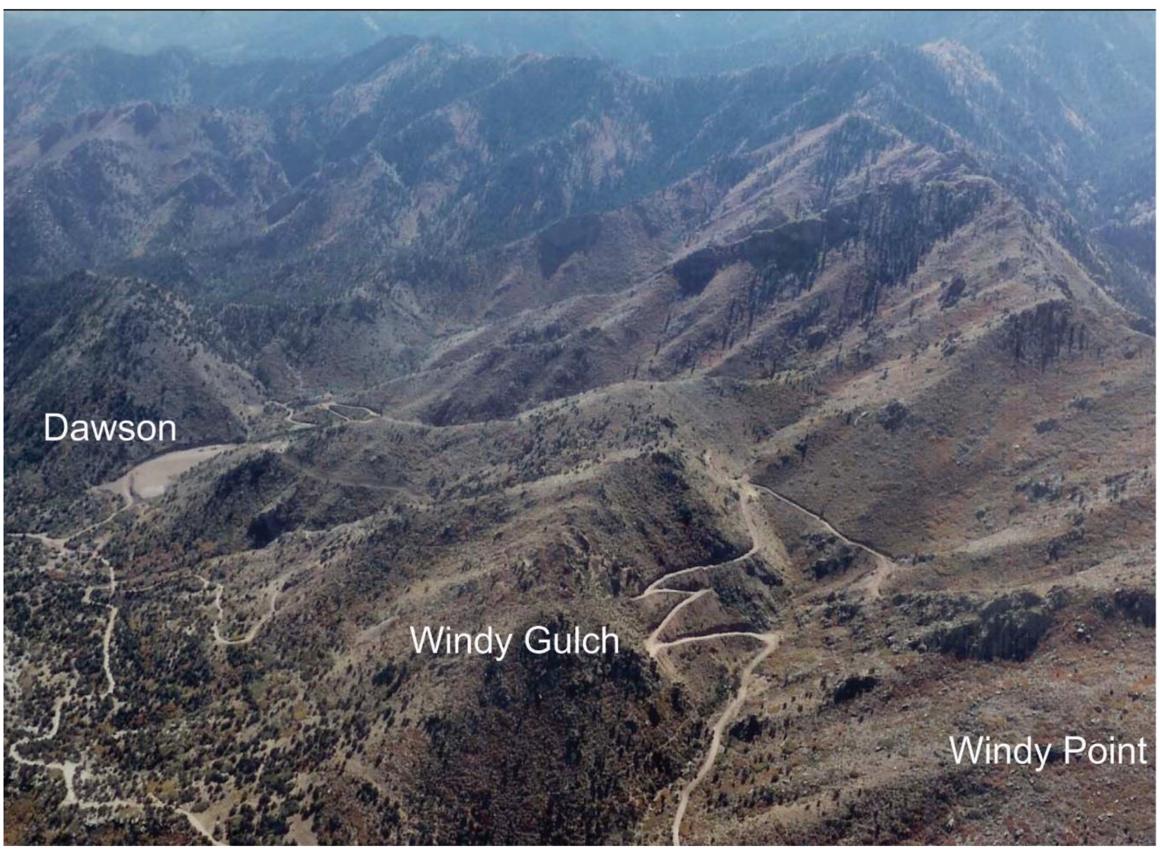
Zephyr Minerals Ltd

Preliminary Mine Design

Dawson Gold Property



Zephyr Minerals Ltd

Mine Design

for the

Dawson Property

located in

Colorado, USA

38°23' N, 105°18' W

Effective Date: August 26, 2015

Filing Date: October 7, 2015

by

Patrick Hannon, M.A.Sc., P.Eng.

Doug Roy, M.A.Sc., P.Eng.

MINETECH INTERNATIONAL LIMITED

CONSULTANT ENGINEERS AND GEOLOGISTS

SINCE 1989

Executive Summary

This preliminary mine plan was co-authored by Patrick Hannon (P.Eng., M.A.Sc.) and W. Douglas Roy (P.Eng., M.A.Sc.), on behalf of Zephyr Minerals Ltd. ('Zephyr'), for the for the Dawson gold deposit ('the Property'). W. Douglas Roy and Patrick Hannon are considered Qualified Persons (QPs) under NI 43-101 definitions.

The purpose of this report is to complete preliminary mine design, estimate underground capital and operating costs for the deposits at a pre-feasibility level of accuracy (-15% / +25%), and to complete a scoping level production schedule.

Preliminary underground design was carried out for the Dawson site, and a preliminary surface pit was designed for the Windy Gulch site.

The Dawson Property is located approximately 10 km southwest of Cañon City, Colorado, USA. Elevations in the area range from approximately 1,870 m to 2,400 m (6,135 ft to 7,875 ft) above mean sea level.

Gold mineralization at the Property is generally hosted by essentially tabular, steeply dipping structures generally ranging from 1 to 4 m (3 to 13 ft) in true thickness and can be as thick as 9-10 metres (29-33 ft). The gold mineralized structures trend northeast and dip between 50° and 70° to the southeast.

Work in this report builds on an existing NI 43-101 compliant Technical Report for the Property (Hilchey, Graves, and Wolfson, Mercator Geological Services, 2013). The existing NI 43-101 compliant resource estimate outlines an inferred resource of approximately 423,000 tonnes (466,000 tons) grading 10.07g/tonne (0.29 opt) gold for the two main segments, Dawson and Windy Gulch. Most of the resource is within the Dawson segment.

After a cost comparison of several mining methods, longhole sublevel stoping was chosen for preliminary mine design. Access would be through a portal and decline. Mining would be nearly fully mechanised.

Advance of approximately 2,500 ft (762m) to the first production level (6125 foot elevation) will take 5-7 months after the portal has been established. The decline will measure 15 ft x 11 ft (4.6m x 3.4m) and a have gradient of -15%. Should a contractor be used for this work, a budget of approximately $US 5-6 million ($1,800/ft) would be appropriate.

Development drilling would be by electric-hydraulic jumbo drill, mucking would be by 1.5 yd^3 and 3 yd^3 load-haul-dump (LHD) equipment and haulage would be by 18 tonne (20 ton) trucks. Stope drilling would be carried out using an in-the-hole longhole drill.

All mobile equipment would be electric or diesel powered. Battery-powered LHDs and trucks were selected as an alternative to diesel. Two underground haul trucks would be required initially and a third truck would be required after the mine reaches the 5862 ft level. At this point, a production shaft should be considered if sufficient resources for a longer mine life have been outlined.

In addition to the Dawson Segment, 13,400 tonnes at 9.2 grams per tonne (14,700 tons grading 0.27 opt) would be mined from a small open cut at Windy Gulch. A contractor would mine the

Windy Gulch deposit for about $US 39-40 per tonne ($36/ton) over a period of three months during Year 1.

Due to four slightly inaccurate drill hole collar surveys, the current open cut design for Windy Gulch is preliminary. Zephyr plans to drill some additional holes with the aim to extend the deposit to the east, and resurvey a few of the historic holes for which more accurate elevations are required. Once this is completed, the deposit will be remodeled and the mine plan will be updated.

The small open cut outlined in this plan at Windy Gulch represents nearly fifty days of milling. This would be mined while Dawson underground development work is underway and stockpiled at the mill. The mill would start to process this stockpile a few weeks before Dawson production begins.

The total capital cost for the underground portion of the project, including Windy Gulch but excluding working capital, is estimated to be $US 12.3 million.

Operating costs were estimated at approximately $US 80 per short ton (US$88 per metric tonne) of mill feed. This cost includes all production costs, development costs, but excludes capital. Additional costs associated with the operation would be processing cost and general and administration (G&A) costs.

It would take most of a year to develop Dawson, and a period of four years of production would follow during which a program dedicated to expanding the resource base and mine life would be undertaken. The small open cut at Windy Gulch could be mined by contractor during year one.

An average milling rate of 272 tonnes (300 tons) per day was targeted, giving a total mine life of 4-5 years assuming no downtime. To account for downtime, a milling capacity of at least 310 tonnes (340 tons) per day is recommended.

On the Dawson Segment, underground production mining would be carried out over five days per week, with a targeted mining rate of 363 to 386 tonnes (400 to 425 tons) of gold mineralized rock per day. Under the proposed mine plan, 407,000 tonnes of diluted mineralized rock would be extracted at a grade of 8.9 g/t (449,000 tons at 0.26 opt). Overall, about 235,000 tonnes (260,000 tons) of waste rock would also be mined.

The total potential mill feed (diluted) for the Dawson Property, including both the Dawson and Windy Gulch Segments, is 420,000 tonnes at an average grade of 8.9 g/tonne, for 121,000 ounces.

Based on the results of this report, a program comprising completion of the mine permitting process, tailings dam design, preliminary mill design, completion of a limited drill program at Windy Gulch and completion of a Preliminary Economic Assessment (PEA) is recommended for the Dawson and Windy Gulch deposits. The budget for this program is estimated at US$ 675,000.

Table of Contents

List of Appendices

List of Tables

List of Figures

Preliminary Mine Design for the Dawson Property

Colorado, USA

1 Introduction

In late March 2015, MineTech International Limited (MineTech) was commissioned by Loren Komperdo, President and CEO of Zephyr Minerals Limited to prepare a preliminary mine plan and schedule for Zephyr Minerals Limited's ("Zephyr") Dawson Property ('Dawson'). Zephyr is a gold exploration and development company, based at Suite 1700, 1959 Upper Water Street, Halifax, Nova Scotia, Canada. The company is listed on the TSX-V exchange, symbol "ZFR".

Zephyr has a current NI 43-101 compliant resource report for the Dawson Property entitled "RESOURCE ESTIMATE TECHNICAL REPORT FOR THE DAWSON PROPERTY FREMONT COUNTY COLORADO, USA". The report was prepared by Mercator Geological Services for Zephyr and was authored by Andrew Hilchey, P.Geo., Mark Graves, P.Geo., and Isobel Wolfson, M.Sc., P.Geo. The report had an effective date of July 19th, 2013. It was filed on SEDAR on September 6, 2013.

1.1 Caution to the Reader

The reader is cautioned that this mine design uses Inferred Mineral Resources which may never be mined. CIM cautions:

> *"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."*

Inferred mineral resources may be used in a Preliminary Economic Analysis or "PEA."

Inferred Mineral Resources are based upon widely spaced samples and are speculative in nature. They may never be part of a mineral reserve. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.2 Scope of Work

The objective of this study is to build upon the 2013 NI 43-101 Technical Report by completing preliminary mine design work for the Dawson gold deposit.

The decline design, capital, and operating costs would be estimated to a preliminary economic analysis (-15% / +25%) level of accuracy.

MineTech was also commissioned to prepare a scoping level, preliminary mine plan and production schedule for the Dawson Property. Preliminary underground design was to be carried out for the Dawson site and a preliminary surface pit would be designed for the Windy Gulch site. Equipment was to be selected, and equipment capital and operating costs were to be estimated using published sources and using previous experience. The mining costs were to be estimated on annual and “per tonne” bases.

Zephyr has completed a mineral resource estimate, has completed some mineral processing and environmental work and has investigated options for tailings disposal.

1.3 Accuracy of Estimate

The accuracy of cost estimation in this report was -15% / +25%. This accuracy is typical for preliminary economic analyses.

For example, in Section 4.1.3, the operating cost was estimated to be $US 80 per ton. Based on the accuracy of this estimate, the actual value, which would only be known for certain after production begins, would likely be within the range of $68 to $100.

1.4 Supplied Data

Zephyr supplied the following data:

- A drill hole database in digital format;
- Wireframes, block models, digital terrain models, solid models, and other computer files from the 2013 resource estimate;
- Previous geological interpretations and technical reports related to the property; and,
- Results and/or reports from mineral processing work.

1.5 Units and Abbreviations

Currency is in United State dollars unless otherwise stated. The exchange rate is approximately $CAN 1 = $US 0.75-0.80 for August 2015.

The vast majority of the supplied data, including the drilling database and the block model, made use of US Customary Units. Because the project is located in the United States, the authors decided to maintain the use of the US Customary system. Most calculations were carried out using that system. Conversions to metric were used when needed. In this report, values are expressed in metric units first, followed by the same value in US Customary units.

The units and abbreviations used in this report are described below in Table 1-1 and Table 1-2.

Table 1-1: Units

Units	Description
kg	kilogram
ton	short ton (2000 lb)
lb	pound (0.454 kg)
tonne	metric tonne (1000 kg)
g/tonne	grams per metric tonne
oz	troy ounce (31.1 grams)
m	metre
in	inch
ft	foot
opt	troy ounce per short ton

Table 1-2: Abbreviations

Abbreviation	Description
LHD	Load-Haul-Dump; low-profile, front-loader.
Level	Level of the mine, in feet above mean sea level
SG	Specific Gravity (density of substance relative to density of water)

1.6 Coordinates

A site compilation drawing ("dawson au project-jms geologic+iw inputs.dxf"), in site grid (feet), shows the UTM coordinates (metres) of two points. The conversion from site grid to UTM involves a translation, a slight rotation, and a scaling from feet to metres. Using that conversion, a UTM-georeferenced satellite photo and other digital data were imported with a good fit.

Table 1-3: UTM to site grid conversion.

Point	Site Grid (East, North) (Feet)	UTM Zone 13S (East, North) (Metres)[1]
Lower Left	41,781.2696, 41,854.4607 ft	472,000, 4,248,000 m
Upper Right	48,339.6726, 48,449.0172 ft	474,000, 4,250,000 m

[1] Presumably WGS84.

1.7 Cross-Sections

The deposit's trend is approximately 060°. Cross-sections were generally cut every 15 m (50 ft) along trend (refer to Appendix 7 for a complete set of cross-sections).

1.8 Mineral Rights

Patented claim boundaries were supplied within the digital file "DawsonClaims-Trails-Creeks-Mines.tab". The original source of this file is not known. Another source for claims survey information is from a Bureau of Land Management ("BLM") drawing named "Patented Claims Map for Sections 13 and 14, Twp 19, Rge 71, W6M.PDF" (refer to Appendix 2). The drawing was georeferenced to site grid using the BLM-surveyed points from Appendix 2 and the UTM-to-site-grid conversion from Section 1.6.

Figure 1-1 shows a comparison between the two sources. The BLM survey (black lines in Figure 1-1) was used for the current work. Land that is peripheral to the patented claims is held under non-patented lode mining claims – refer to the current mineral resource report (Hilchey and Wolfson, 2013) for details.

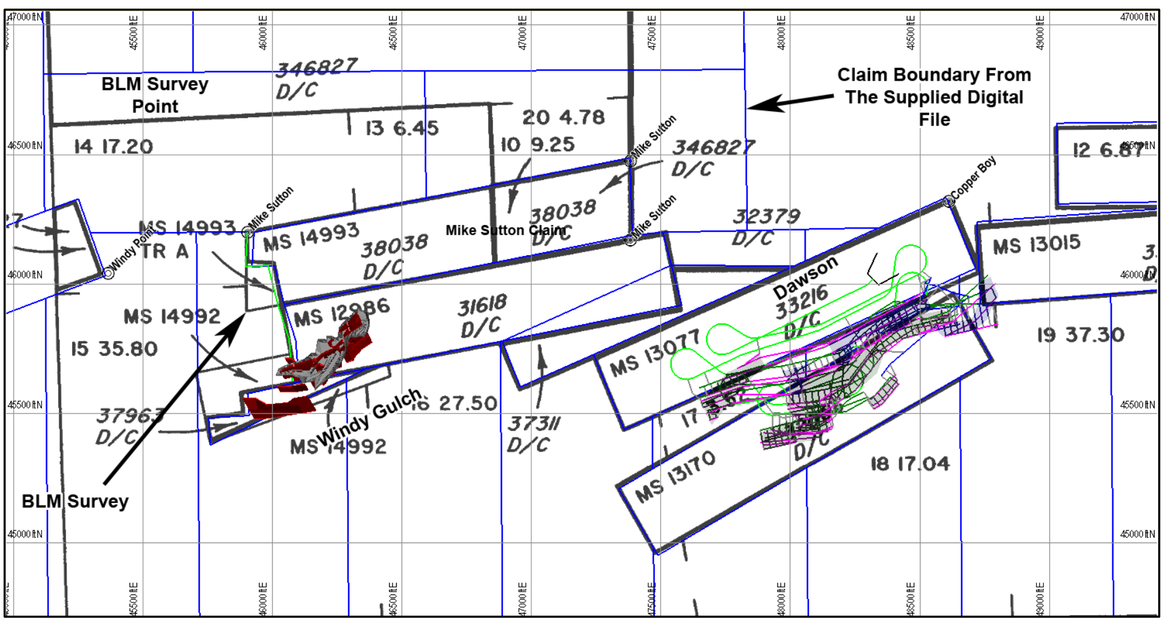


Figure 1-1: Comparison between claim boundaries from various sources (showing patented claims only).

2 Available Mineral Resources

2.1 Resource Estimate

Mercator Geological Services produced a mineral resource report for the Dawson Property in 2013 (Hilchey et al, 2013). This resource was the basis for the current mine planning work. Approximately 423,000 tonnes (466,000 tons) (4.0 g/t cutoff and 40g/t cap) are inferred for the two main segments, most within the Dawson segment. A portion of Table 1.2 from Mercator's 2013 report is shown below as Table 2-1.

Table 2-1: Mineral Resources for the Dawson Project (Hilchey et al, 2013).

Zone	Cutoff Grade, g/tonne	Tonnes	Grade, g/t (40g/t cap)	Grade, g/t (uncapped)
Dawson Segment	4.0	371,000	10.09	11.53
Dawson Segment	5.0	343,000	10.55	12.11
Dawson Segment	6.0	310,000	11.08	12.8
Windy Gulch Segment	4.0	52,000	9.89	10.63
Windy Gulch Segment	5.0	49,000	10.17	10.95
Windy Gulch Segment	6.0	40,000	11.2	12.15
Total	4.0	423,000	10.07	11.42
Total	5.0	392,000	10.50	11.97
Total	6.0	350,000	11.09	12.73

Mercator's Notes on the resource calculation:

(1) Tonnages have been rounded to the nearest 1,000 tonnes.
(2) Ounces have been rounded to the nearest 100 ounces
(3) Contributing assay composites were capped at 40 g/t Au for both the Dawson Segment and Windy Gulch Segment deposits.
(4) Uncapped values were reported using capped reporting threshold values and are provided for general information but are not part of the statement of mineral resources.
(5) The resource statement cut-off grade of 5.00 g/t Au is highlighted in Table 1.2 above reflects underground development potential based on an Au price of $US1,200/ounce.
(6) A density value of 2.63 g/cm3 was used for the Dawson Segment and 2.64 g/cm3 for the Windy Gulch Segment.
(7) Mineral resources were estimated in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum – Standards on Mineral Resources and Reserves – Definitions and Guidelines, as referenced in NI 43-101.
(8) The rounding of tonnes as required by NI 43-101 reporting guidelines may result in apparent differences between tonnes, grade and contained ounces.
(9) Mineral resources are not mineral reserves and do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
(10)The quantities and grades of reported Inferred Mineral Resources are uncertain in nature and further exploration may not result in their upgrading to Indicated or Measured status.

2.2 Block Model

The approximate geometry of the supplied wireframe segments that were used to model the resource are tabulated in Table 2-2.

Table 2-2: Description of mineralized zones over various cross-sections.

Zone	Azimuth	Plunge (SW)	Dip (S)	Range in Thickness	Average Thickness
1a	249°	-14°	56°	2.1-3.7 m (7-12 ft)	2.13 m (7 ft)
1b	232°	-14°	62°	1.5-5.2 m (5-17 ft)	2.44 m (8 ft)
1c	225°	-14°	62°	2.7-5.8 m (9-19 ft)	4.08 m (13.4 ft)
2a	245°	-14°	69°	1.8-3.7 m (6-12 ft)	2.13 m (7 ft)
2b	258°	-14°	75°	1.8-10 m (6-33 ft)	2.96 m (9.7 ft)
	235°	-14°	69°		
	209°	-14°	69°		
	236°	-14°	60°		
	262°	-14°	75°		
2c	222°	-14°	77°	1.5-1.8 m (5-6 ft)	1.62 m (5.3 ft)
3a	217°	-14°	65°	1.2-5.8 m (4-19 ft)	2.07 m (6.8 ft)
	243°	-14°	49°		
	251°	-14°	62°		
	265°	-14°	79°		
3b	254°	-14°	65°	1.2-1.5 m (4-5 ft)	1.34 (4.4 ft)

To aid mine planning, the supplied, “percent-type” model was sub-blocked into a model with irregular block sizes (refer to Table 2-3 and Figure 2-1). The resulting file was “Blocks, Dawson, Sub-blocked.dat”.

Table 2-3: Block model sub-blocking.

Direction	Block Size (feet)	Number of Sub-blocks Added
East	16.5	2
North	5	5
Elevation	16.5	2

Original “Percent-Type” Model with Regular Block Sizes

Sub-Blocked Model

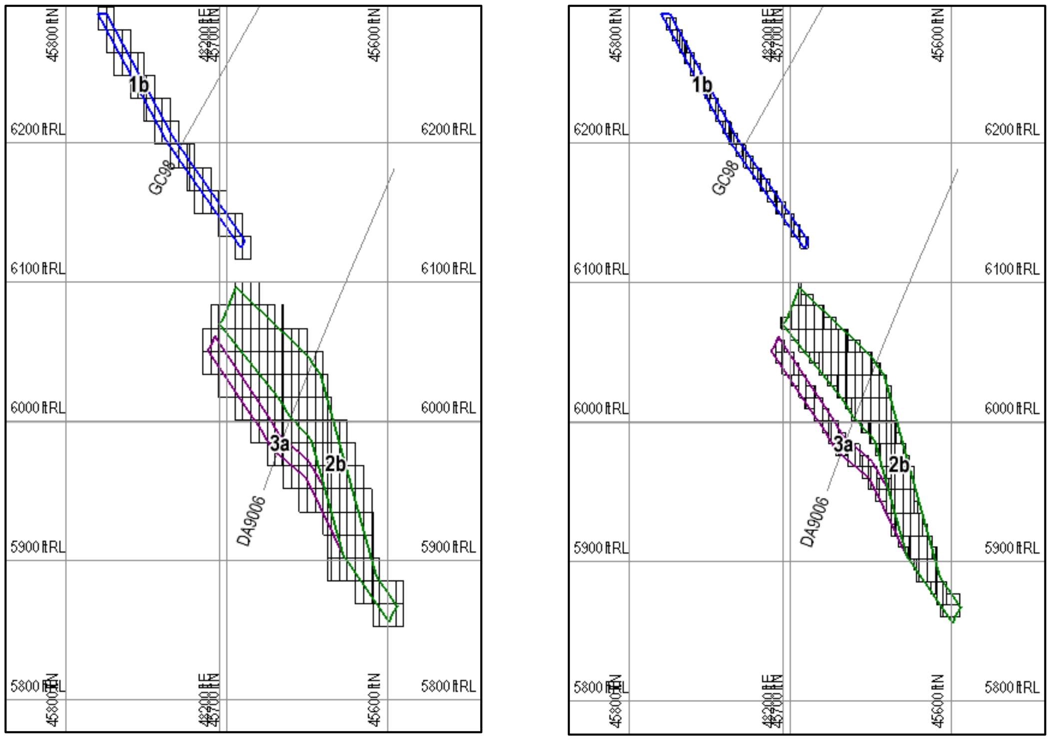


Figure 2-1: Block model before and after sub-blocking (Section G—G').

2.3 Bulk Density

A resource SG of 2.63 was used (Hilchey et al, 2013). This corresponds to a tonnage factor of 12.2 ft^3/ton (cubic feet per ton).

2.4 Dilution

There are two main types of dilution: planned and unplanned (see Figure 2-2). Unplanned dilution is mainly caused by blasting overbreak. Proper blast design and blasting practice can reduce overbreak.

High stress in the rock mass can also cause the walls of the stope to slough or fail, leading to unplanned dilution. Poor blasting practice will also contribute to unplanned dilution.

For this work, unplanned dilution was added to the non-diluted mineral resource as a function of the mining width. A skin of 15 cm (6 in) of rock was added to both sides of the width. When this skin is taken into account, dilution averages about 14.5% with a range from 7.5% to about 23%. Good blasting practice can help to minimize this unplanned dilution.

A diluting grade of 1 g/t (0.0292 oz/ton) was used.

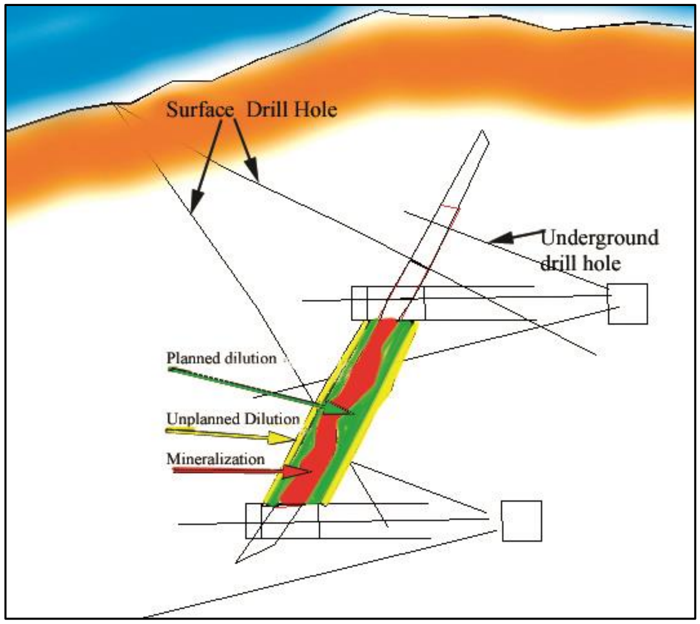


Figure 2-2: Dilution.

Table 2-4 Dilution that is possible from overbreak and blasting damage in the mine stope plan.

Zone	Thickness (ft)	Thickness After Blasting (ft)	Blasting Dilution
1a	7.0	8.0	14.3%
1b	8.0	9.0	12.5%
1c	13.4	14.4	7.5%
2a	7.0	8.0	14.3%
2b	9.7	10.7	10.3%
2c	5.3	6.3	18.9%
3a	6.8	7.8	14.7%
3b	4.4	5.4	22.7%

2.4.1 Opportunities for Improvement

Dilution can be minimized through conscientious blast design and implementation. A minimum mining width of 1.5 metres (5 ft) was used to outline the mineral resources (Hilchey and Wolfson, 2013). This would be considered “planned dilution.” However, longhole blasting utilizing modern, in-the-hole production drills, can be carried out as narrow as one metre (3 ft). This presents an opportunity to reduce the planned dilution, thereby reducing the milling cost and increasing the mill feed grade.

2.5 Mining Recovery

A mining recovery of 95% is used in this report as a percentage of the resource will be left as remnant pillars. Every effort will be made to recover profitable pillars. However, some losses are inevitable.

In the mining plan, approximately 85% of the resource is mined from the stope and another 10% is recovered from the sill pillars. Mining the sill can be problematic if the stope above is filled with loose rock.

To recover the sill between stopes, development of the bottom sill of the stope above could include constructing a concrete beam at the base of the sill. The sill would be driven 3.7 m (12 ft) high. This is because the stope will still be 3 m (10 ft) high after of the concrete floor has been constructed.

Recovery of the sill is done after the stope above has been completed and the stope below has been backfilled. The sill is recovered by drilling up holes from the stope below. Recovery is in retreat fashion, starting at the end of the stope and working back to the raise (Figure 2-3).

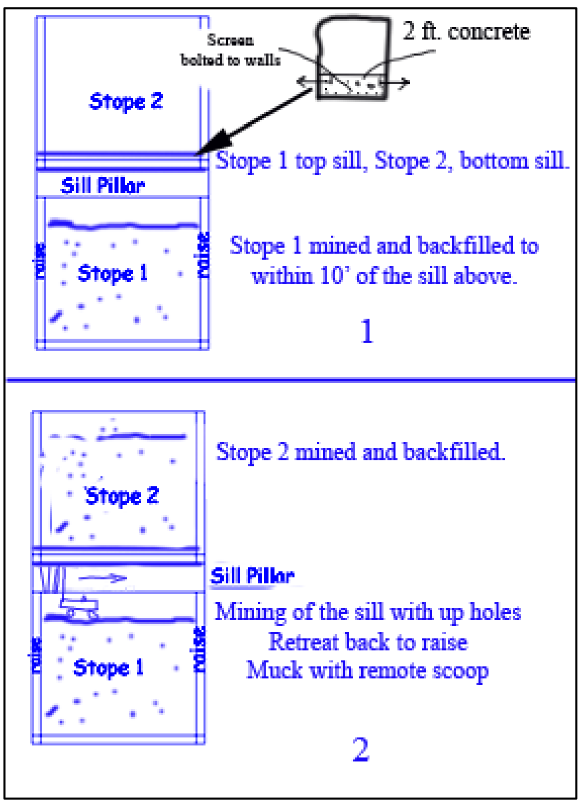


Figure 2-3: Schematic illustrating the steps required to recover the sill pillar.

2.6 Mining Adjacent or Adjoining Zones

Many zones coalesce and bifurcate in places (refer to Figure 2-1 for an example and Appendix 7 for detailed sections). Coalescing zones, or zones with only a thin pillar of “waste rock” separating them would be mined together as one. In many cases, the “waste” between zones is mineralized, but below cut-off. This would be considered “planned dilution.”

As the separation between zones increases, a decision would need to be made, on a case-by-case basis, whether the two zones would be most profitably mined as one or separately, leaving a waste pillar between them. This analysis would take into account that leaving a thin pillar may require additional ground support measures such as backfill or cable bolting.

2.7 Stope Mining Cutoff Grade

A stope cutoff grade of 3 g/t (0.0876 oz/ton) was used to outline the resources that are eligible for mining since a 3 g/t outline gives reasonable mining widths and mineralization continuity. At the time of report writing, 3 g/tonne is *approximately* the breakeven cut-off grade after mining, milling, and G&A (general and administration) costs are tallied. In practice, however, the profitability of each potential stope would be evaluated on a case-by-case basis. This evaluation would also take development into consideration.

If the haulageway is excavated through low grade material a calculation will be made to determine if the material should go to a low grade stockpile rather than the waste pile. The material may be profitable as it only has to pay for processing, the cost of transport to surface and overhead, including profit (breaking costs are sunk). The breakeven cut-off grade for already-broken rock would likely be in the 1.0-1.5 g/tonne range.

2.8 Mining-Recoverable Resources – Dawson Segment

The mining-recoverable resource (i.e.: potential mill feed) for the Dawson Segment totals 407,000 tonnes grading 8.9 g/tonne, for 117,000 ounces delivered to surface (shown in Table 2-5, below). At this point in time, the entire mineral resource is in the Inferred category.

The mining-recoverable resources for the Windy Gulch Segment are discussed in Section 5.

Table 2-5: Dawson Segment Diluted mining-recoverable resources with 5% mining losses.

Main Level	Zone	metric tonnes in volume	less 5% mining losses, metric tonnes	Available tonnes	g/t	Grams in mined material	Wall Dilution	dilution tonnes	dilution g/t	Grams in dilution	Total grams to surface	total metric tonnes delivered to surface	grams/tonne in material delivered to surface
6125	1a	2,520	126	2,394	8.48	20,305	14.3%	342	1.0	342	20647	2,737	7.54
6125	1b	38,060	1,903	36,157	10.3	370,607	12.5%	4,520	1.0	4520	375127	40,676	9.22
6125	1c	2,455	123	2,332	4.07	9,491	7.5%	175	1.0	175	9666	2,507	3.86
6125	2a	21,080	1,054	20,026	12.1	241,509	14.3%	2,864	1.0	2864	244373	22,889	10.7
6125	2b	10,345	517	9,828	10.7	104,665	10.3%	1,012	1.0	1012	105677	10,840	9.75
5862	1a	5,512	276	5,236	8.49	44,454	14.3%	749	1.0	749	45202	5,985	7.55
5862	1b	516	26	490	8.27	4,056	12.5%	61	1.0	61	4117	552	7.46
5862	1c	35,413	1,771	33,642	6.6	222,038	7.5%	2,523	1.0	2523	224561	36,165	6.21
5862	2a	14,647	732	13,915	4.78	66,512	14.3%	1,990	1.0	1990	68502	15,904	4.31
5862	2b	114,012	5,701	108,312	10.7	1,155,685	10.3%	11,156	1.0	11156	1166842	119,468	9.77
5862	3a	53,886	2,694	51,192	10.9	559,525	14.7%	7,525	1.0	7525	567050	58,717	9.66
5862	3b	4,788	239	4,548	4.14	18,831	22.7%	1,032	1.0	1032	19863	5,581	3.56
5687	2a	334	17	317	5.24	1,663	14.3%	45	1.0	45	1708	363	4.71
5687	2b	18,169	908	17,261	6.82	117,720	10.3%	1,778	1.0	1778	119498	19,039	6.28
5687	3a	11,271	564	10,708	24.5	262,336	14.7%	1,574	1.0	1574	263910	12,282	21.5
5425	2b	24,243	1,212	23,031	7.2	165,821	10.3%	2,372	1.0	2372	168193	25,403	6.62
5425	2c	11,129	556	10,573	13.1	137,977	18.9%	1,998	1.0	1998	139976	12,571	11.1
5425	3a	12,461	623	11,838	6.05	71,617	14.7%	1,740	1.0	1740	73357	13,578	5.40
Deeper	2c	1,022	51	971	13.0	12,607	18.9%	184	1.0	184	12790	1,155	11.1
Deeper	3a	243	12	231	5.92	1,366	14.7%	34	1.0	34	1400	265	5.29
Column Totals (rounded)		**382,000**	**19,000**	**363,000**	**9.40**	**3,589,000**		**44,000**		**44,000**	**3,632,460**	**407,000**	**8.92**
											TOTALS	407,000 tonnes	8.9 g/tonne
												449,000 tons	0.26 troy ounces/ton

Table 2-6: Summary of mineral resources and potential diluted mill feed (3 g/tonne block cut-off).

Non Diluted Mineral Resource Blocks						
	tonnes	tons	grams	ounces	g/tonne	oz/ton
Dawson Segment	382 k	421 k	3,778 k	121 k	9.89	0.288
Windy Gulch	12.2 k	13.5 k	137 k	4.4 k	11.2	0.327
Total (Rounded)	**394 k**	**435 k**	**3,920 k**	**126 k**	**9.93**	**0.290**
Diluted and Mineable Potential Mill Feed						
	tonnes	tons	grams	ounces	g/tonne	oz/ton
Dawson Segment	407 k	449 k	3,630 k	117 k	8.9	0.26
Windy Gulch	13.4 k	14.8 k	123 k	4.0 k	9.2	0.27
Total (Rounded)	**420 k**	**464 k**	**3,750 k**	**121 k**	**8.9**	**0.26**
Note: 3 g/tonne block cut-off.						

3 Mine Design

The majority of the Dawson deposit has a true thickness of less than 4 m (13 ft). Most mineralization is steeply dipping, between 50° and 70° to the south.

3.1 Historical Mining Methods

There are historical workings, all of which are shallow (less than 100ft (30m) deep on the property, known as the "Mike Sutton Workings", the "Last Show", the "Haulage Adit Fault", the "Copper Boy Workings" and the "Copper Boy" shaft. Old workings are found in the "Copper King" area where an adit was driven to intersect a southeast plunging (40° – 50°) structure. These old workings, including pits, shafts, and adits, which were primarily targeted on the copper mineralized massive sulphide zone stratigraphically above the gold zones, are described in Hilchey et al. (2013). Shrinkage stoping was likely the method employed at most of these deposits.

No historical workings affect the current mine design in any way. The Dawson segment being targeted for underground mining is a "blind" discovery[2] which is entirely intact and has not been subjected to any historical mining.

3.2 Geotechnical Considerations

The Dawson project is hosted by hard rock that has been folded, faulted and metamorphosed.

There is some debate regarding rock competence with respect to choosing a mining method. In the American Mine Services' pre-feasibility study (AMS, 1991), they chose cut-and-fill mining because "hanging wall rock conditions may prohibit the use of shrinkage stoping." AMS also felt that "the [mineralized rock] zones will be too narrow for mechanized equipment."

Dynatec (1991), on the other hand, were "of the opinion that a backfill system of mining is <u>not</u> required. This opinion was arrived at based on the configuration of the [mineralized rock] and the geologic logs and calculated RQDs from the drill holes…"

Because a site visit and personal examination of the core were beyond the scope of this work, the authors elected to rely on Dynatec's opinion that backfill would not be required and that the rocks are generally strong enough to stand up whilst mining is being completed. Therefore, cut-and-fill stoping was ruled out due to its higher cost.

During operations, the hanging wall rocks will be cable bolted from the top and bottom sills to reduce dilution. Diamond drilling will confirm stope geometry and the rock quality prior to mining.

During the initial mine development, geotechnical sampling for rock pressure should be done to determine the axis of the principal stress in the area. If weak hanging wall rocks are locally encountered that could significantly dilute the mineralized rock, then a cut-and-fill mining method could be substituted where required.

[2] A deposit that has no surface expression or "outcrop."

3.3 Groundwater

Groundwater inflow through the rock mass is not expected to be significant. Fracture porosity may carry some water into the underground workings near the surface. In the absence of detailed groundwater studies, groundwater infiltration is anticipated at no more than about 500 US gallons per minute.

3.4 Proposed Mine Design

Several mining methods could be applied to this deposit, including “Alimak mining,” shrinkage stoping, and longhole sublevel stoping. The various advantages and disadvantages of each method were considered (details in Appendix 1) and longhole sublevel stoping was selected for preliminary mine design.

Mining would be near-completely mechanized. Jack-leg and stoper drilling would be minimized.

A mine plan was developed based on the criteria discussed in previous sections. These parameters are summarized in Table 3-1. The overall plan is shown in Figure 3-1 and Figure 3-2, with detailed cross-sections and level plans shown in Appendix 4.

For the Dawson deposit, the portal will be at elevation 1,981 m (6,500 ft). The base of the known mineralization is at about 1,646 m (5400 ft), a vertical distance of 335 m (1,100 ft).

The sills and the raises are driven in mineralization. The decline, haulage levels and the ventilation raise are all driven in waste rock.

Geological and geotechnical information is collected prior to mining a particular stope. Diamond drill holes, drilled from the footwall ramp provide geological, grade, and geotechnical data prior to stoping and aid the stope design process.

Although there are some historical workings on the site as described in Section 3.1, no existing underground workings affect or are incorporated in this mine plan.

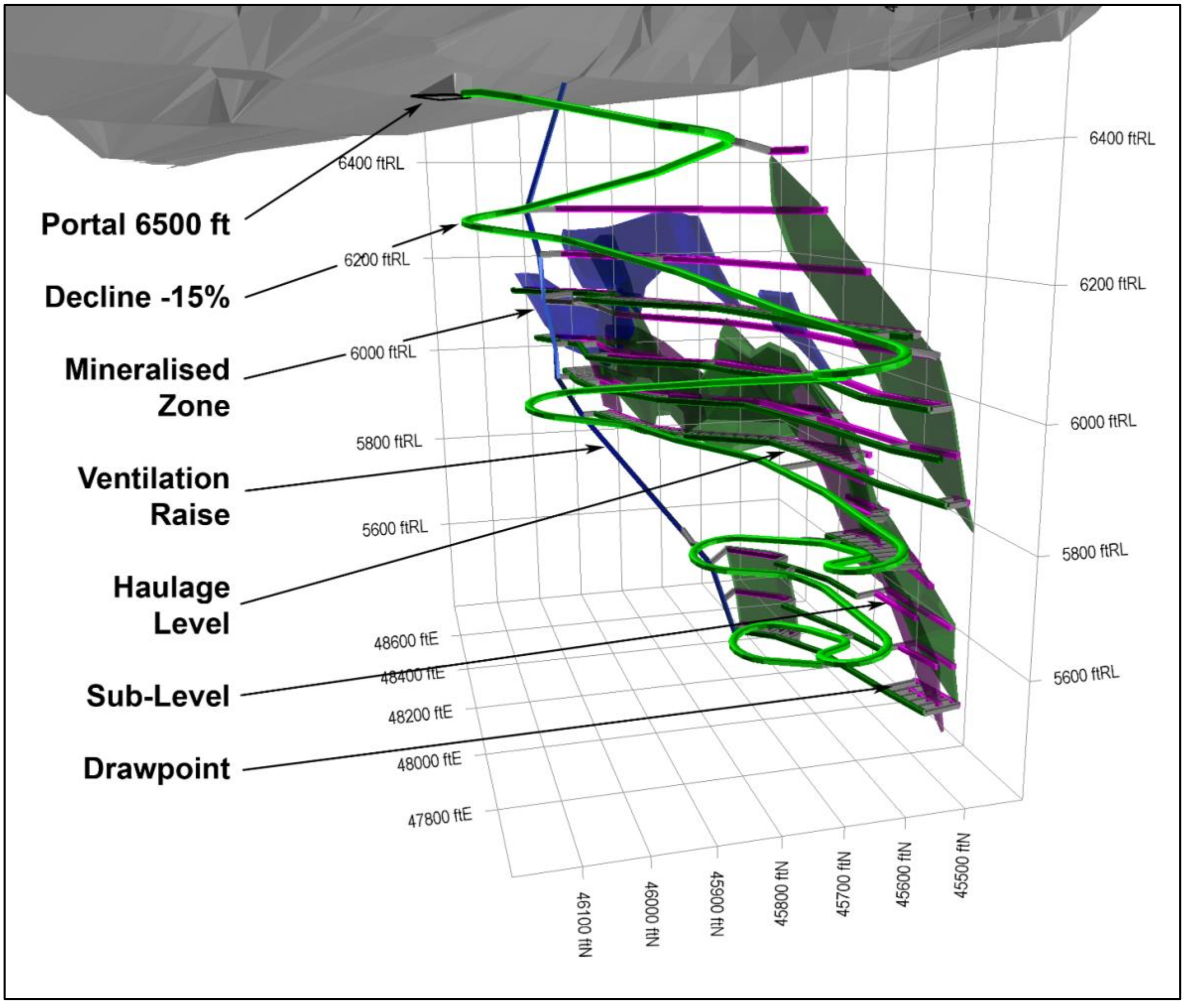
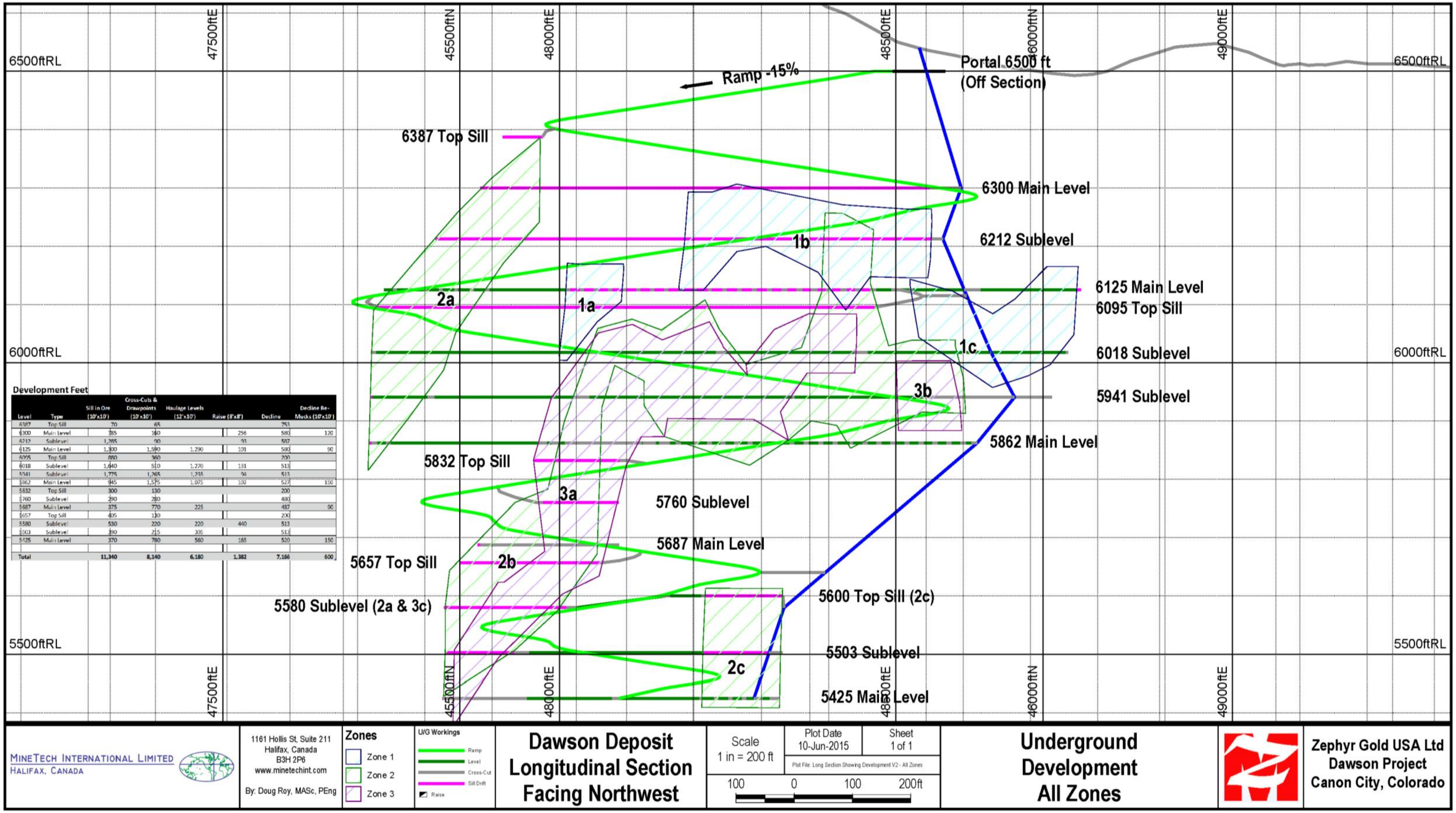


Figure 3-1: Three-dimensional view of Dawson underground development (facing east).



Development Feet

Level	Type	Sill in Ore (10'x10')	Cross-Cuts & Drawpoints (10'x10')	Haulage Levels (12'x10')	Raise (8'x8')	Decline	Decline Re-Mucks (10'x10')
6387	Top Sill	70	65			753	
6300	Main Level	785	160		256	580	120
6212	Sublevel	1,285	90		93	587	
6125	Main Level	1,300	1,590	1,290	101	580	90
6095	Top Sill	880	360			200	
6018	Sublevel	1,640	510	1,270	131	513	
5941	Sublevel	1,775	1,265	1,235	94	513	
5862	Main Level	945	1,575	1,075	102	527	150
5832	Top Sill	300	130			200	
5760	Sublevel	290	280			480	
5687	Main Level	375	770	225		487	90
5657	Top Sill	405	130			200	
5580	Sublevel	530	220	220	440	513	
5503	Sublevel	390	215	305		513	
5425	Main Level	370	780	560	165	520	150
Total		11,340	8,140	6,180	1,382	7,166	600

Figure 3-2: Longitudinal section showing underground development.

3.5 Mine Development

The main development priorities are the decline, the main haulage drifts and the ventilation/escapeway raise.

During the first year's development work, waste rock would have to be hauled out of the mine, to be used for road construction and to flatten out areas on surface for the surface infrastructure. Once mining commences, the waste rock would be used to backfill mined-out stopes.

Trackless equipment would be used. The equipment includes 18 tonne (20 ton) trucks and load-haul-dump (LHD) units with a capacity of 1.15 m^3 and 2.295 m^3 (1.5 cubic yards and 3.0 cubic yards). A drill jumbo and a longhole drill will be used on the ramp and for stope mining respectfully.

The underground mine would be developed from a -15% ramp. Once underground, infill diamond drilling will be carried out on, and between current resource blocks to upgrade the resources to the indicated category. And, drilling will be undertaken on currently identified exploration targets that show potential to expand the resource base.

Table 3-1: Mine design parameters.

Item	Property
Mineralized Zone Width	1.5 to 10 m (5 ft to 33 ft)
Mining Method	Longhole Sublevel Stoping
Available Mineral Resources and Potential Mill Feed	Refer to Table 2-6.
Dilution	0.15 m (6 in) Wall Rock at 1 g/tonne, Averaging 14.5%
Mining Recovery	95% Overall
Milling Rate	Average 272 tonnes (300 tons) per day, 365 days per Year
Mill Downtime	1 Shift per Week Preventive Maintenance, 2 Weeks per Year Breakdowns & Major Maintenance
Mill Nameplate Capacity	310 tonnes (340 tons) per day
Target Mining Rate	363 to 386 tonnes (400 to 425 tons) per day, 5 days per week
Stope Cut-off Grade	3 g/tonne (0.0876 oz/ton)
Haulage Level Spacing	53 vertical metres (175 feet)
Sublevel Spacing	21 m to 27 m (70 ft to 90 ft)
Decline Gradient	-15%
Decline Size	4.6 m x 3.4 m (15 ft x 11 ft) (Arched)
Haulage Drifts (Level Drifts)	3.7 m W x 3.0 m H (12 ft x 10 ft) (Arched)
Raises	2.4 m x 2.4 m (8 ft x 8 ft)
Sill Drifts	3.0 m (10 ft) High, Width of Zone [Minimum 3.0 m (10 ft)]
Cross-Cuts	3.7 m H x 3.0 m W (12 ft x 10 ft)
Draw Drifts	3.0 m x 3.0 m (10 ft x 10 ft)

3.5.1 Portal

A portal was positioned on the footwall side of the deposit, roughly 100 m (328 ft) north-northwest of the deposit.

An accurate elevation survey was not available for portal design work. According to the supplied digital terrain model, the elevation of the selected location is approximately 1,981 m (6,500 ft).

The terrain's gradient in this location is approximately 20°. As an approximate guide, the solid rock overburden thickness over the slope's brow should be approximately 1 to 2 times the decline height, or 1.5 times 3.6 m (12 ft), equal to 5.5 m (18 ft) of rock cover.

It is recommended that approaching the brow, the ground have a +1° slope (1.7%) to the brow so that surface runoff will drain away from the portal.

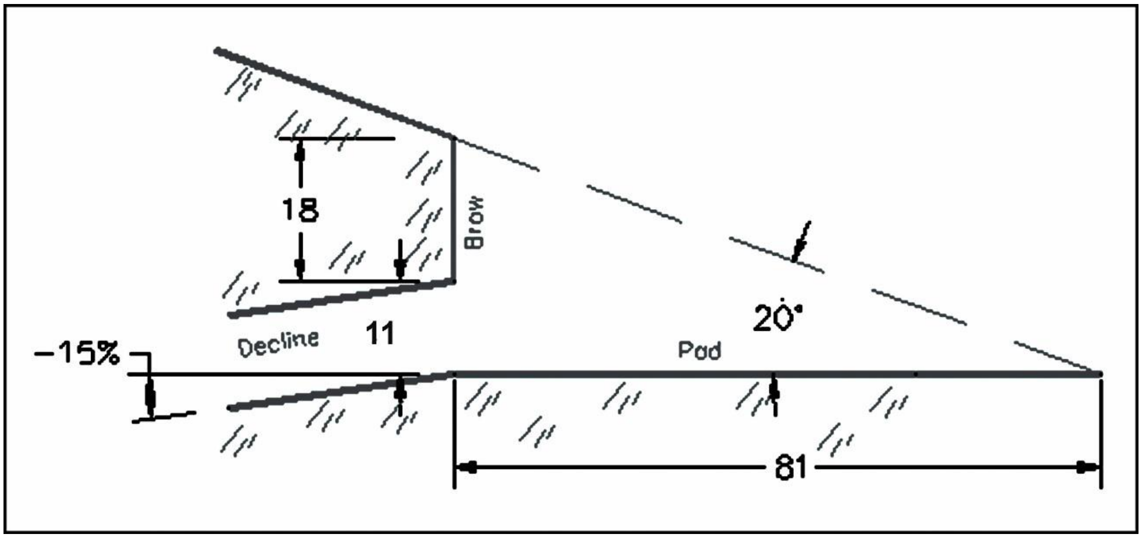


Figure 3-3: Portal cross-section. Dimensions in feet.

3.5.2 Decline

Development of the deposit would be from a decline driven in the footwall gneisses. The decline is designed to stay between 18 m and 30 m (60 ft and 100 ft) from the mineralization by spiraling at the southwest and northeast ends of the deposit and shifting southeast as the deposit dips. Drilling the decline would be by electric-hydraulic jumbo. Mucking and haulage will be by 2.3 m^3 (3 cubic yard) LHD and 18 tonne (20 ton) trucks. Ground support in the ramp will be rock bolts and screen.

For drilling a decline round, 3.7 m (12 ft) drill steel could be used. These rounds would likely break to approximately 3.4 metres (11 ft). Smaller openings such as cross-cuts and sill drifts would likely be drilled using shorter, 2.4-3.0 m (8-10 ft) drill steel because a development round advance is typically in the neighbourhood of 80% of the opening's width.

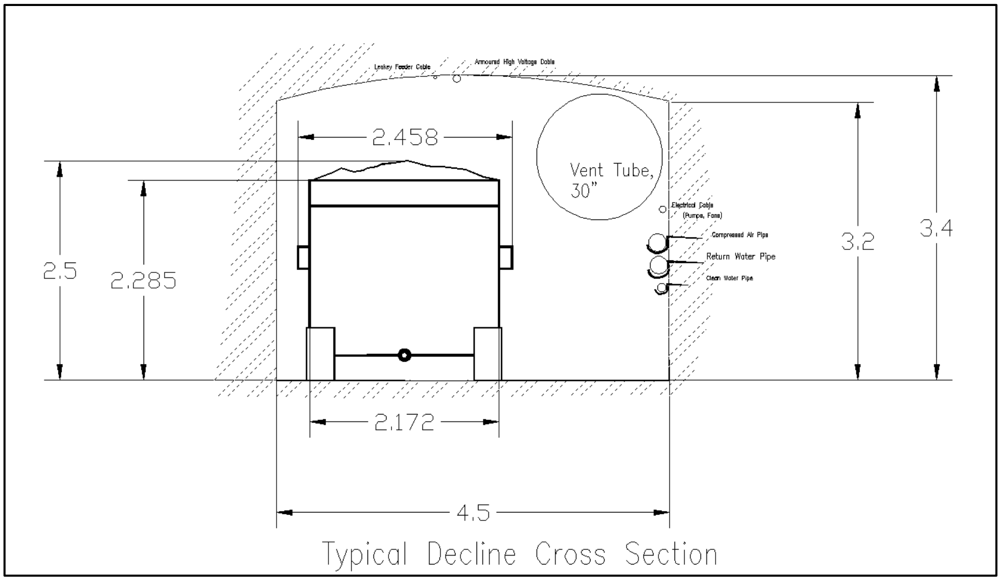


Figure 3-4: Decline cross-section.

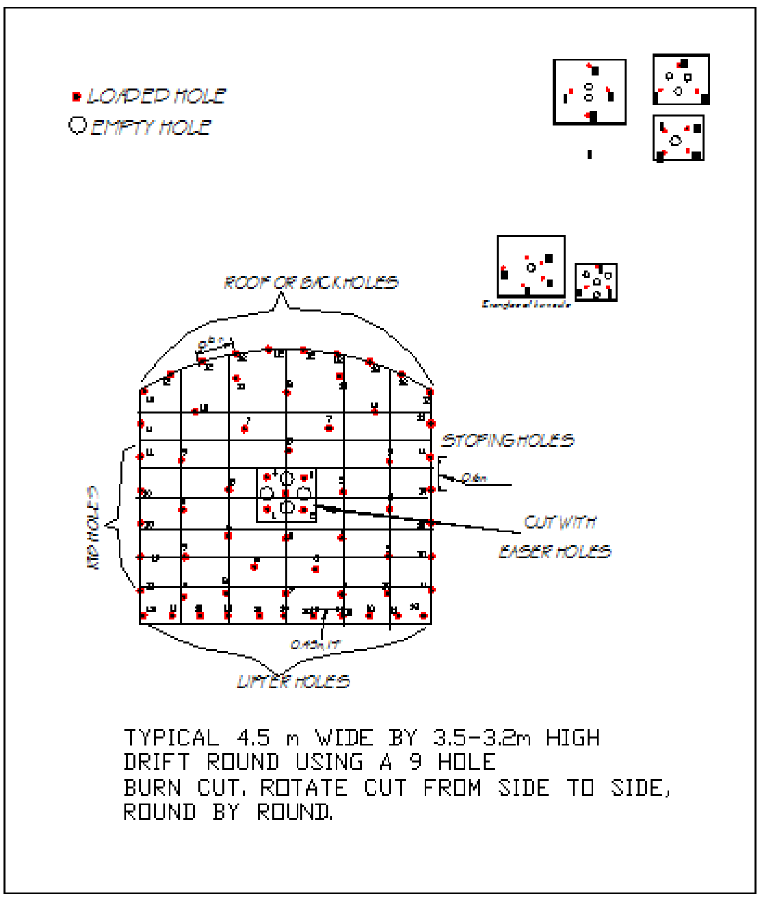


Figure 3-5: Decline blasting pattern.

The decline excavation will generate approximately 15,600 tonnes (17,200 tons) of waste over the 53.3 vertical metres (175 vertical feet) between levels. The decline is planned to have a cross sectional area of 15 m^2 (160 ft^2), being 4.6 m (15 ft) wide, and 3.4 m (11 ft) high. The decline is 3.2 m (10.5 ft) high at the shoulder.

The powder factor used for the decline is 2.2 kg/m^3 (1.7 pounds of explosives per ton). For each round, a total of 68 holes are drilled. The cut holes are not loaded and the rib and back holes are loaded lighter to limit blast damage (see Figure 3-5). There will be a 9 m (30 ft) bay or "remuck" cut into the walls every 90 m (300 ft). The scooptram will load trucks from one of these bays to keep the tramming time under control.

Table 3-2 illustrates the time to advance the decline between the main haulage levels, which will be 53.3 m (175 ft) apart.

Table 3-3 is an estimate of the materials required for ground support for each decline round. As the decline ramp is a semi-permanent structure, the ground support includes regular bolting and screening of the back and upper walls. Rock quality is expected to be good within the footwall gneisses. At this time there is no hard data on rock pressures. In mountainous areas, horizontal pressures are sometimes greater than the vertical rock pressure.

Table 3-2: An estimate of the time to excavate the ramp between levels that are 175 feet vertically apart.

Dawson Decline metres between Haulage Levels.	metres, m^3, tonnes	feet, ft^3, tons
vertical distance	53	175
slope distance	356	1,167
decline remucks	4	4
remuck length	32	105
advance/rnd	3	9
total length of advance	388	1,272
total volume of rock broken, m^3	5,814	205,322
Number of rounds	144	144
nominal width	4.6	15
nominal height (arched back)	3	11
Volume (m3) per round solid	41	1,447
specific gravity	2.63	2.63
Tonnes per round + 5% overbreak	113	125
swell	38%	38%
Loose Volume per round loose, plus 5% overbreak, m^3	57	1,997
required Shifts	146	146
shifts per day	2	2
Minimum number of days to drive decline between main levels:	73	73

Table 3-3: An estimate of the consumables used for each decline round.

Consumables per decline round	metres	Feet	Cost
Ground Control			
Area to be bolted, sides and back	35	377	
Number of bolts required for walls and back	17	17	$185.25
Screen required	25	269	$538.00
pins for screen	8	8	$16.00
Bolt length	2.44	8	
bolt spacing	1.22	4	
Scissor Lift, time, cost	1.2		$49.45
Explosives			
Kg. ANFO, trim powder	97.2		$173.02
caps and detonator for holes	68		$274.96
One way Haulage, metres	1800		
Truck loads/round	6		
Truck return time /load, minutes	65		
truck time, hours	7.1		
Fuel, maint and Lub			$108.61
Drilling			
drill metres: steel and bit cost	183.6		$100.67
Jumbo hours; fuel & maintenance	3.4		$58.65
Power			
Power Use for 12 hr. shift			
Fan, Pumps, Compressor etc. kw, hrs, $./kW	2160	$0.07	$151.20
Compressed Air Power			
Jacklegs for bolting	1.2	$14.16	$17.51
Piping			
2 inch water intake	3	8.9	$34.14
4 inch water discharge	3	8.9	$77.36
4 inch compressed air, Vic couplings	3	8.9	$97.36
Ventilation			
Ventilation tube and accessories	3	8.9	$22.94
Electric Cable	3	8.9	$17.80
Leaky Feeder	3	8.9	$17.80
Subtotal			$1,940.72
Miscellaneous	15%		$291.11
Consumables per 3m round.		total/ per round -->	$2,231.83
$826.60	per m	$251.95	per foot

3.5.3 Diamond Drilling

Once a drilling space is available a number of holes will be drilled to further define the mineralized zones. The operating cost for diamond drilling underground is approximately $23 per metre ($7.00 per foot). At each drill set up, at least three holes would be drilled, as illustrated below in Figure 3-6. This would amount to about 107 m (350 ft) of drilling from each setup for a cost of about $2,500 to $3,000, depending upon the hardness of the rock and amount of assaying.

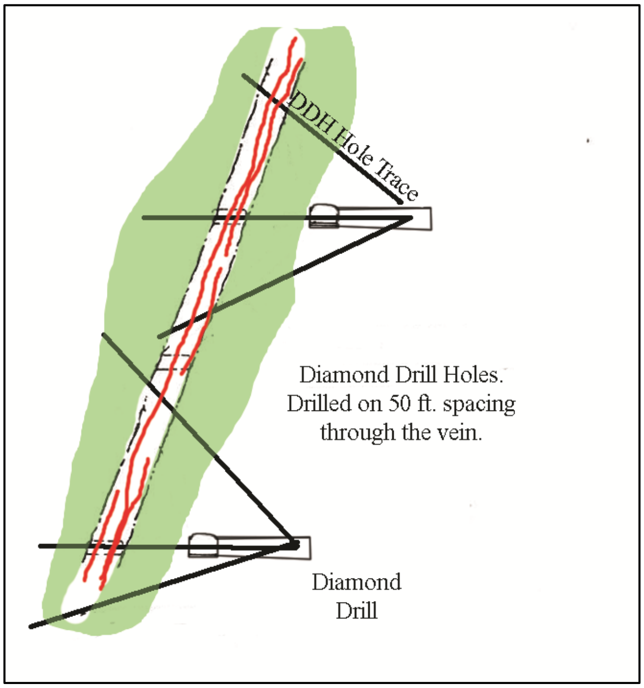


Figure 3-6: Diamond Drill setup from a former remuck station.

3.5.4 Refuge Chambers

The remuck stations near the 6300 Level will be a furnished as a refuge chamber and lunch room until the ventilation raise/escapeway has been completed. If an Alimak raise climber is used, the Alimak nest could then be outfitted as the 6125 Level refuge chamber.

3.5.5 Ventilation Raise

The ventilation raise will be excavated 2.4 m by 2.4 m (8 ft by 8 ft), and will be equipped with ladders and landings.

The ventilation fan air could be downcast to prevent the portal and decline from freezing in winter. In the unlikely event that freezing air in the vent raise causes ice buildup that could impede use of the escapeway, an air heater will be required.

Mine ventilation is further discussed in Section 3.8.

3.5.6 Development Schedule

Development of the decline is on the critical path and should be take priority. A second exit (the ventilation raise) and a refuge station (6125 Level) are required prior to production mining.

Development of the mine to the point at which production mining could start will take about a year. After the ventilation raise has been driven up from the 6125 level and after the refuge station has been established and equipped, production from the 6125 level could begin. Between the portal and the 6125 level, approximately 25,000 tonnes (27,300 tons) of mineralization will have been mined from the sills along with about 59,000 tonnes (65,000 tons) of waste. A summary of development and production milestones is included below as Table 3-4.

Table 3-4: Dawson project development and production schedule.

	months	1	2	3	4	5	6	7	8	9	10	11	12										
BOD Decision to finance Dawson Underground Development																							
Prepare detailed site maps of topography options for plant site, tailings disposal, office site. Complete metallurgical flow sheet for plant, Preliminary feasibility study report.																							
Permitting and Public Meeting		To be completed prior to Month 1.																					
Retain senior mining, hourly and processing staff																							
Survey areas required for tailings area, portal and ventilation raise areas and the plant area, Complete Layout survey of plant, tailings area, access road etc. Prepare portal site, flatten an area for infrastructure.																							
Ancillary services and requirements																							
Complete design of infrastructure. Obtain all necessary municipal permits for construction. Construct sewage disposal field, roads, security fencing and construct office, dry, shop and warehouse around plant.																							
Mining exploration and Development - Underground																							
Detail labour and equipment requirements, cost and replacement schedule. Order underground equipment & inventory. Excavate portal area and ventilation/egress raise area to solid bedrock. Complete concrete work in portal and vent raise area . Portal El 6545 feet.																							
Begin underground work																							
Advance decline to the top of mineralization elevation, about 6300 elevation.																							
Cross cut to 6300 sill and excavate sill																							
Excavate Alimak nest for ventilation raise from x-cut off of 6300 L.																							
Excavate and equip vent raise- escapeway from 6300L to surface, 2 rounds/day																							
Continue Mine Development, Excavate Decline to 6237 elevation, 450 ft., Establish 6237 drll sill. Set up diamond drill in re-muck off ramp, section E, M and R areas.																							
Underground Diamond Drilling set up 1,2,3														**Continue Diamond Drilling exploration and development**									
Complete Resource Estimate, Prepare more detailed mine plan.															BOD - Complete Development & start production								
Decide to Proceed with mine development																							
Excavate 6125, 5862, 5687, and 5425 main levels, sublevels, and topsills, approximatly 10,500 feet development. Vent raises to connect main levels.														**Production Mining starts and Development continues if sufficient resources are demonstrated.**									

3.6 Stoping

Stoping would be carried out using a mobile, in-the-hole longhole drill. The mining cycle includes backfilling the stope as soon as possible before the stope walls deteriorate.

The sequence of stoping will depend upon the grade of the material, rock pressures, continuity of mineralization, and continuity of thickness of the mineralization. After detailed drilling, low grade areas can be identified and left as pillars. Maximum recovery of the deposit will be possible if backfilling is completed as the deposit is mined.

3.6.1 Stope Development

The capital development of a stope includes the decline, the footwall haulageway, cross cuts to the sills, the ventilation raise, refuge chambers, water sumps, electrical substations, powder and cap magazines and various storage places underground.

From the decline, two sill levels are driven approximately 53.3 vertical metres (175 ft) apart. Raises are excavated from the bottom sill to the top sill every 60 m (200 ft) along strike, beginning at the west end of the deposit.

Sill drifts would be 3 m (10 ft) high and the width of the zone, with a minimum width of 3 m (10 ft) to accommodate mobile equipment. These sills are driven in the mineralized horizon at a slight uphill grade of 0.5% (1 foot in 50 feet) so that water will drain back to the decline and loaded vehicles will have a slight advantage going downhill.

The upper sill level will be driven 3 m x 3 m (10 ft x 10 ft) in mineralization, under geological guidance. It will be as wide as the mineralization but in any case, no less than 3 m (10 ft) wide to accommodate the longhole drill. A bottom sill is then driven in mineralization to the west and east under geological control, in order to define the bottom of the stopes.

Once the bottom sill level has been excavated, mapped and surveyed, a haulage level can be excavated in the footwall about 9 m (30 ft) from the bottom sill. Draw points are excavated from the haulage level through the mineralized horizon and one round into the hanging wall.

Sublevel spacing for parallel blast holes drilled in the dip direction would be 15 m to 30 m (50 ft to 100 ft) apart. A 20 m (65 ft) sublevel spacing has been incorporated.

3.6.2 Drop Raises

A raise is needed in each stope to provide a void for the initial production blasts. Drop raising is the proposed method for creating this void.

After the top and bottom sills have been excavated, a drop raise is excavated by drilling and blasting the raise from the top. A large hole, 15 to 20 cm (6 in to 8 in) in diameter, is drilled for relief. Drilling accuracy is necessary for the drop raise method. Once the raise is excavated, the walls are slashed to the full width of the zone and stope mining can begin.

There are several advantages to this method over conventional raise driving. It is safer as no persons are in the raise. It is cheaper since there are no ground support costs and mining can begin as soon as the raise has been excavated. Drillhole accuracy has improved greatly with rigid rods and down-the-hole hammer (DTH) drilling rigs. The dip of the structure will limit the practical length of the drop raise.

Table 3-5: Productivity for drop raising.

4.3 - Drilling

	Drilling time (hours)	
Sub-levels	Drop raises	Long-Hole
1	35.96	19.42
2	35.96	19.42
3	35.96	19.42
0	0.00	22.01
0	0.00	22.01
0	0.00	22.01
	107.87	124.32
# shifts (incl maintenance)	19	22
drilling hours (incl maintenance)	**113.57**	**130.92**
	Average productivity (m/shift)	
Drop raises ->	**44.3**	**69.2**

NOTE: The average productivity includes: drilling, daily maintenance, fixed time per shift and contingencies. <- Long-Hole

	Redrilling time following blast damages (hours)			
	Drop raises		Long-Hole	
	%	Hours	%	Hours
	10%	3.6	1%	0.2
	10%	3.6	1%	0.2
	10%	3.6	1%	0.2
	10%	0.0	1%	0.2
	10%	0.0	1%	0.2
	10%	0.0	1%	0.2
	10%	10.79	1%	1.24
# shifts (incl maintenance)		2		0
drilling hours (incl maintenanc		**11.39**		**1.24**

Productivity (meters/hour)	
Drop raises	Long-Hole
8.9	**13.9**

Mechanical maintenance of drill (for each shift)		
Grease and lubricants	8.0	
Check bolts of table, slide, etc.	4.0	
Check hydraulic oil level	2.0	
Drill cleaning	2.0	
Other verifications	2.0	
Average maintenance time per shift	**18.0**	**min/shift**

3.6.3 Production Rate

It will take at least six months to drive the decline to the 6125 sill elevation and another six months to develop the deposit for production.

A detailed development and production schedule is included as Appendix 7 (a summary is presented in Table 3-4). A summary of drilling and blasting during stoping is included as Table 3-6.

An average milling rate of approximately 272 tonnes (300 tons) per day, 365 days per year was considered. Assuming one shift per week would be devoted to preventive maintenance and two weeks per year would be lost to breakdown and major maintenance, the mill capacity should be 310 tonnes (340 tons) per day.

Production mining would be carried out over five days per week, with a targeted mining rate of 363-386 tonnes (400-425 tons) per day. Waste mining will be approximately 340 tonnes (375 tons) per day, 227 tonnes (250 tons) from the ramp and another 113 tonnes (125 tons) from other development.

At first, the only mineralized rock mined will be from the sills, raises and the swell from the stope. As the mining proceeds upward about 40% of the rock blasted is drawn out of the stope. The remaining rock stays put in the stope until stope mining has been completed.

As shown in Table 3-7, approximately 82% of the time the worker spends underground is productive time.

Table 3-6: Drilling and blasting, under stoping conditions

Item	Qty	Unit
Hole Size	0.066	m
Hole Length	20	
Hole Depth	22	m
Total Hole Volume	0.07527	m^3
Percent of hole depth filled with explosives	0.8	
volume of explosive	0.06021	m^3
bulk density of explosives, average	1150	kg/m^3
Weight of explosive in hole	69.24492	kg
explosive factor, kg /m^3 rock blasted	1.5	
m3 broken per hole	46.2	
total tonnes mineralized rock and waste per year	77,800	tonnes
total m^3 mineralized rock and waste per year	29,000	m^3
total holes per year	628	
total length of hole	13,821	m
bits required	69	
drilling rate while drilling the hole	20	m/hr
actual drilling time required	691	hours
scheduled annual hours	1,400	hours
overall job efficiency	0.8	
mechanical availability	0.8	
annual outage factor	0.95	
production utilization	0.608	
actual production hours	851.2	hours
drills required	0.8	
minimum drills in use or available	**1**	

Table 3-7: Shift metrics for underground production workers.

Shift Metrics	Time (minutes)
Shift length	600
Planned Downtime	
Travel to workplace	15
Safety Talk	5
Break #1	15
Lunch	30
Break #2	15
Total Planned Downtime	80
Unplanned downtime, 5%	30
Total Downtime	110
Available work time	**490**

3.6.4 Production Blasting

In areas of narrow stoping widths, relatively fine fragmentation is required to promote the free flow of rock down the stope to the drawpoints. This can be achieved by employing proper blast design and conscientious blast implementation.

3.6.5 Longhole Stoping Productivity

Table 3-8 and Table 3-9 illustrate the parameters for longhole stoping evaluation and production drilling.

Table 3-8: Longhole stoping evaluation.

Long-Hole Stope Evaluation

RFEFERENCE: Zephyr, Dawson May 8, 2015

1.0 - Physical parameters of the stope

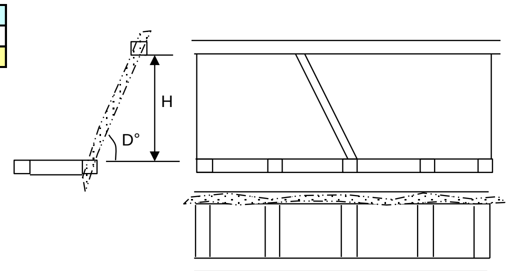
27861.56796

Typical stope

Length (m)	Width (m)	Vertical height	Dip (°)	Slope distance (m)	Density (t/m³)	Swell factor
60	3.0	53.34	65	59	2.63	1.38

number of drawpo	6

in place		Milled tonnes and grade		
Tonnage	Grade (g/t)	Tonnage	Grade (g/t)	Dilution
27,862	10.00	29,255	9.57	5%



2.0 - Stope development

2.1 - Development parameters

	Dimensions					
	Width	Height	T/m advanced	# of men	M advanced /shift	M advanced /manshift
Drift 1	3.7	3.2	31.1	**3**	10.38	**3.46**
Drift 2	3.7	3.2	31.1	**3**	10.38	**3.46**
Drawpoints	3.0	2.7	21.3	**2**	10.65	**5.33**
Sub-level 1	3.0	3.0	23.7	**3**	7.89	**2.63**
Sub-level 2	4.0	3.0	31.6	**3**	10.52	**3.51**

Dawson Deposit Long-Hole Stope

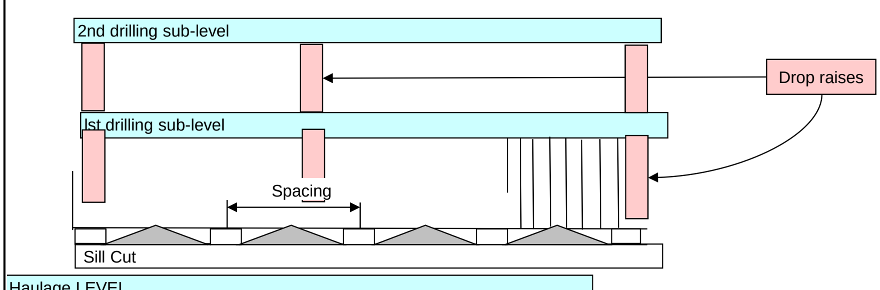


Table 3-9: Longhole production drilling.

3.2 - Stope life calculation						
Broken tonnes/day (in production)	240	tonnes				
Mucking capacity during the blasting phase						
a) muck the excess due to the swell	91	tonnes/day				
b) muck to equipment capacity	250	tonnes/day	**Stope life (production)**			
			94	working days	**4.5**	months
Mucked tonnes/day (in production)	331	tonnes				
Remaining tonnage inside the stope	2127	tonnes				
- Lost ore between drawpoints	50	tonnes				
= Recovered tonnage inside the stope	**2077**	**tonnes**	**Time required to muck the rest of the stope**			
Tonnes/shift (mucking of the stope)	**165.6**	**tonnes/shift**	**6**	working days	**0.3**	months

3.6.6 Backfilling

Once the stope has been emptied, development waste can be dumped from above to provide wall support and to dispose of waste rock (rather than trucking it to surface). When loading waste the LHD operator must ensure that no oversize rock is included as it could cause the fill to hang up in the stope. The largest piece of backfill should be no bigger than 1/8th the width of the stope to backfill.

Table 3-10: Number of rounds to completion by Level and Type of Development

Level		Sill Rounds	X-Cut Rounds	Haulage Levels Rounds	Raise Rounds	Decline & Remuck Rounds
6387	Top Sill	8	8			119
6300	Main Level	89	18		36	83
6212	Sublevel	145	11		13	67
6125	Main Level	146	179	129	14	76
Underground Exploration Can begin once the decline gets to the 6125 level.						
6095	Top Sill	99	41			23
6018	Sublevel	185	58	127	19	58
5941	Sublevel	200	143	124	13	58
5862	Main Level	107	177	108	14	77
5832	Top Sill	34	15			23
5760	Sublevel	33	32			55
5687	Main Level	43	87	23		66
5657	Top Sill	46	15			23
5580	Sublevel	60	25	22	62	58
5503	Sublevel	44	25	31	0	58
5425	Main Level	42	88	56	23	76
	Number of rounds	1281	922	620	196	920
	Days to completion*	**427**	**461**	**310**	**98**	**460**
	Months to completion	14	15	10	3	15

* Assuming 2 rounds/day except 3 rounds/day in sills.

Table 3-11: Tons per Round by Type of Development

Location	Sill Rounds	X-Cut Rounds	Haulage Levels Rounds	Raise Rounds	Decline & Remuck tons
tons/round	73	73	98	37	117

Table 3-12: Time required for decline excavation between main levels.

Dawson Decline metres between Haulage Levels.	metres, m^3, tonnes	feet, ft^3, tons
vertical distance	53	175
slope distance	356	1,167
decline remucks	4	4
remuck metres	32	105
advance/rnd	3	9
total metres drivage	388	1,272
total volume of rock broken, m^3	5,814	205,322
Number of rounds	144	144
nominal width	4.6	15
nominal height (arched back)	3	11
Volume (m^3) per round solid	41	1,447
specific gravity	2.63	2.63
Tonnes per round + 5% overbreak	113	125
swell	38%	38%
Loose Volume per round loose, plus 5% overbreak, m^3 ?	57	1,997
required Shifts	146	146
shifts per day	2	2
Minimum number of days to drive decline between main levels:	73	73

Table 3-13: Drilling required per round for the decline.

Item	Amount (metric)		Amount (US Customary)	
35mm, loaded holes	68			
75mm empty holes	4			
Mass of broken rock	112	tonnes/round	123	tons/round
Volume of broken rock	42.6	m^3/round	1503	ft^3/round
Hole Length	3	metres	10	feet
Length of Drilling	216	metres	709	feet

Table 3-14: Estimated cost of consumables for development.

Level	Type	Sill in Mineralization (10'x10')	Cross-Cuts & Drawpoints (10'x10')	Haulage Levels (12'x10')	Raise (8'x8')	Decline	Decline Re-Mucks (10'x10')	Total	Total Waste Mining, Consumables cost
6387	**Top Sill**	$14,420	$14,235			$216,918		$245,573	$231,153
6300	Main Level	$161,710	$35,040		$43,520	$119,480	$33,750	$393,500	$231,790
6095	Top Sill	$181,280	$78,840			$41,200		$301,320	$120,040
6018	Sublevel	$337,840	$111,690	$278,130	$22,270	$105,678		$855,608	$517,768
6212	Sublevel	$264,710	$19,710		$15,810	$120,922		$421,152	$156,442
6125	Main Level	$267,800	$348,210	$282,510	$17,170	$119,480	$20,250	$1,055,420	$787,620
5941	Sublevel	$365,650	$277,035	$270,465	$15,980	$105,678		$1,034,808	$669,158
5862	Main Level	$194,670	$344,925	$235,425	$17,340	$108,562	$33,750	$934,672	$740,002
5832	Top Sill	$61,800	$28,470			$41,200		$131,470	$69,670
5760	Sublevel	$59,740	$61,320			$98,880		$219,940	$160,200
5687	Main Level	$77,250	$168,630	$49,275		$100,322	$20,250	$415,727	$338,477
5657	Top Sill	$83,430	$28,470			$41,200		$153,100	$69,670
5580	Sublevel	$109,180	$48,180	$48,180	$74,800	$105,678		$386,018	$276,838
5503	Sublevel	$80,340	$47,085	$66,795		$105,678		$299,898	$219,558
5425	Main Level	$76,220	$170,820	$122,640	$28,050	$107,120	$33,750	$538,600	$462,380
		$2,336,040	***$1,782,660***	***$1,353,420***	***$234,940***	***$1,537,996***	***$141,750***	***$7,386,806***	***$5,050,766***

Table 3-15: Production schedule summary.

Item	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Y2 Q1	Y2 Q2	Y2 Q3	Y2 Q4	Year 3	Year 4	Year 5	Year 5	Total (Rounded)
Mining Equipment Capital	$4,000 k									$2,070 k											**$6,070 k**
Initial Development	$283 k	$283 k	$283 k	$283 k	$283 k	$412 k	$487 k	$571 k	$640 k	$706 k			$2,001 k								**$6,230 k**
Production Mining	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$158 k	$554 k	$1,003 k	$1,478 k	$8,131 k	$7,392 k	$10,377 k	$6,838 k	**$35,920 k**
Windy Gulch Mining	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$122 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	$0 k	**$120 k**
Dawson Sill Tons	0 k	0 k	0 k	0 k	0 k	2 k	4 k	6 k	7 k	9 k	9 k	7 k	18 k	12 k	9 k	3 k	0 k	0 k	0 k	0 k	**85 k**
Dawson Stope Tons	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	2 k	7 k	13 k	18 k	102 k	92 k	130 k	85 k	**364 k**
Total Dawson Tons	0 k	0 k	0 k	0 k	0 k	2 k	4 k	6 k	7 k	9 k	9 k	7 k	20 k	19 k	21 k	21 k	102 k	92 k	130 k	85 k	**449 k**
Tonnes	*0 k*	*0 k*	*0 k*	*0 k*	*0 k*	*2 k*	*4 k*	*5 k*	*6 k*	*8 k*	*8 k*	*7 k*	*18 k*	*17 k*	*19 k*	*19 k*	*92 k*	*84 k*	118 k	*78 k*	***407 k***
Windy Gulch Tons	0 k	0 k	0 k	0 k	0 k	7 k	7 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	0 k	**15 k**
Total Tons to surface	0 k	0 k	0 k	0 k	0 k	9 k	11 k	6 k	7 k	9 k	9 k	7 k	20 k	19 k	21 k	21 k	102 k	92 k	130 k	85 k	**464 k**
Tonnes	*0 k*	*0 k*	*0 k*	*0 k*	*0 k*	*9 k*	*10 k*	*5 k*	*6 k*	*8 k*	*8 k*	*7 k*	*18 k*	*17 k*	*19 k*	*19 k*	*92 k*	*84 k*	118 k	*78 k*	***420 k***
Grade (opt)	-	-	-	-	-	0.27	0.27	0.28	0.28	0.28	0.28	0.28	0.28	0.27	0.27	0.26	0.26	0.25	0 k	0.25	**0.26**
g/tonne	-	-	-	-	-	*9.2*	*9.3*	*9.7*	*9.7*	*9.7*	*9.7*	*9.7*	*9.6*	*9.3*	*9.1*	*8.9*	*8.8*	*8.7*	0 k	*8.7*	***8.9***
Total Ounces	0 k	0 k	0 k	0 k	0 k	3 k	3 k	2 k	2 k	3 k	3 k	2 k	6 k	5 k	6 k	6 k	27 k	24 k	29 k	22 k	**121 k**
Grams	*0 k*	*0 k*	*0 k*	*0 k*	*0 k*	*83 k*	*102 k*	*53 k*	*64 k*	*81 k*	*82 k*	*67 k*	*186 k*	*171 k*	*187 k*	*181 k*	*837 k*	*755 k*	*906 k*	*698 k*	***3,755 k***
Development Waste Tons	14 k	10 k	8 k	9 k	3 k	7 k	7 k	9 k	9 k	9 k	9 k	9 k	27 k	27 k	27 k	27 k	50 k	0 k		0 k	**260 k**

Refer to Appendix 6 for more details.

3.7 Equipment

Capital and operating costs for the main pieces of underground mining equipment are given in Section 4.

3.7.1 Power Source

All mobile equipment would be electric or diesel powered. Diesel equipment must be equipped with exhaust scrubbers to ensure that diesel particulate matter is kept below the regulated standard of 400 mg/m^3 (400 micrograms per cubic metre).

Battery-powered LHDs and trucks were selected for this mine design. Working heavily, the batteries last approximately four hours and can be swapped in fifteen minutes. They take two hours to fully charge. This equipment is currently in use at Kirkland Lake Gold's Macassa Mine in Ontario, Canada.

Advantages of battery power over diesel include greatly improved working conditions (much quieter, no emissions), lower energy cost, lower maintenance cost, and reduced ventilation requirements. The main disadvantage is the higher capital cost. In the next feasibility stage, a detailed comparison between the capital and operating "life of equipment" costs of battery- and diesel-powered equipment should be made.

3.7.2 Cycle Times

The cycle time for a truck includes:

- The time it takes the scooptram to load the truck;
- The travel time from the loading point to the dumping point, using average uphill and downhill speeds;
- The time to maneuver and dump the load;
- The time to return to the loading point;
- Waiting time; and,
- Maneuver to load time.

The truck cycle time can be used to ensure the fleet is adequate for the job. A third truck is required after the 5862 level, and 4 trucks are required below the 5500 Level.

Table 3-16 Loading Time for a 3 yard scoop matched with a 20 ton truck.

LHD Loading Truck	**m^3 bucket**	**t/bucket**	**Qty**	**seconds**	**minutes**
Loading bucket size 3 yd (2.25 m^3)	2.25	3.6		20	
Tramming loaded	15	1.8		8	
Manoeuvre and dump				30	
Delays				60	
return	50	2		25	
Cycle time				118	
buckets to load truck			6		
Total time to load truck				657	
Efficiency factor	90%			730	12

Table 3-17: Total truck cycle time from the 5687 foot level.

Truck Haulage to Surface	**To Surface From 5687 Foot Level**	**Haulage Route**	**Distance (m)**	**Speed (m/min)**	**time, minutes**
scoop bucket size, yd	3	Loading Point			
Loading truck					12
metres from Level		Travel distance for truck (m) 1800		83	
		Travel distance for truck (Ft) is 5908 feet			
truck tonnes	20	to Passing Pont	900	83	10.8
m/min ave truck speed	83	at Passing Point			0.0
		to Dumping Point	900	83	10.8
metres travel, one way	1,800	at Dumping Point			2.0
Return time, face-dump-face.		Return			
min return trip travel time	58.4	to Passing Pont	900	83	10.8
dump time		at Passing Point			1
load time		to Loading Point	900	83	10.8
wait time					
min total return trip time	58				
time at 90% efficiency	**65**		3,600		58.4
Available minutes/day	980				
tonnes moved/day	600				
Truck loads possible/day	15				
Possible tonnes/day/truck	300				
Minimum num. of trucks	2	The minimum number of trucks is 2 plus one spare = 3 trucks			

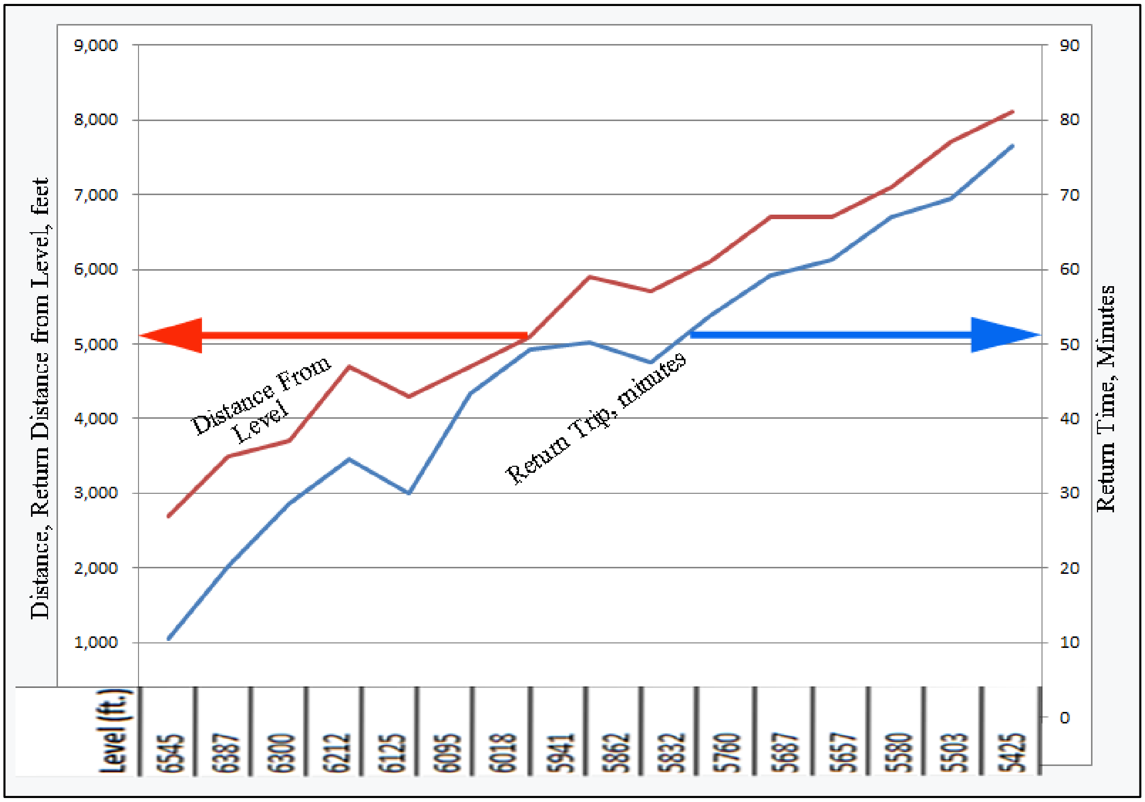


Figure 3-7: Truck cycle times for the different levels.

3.7.3 Drilling Equipment

The main decline and haulage ways would be drilled using a 1 boom electric-hydraulic jumbo. It is presumed that the sills would be drilled using a similar, single boom drill jumbo. However, jack leg drills or "Long Tom" rigs would also be suitable.

3.7.4 Mucking & Loading Equipment

Mucking will be done with battery powered 3 yd^3 LHDs that would load battery powered 18 tonne (20 ton) trucks. Smaller, 1.5 yd^3 LHDs could also be used in narrower openings.

3.7.5 Hauling

Two 18 tonne (20 ton) trucks will haul waste and mineralized rock from the mine. Approximately 544 tonnes (600 tons) will be moved per day, about 272 tonnes (300 tons) per shift.

3.8 Ventilation

Ventilation requirements have been estimated based on providing 100-125 CFM of fresh air per rated horsepower of diesel powered equipment underground plus 100 CFM per person underground. An anticipated utilization rate was incorporated for each piece of mobile equipment.

During the initial development phase, while developing the decline and the ventilation/escape raise, fresh air would be delivered to the working face using a fan and flexible ducting. After the ventilation raise breaks through to surface, a fan arrangement would be installed over the raise in either an intake or exhaust setup. Approximately 150-200 fan horsepower would be required during the production phase.

Preliminary calculations for the initial development phase were carried out. If only diesel equipment is used and appropriate utilization factors are used, the diesel power would total 300-350 HP. For that power, approximately 40,000 cfm would be an appropriate diluting airflow. Using 42-48 inch ducting to supply the air, the total head would be 6-11 inches of water at that elevation. A 100 HP fan would be in the right family of fans for that operating point.

Depending on the equipment that is eventually selected to excavate the decline, it may not be wide enough to accommodate such large ducting. If larger equipment is used, it was determined that the same air flowrate could be accomplished using smaller ducting, a smaller fan at surface, and 1-2 booster fans, at intervals along the ducting, with a similar total power requirement in the neighbourhood of 100 HP.

In reality, if the LHD and haul truck are battery powered as planned, only a small fraction of that airflow would actually be needed because the vast majority of this airflow is needed for diluting diesel engine exhaust.

If a versatile, variable blade pitch fan were used, one single fan, perhaps with a different motor, may be able to be service both the initial development and production phases.

During the production phase, fresh air is channeled to the bottom of the mine and the main levels using air locks, regulators and ventilation raises. Air is delivered to dead-end stopes and haulage drifts using flexible ducting. Small fans, between 10HP and 20HP, deliver air to the stopes.

Once the Dawson deposit has been opened up and prior to installing the main fan, a complete ventilation study should be completed using the data gleaned from the decline and level development.

3.9 Mine Services

3.9.1 Electrical

The main electrical trunk would run down the portal initially, then down the ventilation raise or a dedicated drill hole. A substation will be installed at each level. Electricity will be used to supply power to pumps, fans, electric jumbos, and refuge stations, the jumbo drill and possibly other major mobile equipment. Electrical equipment has considerable advantages over diesel power, including longer machine life, less noise and most importantly, a cleaner underground atmosphere. It may be possible to also use electric-hydraulic jackleg drills.

Electrical substations would be located underground, with one substation serving each main level. After stopes are completed, it would be possible to relocate electrical equipment for reuse in new areas.

3.9.2 Compressed Air

Compressed air would be delivered to the underground using a main line of 100 mm (4 in) steel pipe. The pipe will initially run down the ramp and then down the raise system once the raises are installed.

Branches from the main line would provide compressed air to the stopes by 100 mm (4 in) steel pipe, and 50 mm (2 in) steel and PVC pipes would deliver compressed air to the mining face. Equipment and activities that require compressed air includes:

- Jacklegs and stopers;
- Cleaning blastholes;
- Loading explosives; and,
- Refuge station pressurization.

Small pneumatic pumps would be used to keep the decline face free of water.

3.9.3 Water Supply

Water would be used underground for drilling operations, dust suppression during mucking, and washing walls and backs for scaling and sampling purposes. Production water would be provided by water pumped from the polishing pond, and gravity fed to the mine through the ventilation raise using a 50 mm (2 in) steel pipe. Connections at each level would provide production water to each stope using 50 mm steel or PVC pipes delivering water to the active face.

3.9.4 Water Discharge

Water that accumulates at the active face would be pumped away using a small pneumatic diaphragm pump (Wilden type pump) or a small electric pump, using 50 mm (2 in) steel or PVC pipe from the face to the sump.

Dirty water sumps should be connected by a system of overflow drain holes with the cleaner water being pumped in stages, to the surface for clarification and reuse. The sump system should be designed so that the slimes can be cleaned out periodically.

A permanent pumping station would be constructed at the bottom of the mine that would pump water to the surface settling pond system. The pumping arrangement would be set up as a redundant parallel system, with either side capable of providing mine dewatering without the other.

3.9.5 Communications

A leaky feeder-type radio communication system is planned to be installed along with a wired phone system where necessary.

3.10 Maintenance

An experienced maintenance planner would run the department. Maintenance personnel would consist of mobile mechanics, industrial mechanics, electricians, and a drill doctor. The maintenance facility would consist of a maintenance shop and warehouse storage facilities.

3.10.1 Mobile Maintenance

A maintenance shop would be constructed on surface, near to the location of the underground portal. The shop would have 2 bays, with space for laydown in between bays. Mobile equipment would be brought from underground to surface for servicing, preventative maintenance, and repairs. The mobile maintenance team would have access to a mine utility vehicle, which would be used to access and service equipment underground as needed.

Refueling and lubrication of vehicles will be done on surface.

3.10.2 Ramp Roadbed

The maintenance of the main decline roadbed is important for mine tire life and haulage efficiency. A good ditch to channel water is a must. Occasional grading of the road is required.

3.11 Personnel

Hiring highly skilled hard rock miners with mining experience would be a highly recommended, however some positions could be filled using less experienced miners. Lead hands, jumbo operators, scoop operators, production miners, and bolters would need to be skilled miners, while truck drivers and nippers could be less experienced.

3.11.1 Training

The applicable mining regulations (Coded of Federal Regulations, Title 30, Mineral Resources, Parts 1 to 199, revised as of July 1, 2014) require training for all new miners, and refresher training for all miners every year. At least two mine rescue teams are required to be available when persons are underground at the mine.

3.11.2 Shift Schedule

Initially, it is planned to have four mining development crews to cover two shifts per day on a 7 days on, 7 days off schedule. For production mining, it may be possible to work on a two-shift per day, five days per week schedule and have the mill working 7 days/week with one shift for maintenance.

There will be a gradual buildup of the mine employees.

3.11.3 Engineering, Geology and Surveying

Engineers would provide all plans for mining, using geological advice and guidance. A senior mine engineer would direct the technical services department. A junior engineer would also be hired, along with engineering students for work terms. Surveying would be done by a dedicated surveyor, with 2 employees splitting duties on a rotation. Geologists would maintain grade control and outline the valuable mineralization to keep dilution to a minimum.

3.11.4 Safety and Environment

It is anticipated that a staff person will be dedicated to training, safety and environmental compliance. Two mine rescue teams are required, training and safety must be documented and there are numerous records to be filled out and retained on site or submitted to regulators. It is a full time position.

4 Capital and Operating Costs

4.1.1 Initial Capital Costs

The total capital cost for the underground, excluding working capital, is $12.3 million (Table 4-1-Table 4-2). Working Capital is excluded from Table 4-1. Working capital is usually 3 months operating costs. In the case of Dawson, working capital would be approximately $2 million ($80 per ton, 105,000 tons per year).

A contractor would excavate the decline and stubs for levels for approximately $5 million ($1,800/ft).

Table 4-1: Summary of initial capital costs.

Item	Cost
Mining Equipment	$6.07 M
Underground Development	$5.69 M
Windy Gulch Mining	$0.53 M
Total* (Rounded)	**$12.3 M**

*Excludes working capital.

Table 4-2: Details of underground capital requirements.

Dawson Project, Estimate of Underground Capital Requirements, May 2015, US$, Major Mining Equipment								
	Number of Item(s)	**Motor/type**	**kW**	**Unit Delivery**	**Year 1**	**YEAR 2**	**YEAR 3**	**Total item cost (rounded)**
Drilling								
Jumbo Drill, micro	2	d/e	40	$25,000	$585,000	$585,000		$1,170,000
Bar and Arm Drill	1			$5,000	$65,000			$65,000
Bazooka Drill	1			$1,000	$13,900			$13,900
Bit Sharpener	1			$2,000	$21,500			$21,500
Stopers	10			$750	$75,600			$75,600
Compressor	1	e	100	$10,000	$130,000			$130,000
Booster Compressor	1	e	60	$5,000	$50,000			$50,000
Blasting								
ANFO Loader	1	a		$1,000	$11,000			$11,000
Mucking								
3.5 yd LHD	1			$30,000	$324,000			$324,000
2 yd, remote control	1			$35,000	$275,000			$275,000
Scoop Tires	24			$200	$40,800			$40,800
Bobcat & spare parts	1			$2,000	$17,000			$17,000
Haulage - UG								
20 ton truck	2			$10,000	$1,018,000		$514,000	$1,532,000
Truck Tires	12			$200	$16,400			$16,400
Vehicles								
Service Vehicle	1			$10,000	$90,000			$90,000
Superintendent Vehicle	1			$10,000	$70,000			$70,000
Geology/Survey vehicle	1			$10,000	$70,000			$70,000
Personnel Transport	1			$30,000	$245,000			$245,000
Ventilation								
Main Fan and Motor	1			$10,000	$85,000			$85,000
Spare main fan plus motor	1			$10,000	$70,000			$70,000
Auxiliar fan	5			$2,000	$84,500			$84,500
Vent monitoring equipment	1			$1,000	$8,500			$8,500
Drainage								
High Head 150 kW pump	1			$8,000	$63,000			$63,000
Small pumps 15 kW	5			$2,000	$62,000			$62,000
Anciliary Equipment								
ger hoist, air powered (Stope use)	6			$1,500	$69,000			$69,000
cap lamps	50			$10	$2,450			$2,450
transformers, 500 kva	3			$5,000	$104,900			$104,900
First Aid Station, fully supplied	3			$200	$3,530			$3,530
Refuge Station	1			$4,000	$34,000			$34,000
Miscellaneous Equipment								
Scissor Lift	1			$15,000	$315,000			$315,000
Bolter for Scissor Lift	1			$10,000	$95,000			$95,000
Boom Truck	1			$10,000	$300,000			$300,000
Ground Support								
Shotcrete, 30 m^3/hr capacity, diesel, trailer mounted	1			$10,000	$79,750			$79,750
Grout Pump for cable bolting, air powered	1			$3,000	$21,750			$21,750
Initial inventory of cable bolts, rock bolts, screen, ground	1				$50,000			$50,000
Electrical Substation	1			$4,000	$204,000			$204,000
Battery Chargers	1			$500	$30,500			$30,500
		Yearly capital purchases			$4,801,000	$585,000	$514,000	
		Miscellaneous	15%		$720,000	$88,000	$77,000	
		Contingency	10%		$552,000	$67,000	$59,000	
		Yearly Totals				$740,000	$650,000	
					$6,073,000	initial Capital		

4.1.2 Sustaining Capital

It is estimated that the sustaining capital will be $740,000 in Year 2 and $650,000 in Year 3. Thereafter, the sustaining capital would be between 10% and 20% of the initial capital.

4.1.3 Operating Costs

Operating costs were estimated at approximately $US 80 per ton of mill feed ($US 88 per tonne). This cost includes overhead but not capital. Underground capital is approximately $US 45 per recovered ton.

Refer to Table 4-3 for a summary and Appendix 3 for details of selected operating costs.

Costs were estimated on a monthly basis and summed over the life of the operation, then divided by the mill feed.

The total underground personnel cost is approximately $4.5 million per year or approximately $20 for every ton broken. A two 10-hour shift option gives 7,300 operating hours per year for an average mine labor cost of $600 per operating hour. In terms of 100,000 tons per year production, the labor component will be $45/ton (between $40/ton and $55/ton) delivered to the plant stockpile.

The costs per decline round were examined, using 3.7 m (12 ft) drill steel.

Table 4-3: Summary of selected operating costs.

Item	Operating Cost ($US)
Stoping (Incl diamond drilling, labour, development)	$80 per ton
Yearly Labour Cost at Full Production	$4.5 million
Single Boom Drill Jumbo (Drillmaster 100)	$17.70 per Broken Foot
Longhole Drill (Drillmaster 100 Longhole)	$65-66 per drilling hour
LHD (Muckmaster 300EB)	$57-58 per hour
Haul Truck (Haulmaster 800-20EB)	$31-32 per hour

5 Windy Gulch Pit Design

A relatively small open cut was designed for the Windy Gulch deposit, illustrated in Figure 5-1, Figure 5-2, and Figure 5-3.

The intent is to provide a small amount of gold mineralized material to “run-in” the mill and provide some initial cash flow.

The geometry of Windy Gulch’s mineralized zone models is “inferred” at this time due to slightly inaccurate collar survey elevations for four drill holes. Zephyr’s geologists report that, based on outcrop information, the mineralized zones are much more regular than the computer model suggests. Zephyr plans to correct the collar surveys, drill some additional holes with the aim to extend the strike length of this gold zone to the east, and re-model the deposit. Thus, the current open cut design is preliminary at this point in time.

A larger open cut is economically justifiable with the current block model. Because this is a small project, a contractor would likely be used (refer to Table 5-2).

The terrain is steep. Small, crawler-type drills would likely be used. Excavation would likely be carried out using a relatively small excavator [28 to 50 tonne (25 to 45 ton) range]. Articulated, six-wheel-drive haul trucks would be appropriate for the steep, rough terrain. Any roads that are needed would be kept outside the cut as much as possible.

Waste dilution was estimated as a 0.3 metre (1 ft) skin at zero grade. For an average width of 2.8 m (9 ft), the average dilution is 21%. Mining losses of 10% were assumed. Both of those factors assume selective and conscientious drilling, blasting, and loading.

The in-pit mineral resource is quite insensitive to cut-off grade. Therefore, all modelled mineralized rock within the proposed pit would be sent to the processing plant. This corresponds to the “0 g/tonne cut-off grade” from the “diluted and recovered” part of Table 5-1, which amounts to 13,400 tonnes (14,700 tons) of diluted, recovered rock at an average grade of 9.2 g/tonne, containing nearly 137 kg (4,000 ounces) of gold. The stripping ratio is 2.8:1 ($tons_{waste}$:$tons_{mill\ feed}$). A 45° pit slope angle was used.

The small cut at Windy Gulch represents nearly fifty days of milling. This would be mined while Dawson underground development work is underway and stockpiled at the mill. The mill would start to process this stockpile a few weeks before Dawson production begins. A stockpile of roughly 4500 to 9000 tonnes (5,000 to 10,000 tons) would always be maintained. This would be enough to feed the mill for between three and six weeks at 272 tonnes (300 tons) per day.

Further mining at Windy Gulch would likely be underground.

Table 5-1: Windy Gulch pit mineral resources.

Windy Gulch Preliminary Pit Resources (Non-Diluted)

Cut-off Grade (g/tonne)	Short Tons	Grade (g/tonne)	Ounces	Waste Tons	Stripping Ratio (Tons:Tons)
5	13.1k	11.4	4,360	42.4k	3.2:1
4	13.5k	11.2	4,400	42.1k	3.1:1
3	13.5k	11.2	4,400	42.1k	3.1:1
2	13.5k	11.2	4,400	42.1k	3.1:1
1	13.5k	11.2	4,400	42.1k	3.1:1
0	13.5k	11.2	4,400	42.1k	3.1:1

Windy Gulch Preliminary Pit Resources (Diluted and Recovered)

Cut-off Grade (g/tonne)	Short Tons	Grade (g/tonne)	Ounces	Waste Tons	Stripping Ratio (Tons:Tons)
5	14.4k	9.4	3,920	41.2k	2.9:1
4	14.7k	9.2	3,960	40.8k	2.8:1
3	14.7k	9.2	3,960	40.8k	2.8:1
2	14.7k	9.2	3,960	40.8k	2.8:1
1	14.7k	9.2	3,960	40.8k	2.8:1
0	14.7k	9.2	3,960	40.8k	2.8:1

Notes:

1. Skin dilution of 0.3 m (1 ft) waste at zero grade. Average width 2.8 m (9 ft). Average dilution 21%.

2. Mining losses = 10%.

Figure 5-1: Proposed Windy Gulch open cut, facing northeast.

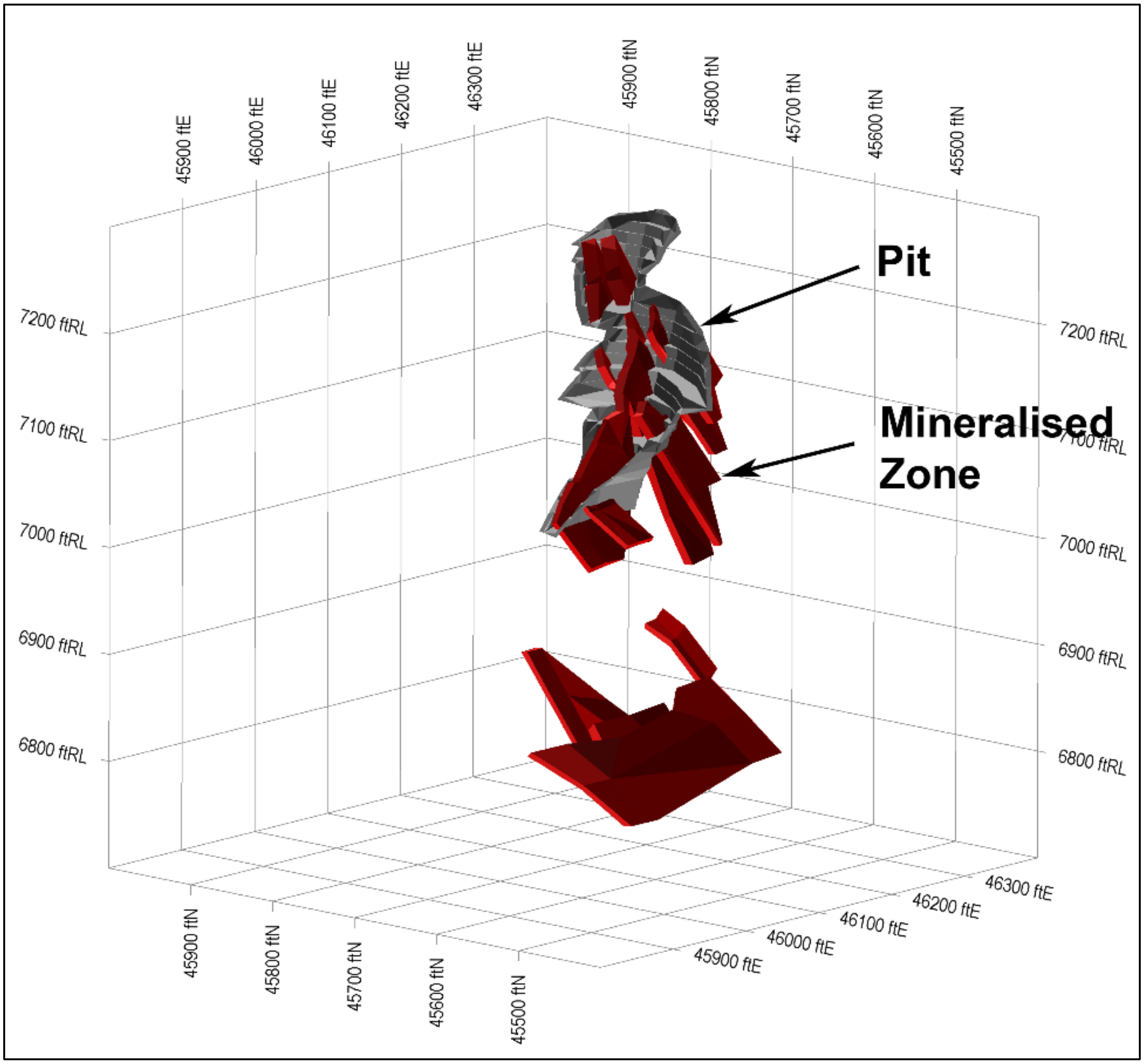
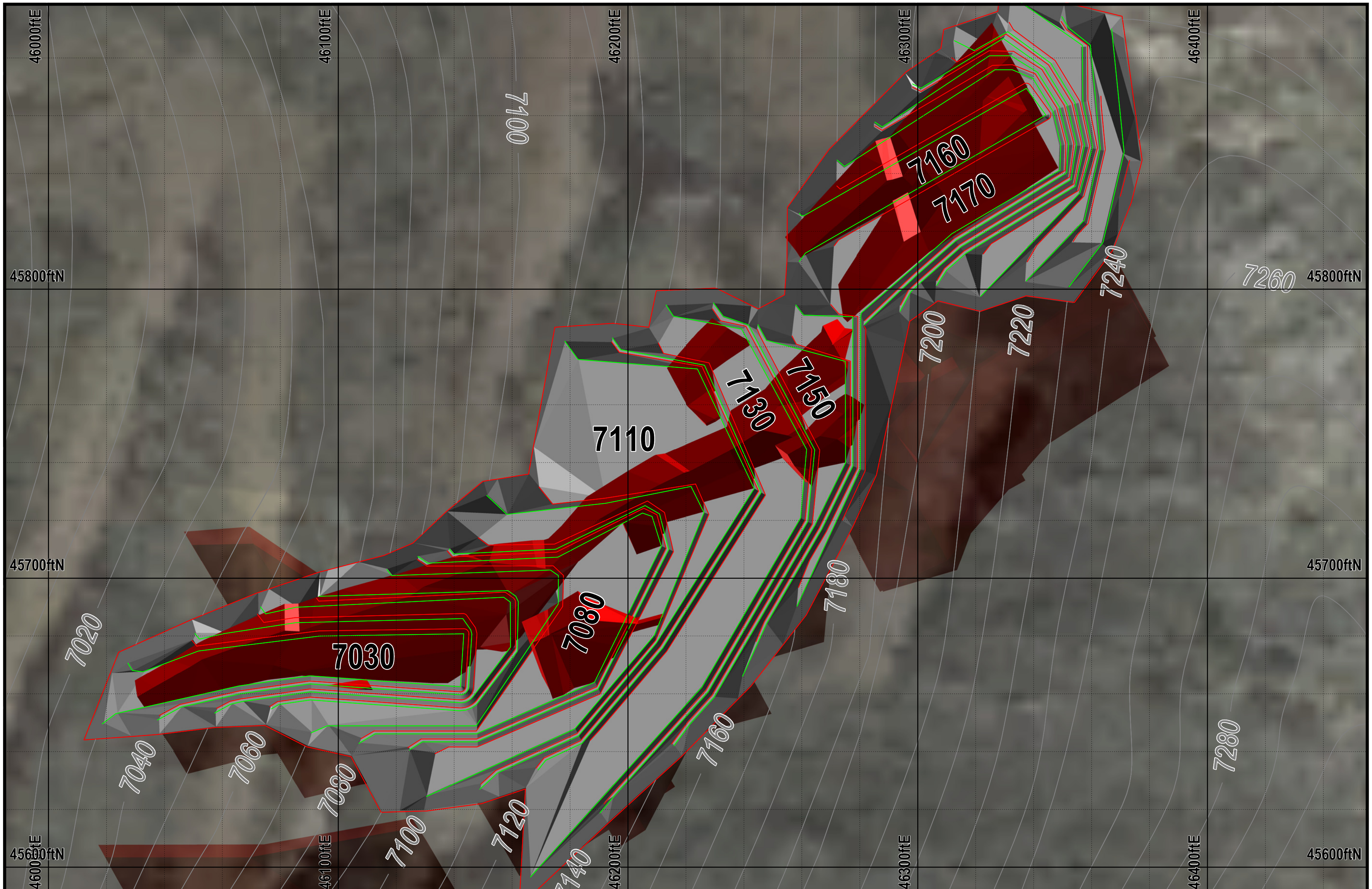


Figure 5-2: Proposed Windy Gulch open cut, facing northeast (showing deeper mineralization).



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Legend	
	Contour
	Crest
	Toe

Scale	Plot Date	Sheet
1 in = 50 ft	28-May-2015	1 of 1
	Plot File: Windy Gulch Pit	

20 0 20 40ft

Plan View of Propsed Windy Gulch Pit

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado

Figure 5-3

Table 5-2: Windy Gulch Contract Mining Costs

Estimate of Contractor Mining Cost				**Days**	**Total $**	**$/ton mineralized rock**	**$/ton broken**	**$/day**
Short tons mineralized rock	$14,700		Mobilization / demobilization		$50,000	$3.40	$0.90	
Short tons waste	$40,800							
Total tons	$55,500		Project Management	60	$60,000	$4.08	$1.08	
			Drilling & Blasting	12	$69,375	$4.72	$1.25	$1,000
Equipment		Fuel truck	Excavation	20	$138,750	$9.44	$2.50	$5,781
		Mechanics truck	Trucking & Stockpiling	40	$111,000	$7.55	$2.00	$6,938
		Excavator	Crushing & Screening	20	$102,900	$7.00	$1.85	$1,000
		Truck						
		Front end loader		Totals	$532,025	$36.19		
		Drill		Cost/ton broken			$9.59	
		Compressor						

6 Conclusions

An underground mine design was developed for the main Dawson Segment on the Dawson Property. A decline would access the mineralized zones, which would be mined using mechanized longhole sublevel stoping. The mining recoverable resource (diluted), all of which is in the Inferred category, is 407,000 tonnes (449,000 tons) at 8.9 g/tonne (0.26 oz/ton), representing 3,630,000 grams (117,000 ounces) delivered to the mill.

The mine design incorporates nearly all of the defined mineral resources. The targeted milling rate averages 272 tonnes (300 tons) per day, 365 days per year. During operations, the mill would operate at a slightly higher rate of approximately 310 tonnes (340 tons) per day to account for downtime. The mine would operate five days a week, averaging 363-386 tonnes (400-425 tons) of mill feed each day.

A shallow open cut was designed for Windy Gulch. This would be contractor-mined while Dawson is being developed. This Inferred resource amounts to 13,400 tonnes (14,800 tons) of diluted, recovered rock at an average grade of 9.2 g/tonne (0.27 oz/ton), containing nearly 123,000 grams (4,000 ounces) of gold. The stripping ratio is 2.8:1 ($tons_{waste}$:$tons_{mill\ feed}$). Further mining at Windy Gulch would likely be underground. The current resource model for Windy Gulch is subject to change pending corrections to four surface diamond drill hole collar elevations.

The combined mill feed (diluted and recoverable) from the Dawson and Windy Gulch Segments would be 420,000 tonnes with a grade of 8.9 g/tonne, for 121,000 ounces.

Initial capital costs for the underground are estimated to be $12.3 million (not including working capital). Mining at Windy Gulch, which would be carried out during the capital construction period, is included in that figure at $0.53 million.

Underground mine operating costs are estimated to be $US 80 per ton of mill feed ($US 88 per tonne). This cost includes overhead but not capital.

7 Recommendations

Based on the results of this report, the following program is recommended for the Dawson and Windy Gulch deposits:

1. Further geotechnical work, for the portions of the portal that are close to surface. Geotechnical drilling is also recommended for the ventilation raise area, the proposed mill area, and the proposed tailings facility;
2. Preliminary mill design work and tailings dam design;
3. Completion of mine permitting requirements;
4. Diamond drilling at the Dawson Segment and Windy Gulch ; and,
5. A Preliminary Economic Analysis. This would include more detailed engineering work. Supplier quotes would be obtained for significant items.

Table 7-1: Budget recommendations.

Item	Budgetary Cost ($USD)
Geotechnical Work (500 metres @ $100 per metre, plus supervision and analysis)	$70 k
Preliminary Mill Design and Tailings Design	$150 k
Mine Permitting Work	$150 k
Diamond Drilling	$200 k
Preliminary Economic Analysis	$105 k
Total	**$675 k**

Prepared by:

"original signed and sealed"

Patrick Hannon, M.A.Sc., P.Eng.

Mining and Geological Engineer

October 7, 2015

"original signed and sealed"

W Douglas Roy, M.A.Sc., P.Eng.

Mining Engineer

October 7, 2015

8 References

P.H. Ferreria, “Improved Technologies in Longhole Blast Hole Drilling, Applied to Drop Raising and Longhole Stoping as well as the Application of a Small Twin Boom Mechanized Drillrig”, *The Journal of the South African Institute of Mining and Metallurgy,* May 2003, pp 233-240.

A. Hilchey and I. Wolfson, Resource Estimate Technical Report for the Dawson Property, Fremont County, Colorado, USA, for Zephyr Minerals Ltd. (Halifax, NS: Mercator Geological Services, 2013)

P.W. MacMillan and B.A. Ferguson, “Principles of Stope Planning and Layout for Ground Control” in Underground Mining Methods, ed. W. Hustrulid and R. Bullock (Englewood, CO, USA: SME, 2001), 526 – 530

H.S. Mitri, R. Hughes, and E. Lecompte, “Factors Influencing Unplanned Ore Dilution in Narrow Vein Longitudinal Mining” (paper presented at the annual meeting of SME, Phoenix, AZ, February 28 – March 3, 2010), Preprint 10-096

I. Wolfson, Technical Report on the Dawson Property, Colorado, United States of America, for Celtic Minerals Ltd. (Sept. 27, 2011)

Code of Federal Regulations, Title 30, Mineral Resources, Parts 1 to 199, July 1, 2014

Final Engineering Report regarding the Review of a Prefeasibility Study for Uranerz USA Inc. (Dynatec, March 25, 1991)

Pre-Feasibility Study of the Dawson Project, Canon City, Colorado, for Uranerz USA Inc. (American Mine Services, February 15, 1991)

Natural Resources Canada, Canmet-Mineral and Mining Sciences Laboratories,2000.

CERTIFICATE OF QUALIFIED PERSON

Patrick James Francis Hannon, M.A.Sc., P.Eng. MineTech International Limited 1161 Hollis Street, Suite 211, Halifax, NS, Canada, B3H 3P3

I, Patrick J.F. Hannon., as a co-author of this report completed for Zephyr Minerals Ltd. and entitled “Mine Design for the Dawson Property located in Colorado, USA, 38°23’ N, 105°18’ W” dated August 26, 2015, co-authored by by Patrick Hannon, M.A.Sc., P.Eng. and Doug Roy, M.A.Sc., P.Eng., do hereby certify that:

1. I am a practising mining and geological engineer with MineTech International Limited of Halifax, Nova Scotia Canada.
2. I am a graduate of the Technical University of Nova Scotia (M.A.Sc., Mining Engineering, 1987), Queen's University at Kingston (B.Sc. (Eng.) 1972) , Geological Engineering) and the Haileybury School of Mines (Senior Mining Technician, 1968).
3. I am a Professional Engineer (Mining and Geological) registered in the Province of Nova Scotia, Ontario, Newfoundland and Labrador and the North West Territories, a fellow of the Canadian Institute of Mining and Metallurgy and the Society of Mining Engineers, AIME. I have worked as an engineer since my graduation from Queen’s. My relevant experience for the purpose of the report is:
   - Between 1972 and 1983 I was employed with Imperial Oil Limited as senior geologist, chief mine geologist, and mine superintendent.
   - Between 1983 and 1987, I was employed by the Nova Scotia Department of Mines and Energy as Manager of Mining Engineering.
   - Between October 1987 and May of 1989, I was employed as Chief Mining Engineer for the consulting firm A.C.A. Howe International Limited.
   - Since May, 1989 I have been President of MineTech International Limited. During this time I have had mining assignments in various parts of the world. These include being mine manager at an open pit mine in Newfoundland, technical consultant on mine safety regulations for Malaysia and Nova Scotia, team leader for various mine feasibility studies, VP Exploration for Claude Resources Inc. and President of Scozinc Inc.
4. I have read the definition of "qualified person" set out in National Instrument 43 101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43 101.
5. I am responsible for selection of the base case mining method and the estimation of the mining consumables, labour cost and underground mine infrastructure.
6. I have not visited the Dawson property, the subject of this report.
7. I am independent of the Issuer applying the test set out in Section 1.4 of National Instrument 43101.
8. I have had no prior involvement with the property that is the subject of the Report.
9. I have read National Instrument 43-101F1, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
10. To the best of my knowledge, information, and belief, the Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

“original signed and sealed”

____

Patrick J.F. Hannon, M.A.Sc., P.Eng.

October 7, 2015

CERTIFICATE OF QUALIFIED PERSON

William Douglas Roy, M.A.Sc., P.Eng. MineTech International Limited 1161 Hollis Street, Suite 211, Halifax, NS, Canada, B3H 3P3

I, Douglas Roy, as a co-author of this report completed for Zephyr Minerals Ltd. and entitled "Mine Design for the Dawson Property located in Colorado, USA, 38°23' N, 105°18' W" dated August 26, 2015, co-authored by by Patrick Hannon, M.A.Sc., P.Eng. and Doug Roy, M.A.Sc., P.Eng., do hereby certify that:

1. I am a practising mining engineer with MineTech International Limited of Halifax, Nova Scotia Canada.
2. I graduated with a Bachelor of Engineering ("B.Eng.") degree in Mining Engineering from the Technical University of Nova Scotia (now Dalhousie University) in 1997 and with a Master of Applied Science ("M.A.Sc.") degree in Mining Engineering from Dalhousie University in 2000.
3. I am a Professional Mining Engineer registered with the Association of Professional Engineers of Nova Scotia (Registered Professional Engineer, No. 7472). I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), the Prospectors and Developers Association of Canada ("PDAC"), and the Society of Mining, Metallurgy, and Exploration ("SME" - USA).
4. I have worked as a mining engineer for more than fifteen years since graduating from university. This work has included the estimation of mineral resources and mineral reserves for precious metals, base metals and industrial minerals, surface and underground mine design, and mine feasibility studies.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible the mine design (Dawson and Windy Gulch) sections of this report.
7. I have not visited the Dawson property, the subject of this report.
8. I am independent of the Issuer applying the test set out in Section 1.4 of National Instrument 43-101.
9. I have had no prior involvement with the property that is the subject of the Report.
10. I have read National Instrument 43-101F1, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
11. To the best of my knowledge, information, and belief, the Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"original signed and sealed"

William Douglas Roy, M.A.Sc., P.Eng.

Mining Engineer

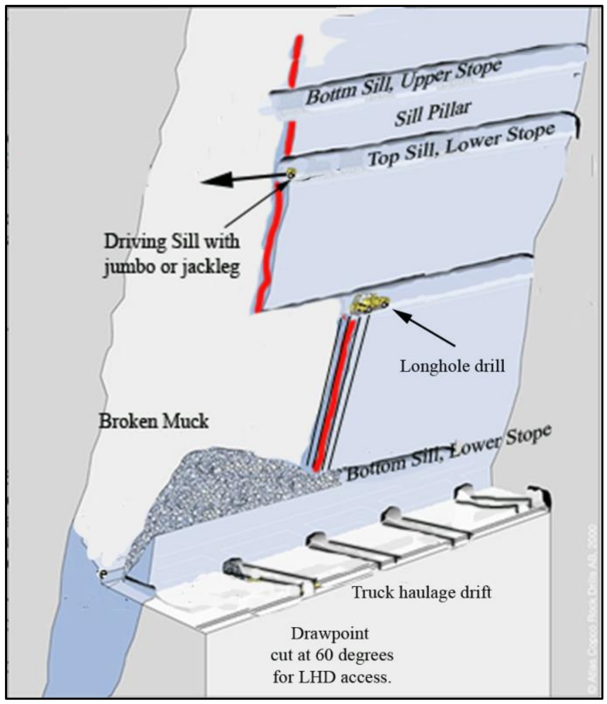
October 7, 2015

Appendix 1: Potential Mining Methods

Longhole Sublevel Stoping

If the plane of the material to be mined is regular in shape and is at least 1 m (3 ft) thick, stoping can be done with longhole drills from horizontal sill drifts cut through the deposit every 15 to 18 m (50 to 60 ft) vertically. This is the recommended method.

Longhole sublevel stoping is desireable because (a) stope development is in potential mill feed, and (b) the stope is not meant to be occupied as it would be in the other methods described in this section.



Schematic illustrating the longhole mining method.

A bottom sill is driven horizontally along the mineralization. Once the bottom sill is in, a haulage drift leading to the ramp is established about 9 m (30 ft) in the footwall. Drawpoints are excavated at an angle of about 60° to the haulage drift to connect the bottom sill and the haulage drift. A drill sill is established above the bottom sill. Once the drill sill has been excavated, a drop raise or slot is drilled and blasted at the extreme end of the drill sill. This creates a space into which the mineralized rock can be blasted. Approximately 40% of the blasted material is removed as mining proceeds to account for swelling.[3] The remaining mineralized rock is left in the stope until mining is complete, at which time all the remaining material is pulled from the stope and the stope may be backfilled.

[3] IE, the same mass of blasted rock occupies 40% more volume than *in situ* rock.

Drilling and blasting are carried out from a drill sill drift above the bottom sill of the stope. At the same time another sill drift is being advanced, by jumbo or by using jacklegs, on the sill above. Once the mineralization between the bottom sill and the one above has been mined, the drill moves to the drill sill above and again retreats from the far end of the stope back towards the sill access cross cut. Sills are kept about 20 m (65 ft) apart vertically, depending upon the width and continuity of the structure. Narrower, less continuous mineralization would require sills to be closer together.

Broken muck is excavated using an LHD, loaded into trucks and transported to the surface for processing.

It is difficult to fully mechanize narrow vein mining, especially when the mineralized horizons pinch and swell along strike and down dip. A longhole drill can drill a reasonably straight hole for a distance up to 30 m (100 ft), depending on the ground conditions and the machine. With this type of drill and with good blast design, a 1 m (3 ft) wide steeply dipping vein or mineralized horizon can be excavated with minimal dilution. At Dawson, the mineralization is 1 to 4 m (3 ft to 13 ft) true thickness. Dilution will vary from 10% to 30%, depending upon the thickness of the zone being mined.

"Alimak" Mining

Another applicable mining method uses Alimak-style raise climbing equipment. In the case of Alimak mining of a dipping structure, the mining approach is from the hanging wall side and a series of finger raises connect the bottom sill and the footwall haulage drift.

Alimak mining is becoming a popular method of mining narrow, steeply dipping veins and might be applicable for the Dawson deposit. An Alimak raise climbing unit is comprised of a light weight longhole drill which sits on the platform. A unit for loading blast holes and for stripping the Alimak track is suspended below the working deck.

For this method, an Alimak nest is excavated on the hanging wall side of the deposit. The raise is drilled from the Alimak deck, blasted and mucked out. Each round advances the raise about 8 feet. The Alimak mining sequence is as follows:

- Once the raise has been blasted the raise is scaled and rock bolts are installed on all four sides;
- The raise is driven to break through in the top sill;
- Screen is installed on all 4 walls of the raise and from top to bottom;
- From the Alimak deck, four cable bolts in a radial pattern are grouted into the hanging wall every 7.62 m (25 ft) up the raise;
- After all cable bolts have been installed, the blast holes are drilled along strike in mineralization. All blast holes, from the bottom of the raise to within 15.24 m (50 ft) of the top of the raise, are drilled. These holes are drilled outwards on both sides of the raise, about 12.20 m (40 ft) from the raise and down about 5° from the horizontal;
- Once all horizontal holes have been drilled, a stripping deck is installed below the Alimak;
- The crew loads the holes to be blasted and then strip the rail from that portion to be blasted;
- The loaded holes are blasted;
- Some of the muck is removed; most of the muck stays in the stope to support stope walls;
- The load, strip, and blast sequence is continued until the horizontal holes have all been blasted, and then the Alimak raise climber is removed from the stope;

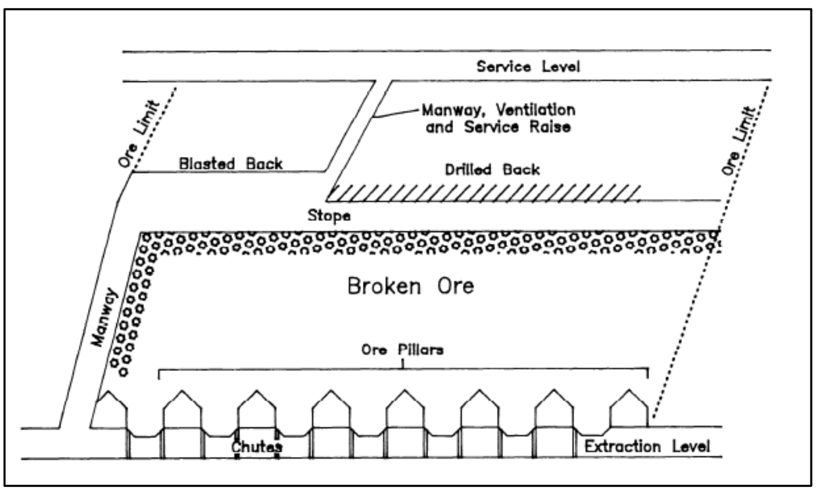
- The top 15.24 m (50 ft) of the stope are drilled off with vertical holes;
- Once the top holes have been blasted, the remaining muck in the stope is quickly and steadily removed to minimize dilution; and,
- Once all the muck has been removed, the stope is backfilled with development rock.

Alimak stoping requires less development than the other methods examined and should remain an option if strike and dip continuity is consistent over 24 m (80 ft) or more.

Shrinkage Stoping

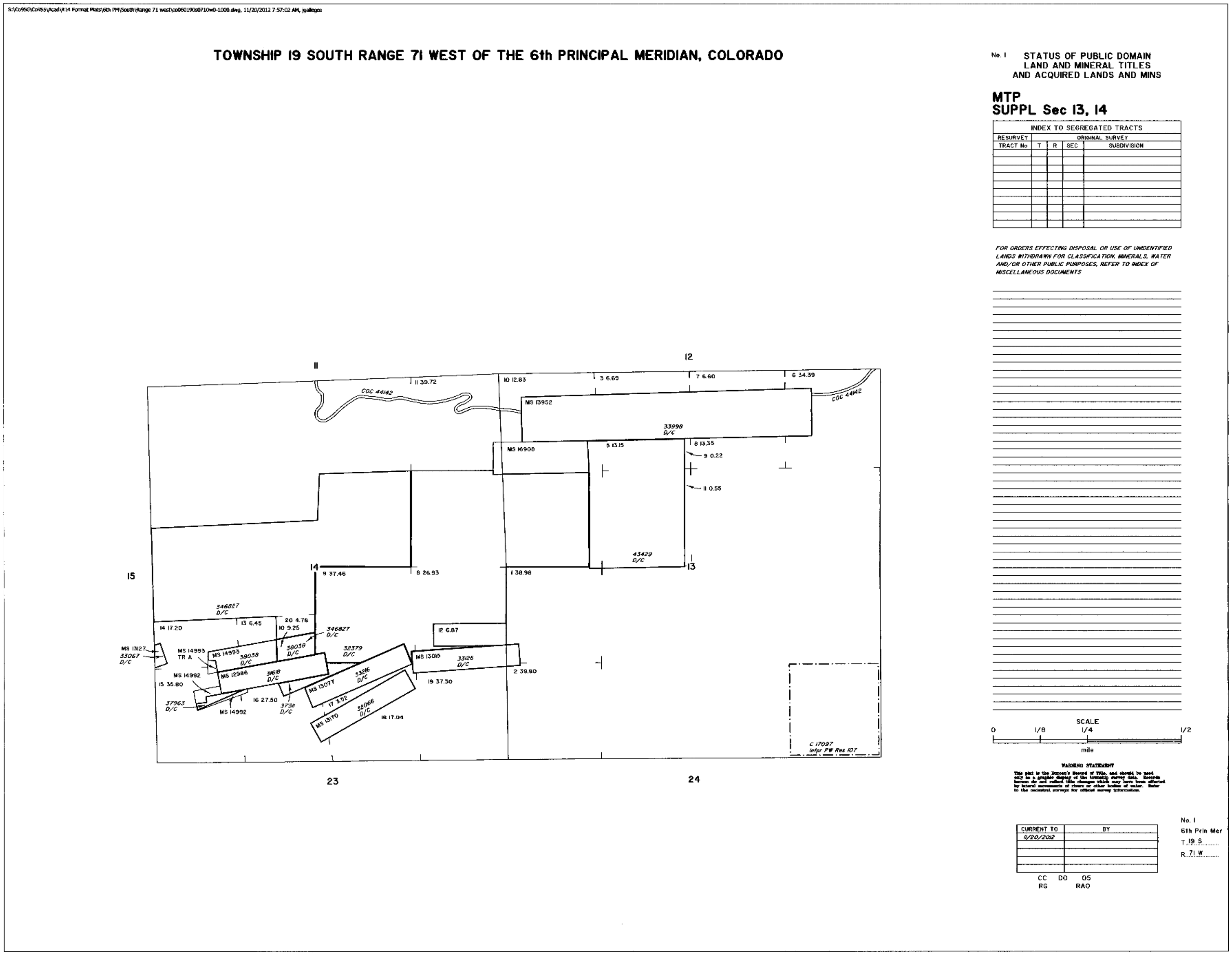
Shrinkage stoping is a vertical, overhand mining method whereby most of the broken mineralized rock remains in the stope to form a working floor for the miners. Another reason for leaving the broken mineralized rock in the stope is to provide additional wall support until the stope is completed and ready for drawdown. Stopes are mined upward in horizontal slices. Normally, about 35% of the mineralized rock derived from the stope cuts - the swell - can be drawn off ("shrunk") as mining progresses. As a consequence, no revenues can be obtained from the mineralized rock remaining in the stope until it is finally extracted ("pulled") and processed.

Shrinkage stoping is labour intensive and cannot be readily mechanized. It is usually applied to narrow veins deposits where other methods cannot be used or might be impractical or uneconomical. The method can readily be applied to zones of mineralized rock as narrow as 1.2 m (4 ft), but can also be successfully used in widths of mineralized rock up to 30 m (100 ft).



Shrinkage stoping schematic.

Appendix 2: Bureau of Land Management claims drawing (showing patented claims only), from 2012.



MINETECH INTERNATIONAL LIMITED
CONSULTANT ENGINEERS AND GEOLOGISTS
SINCE 1989

BLM-surveyed Points

1/MS 13077, Copper Boy Lode, a firmly set, marked, pink granite stone, Lat. 38-23-31.21N, Long. 105-17-48.47W.

1/MS 14993, Mike Sutton Lode, a firmly set, marked, pink granite stone, Lat. 38-23-29.74N, Long. 105-18-03.90W.

2/MS 14993, Mike Sutton Lode, a firmly set, marked but broke sandstone, Lat. 38-23-32.76N, Long. 105-18-03.90W.

3/MS 14993, Mike Sutton Lode, a firmly set , marked, granite stone, Lat. 38-23-30.04N, Long. 105-18-22.42W.

1/MS 13127, Windy Point Lode, a firmly set, marked, granite stone, Lat. 38-23-28.47N, Long. 105-18-29.16W.

A sandstone, marked 1\4, Lat. 38-23-41.83N, Long. 105-18-37.04W, this was found using a tie from 1/13127 Windy Point.

1/MS 16908, Great American Placer, a firmly set, marked, granite stone, Lat. 38-23-41.71N, Long. 105-16-57.91W.

3/MS 16908, Great American Placer, a firmly set, marked , broke granite stone, Lat. 38-23-54.67N, Long. 105-17-14.61W.

4/MS 16908, Great American Placer, a firmly set, marked stone, Lat. 38-23-54.57N, Long. 105-17-31.20W.

5/MS 16908, Great American Placer, a firmly set, marked granite stone, Lat. 38-23-58.87N, Long. 105-17-31.24W.

6/MS 16908, Great American Placer, a firmly set, marked limestone, Lat. 38-23-59.06N, Long. 105-16-58.08W.

1/MS 13952, Diamond Placer, a firmly set, marked granite stone, Lat. 38-23-59.18N, Long. 105-16-36.41W.

Appendix 3: Details of selected operating costs.

Life-of-mine longhole mining cost (direct costs).

Task	Life of mine cost	cost/recovered ton
Diamond Drilling	$900,000	$2.00
Mining Labour	$23,840,000	$53.80
Cross Cuts and Drawpoints	$1,783,000	$4.00
Haulage Levels	$1,353,000	$3.10
Raises	$235,000	$0.50
Decline	$1,538,000	$3.50
Remucks	$142,000	$0.30
Stope mining consumables	$2,164,400	$4.90
Sill mining	2336000	$5.30
Totals	**$29,791,000**	**$77.40**
260,000	tons waste	
93,000	sills	
350,000	stopes	
443,000	tons mined & hauled to pad	

Yearly labour costs at full production.

Mining Crew		Cost	25% Burden	20% Incentive	Total
Mine Super	1	$125,000	$31,250	$25,000	$181,250
Lead Hand	4	75000	$18,750	$15,000	$435,000
Miner 1	4	60000	$15,000	$12,000	$348,000
raise Miners	4	60000	$15,000	$12,000	$348,000
Truck Drivers	8	40000	$10,000	$8,000	$464,000
Nipper	4	40000	$10,000	$8,000	$232,000
Warehouse	2	50000	$12,500	$10,000	$145,000
Mechanic	4	60000	$15,000	$12,000	$348,000
Electrician	2	60000	$15,000	$12,000	$174,000
Labour	4	35000	$8,750	$7,000	$203,000
Bolting Crew	4	50000	$12,500	$10,000	$290,000
LHD operator	4	50000	$12,500	$10,000	$290,000
Miners in stope	12	60000	$15,000	$12,000	$1,044,000
Total	57				$4,502,250

Drill Jumbo (Drillmaster 100) Operating Costs

Item	Description	Cost Per Unit	Cost per 12ft Round
Bits	4 rounds per bit	$ 75.00	$ 18.75
Reamer Bits	20 rounds per bit	$ 275.00	$ 13.75
Steel	5 rounds per steel	$ 384.00	$ 76.80
Striking Bar	20 rounds per bar	$ 285.00	$ 14.25
Drilling Oil	4L/rnd	$ 3.50	$ 14.00
Hydraulic Oil	4L/rnd	$ 3.50	$ 14.00
Air Hose	10 rounds per hose	$ 72.00	$ 7.20
Water Hose	10 rounds per hose	$ 67.00	$ 6.70
Electricity	80 kW, 2.5 hrs per round	$ 0.08	$ 16.00
Diesel (Tramming)	5 L per round	$ 1.00	$ 5.00
Maintenance Supplies	$3.50 per hour, 2.5 hrs per round	$ 3.25	$ 8.13
Total Per 12 ft Round			$ 194.58
Total Per **Broken Foot** (11 Broken Feet Per Round)			$ 17.69

Longhole Drill Operating Costs (Drillmaster 100 Longhole)

Item	Description	Cost Per Unit	Cost per Drilling Hour
Bits	300 m/bit	75.00	18.00
Steel	1200 metres per Steel	384.00	16.00
Drilling Oil	1 L/hr	3.50	3.50
Hydraulic Oil	1 L/hr	3.50	3.50
Air Hose	10 rounds per hose	72.00	7.20
Water Hose	10 rounds per hose	67.00	6.70
Electricity	80 kW	0.08	6.40
Diesel (Tramming)	5 L per Shift	1.00	0.63
Maintenance Supplies		3.50	3.50
Total Cost per Drilling Hour			65.43
Penetration Rate (m/min)	*1.2 (50 m/hr)*		
Tonnes Per Metre Drilled	*2*		

Scoop Operating Costs (Muckmaster 300EB)

Item	Description	Cost Per Unit	Cost per Hour
Electricity	150 kW Average	0.08	12.00
Tires	1000 hr/set	20,000.00	20.00
Wear Plates	1000 hr/set	12,000.00	12.00
Lube & Fluids	0.5 L/hr	3.50	1.75
Hydraulic Oil	0.5 L/hr	3.50	1.75
Maintenance Supplies		10.00	10.00
Total Per Hour			57.50

Truck Operating Costs (Haulmaster 800-20EB)

Item	Description	Cost Per Unit	Cost per Hour
Electricity	150 kW Average	0.08	12.00
Tires	2000 hr/set	20,000.00	10.00
Lube & Fluids	0.5 L/hr	3.50	1.75
Hydraulic Oil	0.5 L/hr	3.50	1.75
Maintenance Supplies		6.00	6.00
Total Per Hour			31.50
Capital			

Appendix 4: Production shaft preliminary design.

As the mine gets deeper, a small production shaft should be considered. The capital cost may be justified if additional resources are found at depth. The system described below could hoist over 600 tons in 8 hours hoisting.

Capital & Operating Cost (InfoMine2012)
Mine Hoist

SPECIFICATIONS	Estimated costs include hoist mechanicals, electrical drive, and control systems. The table considers drum-type hoists only and assumes skips are in balance and hoisting vertically from one level. The table does not necessarily indicate the optimum hoist type or parameters for the production rate indicated. For further information contact: Ken Nelson, Siemag Inc., 2169 S. Chase Ave., Milwaukee, Wisconsin, 53207 Phone (414) 727-5725, Fax (414) 727-5710, mining@siemag-inc.com

DESCRIPTION										
Hoisting Distance Feet	Production Rate, tons/hr	Drum Dia. (in.	Velocity, ft./sec.	p Load (L	OR TYPE/	HP	oist Capital Co	nstallation Co	otal Capital Co	tal Reco
1000	150	60	20	6000	electric	250	$540,000	$587,000	$1,127,000	$28.19
1000	300	80	20	12000	electric	500	$700,000	$741,000	$1,441,000	$36.04

Hourly Operating Costs							
	Overhaul		Maintenance				Total
Overhead	Parts	Labour	Parts	Labour	Power	Lube	
$0.99	$4.97	$6.33	$9.24	$11.75	$10.46	$4.03	$46.78
$1.26	$6.36	$8.09	$11.81	$15.02	$20.93	$5.15	$67.36

Power Calculation for Single Drum Hoist					Minimum Factor of Safety for Wire Ropes, people		Length of Rope
Item	Number	Imperial	SI		Vertical Hoist	6	1000
Shaft Depth		1000	304.8		Slope Hoist	6	
Skip Live Load	1	6000	2724		Koepe Hoist	6.5	
Skip Dead Load	1	7200	3268.8		Production Only	5	
Skip dead-load/live-load ratio		1.2	1.2		Maximum static rope pull	16600	
Hoist Ropes	1	1.375	0.0254		MSRP* Factor of Safety	99600	
Sheave Diameter		9.17	3.14		Breaking strength of rope chosen	155200	
Drum Diametre standard rope	6.67				Factor of Safety	9.35	
Drum Diametre locked coil rope,	8.33				**tonnes/day**		**8-hr shiift**
Single Drum Face Width, single	90.59				300	TPH =	37.5
						Depth =	1000
						Rope Velocity at Depth	20
						Recommended Skip Load	6000
						Rope Size =	1.375
							flattened Strand
Hoisting Velocity, ft/s; m/s		20	6.1				
Hoisting Cycle (seconds)							
t_a	10					28800	seconds/8hrs
t_v	39.75					213	trips
t_r	8					638	tons/8hrs
t_d	10						
Hoist efficiency, n	0.9						
Counterweight, skip+1/2 load	1	10200					
Tail rope	1						

Hoisting Cycle				
Acceleration time	$t_a = V/a$			
Acceleration Distance	$h_a = 1/2\ (a*t_a)$^2			
	= retadation distance, hr			
constant velocity distance	$h_v=h_f-h_a-h_r$			
where h_t is the total hoisting distance, from loading pocket to headframe bin.				
constant velocity time	$t_v = h_v/V$			
cycle time =	$t_t = 2(t_a+t_v+t_r+t_d)$		136	136
per round trip, where t_d is the load or dump time				
Duty Cycle				
Rope Weight	$W_r=w_r(h_t+h_h)$		3400	1544
w_r is the rope weight and hh is the distance from Sheave to Loading Pocket				
Total weight of load	$W_l = W_r + W_s+W_o$		16600	7536
	where W_s is skip dead weight & W_o is skip live load			
Design Load	$L=FS \times W_l$			
	where FS is the factor of safety.			
Rope Strength	$S>= L$			
Equivalent effective weight	from Figure 9.19, p 330, Hartman		45000	20430
Total Suspended Load	$W=W_e+W_o+2W_s+2W_r$		65000	29510
Key points on duty cycle	$P_1 = (WV^{2)}/(550gt_a) = WV^2/17{,}177t_a$ in horsepower		147	110
	P is the power in hp, g is the acceleration due to gravity,			
	$P_2 = WV^2/17700t_r$		-184	-137
	$P_3=W_oV/550$		218	163
	$P_4=(W_oV/550)*(1-n/n)$		24	
	where 'n' is the efficiency			
	PA=P1+P3+P4		389	290
	PB=P3+P4		242	181
	PC=P2+P3+P4		59	44
	$P5=1.2(0.75P_A)/t_a = 0.9P_A/t_a$		35	26
	$P6=0.9P_A/t_r$		-44	-33
	$P_D=P_A+P_5$		424	316
	$P_E= P_C + P6$		15	11
RMS (root mean square) power **fo**	$P_{rms}= ((P^2{}_Dt_a + P^2{}_Bt_v+P^2{}_Et_r)/(0.5t_a+t_v+0.5t_r+0.25t_d)$		284	212
RMS (root mean square) power **fo**	$P_{rms}= ((P^2{}_Dt_a + P^2{}_Bt_v+P^2{}_Et_r)/(0.75t_a+t_v+0.75t_r+0.5t_d)$		267	199
Approximate energy consumption for duty cycle	(kW-hr/trip)= E=(0.7457PB(ta+tv))/(3600* n)		2.78	2.78

Appendix 5: Glossary

Selected definitions from "A Dictionary of Mining, Mineral and Related Terms", Paul W. Thrush, Ed., US Dept of the Interior, Bureau of Mines, Washington, DC.

cut-and-fill stoping. A stoping method in which the ore is excavated by successive flat or inclined slices, working upward from the level, as in shrinkage stoping. However, after each slice is blasted down all broken ore is removed, and the stope is filled with waste up to within a few feet of the back before the next slice is taken out, just enough room being left between the top of the waste pile and the back of the stope to provide working space. The term cut-and·fill stoping implies a definite and characteristic sequence of operations: (1) breaking a slice of ore from the back; (2) removing the broken ore; and (3) introducing filling.

haulage level. Underground level either along and inside the ore body or closely parallel to it, usually in the f001- wall. In this level the mineral gravitated or drawn (slushed) down from overhand stopes or raised from underhand stopes is loaded into trams (tubs, trucks, cocopans) and sent out to the hoisting shaft. Haulageways include levels and connecting passes (crosscuts). and are also used to transport supplies, waste rock, and for movement of miners.

open-stope method. Sloping in which no regular artificial method of support is employed, although occasional props or cribs may be used to hold local patches of insecure ground. The walls and roof are self-supporting, and open stapes can be used only where the ore and wall rocks are firm. The simplest open stopes are those in which the entire ore body is removed from wall to wall without leaving any pillars. The sloping of ore in this manner is usually confined to relatively small ore bodies, since regardless of the firmness of the ground, there is a limit to the length of unsupported span which will stand without breaking.

rib. The side of a pillar or the wall of an entry.

round. Planned pattern of drill holes fired in sequence in tunncling, shaft sinking, or sloping. First the cut holes are fired, followed by relief, lifter, and rib holes.

scaling. The removal of loose rocks from the roof or walls. This work is dangerous and a long scaling bar is often used.

shrinkage stoping. In this method of stoping the ore is mined out in successive flat or inclined 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 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. Shrinkage stopes often are excavated by taking slices along the vein (especially in narrow veins) from one er.d of an ore shoot to the other, without leaving any pillars for supporting the walls. Sometimes (especially in wide veins) the ore is mined in a series of transverse slopes of limited size, each stope being separated from the next by a pillar of solid ore to reduce the length of the unsupported span.

In some instances, casual pillars may be left to support local area where the walls are weak; in other instances, pillars of lean ore or waste within the ore body are left. The latter are left primarily because it does not pay to mine them, but at the same time they reduce the length of the unsupported span and assist in supporting the walls and back.

stull. A timber prop set between the walls of a stope.

stull stoping. The walls of narrow veins frequently are supported by stull timbers placed between the foot and hanging walls, which constitute the only artificial support provided during the excavation of the stopes. Stulls may be placed at irregular intervals to support local patches of insecure ground, in which case the stopes are virtually open stopes.

Sometimes the stulls are placed at regular intervals both along the stope and vertically, in which case stull stoping should be considered a distinctive method.

sublevel. A secondary level for working ore.

sublevel sloping. A mining method involving overhand, underhand, and shrinkage stoping. Its characteristic feature is the use of sublevels. The sublevels are worked simultaneously, the lowest on a given block being farthest advanced and the subs above following one another at short intervals. The uppermost sublevel underneath lhe cover is partly caved. The caved cover follows down upon the caved ore. The broken ore is in part drawn from the level, and a part remains in the stope in order to give lateral support to the walls and to prevent ad· mixture of cover and ore. The breaking faces are developed by crosscuts, which are extended from wall to wall from the end of the sublevel. The method can also be looked upon as a retreating method, the ore body being worked from the top down, and the individual blocks upon a given level being worked from their ends to the center.

sump. An excavation made underground to collect water, from which water is pumped to the surface or to another sump nearer the surface. Sumps arc placed at the bottom of a shaft, near the shaft on a level, or at some interior point.

ventilation. The provision of an adequate flow of fresh air along all roadways, traveling roads, workings, and service points underground. Ventilation is an essential factor in safety, health, and working efficiency and is also necessary to dilute and remove noxious or flammable gases and to abate such problems as dust and high temperatures.

ventilation efficiency. One measure of the efficiency of a mine ventilation system is the ratio of the total amount [volume in cubic meters per second (cubic feet per minute)] of air handled by the fan to the total amount of air actually getting to the working faces. If 94 ffi'1/s (200,000 cfm) are handled by the fan and only 47 (100,000) get to the working faces, the efficiency is only 50%. ventilation planning. When a new mine is projected or a new seam to be worked from an existing mine, plans are prepared to show the proposed ventilating system, including the quantities of air and pressures and the principal appliances to control and distribute the air. Investigations and calculations are made to select a fan of the necessary type and size for the ventilation required. All this very important work comes within the general term ventilation planning.

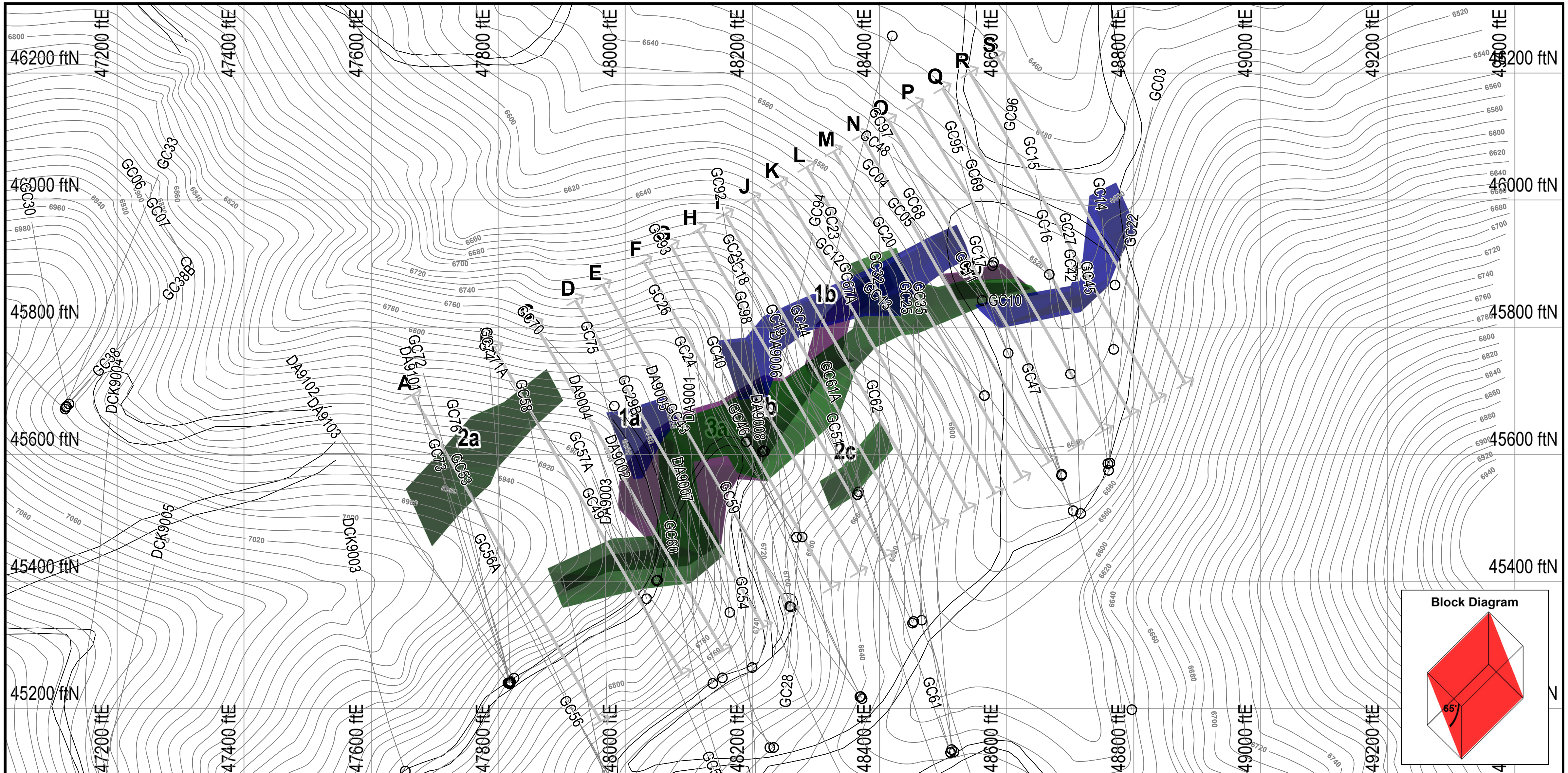
ventilation survey. In order to distribute the air in a mine efficiently and economically, ventilation surveys are conducted.

Appendix 6: Development and Production Schedule

Month	1	2	3	4	5	6	7	8	9	10	11	12	Yr2 Q1	Yr 2 Q2	Yr 2 Q3	Yr 2 Q4	Year 3	Year 4	Year 5
portal	6545																		
		119	83	67	76	23	58	58	77	23	55	66							
		$217,338	$151,589	$122,367	$138,804	$42,007	$105,929	$105,929	$140,630	$42,007	$100,450	$120,540							
Level starts (Months from start)		2	4	5	6	7	8	9	10	10	11	13							
		89417	sill tons			7081	10585	17885	13505	10439	10358	2409							
	days from star	70	42	34	38	12	29	29	39	12	28	33							
					X-Cut	sill	Haulage	Duration (months)	End time, months from start										
6387				16	8	8	0	0.55	5.5										
					$11,189	$14,676	$ -												
6300				107	18	89	0	3.69	8.7										
					$25,176	$163,270	$ -												
6212					156	11	145	0	5.38	11.67									
						$15,385	$266,001	$ -											
6125						454	179	146	129	15.67	22.4								
							$250,364	$267,836	$ 283,613										
6095						140	41	99	0	4.83	12.5								
							$57,346	$181,615	$ -										
6018							370	58	185	127	12.77	20.5							
								$81,123	$339,381	$ 279,216									
5941								467	143	143	124	16.10							
									$200,011	$262,332	$ 271,521								
5862									392	177	108	77							
										$247,566	$197,933	$ 168,668							
5832									49	15	34	0							
										$20,980	$62,373	$ -							
5760										65	32	33							
											$44,758	$60,538							
5687											153	87							
												$121,685							
5657											61	15							
												$20,980							
5580												107							
5503																			
5425																			
opment places ea	1	1	1	3	3	5	6	6	8	8	10	11							
Alimak Raise for Ventilation/Escapeway				shifts	101	19	13	14			62								
		Duration (months)			1.73	0.32	0.23	0.25	0.00	0.00	1.08	0.00							
		End time, months from star			8	8	9	9	9	9	10	10							
UG Diamond Drilling																			
Stope 1																			
Stope 2																			
Stope 3																			
Stope 4																			

	Month	1	2	3	4	5	6	7	8	9	10	11	12	Yr2 Q1	Yr 2 Q2	Yr 2 Q3	Yr 2 Q4	Year 3	Year 4	Year 5		
Development tons		6962	6962	4856	3920	4446	1346	3393	3393	4505	1346	3218	3861	8132	4446	0	0					
	Decline	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$105,020	$315,060	$105,020	$0	$0					
	6387						$25,865															
	6300						$51,074	$51,074	$51,074	$35,241												
	6212						$52,309	$52,309	$52,309	$52,309	$52,309	$20,401										
	6125							$51,164	$51,164	$51,164	$51,164	$51,164	$51,164	$153,491	$153,491	$153,491	$34,280					
	6095							$49,499	$49,499	$49,499	$49,499	$41,084		$0	$0	$0	$0					
	6018								$54,774	$54,774	$54,774	$54,774	$54,774	$164,321	$164,321	$96,950	$0					
	5941									$45,576	$45,576	$45,576	$45,576	$136,729	$136,729	$136,729	$136,729					
	5862										$45,448	$45,448	$45,448	$136,344	$136,344	$136,344	$68,626					
	5832										$49,331	$34,039		$0	$0	$0	$0					
	5760											$46,978	$46,978	$0	$0	$0	$0					
	5687												$47,522	$142,566	$59,878	$0	$0					
	5657												$50,092	$55,102	$0	$0	$0					
	5580													$157,135	$36,141	$0	$0					
	5503													$106,402	$76,077	$0	$0					
	5425													$100,687	$151,031	$71,488	$0					
	Totals	$105,020	$105,020	$105,020	$105,020	$105,020	$234,269	$309,066	$363,839	$393,583	$453,121	$444,483	$446,574	$1,467,837	$1,019,032	$595,001	$239,635					
Sill tons	Sill Level					duration	tons															
584	6387					0.55	584															
6497	6300					3.69	1,461	1,761	1,761	1,215												
10585	6212					5.38		1,604	1,785	1,749	1,690	1,685	518	0	0	0	0					
10658	6125					15.67		680	680	680	680	680	680	1395	1434	1362	485					
7227	6095					4.83			1,497	1,497	1,497	1,497	1,243	0	0	0	0					
13505	6018					12.77				1,057	1,057	1,057	1,057	3171	3171	2928	0					
10439	5941					16.10				648	648	648	648	1945	1945	1945	1498					
7876	5862					13.51					583	583	583	1749	1749	1749	880					
2482	5832					1.69					1,469	1,014		0	0	0	0					
2409	5760					2.24					1,075	1,075	258	0	0	0	0					
3139	5687					5.26						597	597	1790	155	0	0					
3358	5657					2.10							1,596	1756	0	0	0					
4380	5580					3.69								3559	818	0	0					
3212	5503					3.43								1870	1337	0	0					
3066	5425					6.42								955	1433	678	0					
							Grams	Grams	Grams	Grams	Grams	Grams	Grams									
	6387						5,747	0	0	0	0	0	0	0	0	0	0					
	6300						14,373	17,327	17,327	11,956	0	0	0	0	0	0	0					
	6212						0	15,779	17,563	17,209	16,627	16,584	5,100	0	0	0	0					
	6125						0	6,691	6,691	6,691	6,691	6,691	6,691	13731	14108	13406	4769					
	6095						0	0	14,731	14,731	14,731	14,731	12,226	0	0	0	0					
	6018						0	0	0	10,402	10,402	10,402	10,402	31207	31207	28815	0					
	5941						0	0	0	6,379	6,379	6,379	6,379	19138	19138	19138	14741					
	5862						0	0	0	0	5,735	5,735	5,735	17206	17206	17206	8660					
	5832						0	0	0	0	14,454	9,974	0	0	0	0	0					
	5760						0	0	0	0	10,576	10,576	2,538	0	0	0	0					
	5687						0	0	0	0	0	5,871	5,871	17612	1526	0	0					
	5657						0	0	0	0	0	0	15,709	17280	0	0	0					
	5580						0	0	0	0	0	0	0	35017	8054	0	0					
	5503						0	0	0	0	0	0	0	18403	13158	0	0					
	5425						0	0	0	0	0	0	0	9397	14096	6672	0					
														0	0	0	0					
						Totals	20,119	39,797	56,312	67,369	85,596	86,943	70,652	178990	118493	85237	28170	0				
Month		**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**Yr2 Q1**	**Yr 2 Q2**	**Yr 2 Q3**	**Yr 2 Q4**	**Year 3**	**Year 4**	**Year 5**		**TOTALS**
Development Consumables cost Totals		$105,020	$105,020	$105,020	$105,020	$105,020	$234,269	$309,066	$363,839	$393,583	$453,121	$444,483	$446,574	$1,467,837	$1,019,032	$595,001	$239,635					**$5,966,400**
														$0	$0	$0	$0					**$0**
Diamond Drilling										$15,000	$15,000	$15,000	$15,000	$45,000	$45,000	$45,000	$45,000	$180,000	$180,000	$180,000	$120,000	**$900,000**
Total Personnel cost/month		$178,231	$178,231	$178,231	$178,231	$178,231	$178,231	$178,231	$207,231	$252,440	$252,440	$252,440	$252,440	$812,902	$931,319	$1,121,029	$1,181,450	$4,725,798	$4,725,798	$4,725,798	$3,150,532	**$22,948,100**
														$0	$0	$0	$0					
														$0	$0	$0	$0					
Stope tons consumables cost		0	0	0	0	0	0	0	0	0	0	0	0	$12,427	$43,495	$78,706	$115,987	$637,929	$579,936	$536,440	$159,482	**$2,164,400**
Windy Gulch Costs						$177,350	$177,350	$177,350														
Windy Gulch tons							7350	7350														
Windy Gulch Grams							67620	67620														
Stope Mining Cost	Stope Tons	0	0	0	0	0	0	0	0	0	0	0	0	1,980	6,930	12,540	18,480	101,640	92,400	85,470	25,410	**344,900**
	Stope tonnes													1,796	6,286	11,374	16,761	92,187	83,807	77,521		**311,000**
Stope grams using 8.9 grams/tonne														17,622	61,677	111,606	164,472	901,661	819,499	758,179	225,362	**3,060,100**
	sill grams						20,119	39,797	56,312	67,369	85,596	86,943	70,652	178,990	118,493	85,237	28,170					**842,200**
						tons	9,695	11,759	5,906	7,065	8,977	9,118	7,410	20,815	19,578	21,880	21,761	101,640	92,400	85,470	25,410	**449,600**
Sill Tons							2,045	4,044	5,723	6,846	8,699	8,836	7,180	18,190	12,042	8,662	2,863					**85,600**
Sill Grade							9.8	9.8	9.8	9.8	9.8	9.8	9.8	9.8	9.8	9.8	9.8					
Total tons		0	0	0	0	0	9,995	12,123	6,089	7,284	9,255	9,400	7,639	21,459	20,185	22,557	22,435	104,788	95,261	88,117	26,197	**463,500**
total grams							83,255	101,896	53,345	63,820	81,087	82,362	66,930	186,309	170,869	186,819	180,616	836,843	754,911	698,292	207,600	**3,755,000**
total ounces							2,677	3,276	1,715	2,052	2,607	2,648	2,152	5,991	5,494	6,007	5,808	26,908	24,274	22,453	6,675	**120,740**

Appendix 7: Cross- and level sections showing underground mining.



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

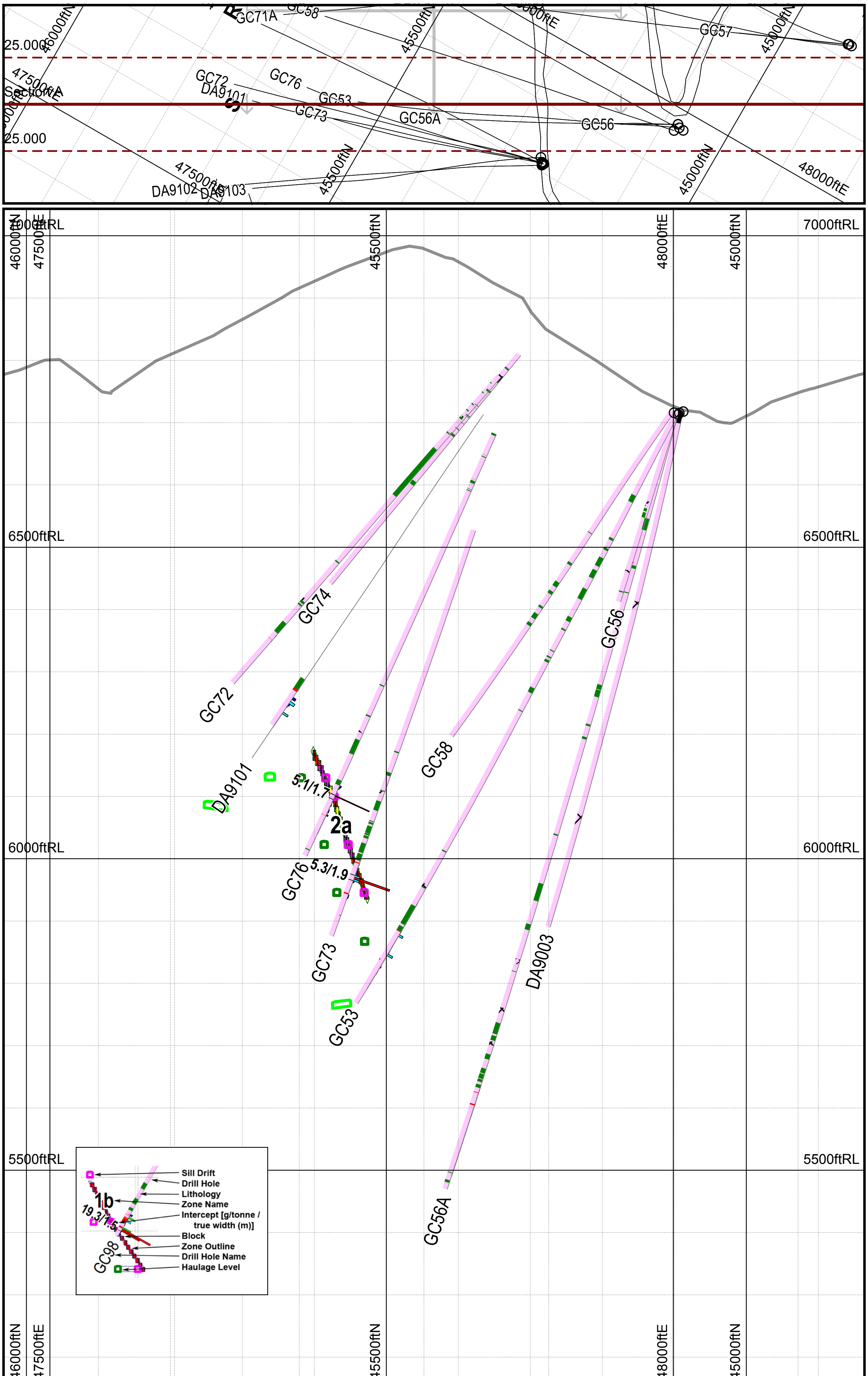
Zones
- Zone 1
- Zone 2
- Zone 3

Scale	Plot Date	Sheet
1 in = 200 ft	30-Apr-2015	1 of 1
	Plot File: Plan View of Dawson Zone Drilling II	

100 0 100 200ft

Plan View of Drilling Dawson Deposit

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA		BX	
DYK		GOU	
MB		NC	
QVN		GOS	
SUL		CAS	
SZ		OVB	
FLT			

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

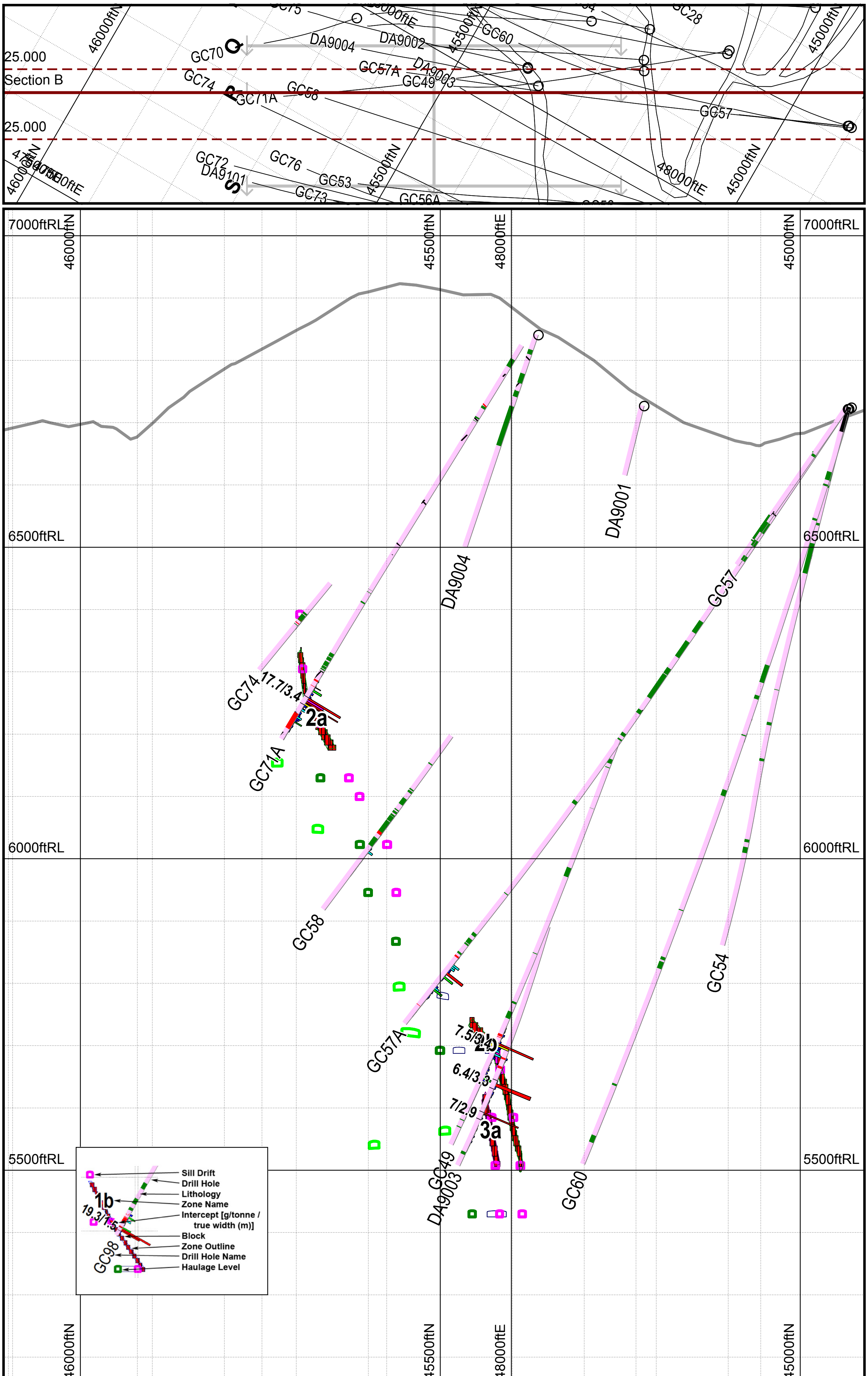
Plot File: Section A

50 0 50 100ft

West Facing Cross-Sections

Section A

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

	MA		BX
	DYK		GOU
	MB		NC
	QVN		GOS
	SUL		CAS
	SZ		OVB
	FLT		

Grade (g/tonne)

	< 1
	1 to 2
	2 to 3
	3 to 4
	4 to 5
	>= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

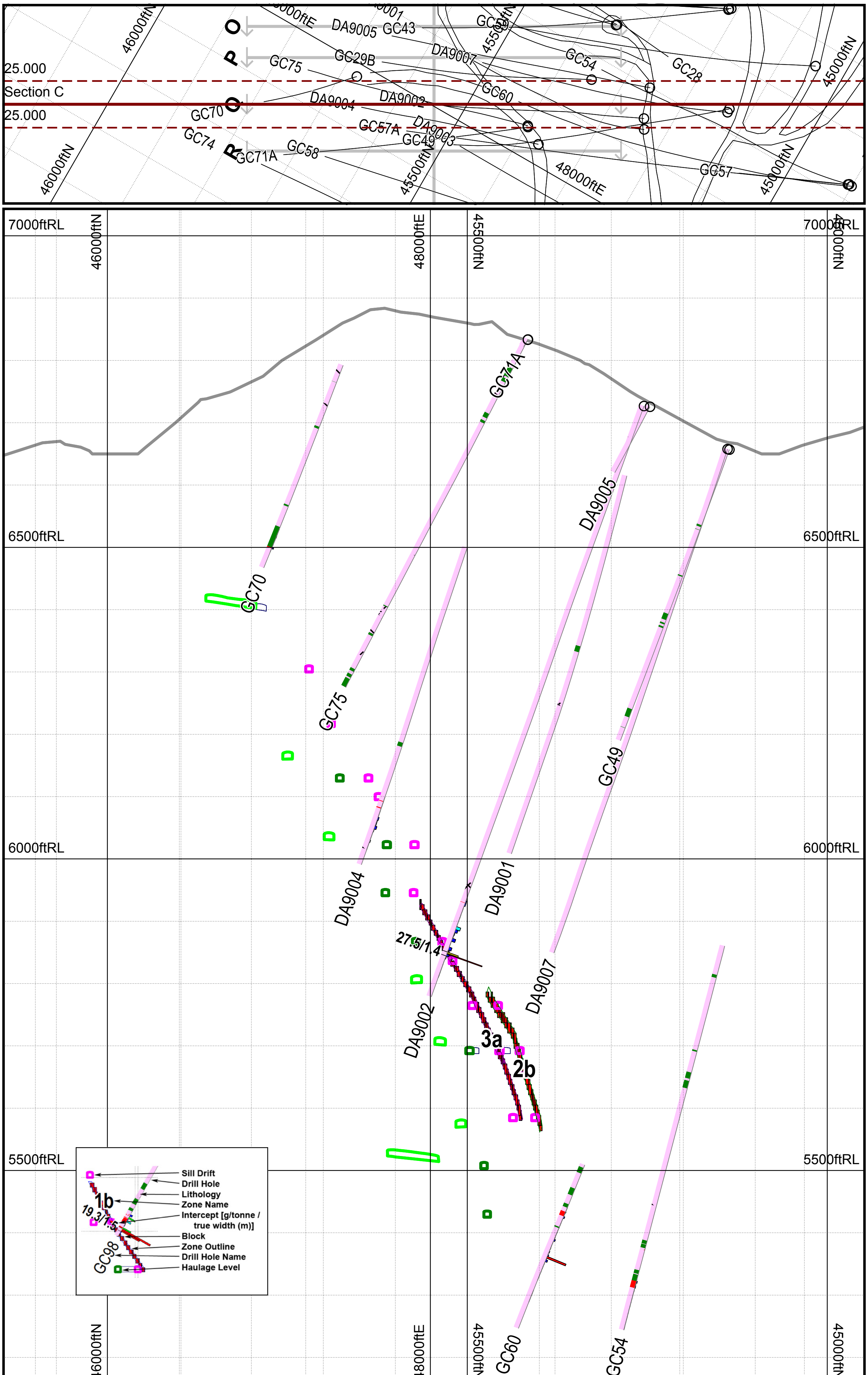
Plot File: Section B

50 0 50 100ft

West Facing Cross-Sections

Section B

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology			
MA		BX	
DYK		GOU	
MB		NC	
QVN		GOS	
SUL		CAS	
SZ		OVB	
FLT			

Grade (g/tonne)
< 1
1 to 2
2 to 3
3 to 4
4 to 5
>= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

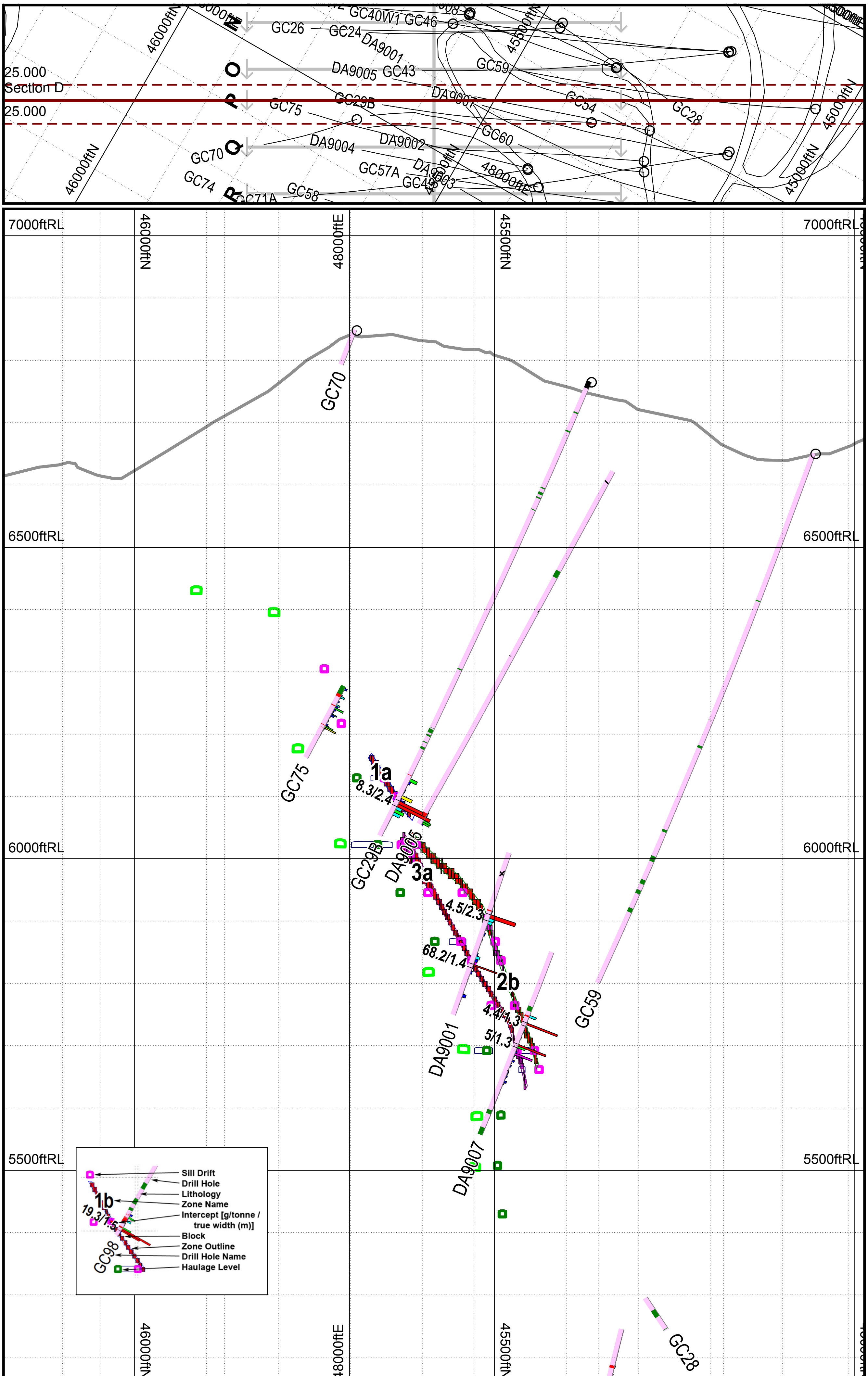
Plot File: Section C

50 0 50 100ft

West Facing Cross-Sections

Section C

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

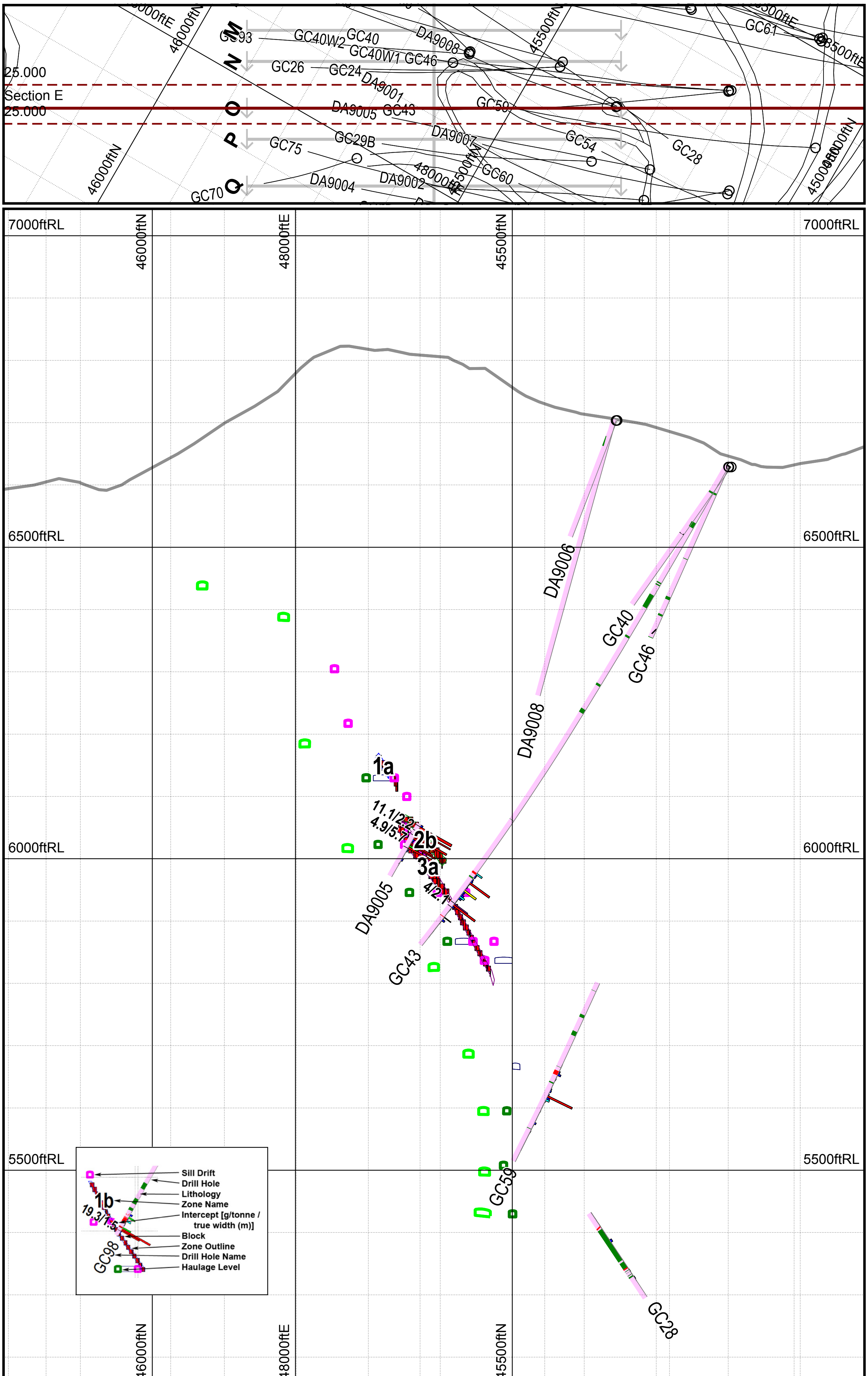
Plot File: Section D

50 0 50 100ft

West Facing Cross-Sections

Section D

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

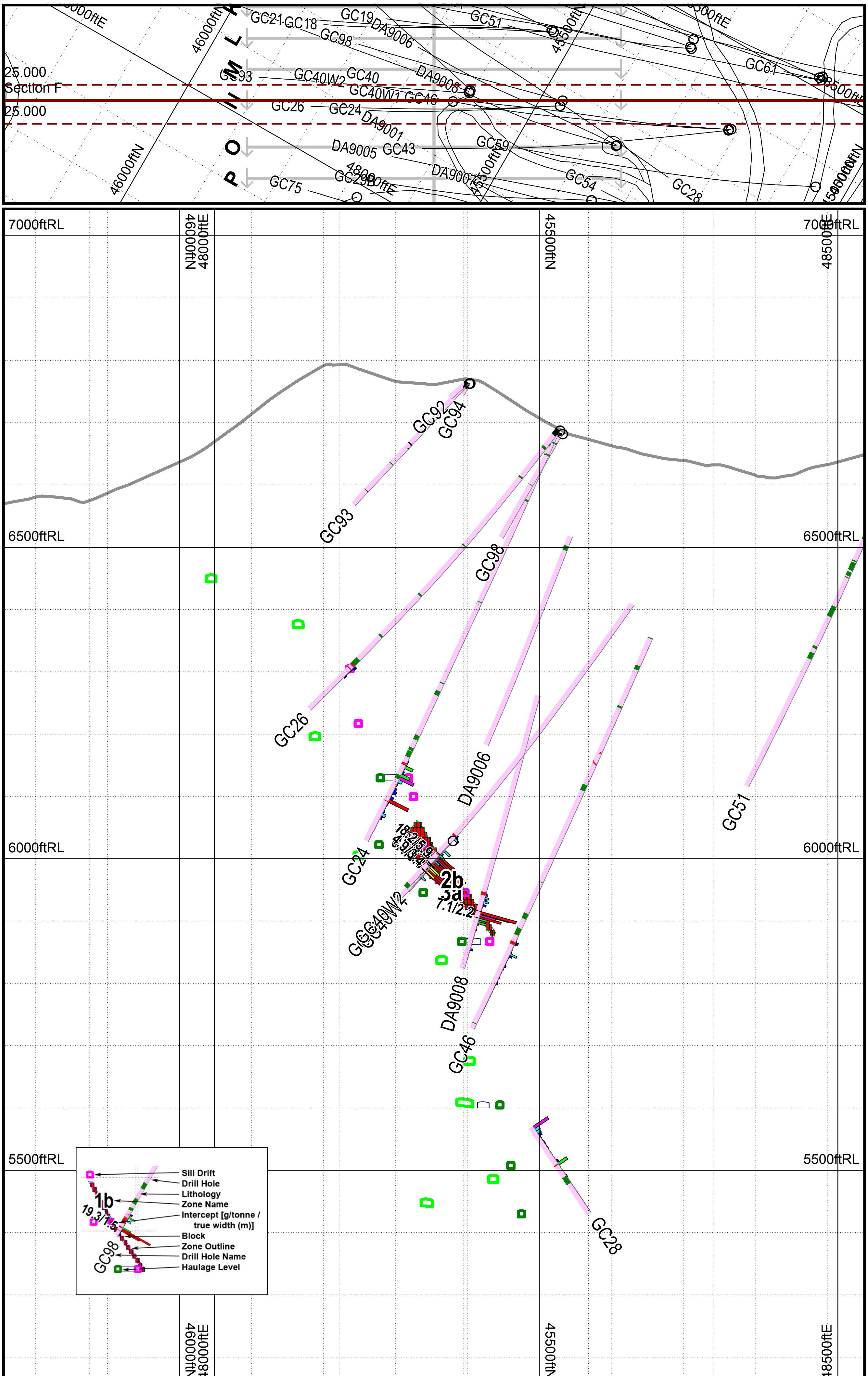
Plot File: Section E

50 0 50 100ft

West Facing Cross-Sections

Section E

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA		BX	
DYK		GOU	
MB		NC	
QVN		GOS	
SUL		CAS	
SZ		OVB	
FLT			

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

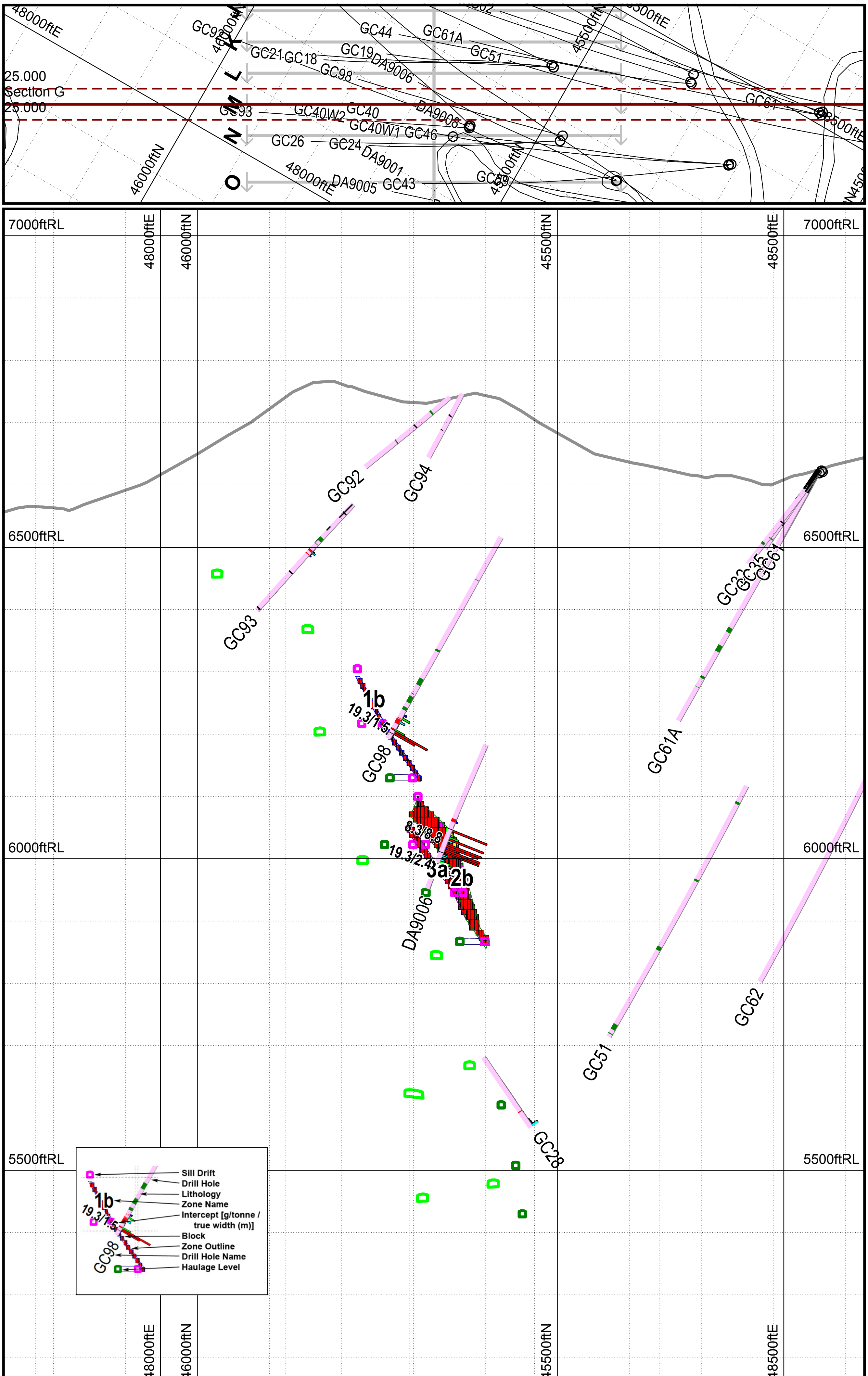
Plot File: Section F

50 0 50 100ft

West Facing Cross-Sections

Section F

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

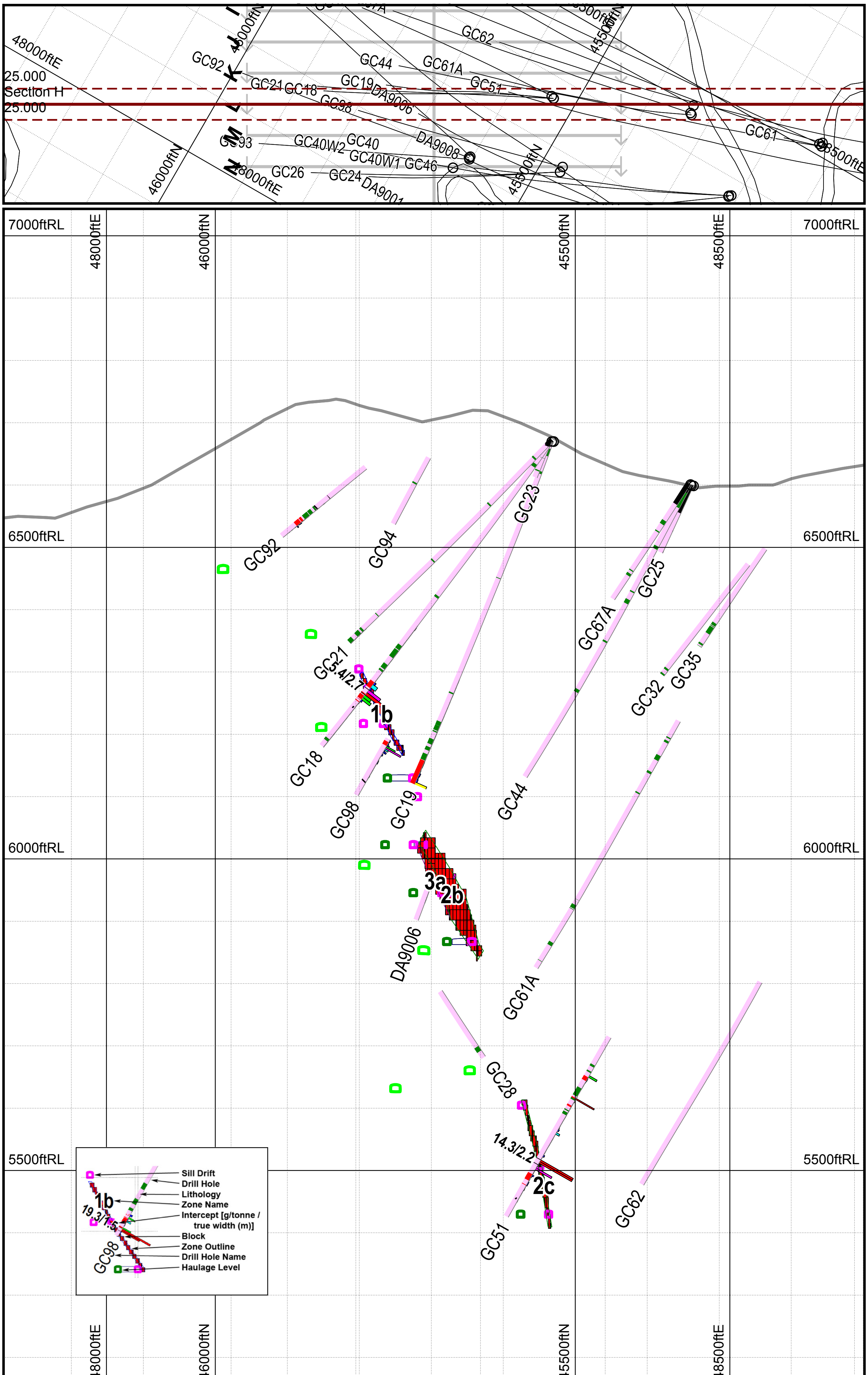
Plot File: Section G

50 0 50 100ft

West Facing Cross-Sections

Section G

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA		BX	
DYK		GOU	
MB		NC	
QVN		GOS	
SUL		CAS	
SZ		OVB	
FLT			

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

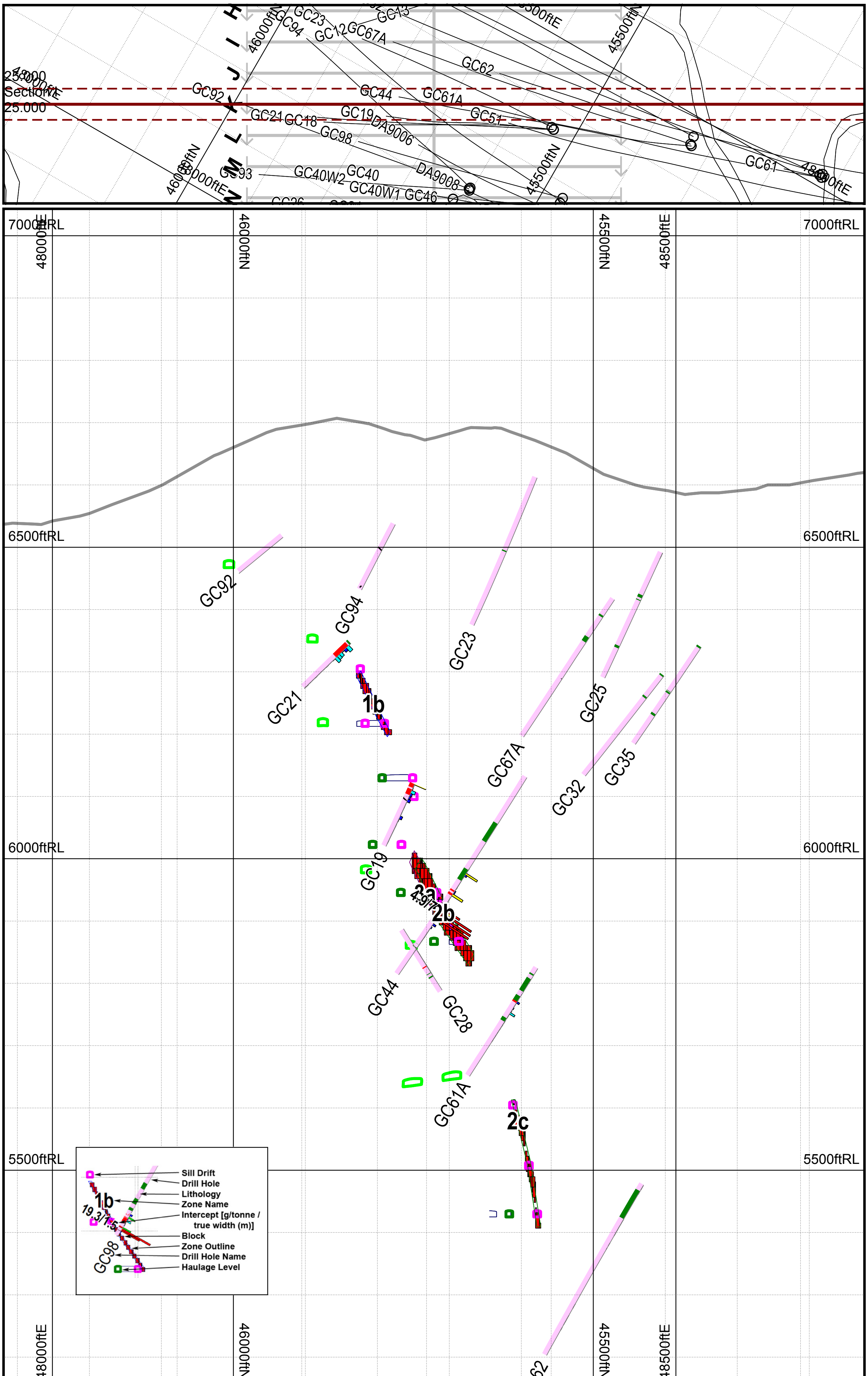
Plot File: Section H

50 0 50 100ft

West Facing Cross-Sections

Section H

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA		BX	
DYK		GOU	
MB		NC	
QVN		GOS	
SUL		CAS	
SZ		OVB	
FLT			

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

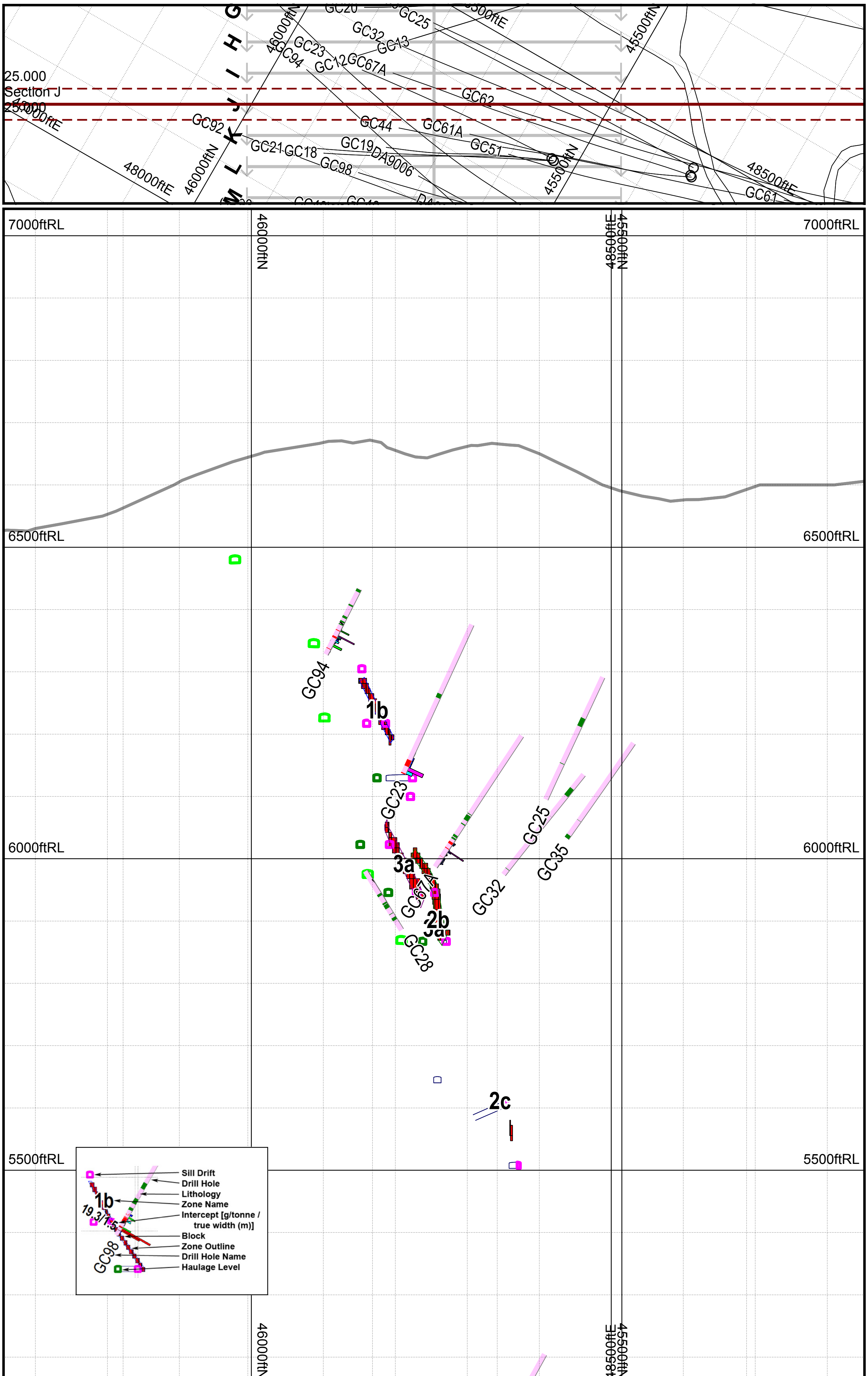
Plot File: Section I

50 0 50 100ft

West Facing Cross-Sections

Section I

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

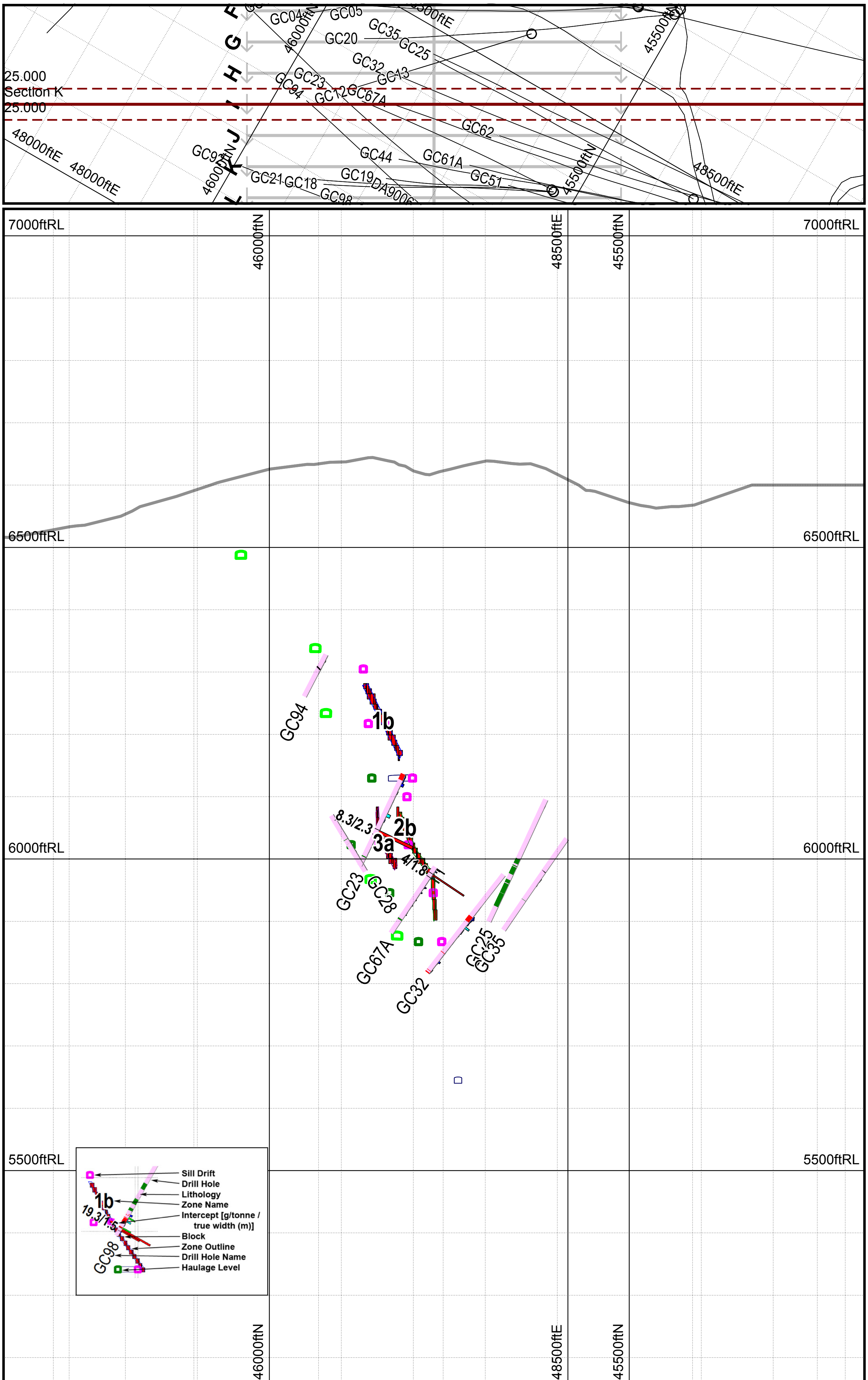
Plot File: Section J

50 0 50 100ft

West Facing Cross-Sections

Section J

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

50 0 50 100ft

Plot Date
29-Sep-2015

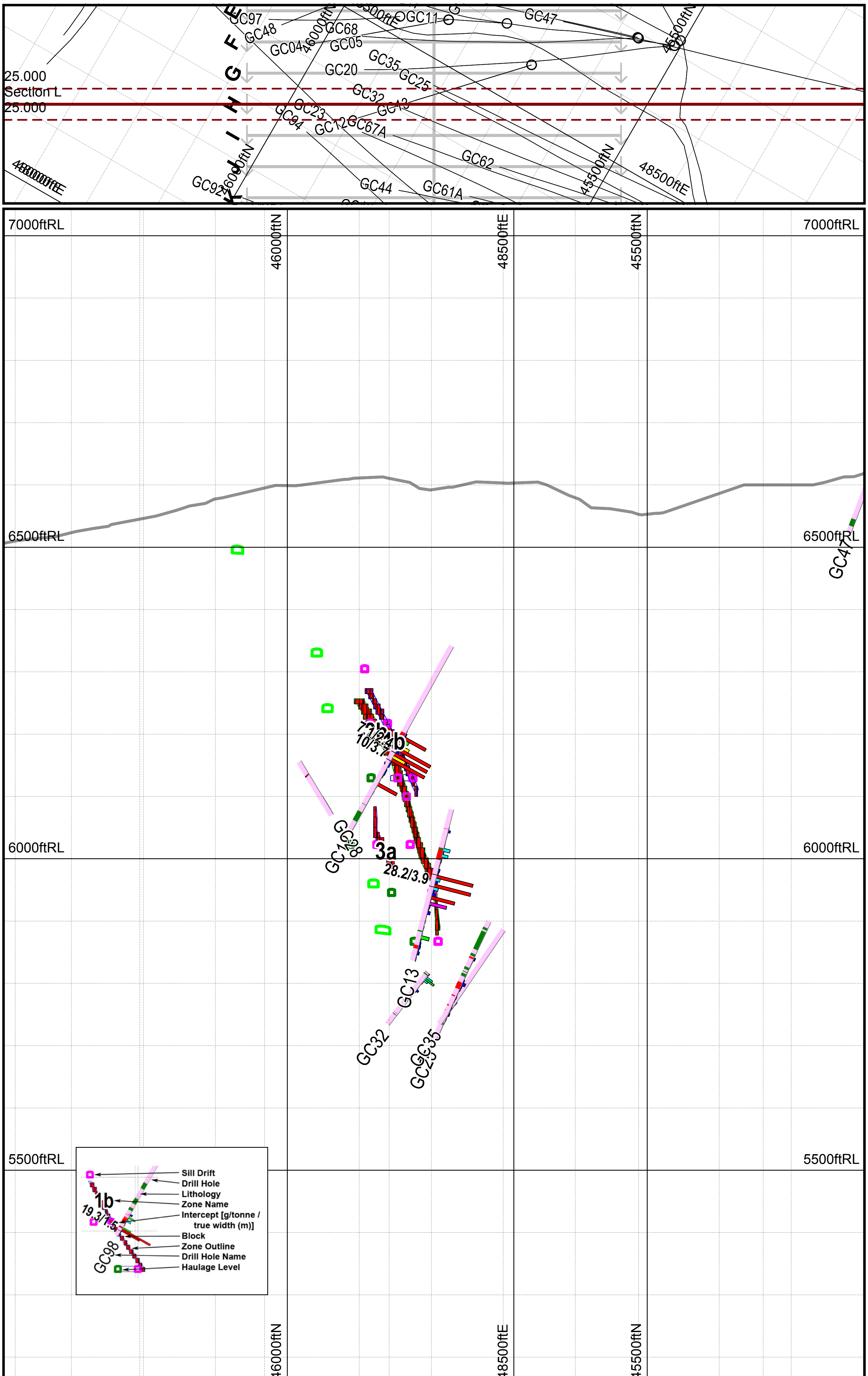
Sheet
1 of 1

Plot File: Section K

West Facing Cross-Sections

Section K

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA
DYK
MB
QVN
SUL
SZ
FLT
BX
GOU
NC
GOS
CAS
OVB

Grade (g/tonne)

< 1
1 to 2
2 to 3
3 to 4
4 to 5
>= 5

Scale
1 in = 200 ft

50 0 50 100ft

Plot Date
29-Sep-2015

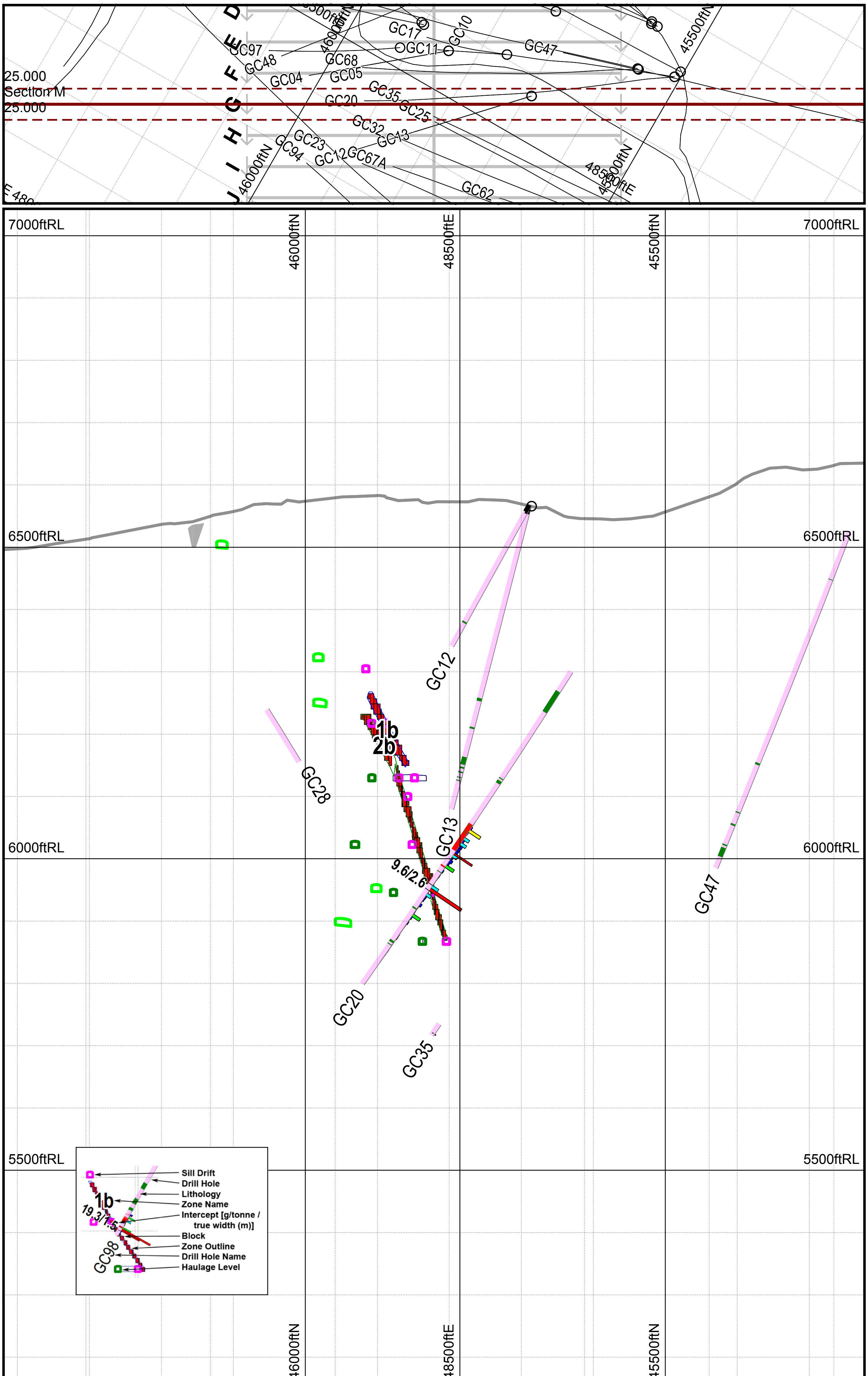
Sheet
1 of 1

Plot File: Section L

West Facing Cross-Sections

Section L

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA, DYK, MB, QVN, SUL, SZ, FLT, BX, GOU, NC, GOS, CAS, OVB

Grade (g/tonne)

< 1
1 to 2
2 to 3
3 to 4
4 to 5
>= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

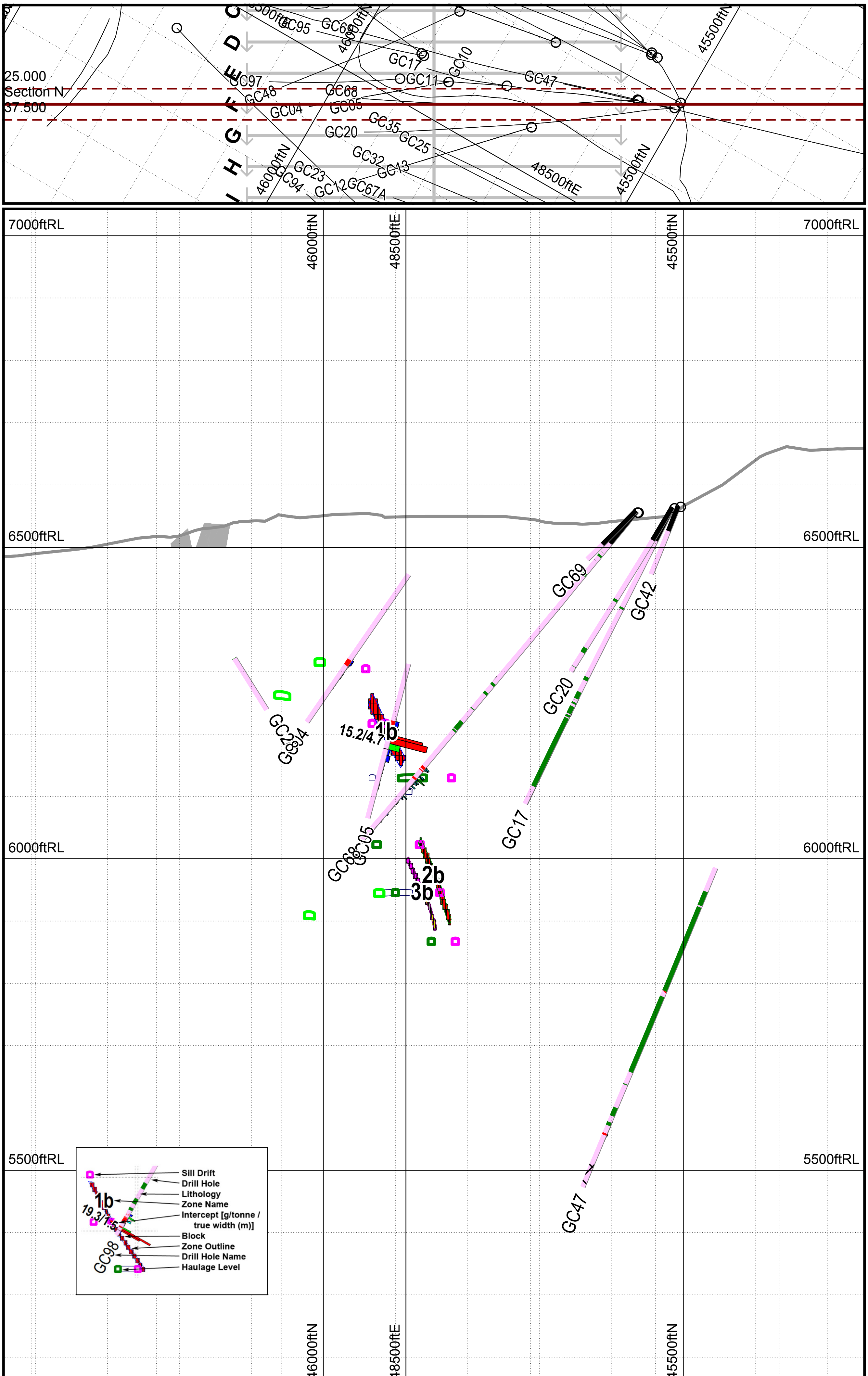
Plot File: Section M

50 0 50 100ft

West Facing Cross-Sections

Section M

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

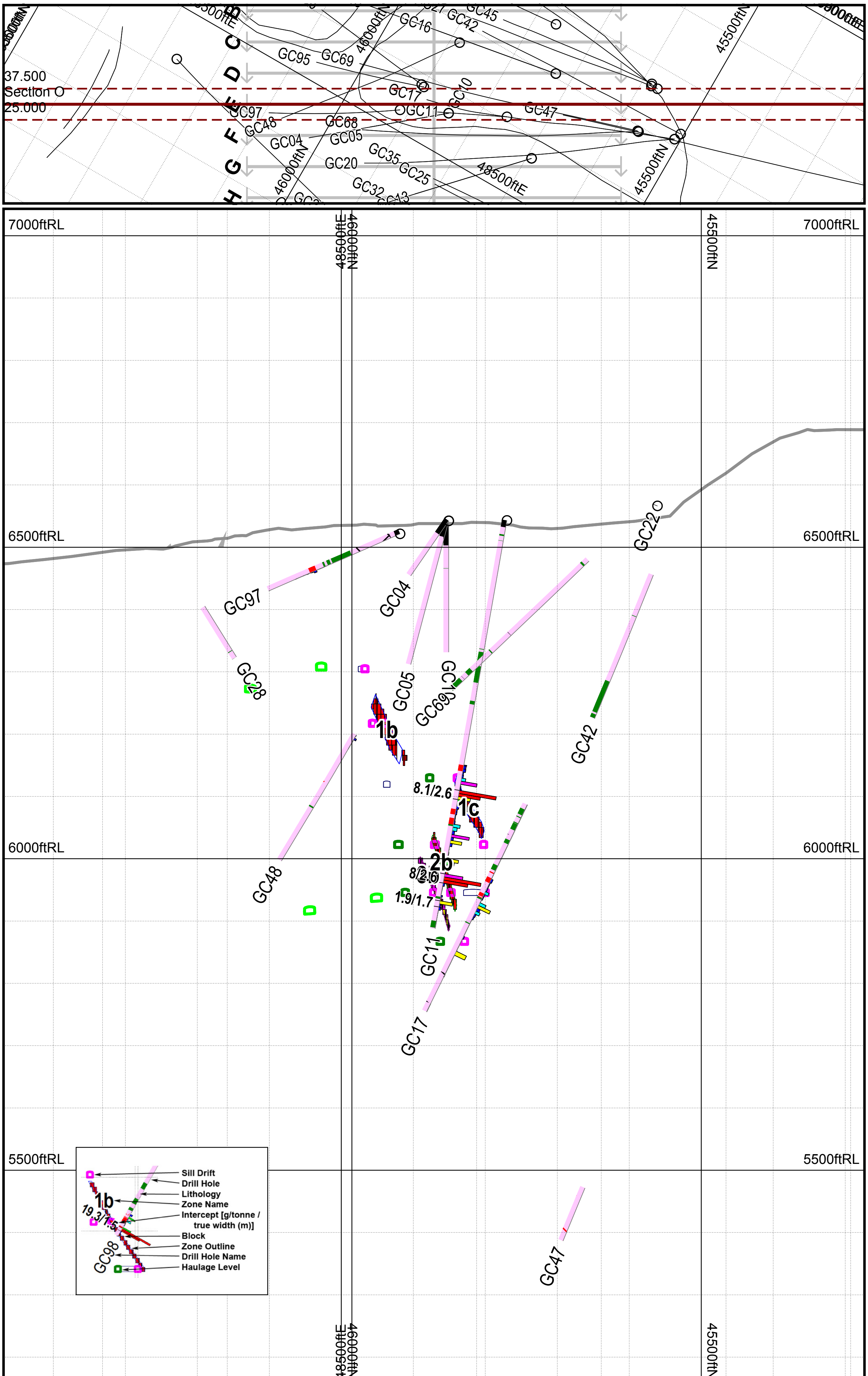
Plot File: Section N

50 0 50 100ft

West Facing Cross-Sections

Section N

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

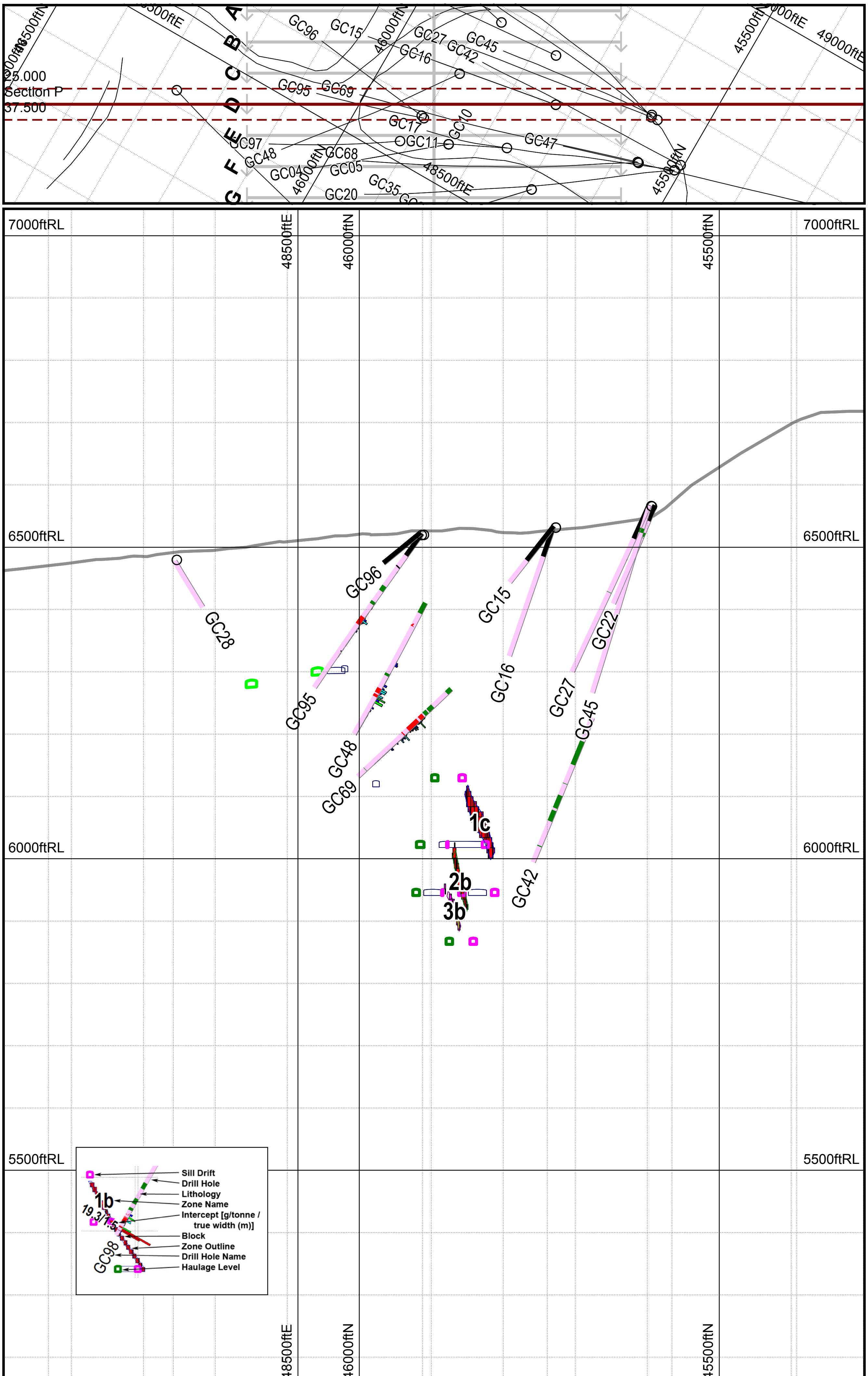
Plot File: Section O

50 0 50 100ft

West Facing Cross-Sections

Section O

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA
DYK
MB
QVN
SUL
SZ
FLT
BX
GOU
NC
GOS
CAS
OVB

Grade (g/tonne)

< 1
1 to 2
2 to 3
3 to 4
4 to 5
>= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

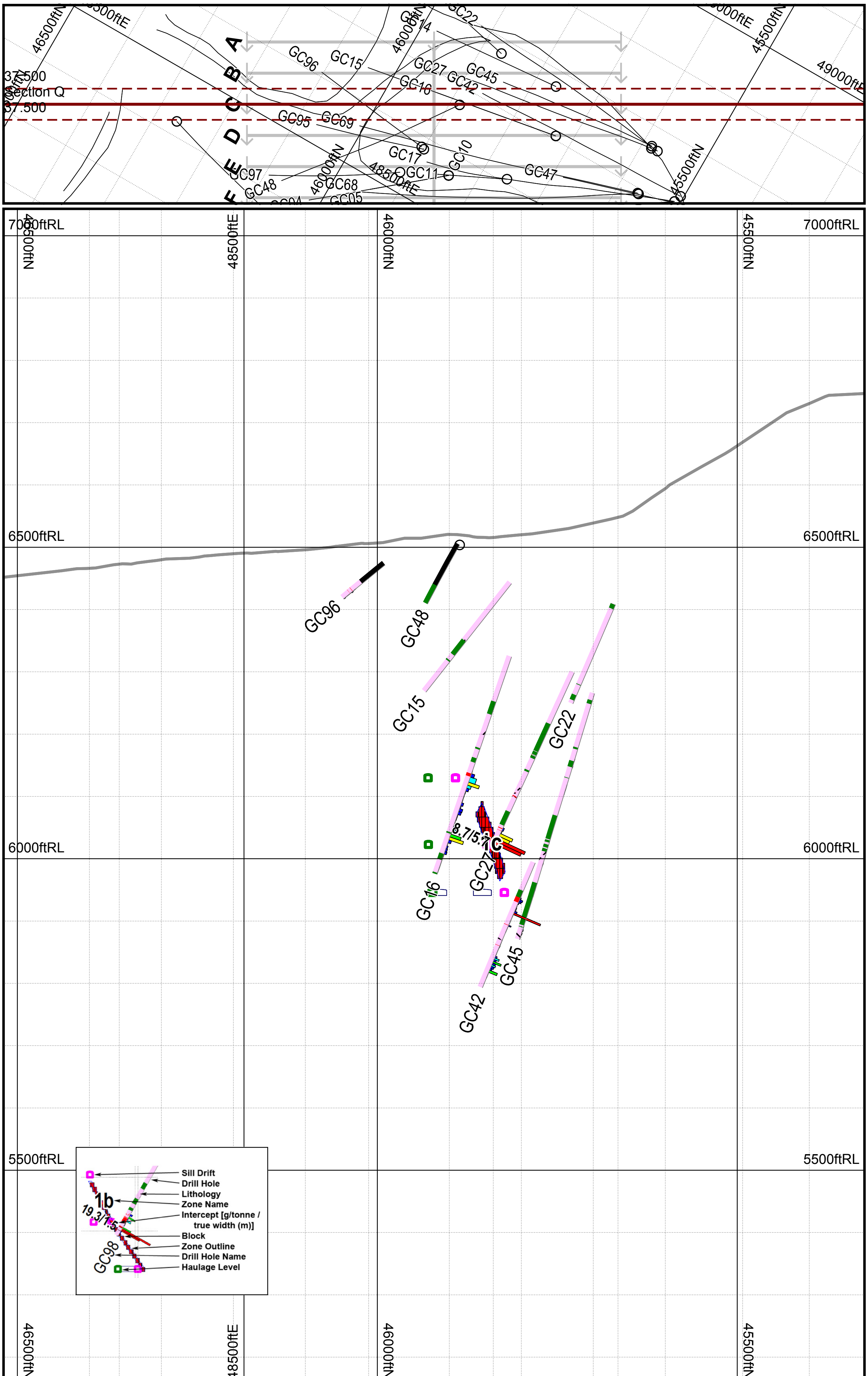
Plot File: Section P

50 0 50 100ft

West Facing Cross-Sections

Section P

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

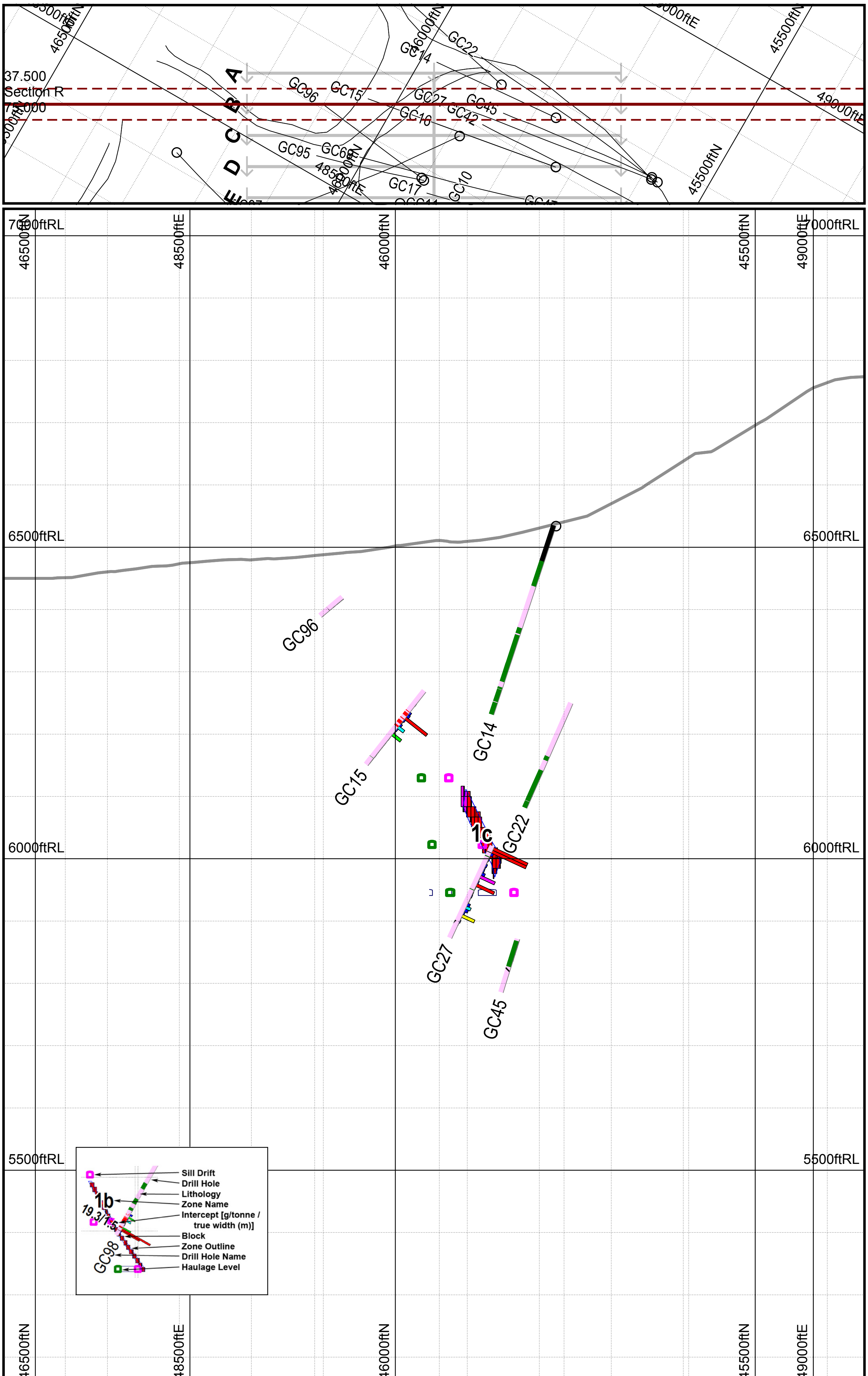
Plot File: Section Q

50 0 50 100ft

West Facing Cross-Sections

Section Q

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA	BX
DYK	GOU
MB	NC
QVN	GOS
SUL	CAS
SZ	OVB
FLT	

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

Plot Date
29-Sep-2015

Sheet
1 of 1

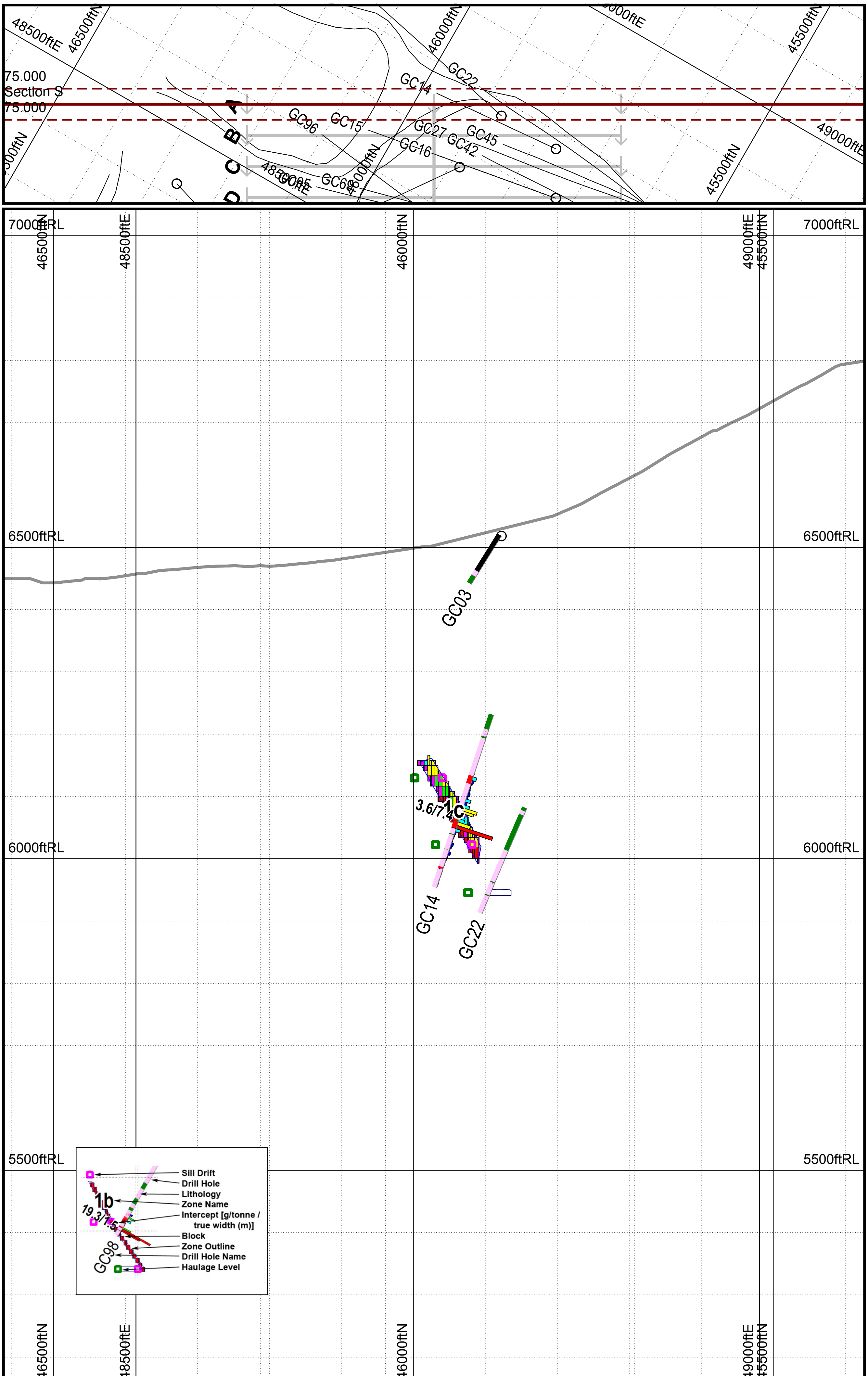
Plot File: Section R

50 0 50 100ft

West Facing Cross-Sections

Section R

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Lithology

MA, DYK, MB, QVN, SUL, SZ, FLT, BX, GOU, NC, GOS, CAS, OVB

Grade (g/tonne)

- < 1
- 1 to 2
- 2 to 3
- 3 to 4
- 4 to 5
- >= 5

Scale
1 in = 200 ft

50 0 50 100ft

Plot Date
29-Sep-2015

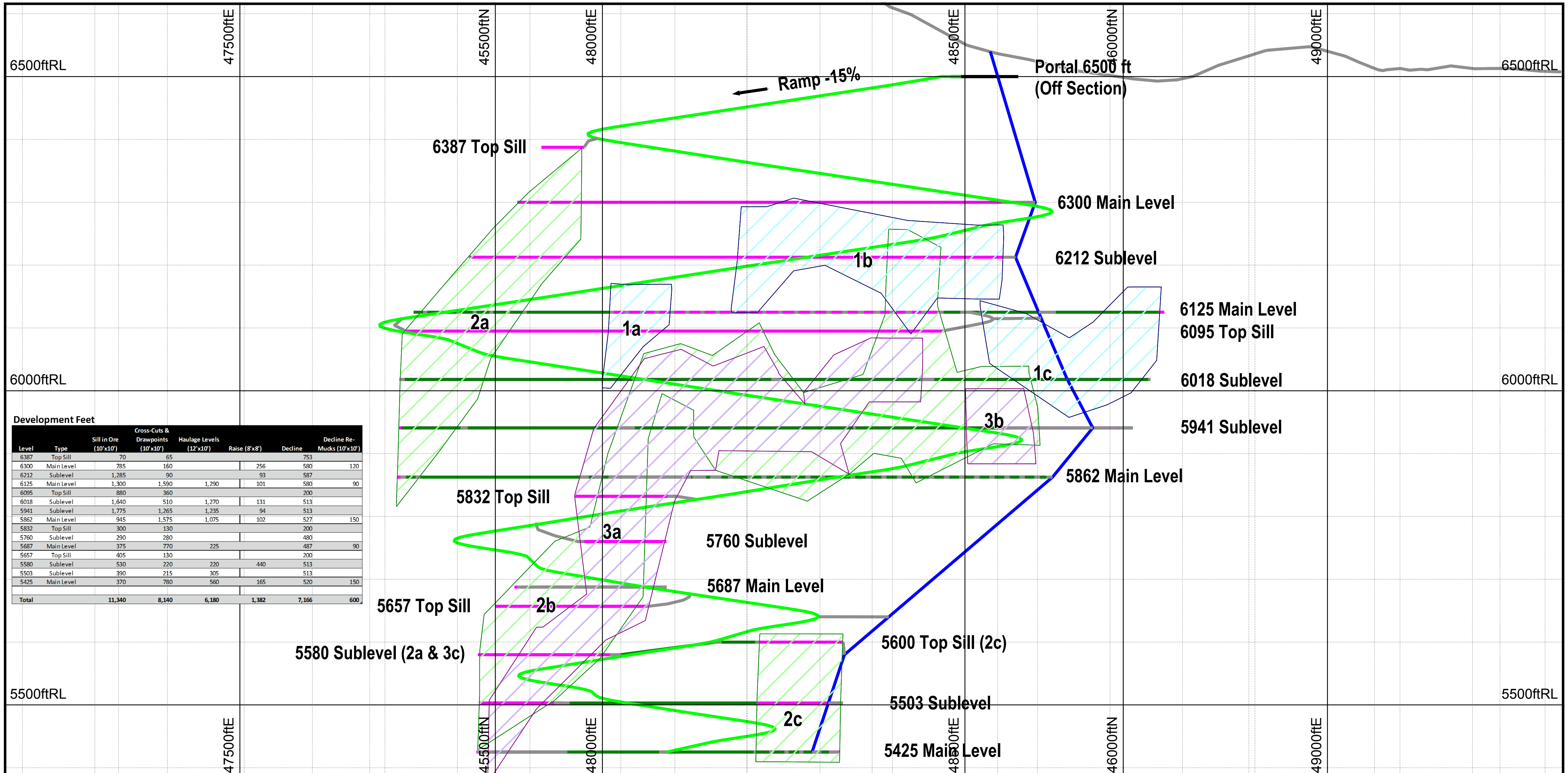
Sheet
1 of 1

Plot File: Section S

West Facing Cross-Sections

Section S

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



Development Feet

Level	Type	Sill in Ore (10'x10')	Cross-Cuts & Drawpoints (10'x10')	Haulage Levels (12'x10')	Raise (8'x8')	Decline	Decline Re-Mucks (10'x10')
6387	Top Sill	70	65			753	
6300	Main Level	785	160		256	580	120
6212	Sublevel	1,285	90		93	587	
6125	Main Level	1,300	1,590	1,290	101	580	90
6095	Top Sill	880	360			200	
6018	Sublevel	1,640	510	1,270	131	513	
5941	Sublevel	1,775	1,265	1,235	94	513	
5862	Main Level	945	1,575	1,075	102	527	150
5832	Top Sill	300	130			200	
5760	Sublevel	290	280			480	
5687	Main Level	375	770	225		487	90
5657	Top Sill	405	130			200	
5580	Sublevel	530	220	220	440	513	
5503	Sublevel	390	215	305		513	
5425	Main Level	370	780	560	165	520	150
Total		**11,340**	**8,140**	**6,180**	**1,382**	**7,166**	**600**

MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

Zones

- Zone 1
- Zone 2
- Zone 3

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

Dawson Deposit Longitudinal Section Facing Northwest

Scale: 1 in = 200 ft

100 0 100 200ft

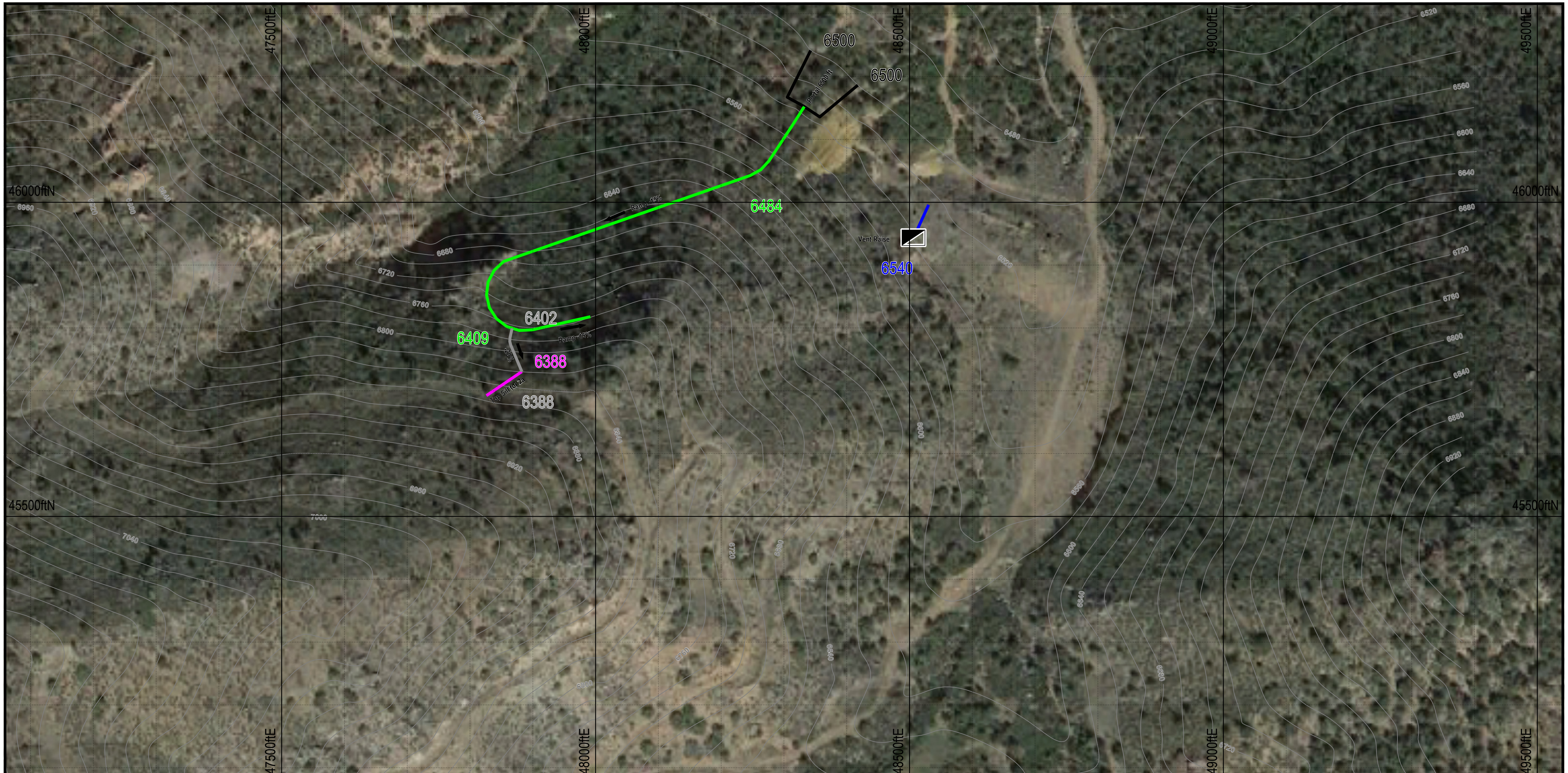
Plot Date: 10-Jun-2015

Sheet: 1 of 1

Plot File: Long Section Showing Development V2 - All Zones

Underground Development All Zones

Zephyr Gold USA Ltd
Dawson Project
Canon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

Scale	Plot Date	Sheet
1 in = 200 ft	10-Jun-2015	1 of 1
	Plot File: 6387SL	

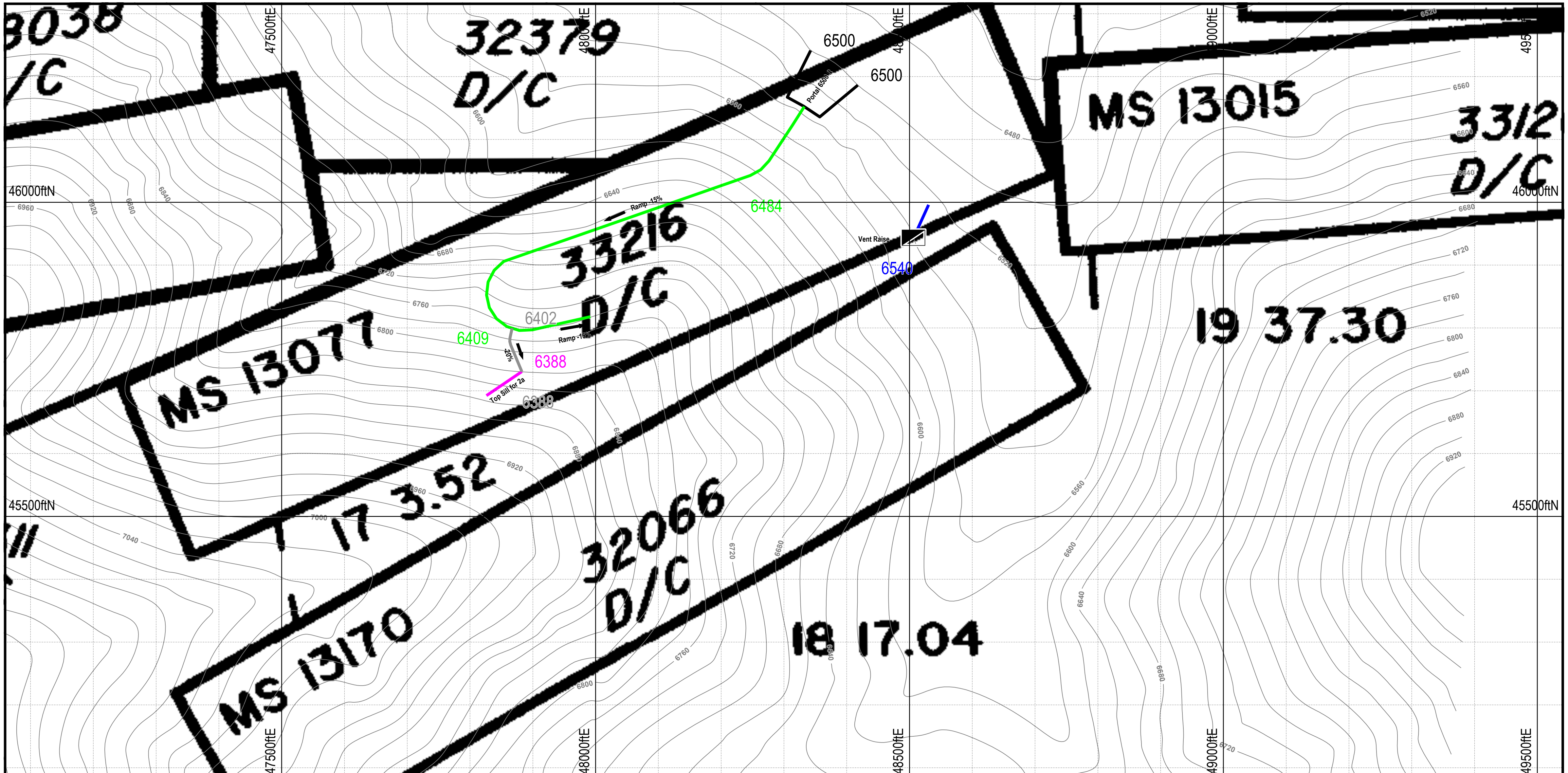
100 0 100 200ft

Level Sections Showing Underground Development

6387SL - Top Sill

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado

Showing Air Photo



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

Scale	Plot Date	Sheet
1 in = 200 ft	10-Jun-2015	1 of 1
	Plot File: 6387SL	

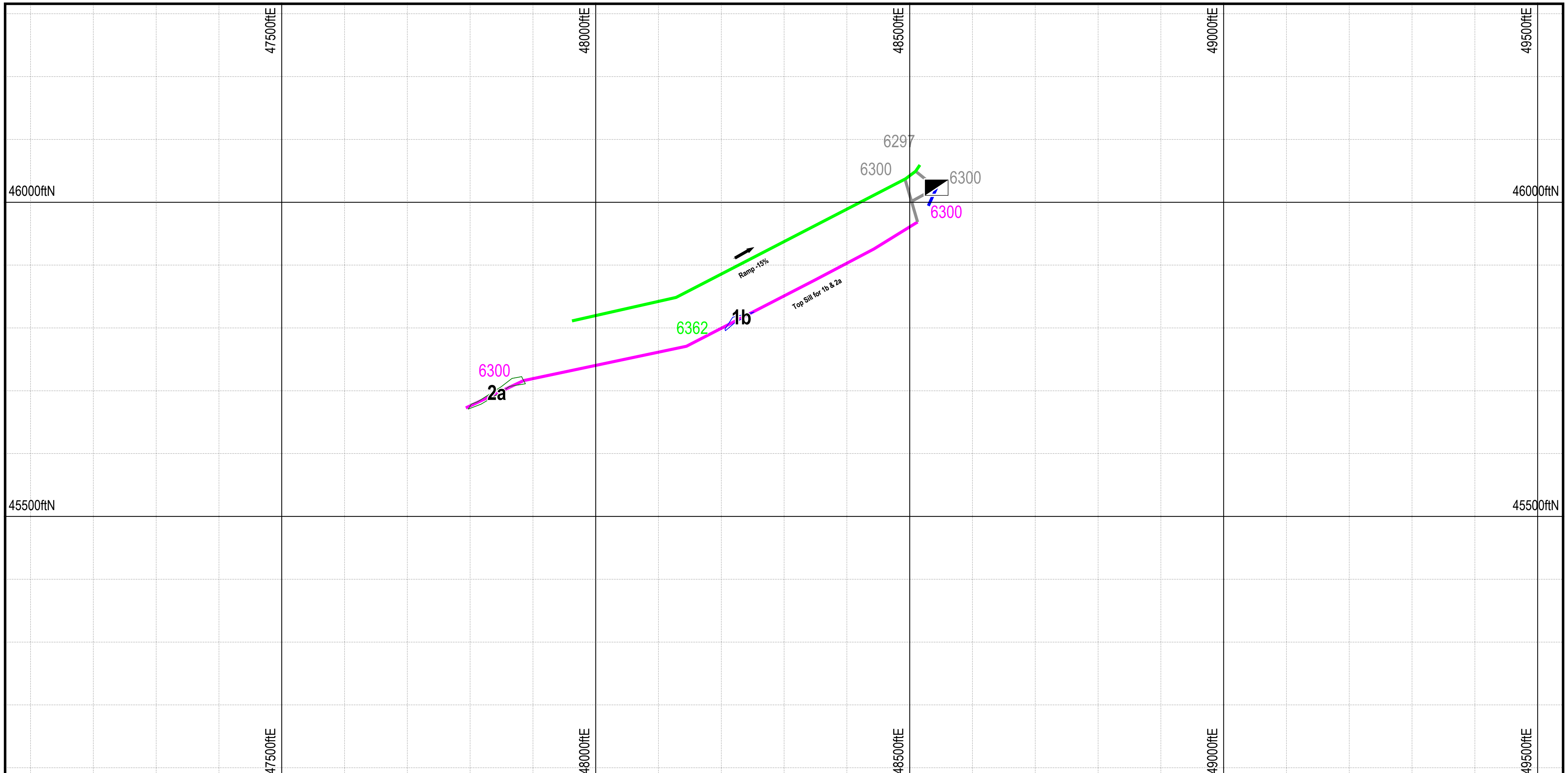
100 0 100 200ft

Level Sections Showing Underground Development

6387SL - Top Sill

Showing Claims

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

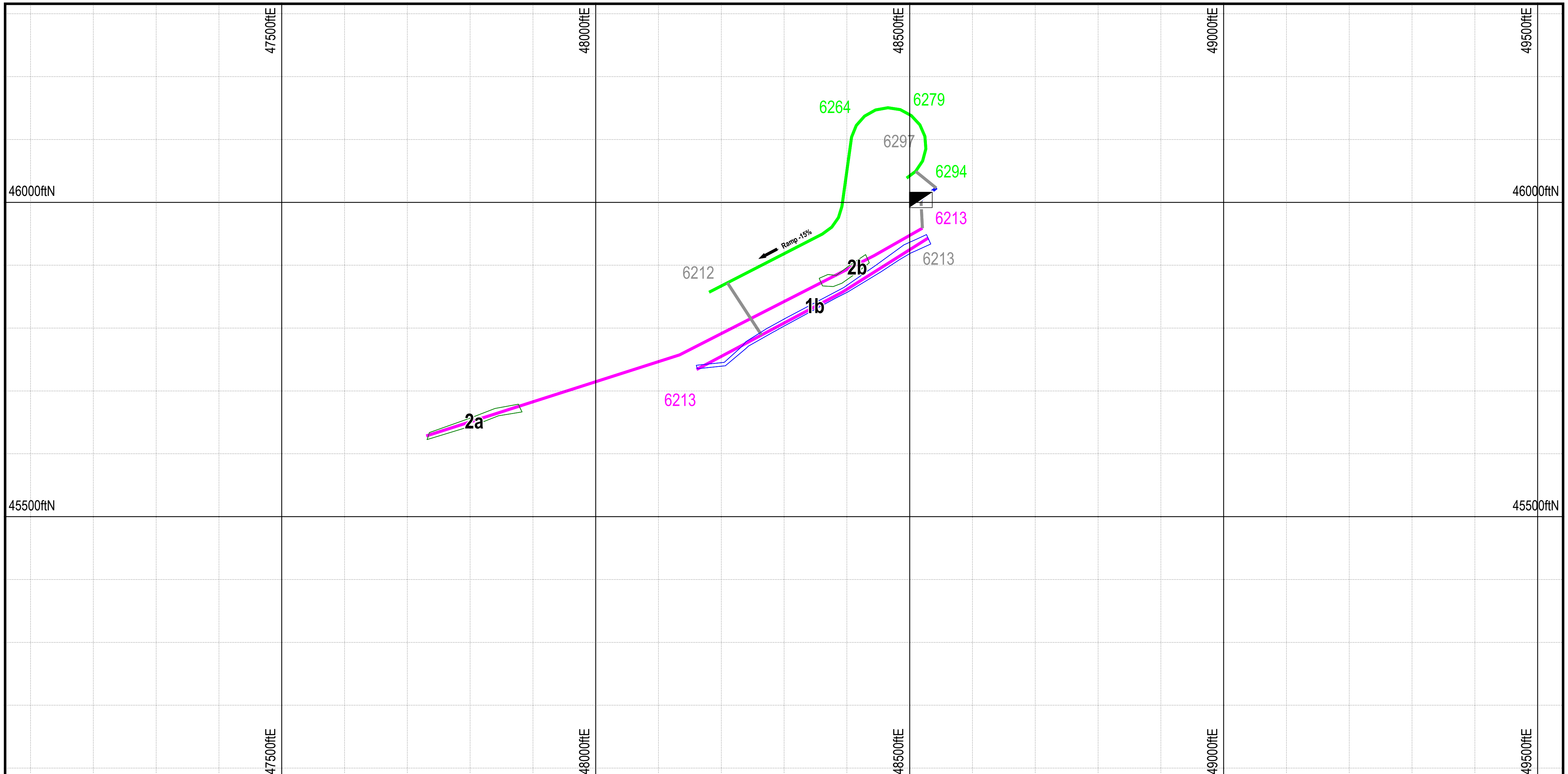
Scale	Plot Date	Sheet
1 in = 200 ft	10-Jun-2015	1 of 1
	Plot File: 6300ML	

100 0 100 200ft

Level Sections Showing Underground Development

6300ML - Main Level

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

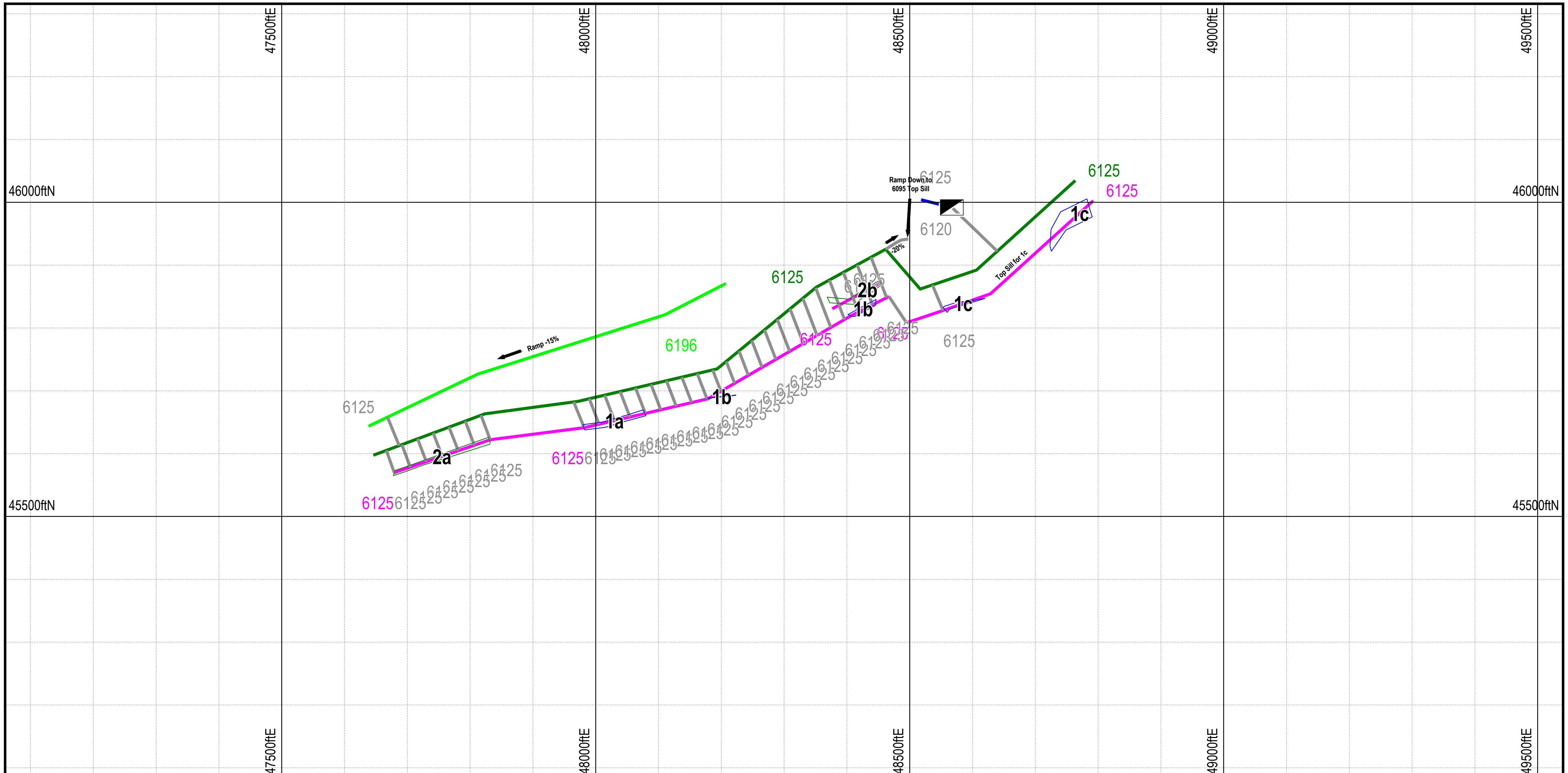
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 6212SL	

100 0 100 200ft

Level Sections Showing Underground Development

6212SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

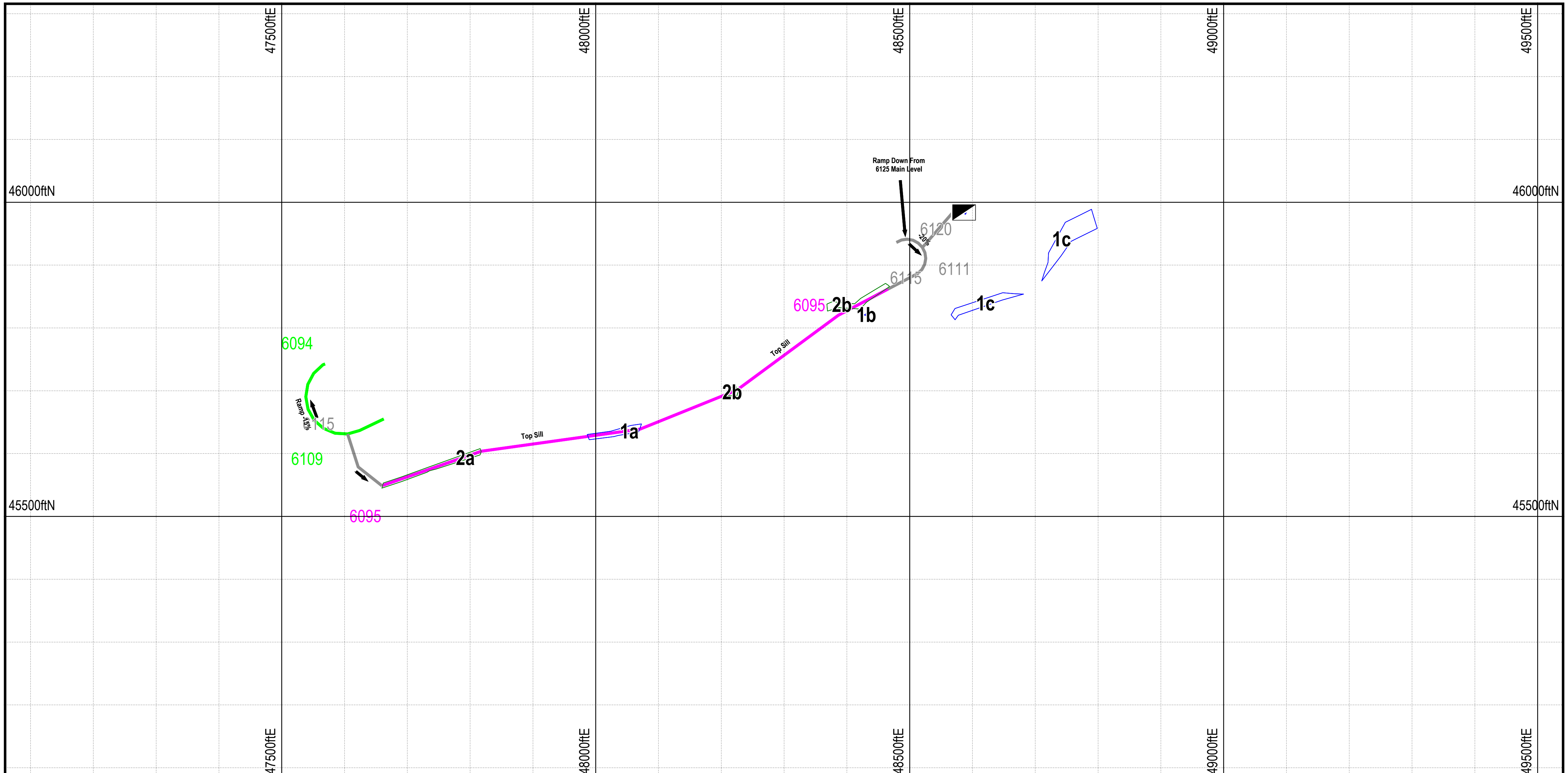
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 6125ML	

100 0 100 200ft

Level Sections Showing Underground Development

6125ML - Main Level

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

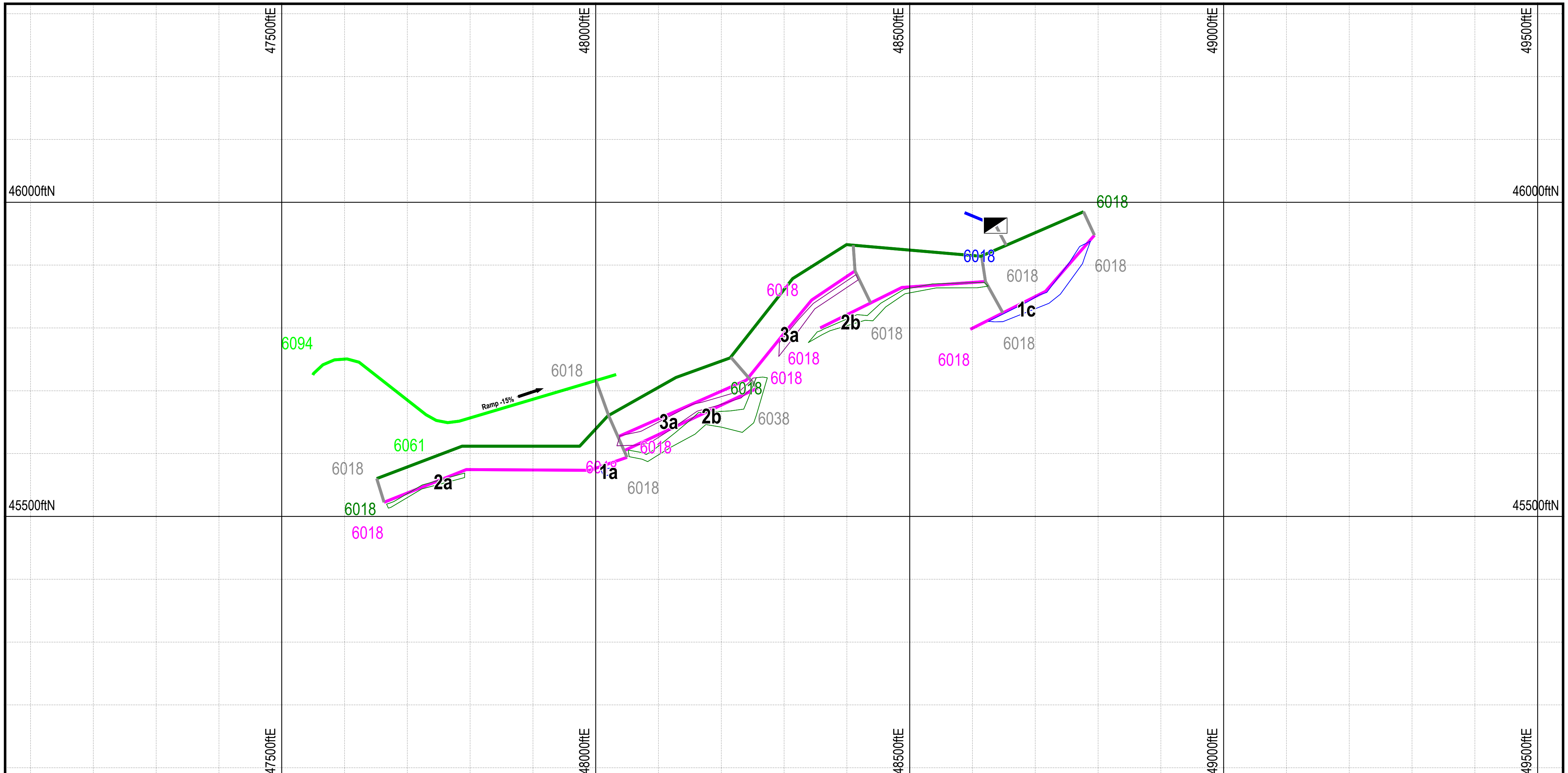
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 6095SL	

100 0 100 200ft

Level Sections Showing Underground Development

6095SL - Top Sill

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

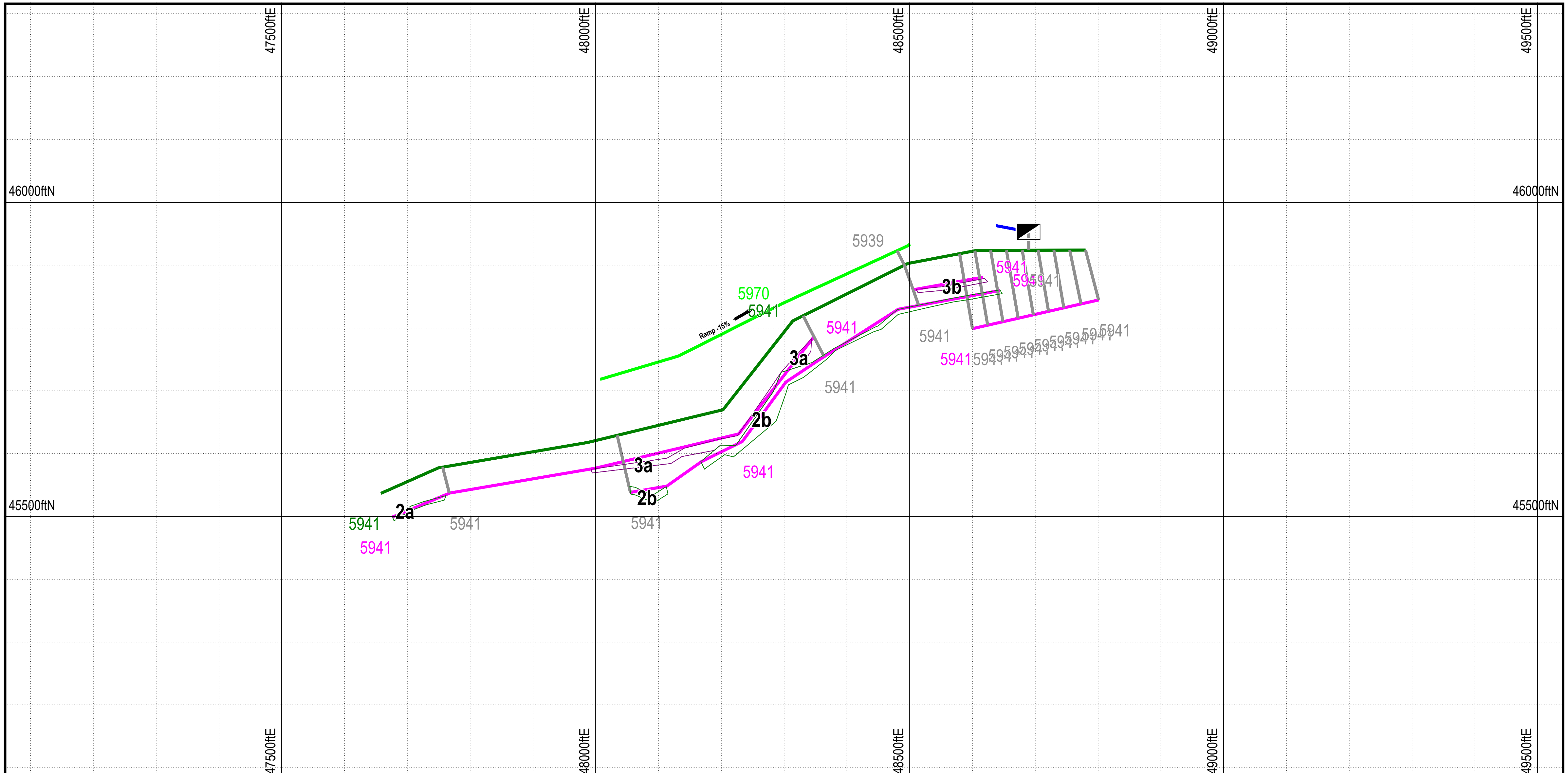
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 6018SL	

100 0 100 200ft

Level Sections Showing Underground Development

6018SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

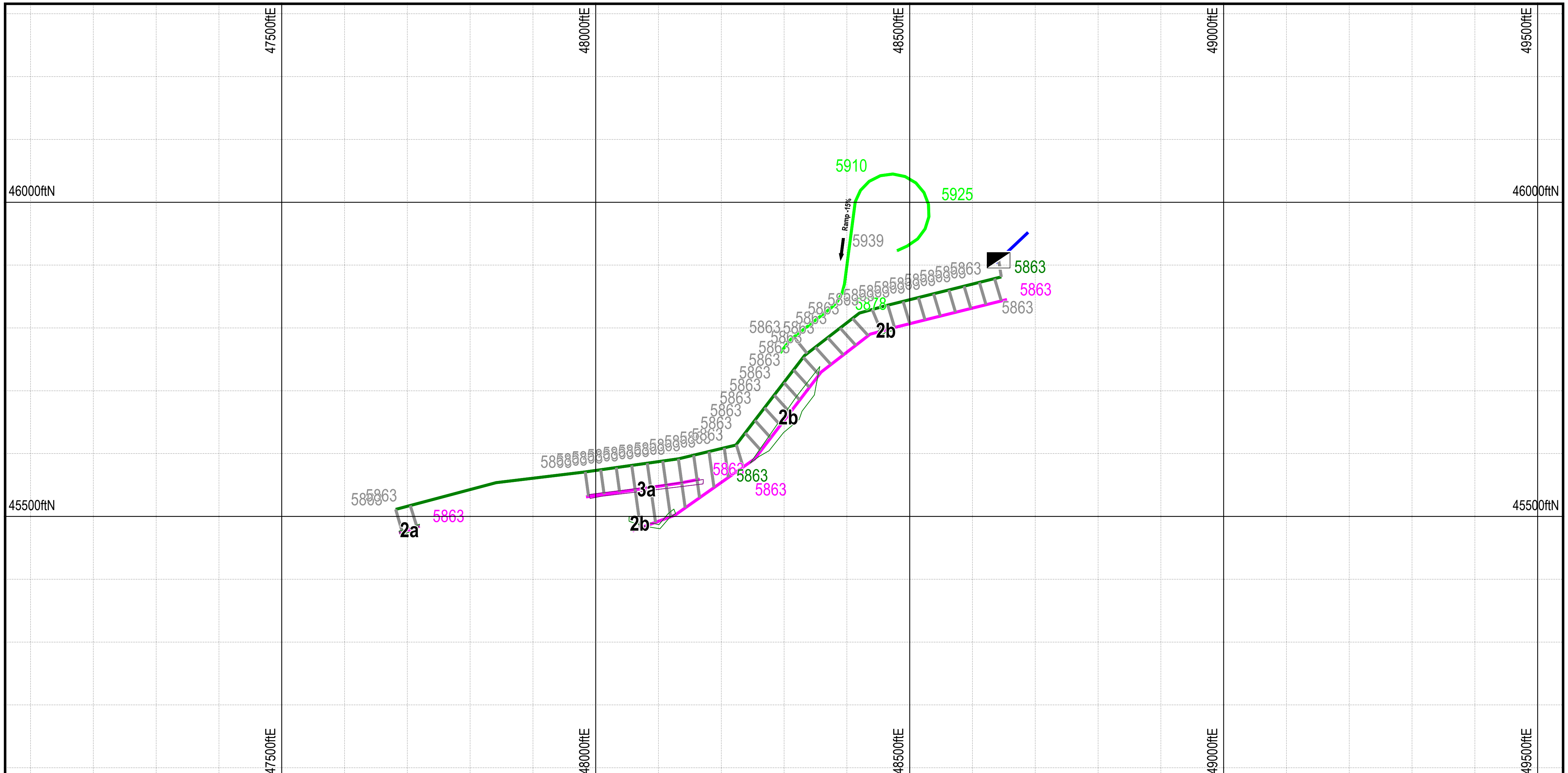
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5941SL	

100 0 100 200ft

Level Sections Showing Underground Development

5941SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

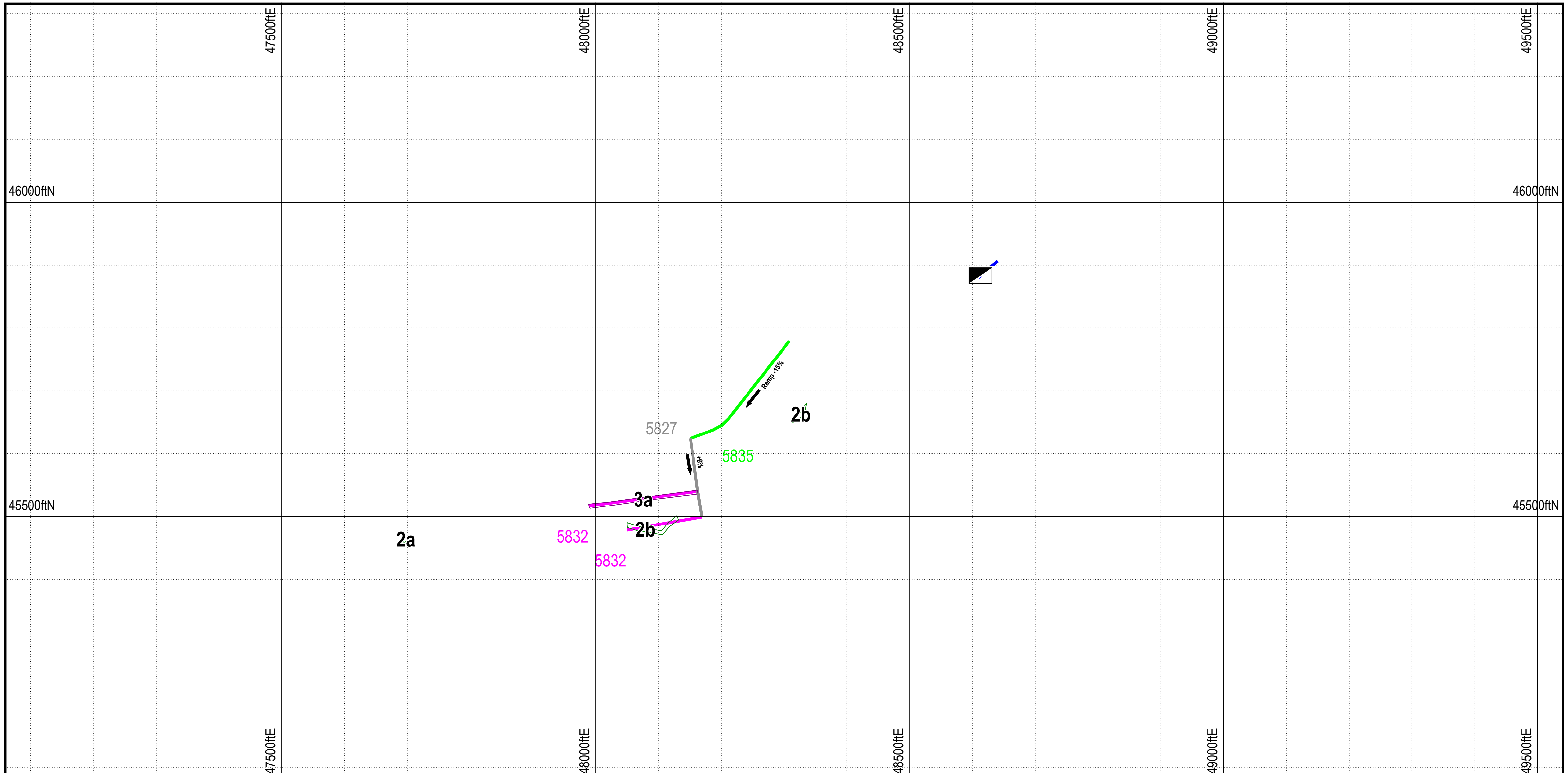
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5862ML	

100 0 100 200ft

Level Sections Showing
Underground Development

5862ML - Main Level

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

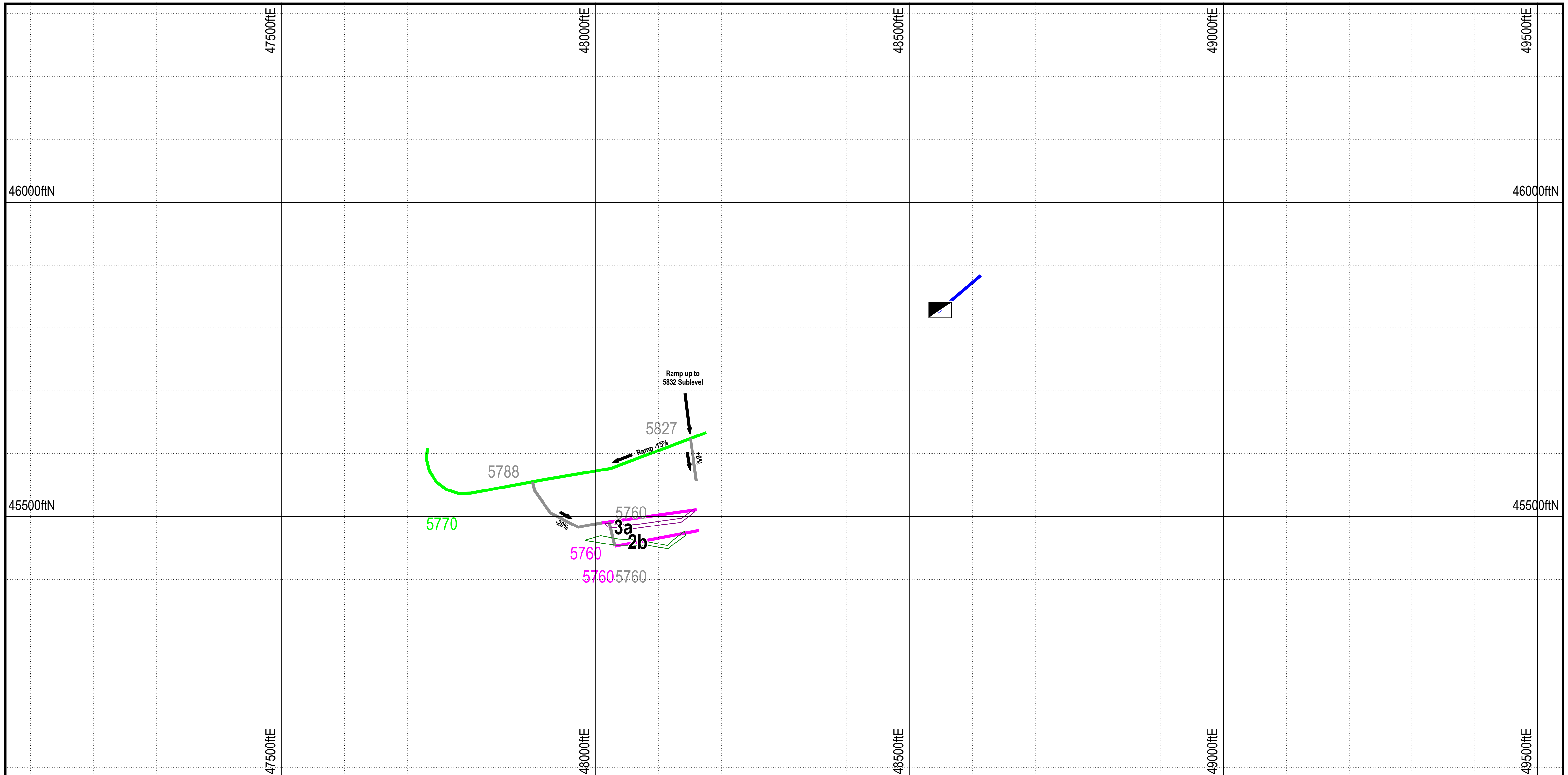
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5832SL	

100 0 100 200ft

Level Sections Showing Underground Development

5832SL - Top Sill

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings

- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5760SL	

100 0 100 200ft

Level Sections Showing Underground Development

5760SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado

MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

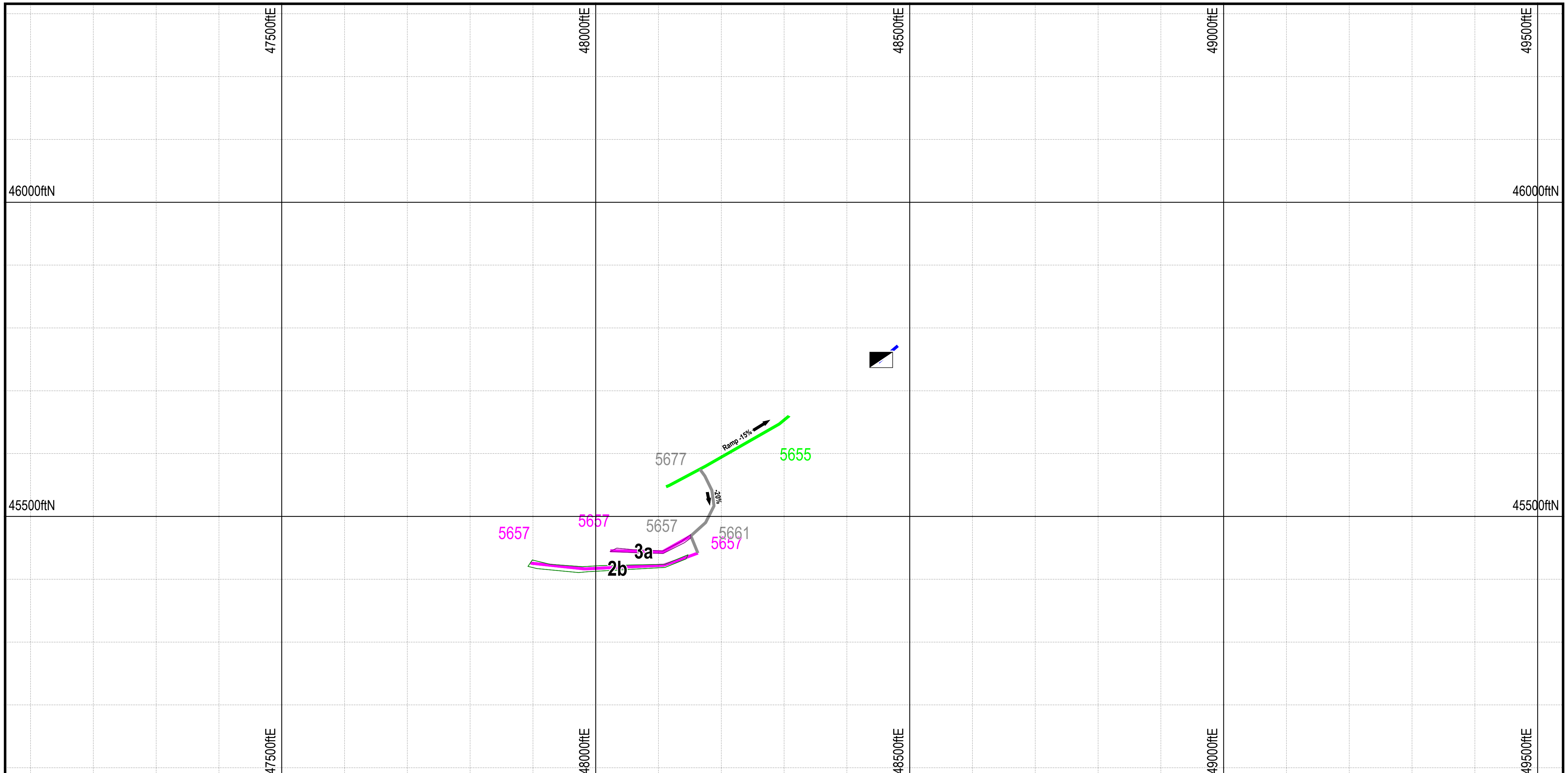
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5687ML	

100 0 100 200ft

Level Sections Showing Underground Development

5687ML - Main Level

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

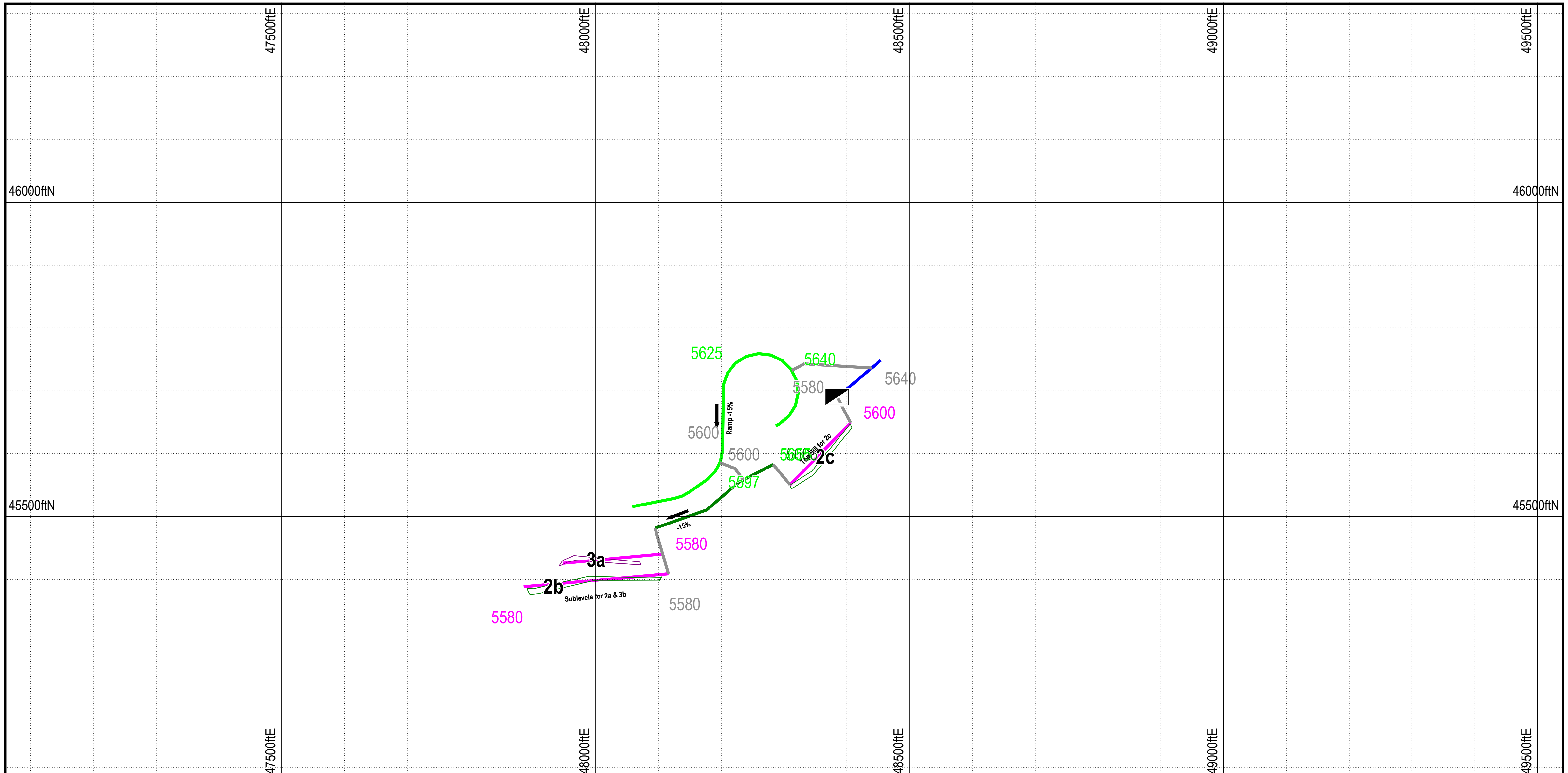
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5657SL	

100 0 100 200ft

Level Sections Showing Underground Development

5657SL - Top Sill

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MineTech International Limited
Halifax, Canada

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

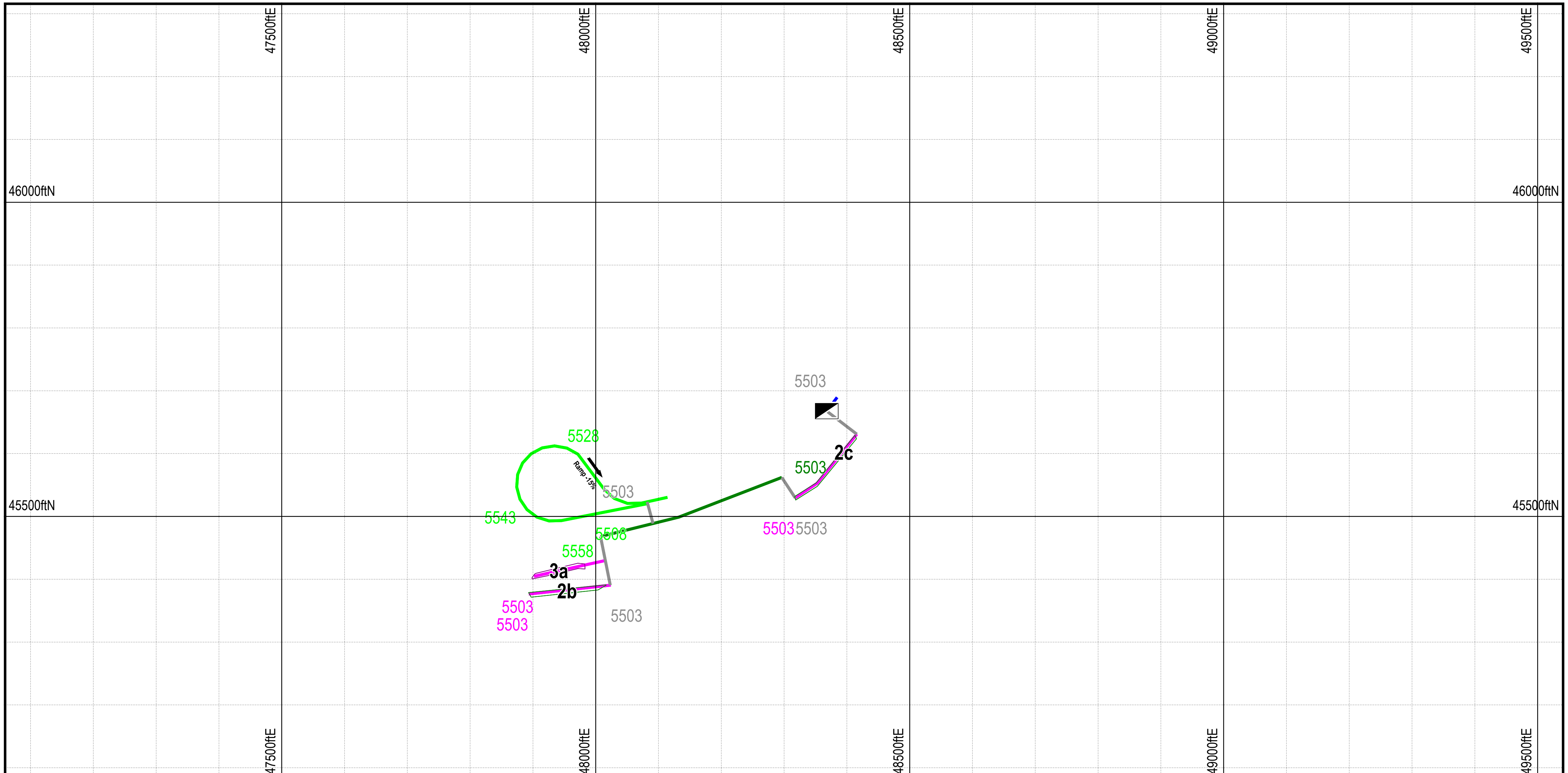
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5580SL	

100 0 100 200ft

Level Sections Showing Underground Development

5580SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
Ramp
Level
Cross-Cut
Sill Drift
Raise

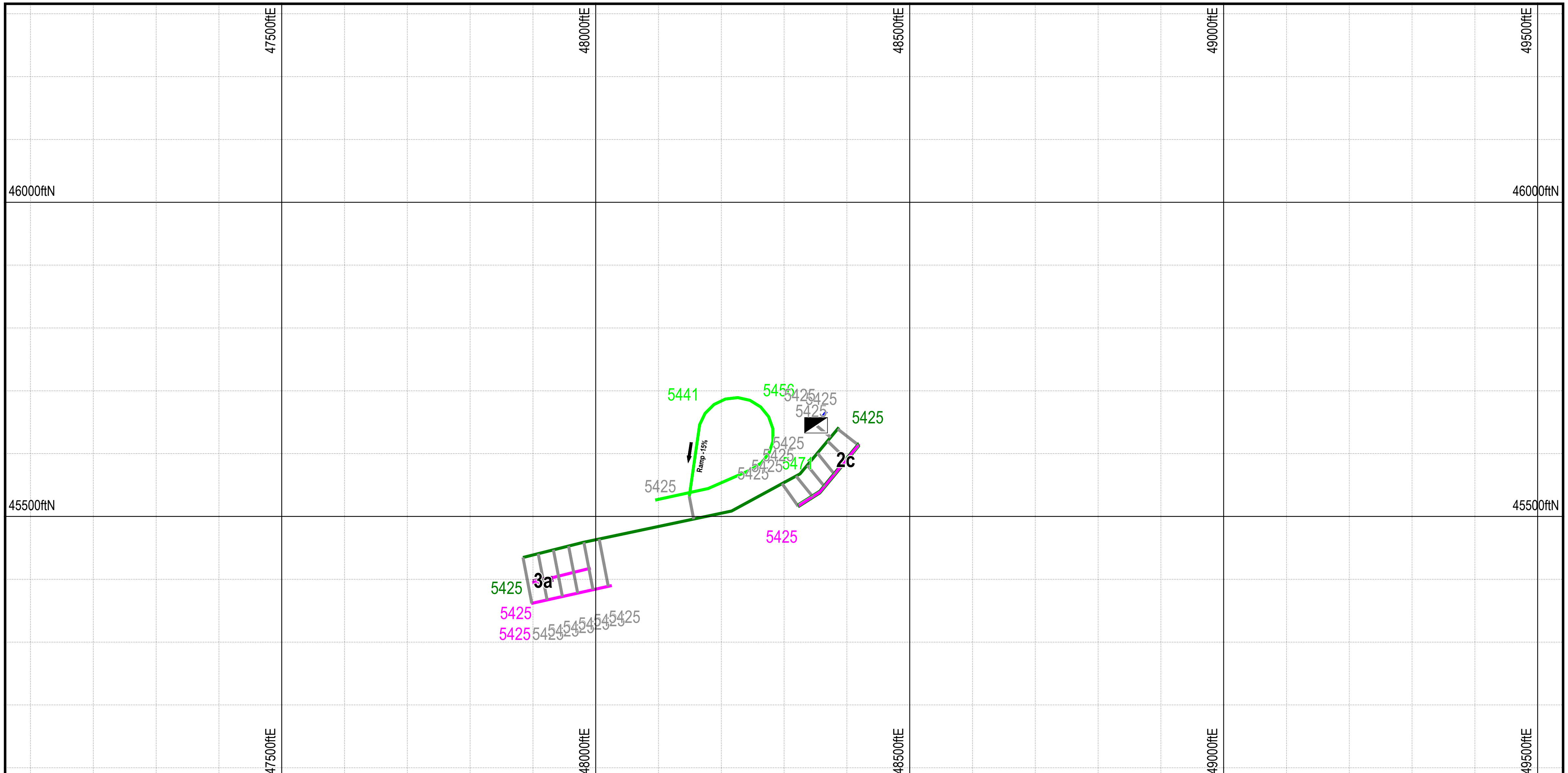
Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5503SL	

100 0 100 200ft

Level Sections Showing Underground Development

5503SL - Sublevel

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado



MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

1161 Hollis St, Suite 211
Halifax, Canada
B3H 2P6
www.minetechint.com

By: Doug Roy, MASc, PEng

U/G Workings
- Ramp
- Level
- Cross-Cut
- Sill Drift
- Raise

Scale	Plot Date	Sheet
1 in = 200 ft	22-May-2015	1 of 1
	Plot File: 5425ML	

100 0 100 200ft

Level Sections Showing Underground Development

5425ML - Main Level

Zephyr Gold USA Ltd
Dawson Project
Cañon City, Colorado