APPENDIX 2-C

Engineering Studies

Part 1 of 2

APPENDIX 2-C-1

Feasibility Design Report for

Coffee Gold Heap Leach Facility

Yukon Territory, Canada

FEASIBILITY DESIGN REPORT FOR COFFEE GOLD HEAP LEACH FACILITY YUKON TERRITORY, CANADA

Prepared for:

Kaminak Gold Corporation
800 W Pender Street, Suite 1020
Vancouver, BC V6C 2V6
Canada

Prepared by:

1325 Airmotive Way Suite 175U
Reno, Nevada 89502
United States of America

Report No. 15-78-01 Rev-A
January 2016

FEASIBILITY DESIGN REPORT FOR
COFFEE GOLD HEAP LEACH FACILITY
YUKON TERRITORY, CANADA

The following documents have been prepared by the staff of The MINES Group, Inc. (MINES) under the professional supervision of the engineer whose seal and signature appear hereon.

The findings and final design report are presented, within the limitations prescribed by Kaminak Gold Corporation, after being prepared in accordance with generally accepted professional engineering principles and practices.

Anthony E.W. Crews, P.E. January 30, 2016
Principal

TABLE OF CONTENTS

SECTION **PAGE**

LIST OF TABLES, FIGURES, AND APPENDICES

TABLES

FIGURES

APPENDICES

1 INTRODUCTION

1.1 GENERAL

The MINES Group, Inc. (MINES) was commissioned by Kaminak Gold Corporation (Kaminak) to carry out a Feasibility Study (FS) for the heap leach facility and related components of the Coffee Gold Project. Kaminak Gold Corporation (KAM) is a wholly owned resource development and gold exploration company headquartered in Vancouver, British Columbia. The Coffee Gold Project is located in the White Gold district of west-central Yukon, approximately 130 km south of Dawson City.

This report presents the results of the Feasibility Study for the heap leach pad and related facilities and was prepared in advance of the NI 43-101 technical report and following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

Figure 1-1: Project Location

1.2 PROJECT DESCRIPTION

The proposed Coffee Gold mine is located about 130 km south of Dawson City and 180 kilometers northwest of Whitehorse, Yukon Territory, Canada. Standard heap leaching technology, extensively used throughout the international mining community, is being proposed for the recovery of gold and precious metal resources. The project encloses several gold occurrences within an exploration concession covering an area of more than 600 square kilometers. The mine is expecting to process a total of 47 million tonnes of ore bearing material which will initially comprise predominantly oxide ore material (P_{80} -50mm), hauled and stacked on the lined leach pad facility in nominal 10-meter high lifts, and irrigated with a dilute solution of sodium cyanide to extract precious metals. The cyanide solutions will be fully contained within the pad area using a double layer of synthetic liners. Pregnant solutions will be gravity fed to the ADR plant via dual containment piping from the pad, with the ability to divert solutions to the lined event containment ponds in the event of an emergency or other upset condition. Following metal extraction, the barren solution will be routed to a barren tank for solution re-buffering prior to recycling to the heap for re-application.

The leach pad facility will consist of the following:

- A leach pad, constructed in multiple stages on an as needed basis, to accommodate a total of approximately 47+ million tonnes (Mt) of crushed ore. The leach pad is expandable to over 60 Mt. The leach pad containment system will consist of five layer, as follows (from the top down):
    - Overliner drainage layer consisting of a 500mm thick blanket of crushed, durable gravel and single wall drainage pipes (corrugated, perforated polyethylene);
    - Primary or top liner consisting of 2.0mm thick Liner low density polyethylene (LLDPE) geomembrane with textured bottom;
    - Secondary or bottom liner consisting of a heavily reinforced geosynthetic clay liner (GCL);
    - Bedding soil and/or prepared subgrade; and,
    - A leak detection system which reports to the solution collection ditch for sampling, visual observation and inspection. This will also provide recovery of fugitive solutions.
- Pregnant and barren solution tanks will be located at the process facility.
- Three event ponds will be constructed for the ultimate heap leach facility, two event ponds (EP-1N and EP-1S) for the initial Stage-1 pad, which will satisfy containment requirements up to the end of year 3 of operations. The third pond (EP-2) will be constructed and put into service in year 4. The event ponds shall be constructed as emergency overflow ponds designed for runoff from the active, exposed and non-reclaimed leach pad area. The event ponds include containment of the volume of estimated runoff for the Probable Maximum Precipitation event (PMP), and to contain solution drain-down from the heap in the event of an extended power outage. Solution draining from the heap can be directed to any event pond at any time, and solution stored in the EPs can be used as make-up water to the plant or heap. EP-2 should rarely see water as it is sized to contain the probable maximum precipitation. EP-1S and EP-1N will have four synthetic layers over prepared subgrade and the foundation, which are: 2.5mm thick HDPE top (primary) liner; drainage layer for leak detection and recovery (the leak detection layer will probably be combined with the

secondary liner); 1.5mm thick HDPE lower (secondary) liner; reinforced GCL. EP-2 (and the rain water pond) will have 2 synthetic layers, excluding the leak detection and recovery layer and the 1.5 mm thick HDPE secondary liner. The pond leak detection systems will report to a leak detection sump, which shall be monitored on a regular basis.
- Temporary geomembrane covers ("raincoats") will be used over the heap, beginning in year 4, to reduce the volume of meteorological water entering the process circuit. One clean water (raincoat) pond will be constructed between EP-1S and EP-1N to contain the diverted clean water for either use as make-up water or to be released to the environment after verification testing.
- An ADR process plant will be located to the east of the heap leach pad. Solutions from the leach pad and event ponds will gravity flow to solution tanks at the process plant.

Design requirements for the proposed leach pad facility are outlined in Section 7.0 of this report.

1.3 ACCESS TO PROJECT AREA

Access to the site will be from Dawson city via road and barge on the Yukon river with various crossings. An airstrip will also be constructed for personnel site access and other sundry supplies for the site.

1.4 SCOPE OF SERVICES

To complete the feasibility design for the Coffee leach pad, MINES oversaw the following scope of work.

1. Mr. Anthony Crews (MINES) visited the site to assess the field conditions and suitability of the proposed leach pad site.
2. Steffen Robertson & Kirsten (SRK) performed geotechnical drilling and test pitting to characterize the geological conditions underneath the proposed leach pad facility. The geological drill holes were logged by SRK field personnel and reviewed by MINES geotechnical engineer. Additionally, MINES reviewed the available applicable monitoring well and Kaminak core drill logs.
3. Laboratory permeability, triaxial shear testing, and classification testing were performed on soil samples obtained during the site investigation, and on proposed materials to be used in the design of the heap leach facility such as the liner interface shear testing.
4. Laboratory permeability, triaxial shear, large scale direct shear, liner puncturing and classification tests were performed on samples of spent or leached ore from of the metallurgical column testing.
5. The feasibility design for the proposed Stage-1 leach pad facility included sizing and grading for the ultimate pad capacity of 60 Mt and reduced to the 47 Mt capacity for the feasibility study. Design for Stage-1 leach pad was performed, including sizing and layout of the pad cells and phases, event ponds, sizing and layout of the solution collection piping and ditches, layout of leak detection systems underneath the solution collection cell divider ditches and the event ponds, as well as the sizing and layout of storm runoff water diversion on the heap leach pad. SRK was tasked with the management system for site-wide surface waters and related sediment control structures.
6. The feasibility design for the leach pad facility is presented on a set of drawings included in

Appendix J at the end of this design report. The drawings were prepared using AutoCAD Release 2016 format.

7. Construction quantity estimates and a capital cost estimate to construct the Stage-1 facility were prepared from the set of drawings and will be submitted as a separate item.
8. The feasibility design for Stage-1 leach pad is summarized in this report.
9. The water balance study was included and expanded to include the use of raincoats for solution management during operations and for final closure of the heap leach facility.
10. A detailed seismic evaluation was also included in the scope of work.

The following tasks were not included MINES scope of work:

- Design of the leach pad loading system
- Design of the ADR processing plant
- Design of pumping and piping systems to and from the leach pad.
- Design of sprinkling systems on the heap.
- Design of crushing and process facilities.
- Design of electrical or mechanical aspects of the project
- Power supply
- Infrastructure
- Design of haul and access roads for the leach pad leach facility.

2 AVAILABLE DATA

2.1 TOPOGRAPHY

The regional study area is characterized topographically by rolling hills with elevations ranging from 850 m above sea level (ASL) at the base of the NW waste rock dump to 1370 m ASL on the hilltops adjacent to the west side of the heap leach pad facility.

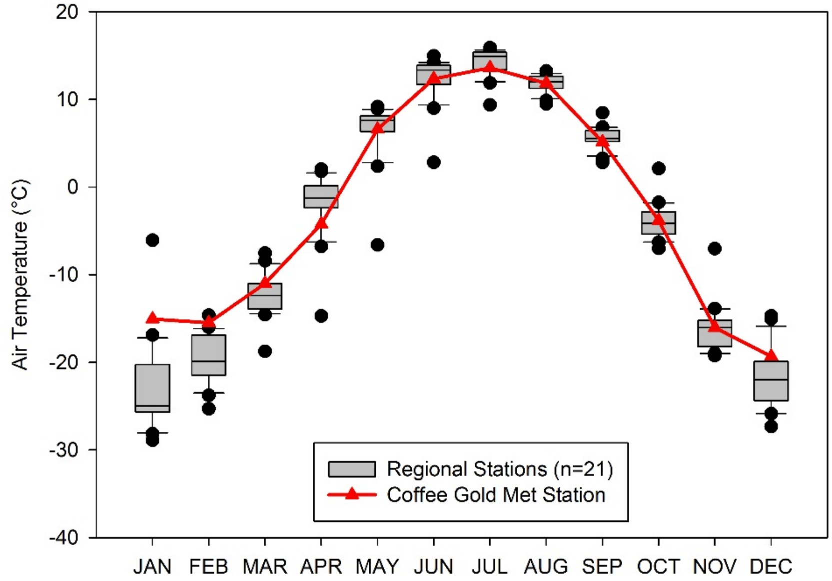
2.2 METEOROLOGY AND CLIMATE

This section summarizes data assimilated and reported by Lorax Environmental Consultants (Lorax) (Lorax Environmental Consultants, 2016)and is also used in the water balance report in Appendix G. A more detailed report by Lorax Environmental Consultants is available as a separate report.

2.2.1 Air temperature

Mean annual air temperatures in the region range from -5.2 to -2.8°C. the coldest month is typically January, with the regional mean temperature of -23°C, and the warmest month is July, with a regional mean of 13.1°C. Monthly average daily minimum, mean and maximum temperatures at the Coffee Gold Project site are generally consistent with the regional signal, with notable exception of January to March, where the project site temperatures are higher than the regional stations (see ure 2-2 and Figure 2-3). This is likely due to valley inversions, as Coffee Gold Project site is consistently warmer than the regional stations in the winter.
These inversions result in a reversal of the normal lapse rate (decreasing temperature with increasing elevation), and are commonly caused by cold Arctic air masses pooling in the valley bottoms from late October to early March. Following an inspection of site- and regional climate data, separate lapse rates are recommended by Lorax for the winter and summer seasons to account for the dominant inversion conditions during the winter at the project site. The lapse rates are not used in the heap leach pad design or as part of the criteria.

Figure 2-1: Monthly average temperatures from the surrounding climate stations (period of record), and from Coffee Gold (2012-14)



ure 2-2: Average daily minimum temperatures for the Coffee Gold Project site (2012-2014) and representative regional climate stations (1981-2010 normals).

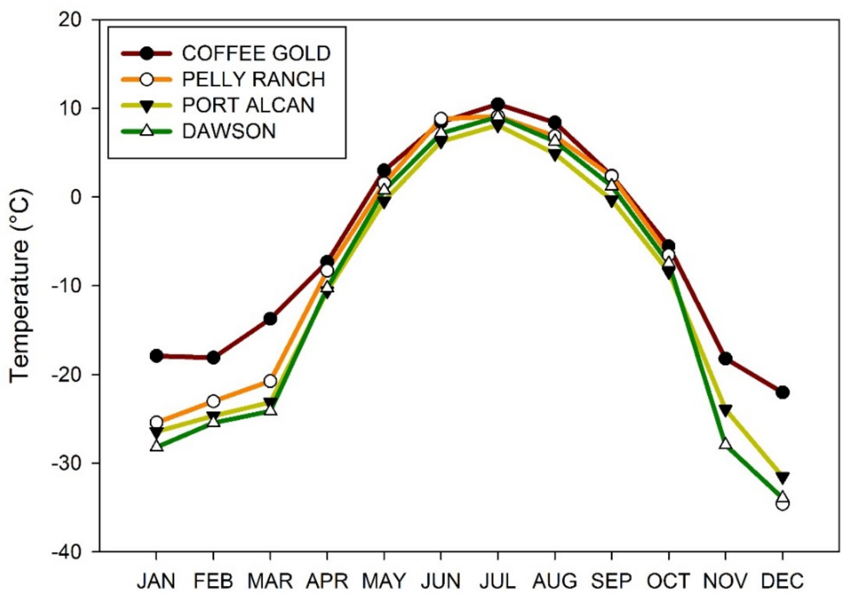


Figure 2-3: Average daily mean temperatures for the Coffee Gold Project site (2012-2014) and representative regional climate stations (1981-2010 normals).

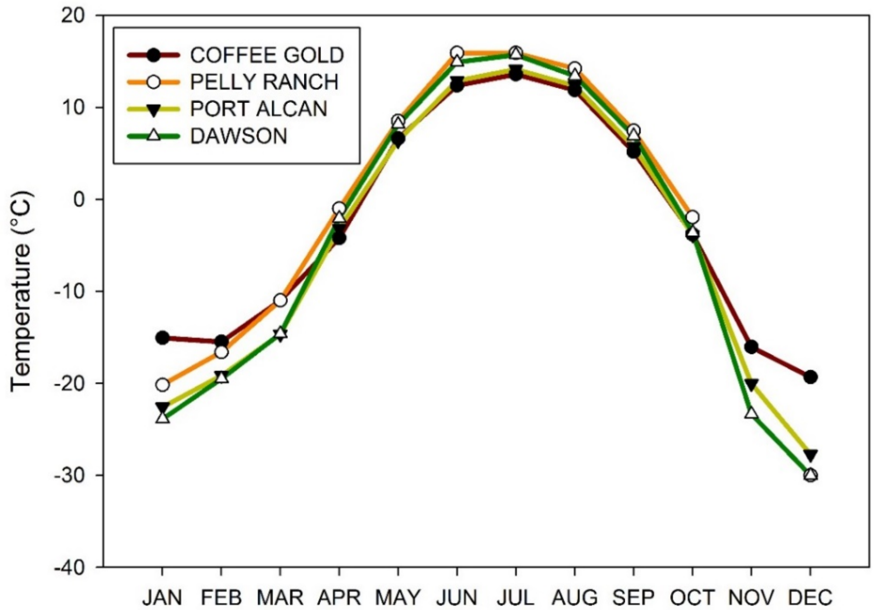
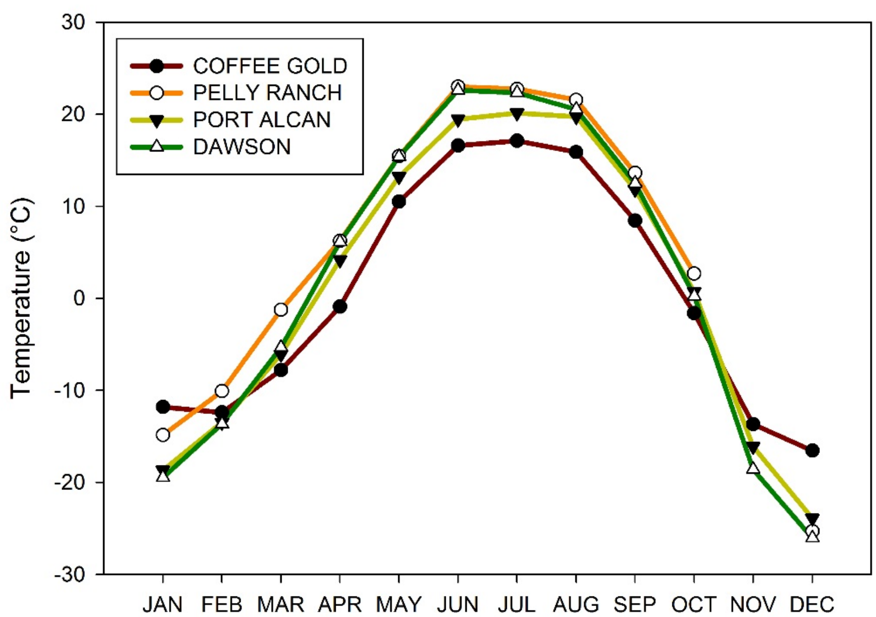


Figure 2-4: Average daily maximum temperatures for the Coffee Gold Project site (2012-2014) and representative regional climate stations (1981-2010 normals).



2.2.2 Precipitation

The study area is characterized by a cold, continental climate. Large frontal systems generated by the Aleutian Low are blocked by the high topography along the coast, and hence the project lies in the leeward rain-shadow (Figure 2-5). Annual precipitation in the study area is dominated by summer rainfall, with relatively lower amounts falling as snow during the winter, averaging 65% and 35% of the annual total, respectively (Figure 2-6).

Figure 2-5: Annual precipitation (1980-2009) for the Yukon River Basin.

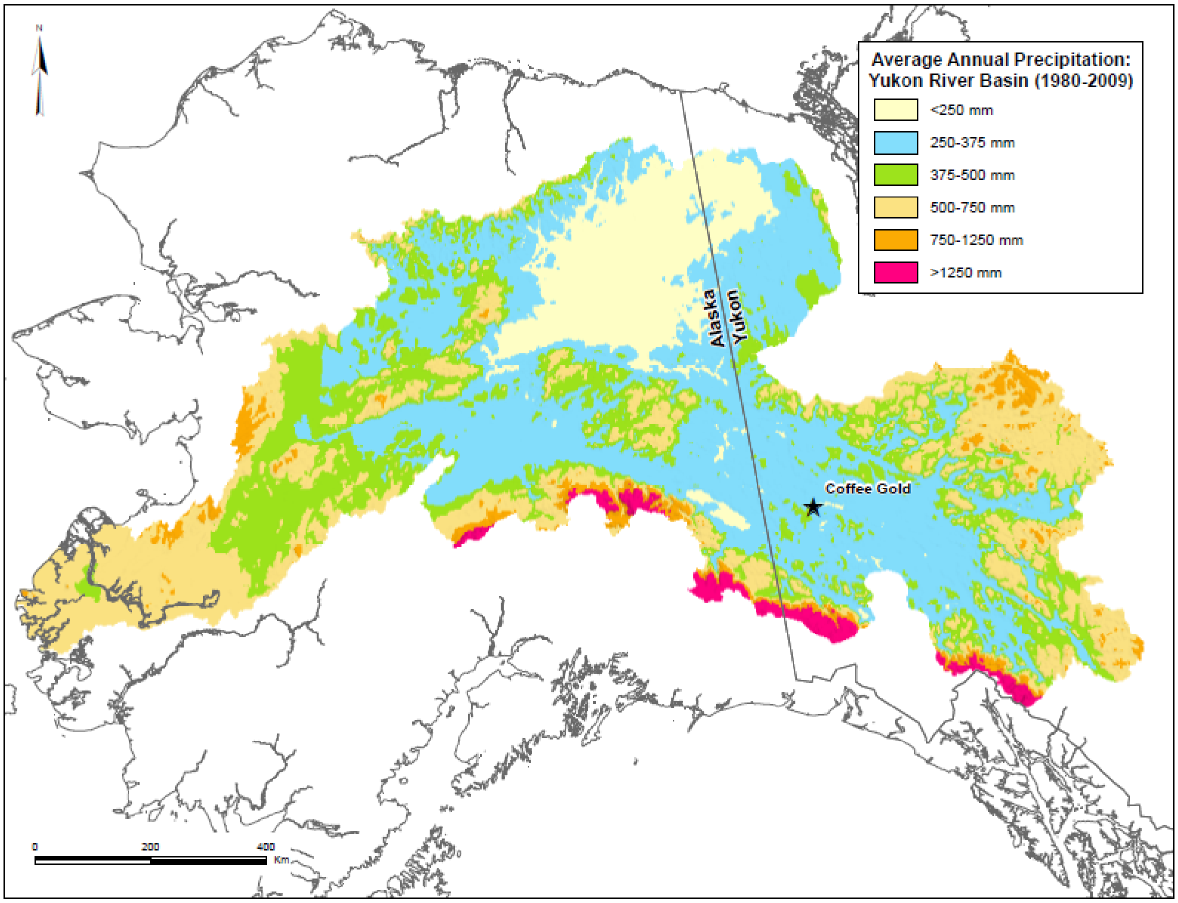
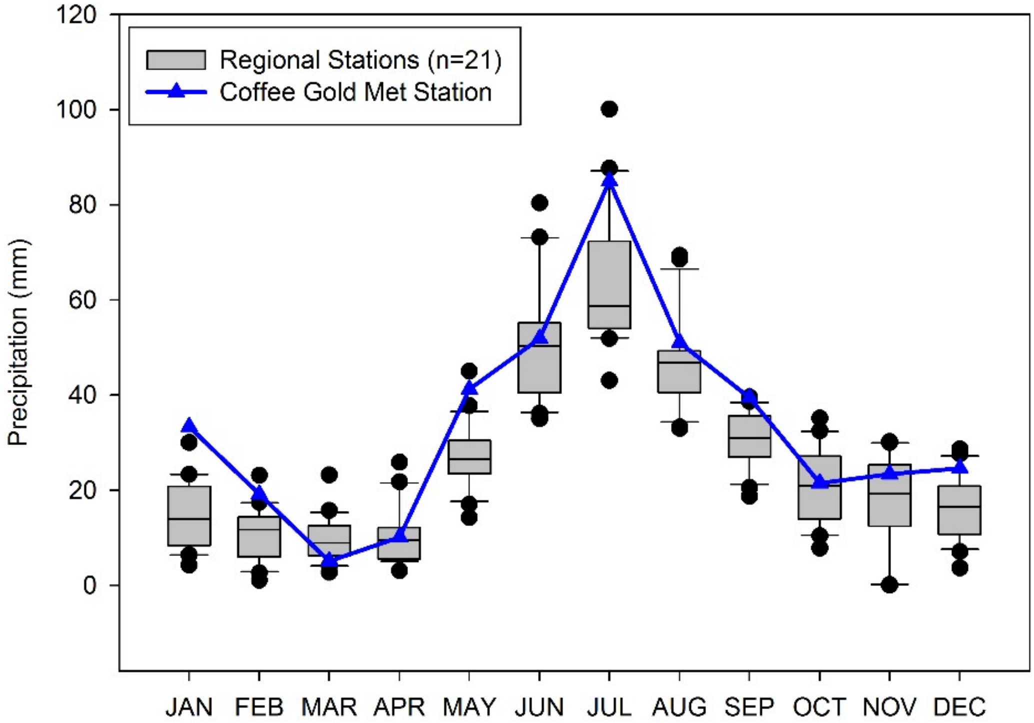


Figure 2-6: Monthly average precipitation from the surrounding climate stations (period of record) and from Coffee Gold (2012-14).



Precipitation has been measured at the Coffee property since July 2012, and in this time, has averaged 406 mm/year. Approximately 65% of the total precipitation falls as rain during the May-September period, and 35% as snow, as measured by the precipitation gauge and the site snow surveys located near the topographic highpoints (Figure 2-6 and Figure 2-7). Precipitation data collected at site is compared to regional data for corresponding time period in Figure 2-8 and to long-term average (1981-2010) precipitation in Figure 2-9.

The 2012-2014 study period was wetter at both regional stations analyzed (Pelly Ranch and Dawson), by 29 mm on average (annual). The intra-annual signal varies, both between months, and between stations in the same month, with regional stations occasionally showing deviations of opposing signs (for May and June in particular). On average, January was wetter (~15 mm more precipitation than the 1981-2010 normal), February, May and June being slightly drier (0.2 – 4 mm less), and March, April and July to December 1.5 – 6 mm wetter (see again Figure 2-10).

Figure 2-7: Daily precipitation totals measured at the site climate station.

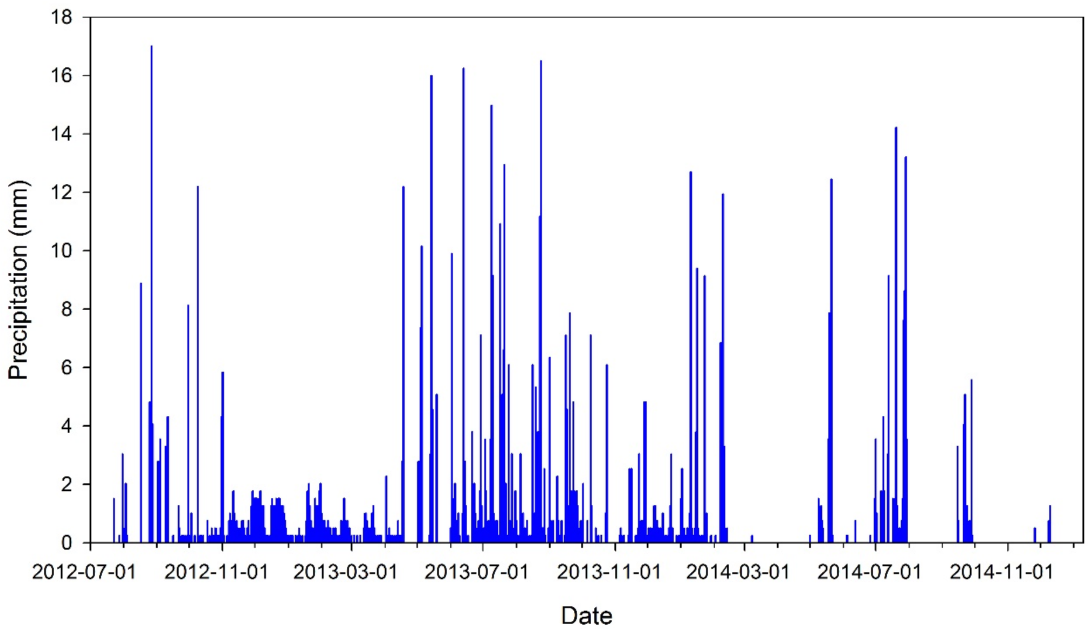


Figure 2-8: Monthly average precipitation measured at site and regional climate stations in 2012-2014.

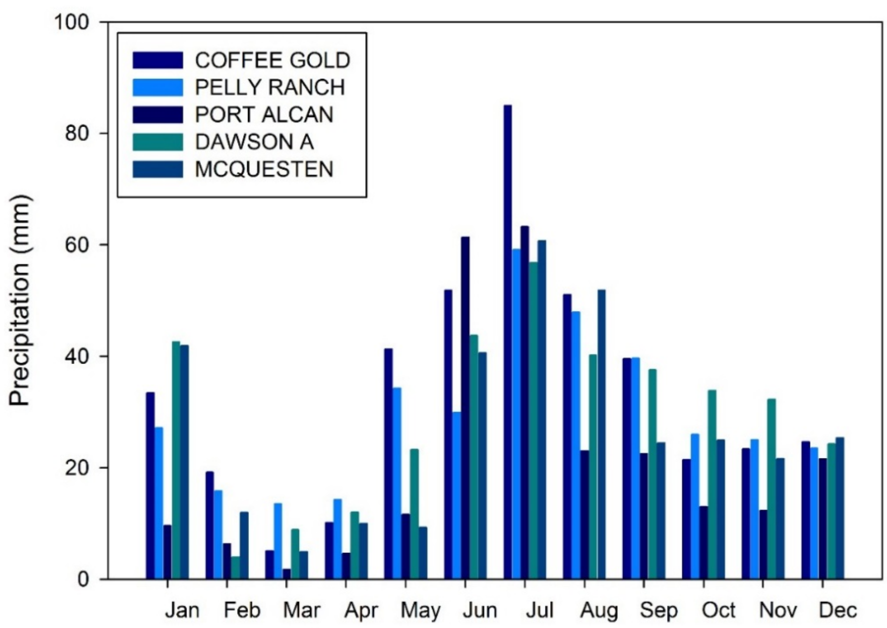


Figure 2-9: Monthly average precipitation measured at site (2012-2014) and regional climate stations (long-term average).

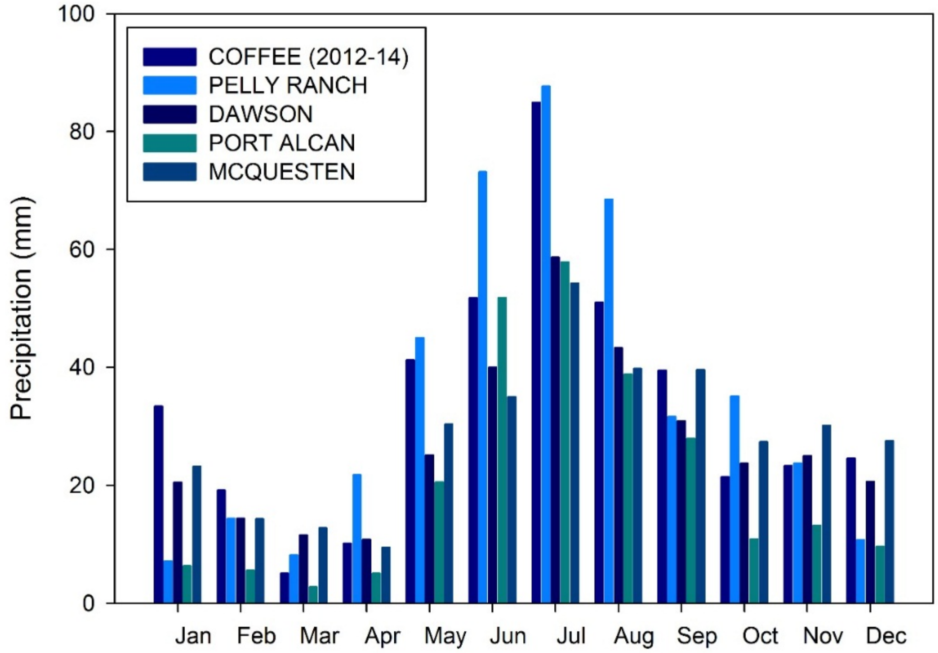
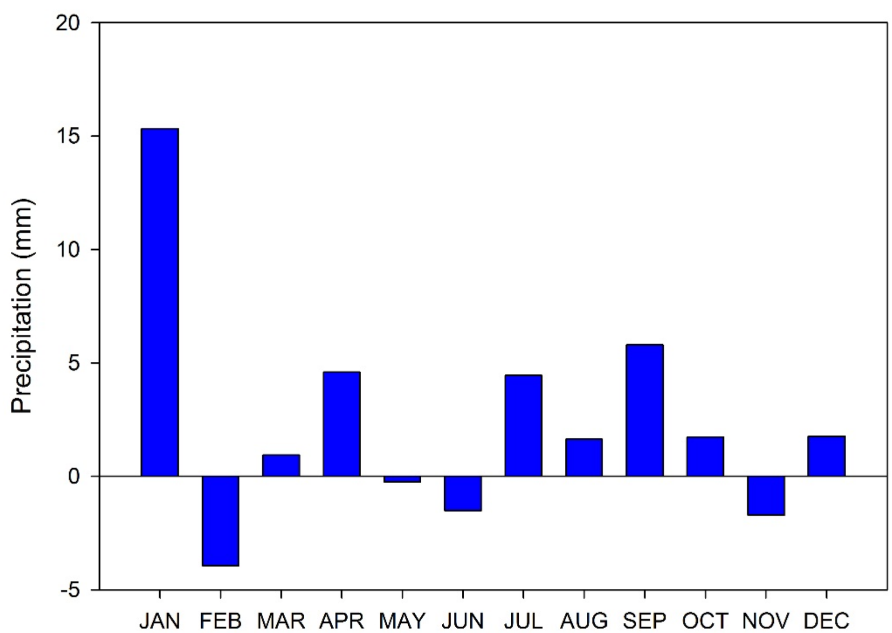


Figure 2-10: Regional average deviation of the 2012-2014 period from 1981-2010 normal precipitation, as measured at the Dawson and Pelly Ranch climate stations.



2.2.3 Snow Water Equivalent – Site Data and Regional Setting

The Casino Creek, CC-east, -north, -south and –west snow courses are aggregates of between two and three snow courses that were sampled within 200 m of each other. The SWE values presented below are averages of these groups, organized by site aspect. These data are presented in Table 2-1.

The Kaminak, KAM-SS series stations were initiated in 2014, and there is no spatial overlap between these sites and the sites previously surveyed in 2013 and 2014. The exceptions are stations IC-1.5/0.5-SS and KAM-SS-T1, which are co-located.

Sufficient data do not exist to determine when the peak snowpack occurred in 2012 at all sites, but the CC-east group recorded the maximum annual SWE at the end of March. In 2013, the maximum SWE occurred in early March at the high-elevation sites, and in mid-February at the one low-elevation site with sufficient data (IC-1.5/0.5 –SS). Only one survey (KAM-SS-T1) was conducted in 2014 with sufficient temporal coverage to define the occurrence of maximum SWE, which occurred in early April.

Table 2-1: Coffee Gold Snow Survey Sites (all values are SWE in mm)

	Elevation (m asl)	2012		2013			2014			
Station		**Mar**	**Apr**	**Feb**	**Mar**	**Apr**	**Jan**	**Feb**	**Mar**	**Apr**
IC-4.5-SS Flat	399	123	--	100	--	113				
IC-1.5 / 0.5 -SS	524	143	--	157	117	97				
KAM-SS-T1	528						71	55	83	84
CC-1.5-SS	733	97		63						
KAM-SS-EAST	987									78
KAM-SS-NORTH	994									77
KAM-SS-WX	996									88
KAM-SS-WEST	1020									76
KAM-SS-HIGH	1184									117
CC-east	1203	142	101	117	136	108				
CC-north	1224	--	156	110	170	153				
CC-south	1253	160	--	122	130	110				
CC-west	1256	--	100	--	132	108				

The Yukon territorial government maintains an extensive snow survey network for flood and water supply forecasting purposes (http://www.env.gov.yk.ca/air-water-waste/snow_survey.php). The nearest representative high-elevation station from this network is located at Casino Creek (09CD-SC01; 1065 m – see Table 2-2) and has a record period spanning 1977-2014. This site is surveyed during a one-week window centered on the first of the month in March, April and May, and occasionally on May 15th, depending on snowpack magnitude and flood risk. The nearest station that can be considered representative of valley bottom snowpack accumulations is located at Pelly Farm (09CD-SC03; 472 m; Table 2-2).

The Casino Creek snow survey site, average maximum SWE is 142 mm over the period of record, with approximately half of the annual maximum snowpack occurrences recorded in early April, and the other half in early May. In almost all years with available data, the annual snowpack melts out completely by May 15.

Table 2-2: Regional Snow Survey Sites

Station Name	ID	Elevation	Dist. From Project	Record Period	Province/ State	Latitude	Longitude	Mean Annual
		(m asl)	(km)			(°)	(°)	SWE (mm)
CASINO CREEK	09CD-SC01	1065	32	1977-2014	YT	62.7333	-138.8	142
PELLY FARM	09CD-SC03	472	81	1986-2014	YT	62.8167	-137.3667	82

2.2.4 Wind

The mean wind speed for April to September for the period of record (2012-2014) was 2.54 m/s, with calm conditions only recorded 1.35% of the time (Figure 2-11). The primary wind vector was from the northwest, with a secondary vector from the south.

The mean October to March wind speed was 1.9 m/s, with calm conditions recorded 8.1% of the time (Figure 2-12). The strongest winds were recorded blowing from the south, but with relatively greater frequency from the northeast.

The annual average wind speed for the period of record is 2.34 m/s, with a maximum recorded gust speed of 18 m/s.

Figure 2-11: Wind rose for the April to October period (2012-2014) for the Coffee Gold Project site. (Flow vectors are in the direction that winds are blowing to).

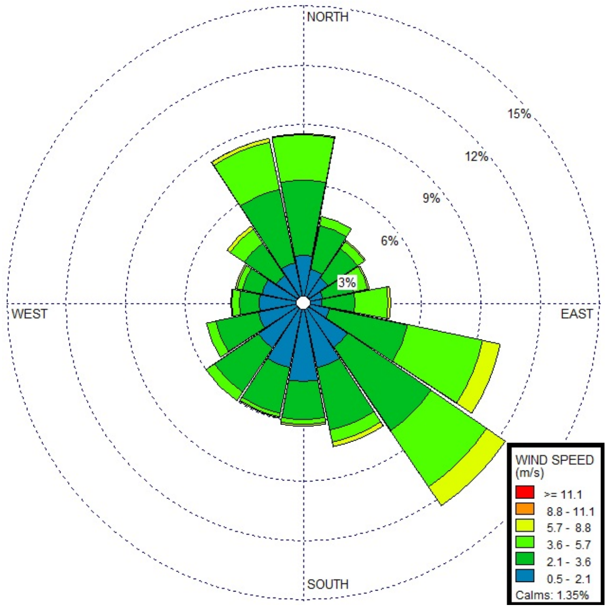
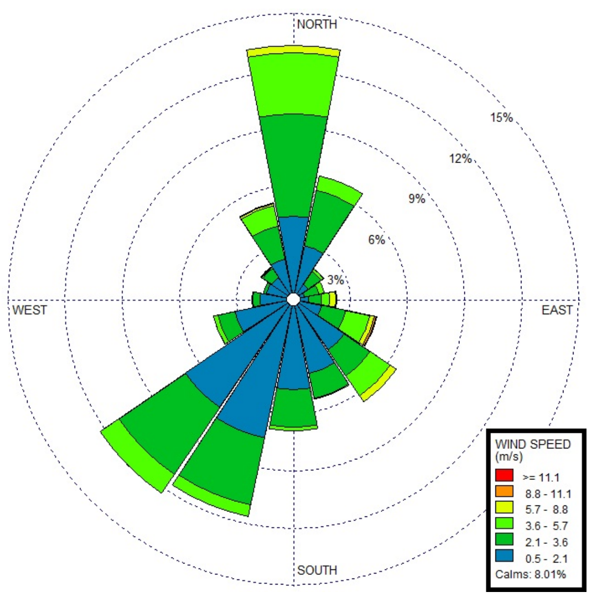


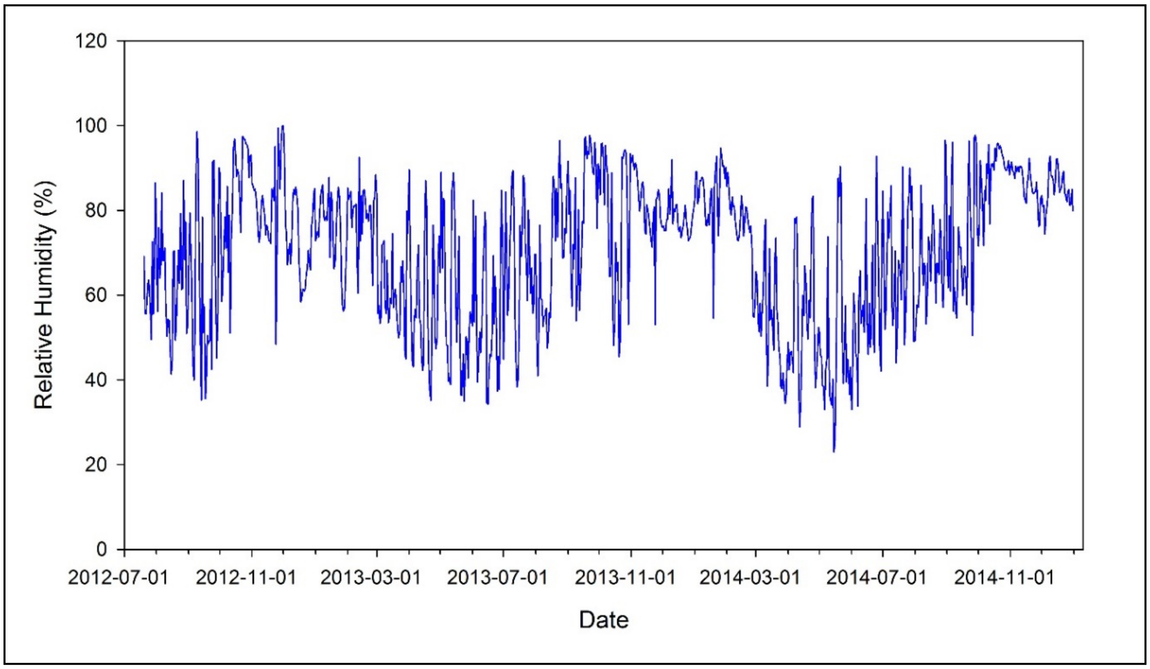
Figure 2-12: Wind rose for the November to March period (2012-2014) for the Coffee Gold Project site.



2.2.5 Relative Humidity

Monthly average relative humidity measured at the site climate station varies from lower values in the spring months (lowest is May; 53%) to higher values in the fall and winter of 76-80%. A relative humidity value of 100% was measured on 84 occasions (hourly), and the minimum measured value was 12.8% on May 31, 2014. The Coffee Creek relative humidity record is shown in Figure 2-13

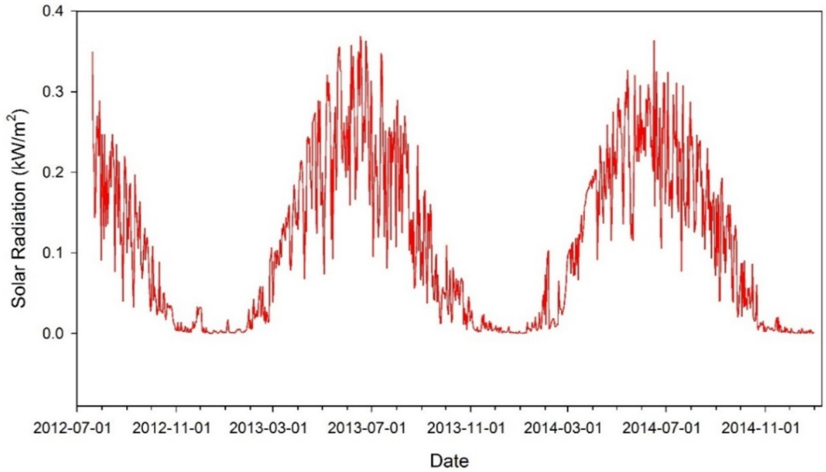
Figure 2-13: Mean and Minimum Relative Humidity



2.2.6 Solar Radiation

Solar radiation varies from a monthly average high in June of 0.26 kW/m^2 to a low of 0.004 kW/m^2 in December, with the highest recorded hourly value of 0.99 kw/m^2. The Coffee Creek solar radiation record is shown in Figure 2-14.

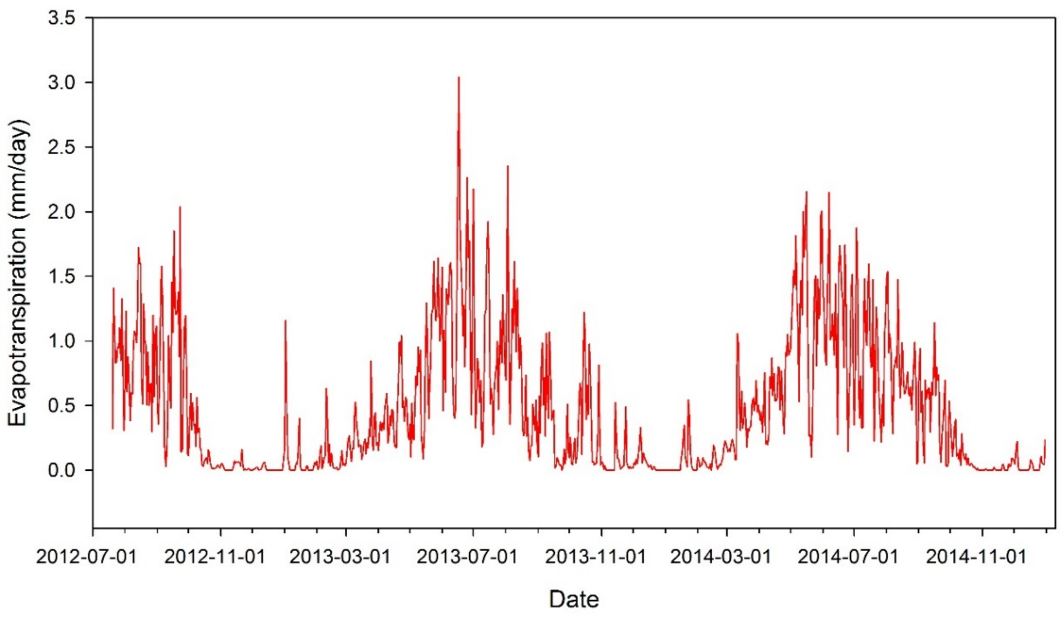
Figure 2-14: Daily average solar radiation measured at the site climate station.



2.2.7 Evaporation

Figure 2-15 shows the site evapotranspiration record for the available period of record (*i.e.*, July 2012 to December 2014). For the May-Sept, average evapotranspiration is estimated to be 180 mm for the site.

Figure 2-15: Daily evapotranspiration calculated from measured meteorological data at the site climate station.



2.2.8 24-Hour Precipitation Return Periods

Lorax evaluated the precipitation return periods and exceedance probability for the Coffee site for the 24-hour precipitation event (Lorax Environmental Consultants, 2016). The summary results are shown in Table 2-3 below.

Table 2-3 24-Hour Precipitation for various Return Periods for various Elevations

Gradient (%)	Elevation (m)	Annual exceedance probability (1:n years, 24-hour)						
		1:2 yr	1:5 yr	1:10 yr	1:25 yr	1:50 yr	1:100 yr	1:200 yr
	600	28	40	48	56	63	69	77
2.2	700	28	41	49	58	64	70	79
2.2	800	29	42	50	59	66	72	81
2.1	900	30	43	51	60	67	73	83
2.1	1000	30	44	52	62	68	75	84
2.1	1100	31	45	53	63	70	76	86
2.0	1200	32	45	54	64	71	78	88
2.0	1300	32	46	55	65	73	79	90
	Upper bound approach	31	44	51	63	75	88	108

3 GEOLOGY

3.1 REGIONAL GEOLOGY

The Coffee Gold Project is located in the Yukon-Tanana Terrane (YTT), an accreted pericratonic rock
sequence that covers a large portion of the Omineca Belt in the Yukon and extends into Alaska and British Columbia. The YTT underlies part of the Tintina gold belt and hosts multiple gold deposits, including the Sonora Gulch gold deposit, the Casino copper-gold-molybdenum porphyry, the Boulevard gold prospect, and the Golden Saddle gold deposit (Bennett et al., 2010; Allan et al, 2013);. The YTT also hosts volcanogenic massive sulphide (VMS) and Mississippi Valley-type (MVT) deposits.

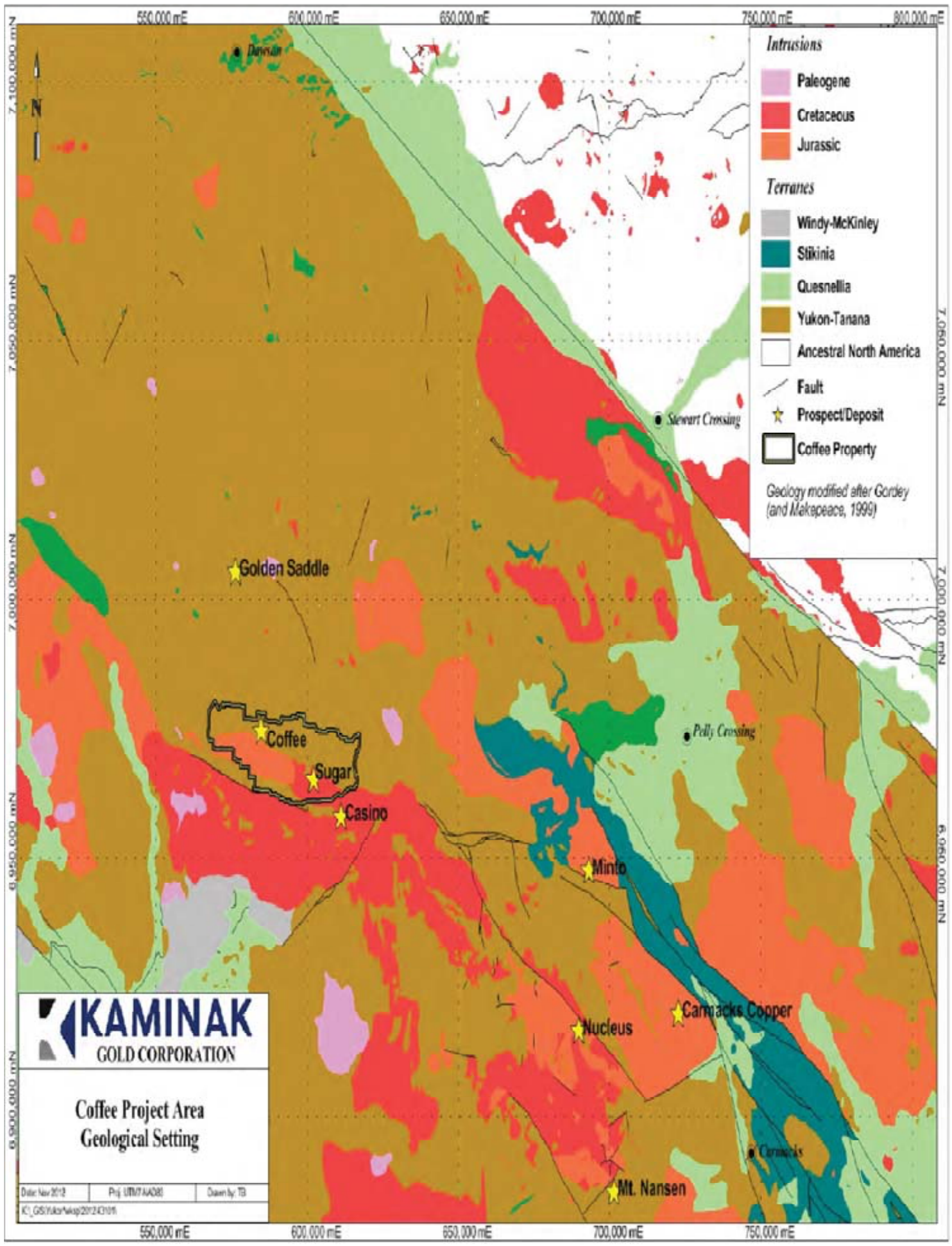
The YTT is composed of a basal metasiliclastic sequence overlain by three subsequent volcanic arcs. The oldest component of the Yukon-Tanana terrane is the pre-Late Devonian Snowcap assemblage, which consists of metasediments including psammitic schist, quartzite, and carbonaceous schist in addition to local amphibolite, greenstone, and ultramafic rocks (Piercey, 2009). The Snowcap assemblage was deposited on the ancient Laurentian margin in a passive marine setting (Piercey, 2009). The beginning of eastward subduction of the Paleo-Pacific plate led to the formation of a magmatic arc at approximately 365 Ma (Colpron, M., Nelson, J.L. and Murphy, D.C., M, 2006). Rapid westward slab rollback caused significant extension, which initiated the formation of the Slide Mountain Ocean back-arc basin by ~360 Ma (Colpron, M., Nelson, J., and Murphy, D.C., 2007). Arc volcanism during the Wolverine-Finlayson magmatic cycle (365-342 Ma) deposited submarine mafic and felsic volcanic rocks of the widespread Finlayson assemblage onto the Snowcap assemblage (Colpron, M., Nelson, J.L. and Murphy, D.C., M, 2006).

A reversal of subduction polarity during the Late Permian resulted in the western margin of Slide Mountain Ocean subducting beneath the evolving YTT (Erdmer, 1998). This subduction initiated a magmatic arc which was active from 269-253 Ma and formed the Klondike arc assemblage, the youngest member of the outboard Yukon-Tanana terrane (Allan et al, 2013); (Colpron, M., Nelson, J.L. and Murphy, D.C., M, 2006). Closure of the Slide Mountain Ocean by the Latest Permian led to the obduction of the YTT onto the Laurentian margin, causing a collisional event responsible for lower amphibolite facies metamorphism in the Coffee project area (Beranek, L.P., and Mortensen, J.K., 2011). In addition, collision resulted in the development of a low-angle transpositional foliation recognized throughout the Yukon-Tanana Terrane (S2 of Berman et al., 2007).

East-dipping subduction along the now docked YTT caused intra-arc shortening and contractional deformation. In the Klondike and the area of the Coffee project, thrust fault-bounded panels of Slide Mountain assemblage greenstone and serpentinized ultramafic occur within the tectonic stratigraphy of the YTT (Buitenhuis, 2014) (MacKenzie, D.J., Craw, D. and Mortensen, J., 2008). These thrust-emplaced slices are generally less than 100 metres in thickness, dip to the southwest, and persist for tens of kilometres in some areas (MacKenzie, 2010) (MacKenzie, D.J. and Craw, D., 2012). The emplacement of these slices is contemporaneous with northeast-vergent, open to tight folding dated between 195 and 187 Ma (Berman, 2007).

Beginning in the early to mid-Cretaceous, localized rapid uplift and exhumation occurred throughout the YTT in Yukon and Alaska, including within the Dawson Range (McCausland, 2006), (Dusel-Bacon, C., Lanphere, M.A., Sharp, W.D., Layer, P.W., and Hansen, V.L., 2002), (Gabrielse H. and Yorath, C.J., 1991., 1991). Extension and unroofing of the Dawson Range was accompanied by the emplacement of the Coffee Creek granite and Dawson Range batholith ~110-90 Ma (McKenzie, 2013) (Wainwright, A.J., Simmons, A.T., Finnigan, C.S., Smith, T.R., and Carpenter, R.L., 2011., 2011) (Colpron, M., Nelson, J.L. and Murphy, D.C., M, 2006) (Mortensen J.K., 1992). This localized extension and exhumation is recorded by an apparent age-resetting event observed in white mica in western Yukon Tanana at roughly 90 Ma (Douglas, 2002), in rhenium-osmium dates in molybdenite (92.4 Ma), and U-Pb dates in monazite (92.5 Ma) from plutons in east-central Alaskan YTT (Selby, 2002). At the Coffee property, this extension resulted in the activation of the Coffee Creek fault system, a set of dextral strike-slip faults and associated north-to-northeast brittle faults interpreted as splays off of the regional Big Creek fault to the south east (Sánchez, 2013) (Johnston, 1999).

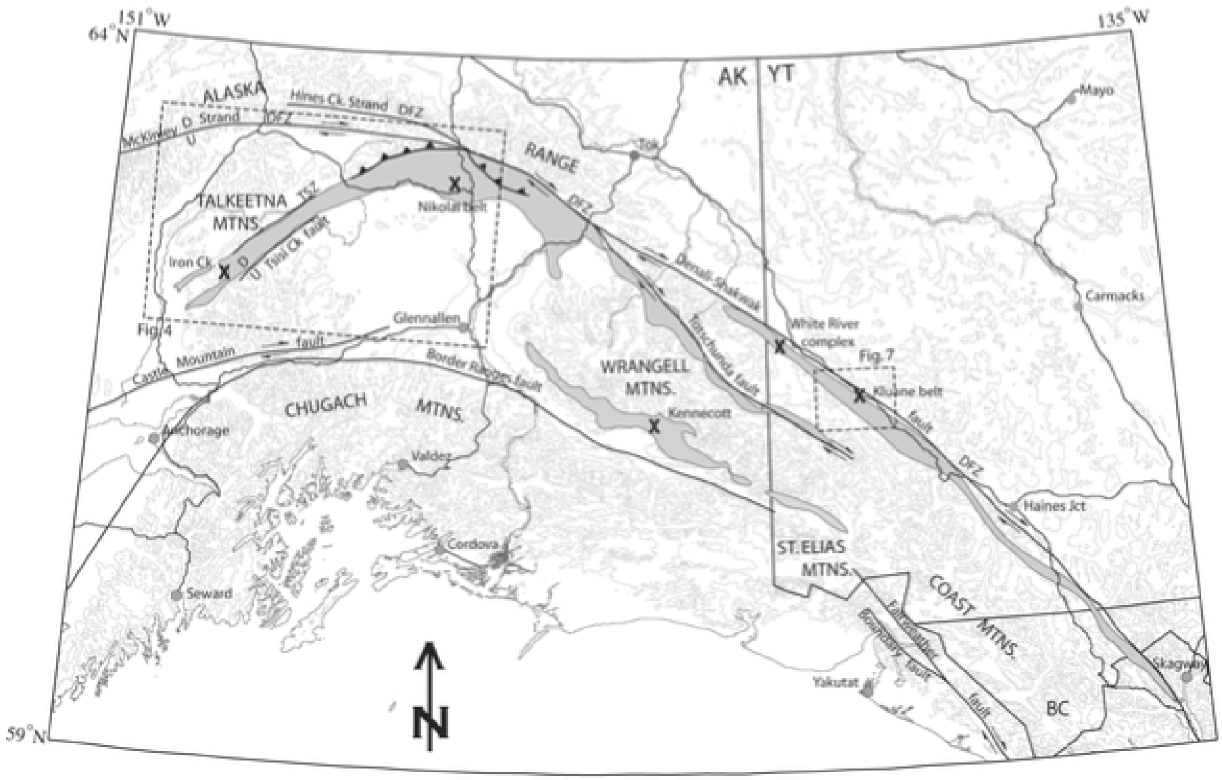
Figure 3-1: Regional Geological Setting



3.2 FAULTING

The regional fault system surrounding the site is shown in Figure 3-2. There is no identified active faulting within the Coffee Project site and no evidence of any Holcene faulting (Active). The regional fault system and site seismicity is discussed in more detail in the Seismic Report included in Appendix C.

Figure 3-2: Regional Fault Map



4 GEOTECHNICAL INVESTIGATION

4.1 SITE CONDITIONS

4.1.1 Surface Conditions

The leach pad facility will be constructed along a ridge to the west of the open pit and process areas at the head of drainage courses flowing to Upper Coffee Creek to the south and Halfway Creek to the north. The site is characterized by a very thin layer of organic topsoil material overlying a weathered transported soil horizon overlying weathered bedrock which transitions into granite bedrock. Bedrock outcrops on both the east and west ends of the proposed heap leach pad area. The existing ground slopes of the pad area vary from 7.5 to 25 percent with the steeper slopes been localized. There are no defined drainage courses within the proposed leach pad area. Surface runoff from winter snow melt and summer rain storms occurs as overland flow and is collected in the Coffee Creek and Halfway Creek drainages, which are both tributaries to the Yukon River.

The Coffee project site is located in a permafrost zone with an "active layer" subject to freeze /thaw cycles. Permafrost is soil, rock or sediment that is frozen for more than two consecutive years. Where there is no insulating layer such as ice or snow cover, it exists beneath a layer of soil, rock or sediment, which freezes and thaws annually and is called the "active layer." Active layer thickness varies with the season, but is generally 0.3 to 4 meters thick. Below the active layer, the underlying soils and rock materials remain permanently frozen. In the context of this report, permafrost generally is meant to refer to those soils and rock materials that have visible ice or non-visible excess ice within the "active layer" or in the permafrost materials underlying the active layer.

Surface soils at the site are of very limited depth with a thin veneer of topsoil and "active layer" extending to generally about a depth of 2 meters. Below the active layer permafrost extends into the weathered bedrock and bedrock materials.

4.1.2 Site Subsurface Geology and Hydrogeology

The leach pad site is underlain by the Coffee Creek granites with occasional surface outcrop expressions. These granites have shallow insitu weathered horizons which transition to moderately competent bedrock material within 2 to 10 meters of the ground surface.

The hydrogeology is addressed in a separate report by Lorax (Lorax Environmental Consultants, 2016). The permafrost depth for the site exceeds 100 meters with the ice-rich soils generally extending to depths of 2 meters for the proposed leach pad ridge location.

4.2 DRILLING - HEAP LEACH PAD AREA

The on-site field geotechnical investigation work was performed by SRK with collaboration from The MINES Group, Inc. Geo-Logic Associates (GLA) of Grass Valley California did geotechnical testing on the various ore materials from the different pits and crush sizes. The GLA laboratory test results are summarized in section 5.3 below and are included in Appendix B.

During the drilling and sampling programs, soil samples were collected for laboratory testing and classification. The laboratory testing methods used are presented in Section 5.3, Geotechnical Laboratory Testing, and the summary results are presented in Appendix B.

4.2.1 Sonic Drill Holes

The drilling program for the leach pad and ponds consisted of 15 sonic drill holes, SRK-15S-13A through SRK-15S-25 and SRK-15S-31 through 34, completed to varying depths from 1.53 meters in SRK-15S-16 to a maximum depth of 9.6 meters below the ground surface in SRK-15S-13A. Sonic drill holes SRK-15S-22 and -24 originally planned for the heap leach facility now fall into the process plant area, but are included in Appendix A.1. The Sonic drill logs are included in Appendix A1which also includes a location figure for the drill logs.

Sonic drilling produces a continuous core that can be logged, and from which samples can be taken for laboratory testing. The sample locations for geotechnical testing are shown on the drill logs.

Most of the leach pad site is covered with an organic topsoil layer that extends to a maximum depth of 0.4 meters, with an average depth of approximately 0.3 meter. This layer of topsoil is to stripped and saved for future reclamation of the project site.

Below the topsoil layer, is a layer of transported silty sand/sandy silt materials with gravel lenses and occasional boulders. In some of the Sonic drill holes this transported layer is absent. Where present, the sandy silts exhibit non-plastic to moderate plasticity indices and also excess ice in the form of lenses and visible ice inclusions. This transported layer ranges in thickness from nonexistent to 2.2 meters, but generally is between 0.4 to 1.5 meters thick. This layer is the subject to freeze thaw conditions.

Below the transported layer, a grey to brown silty sandy layer with gravel and trace cobbles is encountered with interspersed gravel layers. The silty sand materials exhibit mostly non- to low plasticity indices. The depths of this material are variable up to 9.6 meters (in SRK-15S-13A drill hole).

Below the silty sand and gravel layer, the materials transition into a weathered bedrock horizon which in turn transitions into more competent granite bedrock material and varying depths.

No water tables were observed during the sonic drilling program. The only water observed was the thaw water from the frozen core material which was in part generated from frictional heat during the drilling process and from the thawing of the upper near-surface frozen materials.

4.2.2 Test pits

Test pits (27) were excavated for the leach pad and pond areas and are designated SRK-15TP-1 through -21, SRK-15TP-41 through -44, and SRK-15TP-65 through -66. The test pit logs are included in Appendix A.2 together with a location figure.

.

The test pits were excavated with a CAT 312 excavator with limited depth capabilities up to 2.2

meters.

The test pits logs exhibit the similar soils profiles to those observed from the sonic drill holes. The organic topsoil layer varied from 0.1 to 0.5 meters thick and was generally observed to be 0.2 to 0.3 meters thick across the leach pad and event pond areas. Below the topsoil horizon, a sandy silty with gravel and cobbles was observed with the fines being non-plastic to low plasticity. This soil horizon is variable from sandy silt with gravel and cobbles, to well graded gravel with silty sand, to silty sand with gravel, to sandy silt materials. This is typical of a transported soil horizon.

Bulk samples were taken during the test pit excavations. The sample locations are shown on the test pits logs in Appendix A.2.

4.2.3 Monitoring, Exploration and Condemnation Drill holes

Kaminak drilled 19 exploration holes using reverse circulation and diamond core methods with a total of 1828.99 meters of drilling been done. These condemnation holes on the western side and center of the heap leach facility served to verify that the area for construction of the heap leach facility did not contain significant mineralized ground. The locations of the drill holes and a spread sheet of the summarized exploration logs are included in Appendix A.3.

These condemnation holes were drilled using both diamond and reverse circulation drilling methods. The condemnation holes were drilled at various dip and dip directions including vertical holes. The length of these drill holes varied from 20 meters in drill hole CFD0597 to 112.78 meters in drill hole CFR0972. Visual observation of core log for CFD0597 (located near the center of leach pad) showed core to be highly fractured and jointed indicating RQD values to be low to moderate; see end of Appendix A.3 for photograph of the core log of CFD0597.

The condemnation holes were not specifically logged for organic topsoil materials. The topsoil was probably removed for drill pad construction. The underlying overburden material logged was a combination of the transported soil horizons and some of the loose weathered bedrock material. The condemnation holes were more focused on the underlying bedrock material, which were almost exclusively in granite material. Hole CFB013 did encounter a massive near surface andesite dyke from 1.52 to 10.67 meter depths.

The underlying granite bedrock material can be characterized as a moderately competent and jointed material, with weak to moderate sericite and clay alteration, and weak oxidation.

4.3 GEOTECHNICAL LABORATORY TESTING

4.3.1 General

The majority of the tested samples were taken from the test pit program. Samples of the soil horizons below the organic topsoil layer were taken at for characterization and to determine the soil properties in the transported and weathered bedrock zones. This sampling and testing program for soil samples was performed to assess geotechnical parameters for use in engineering design for the proposed leach pad project.

The sonic drilling program was limited by the ability of the drill to penetrate boulders and bedrock materials. Hence little core material for geotechnical testing was produced, with limited testing as reported in SRK site geotechnical report (Steffen Robertson & Kirsten, 2016). No geotechnical testing was performed for the granite bedrock material underlying the leach pad site.

There was little evidence during the site investigation by SRK of an abundance of clay material suitable for use as a construction material for a composite lining system for the leach pad.

The laboratory testing program for samples obtained during the SRK site investigation included moisture determinations, grain-size analyses, Atterberg limit tests, moisture/density, compaction testing, permeability tests, and direct shear strength tests. All tests were performed in accordance with the American Society of Testing and Materials (ASTM) standard test procedures where applicable. The summary of geotechnical test results for the leach pad are included below and are also reported the SRK report, 2015 Geotechnical Field Investigation, Coffee Gold Project, Yukon, Canada (Steffen Robertson & Kirsten, 2016). Geologic Associates (GLA) of Grass Valley, California also provide laboratory testing for samples of ore materials which are included in the summary tables below and attached in Appendix B.2.

4.3.2 Specific Gravity

One specific gravity tests were conducted on sample taken from Sonic Drill hole SRK-15S-20 at a depth of 0.9 meters with performed on samples from ore testing. The test results are summarized in Table 4-1

Table 4-1: Summary of Specific Gravity Results for Leach Pad

Hole ID	Sample Number	Sample Depth (m)	Sample Description	Specific Gravity
SRK-15S-20	17722	0.9	SILT and SAND, some gravel, trace clay, brown.	2.63
Latte 583150	72114*	Trench	Brown poorly Graded Gravel with Silt and Sand (GP-GM)	2.72
Latte Oxide West	72147*	-	Poorly Graded Gravel w/ Sand (GP)	2.75
Supremo #1	73007*	-	Brown Poorly Graded Gravel w/ Silty Clay and Sand (GP-GC)	2.72
Supremo #2	73016*	-	Brown Poorly Graded Gravel w/ Clay (GP-GC	2.69

*GLA test results on leached ore residues from the column testing

4.3.3 Grain Size Characteristics

Sieve analyses were performed on selected samples from the site investigation by SRK, and from the ore characterization testing by GLA to determine grain size distributions and to aid in soil classifications and determination of engineering properties. The tests were performed in accordance with ASTM D-422. The grain size data are summarized in Table 4-2.

Table 4-2: Grain size Summary of Samples from Leach Pad Area

Location ID	Sample Number	Sample Depth (m)	% Clay	% Silt	% Sand	% Gravel
SRK-15S-13A	17718	0.9	6	44	35	15
SRK-15S-13A	17719	4	21		73	6
SRK-15S-16	17715	0	14	32	39	15
SRK-15S-17	17717	1.5-1.8	7	32	32	29
SRK-15S-19	17713	2.7	3	37	43	7
SRK-15S-20	17722	0.9	5	39	38	18
SRK-15S-23	17721	0.6	40		36	24
SRK-15S-25	17723	0.6	7	27	35	31
SRK-15S-25	17724	1.8	7	18	37	39
SRK-15TP-03	17565	0.5-1	3	14	37	46
SRK-15TP-04	17563	0.6	4	24	31	40
SRK-15TP-06	17559	0.7	3	9	48	40
SRK-15TP-08	17581	0.85	4	13	37	47
SRK-15TP-09	17579	0.75	4	31	36	29
SRK-15TP-11	17575	0.7	6	28	40	27
SRK-15TP-12	17574	0.6	1	23	40	36
SRK-15TP-13A	17623	0.6	6	35	41	19
SRK-15TP-14	17572	0.85	1	18	42	40
SRK-15TP-16	17570	0.9	4	22	36	39
SRK-15TP-17	17566	0.4	3	22	39	36
SRK-15TP-18	17576	0.3-0.5	5	36	27	32
SRK-15TP-20	17564	0.6	4	34	39	24
SRK-15TP-21	17562	0.5	3	27	48	22
SRK-15TP-43	17560	0.6	9	26	44	21
Latte 583150	72114*	Trench	7.2		19.5	73.3
Latte 583350	72120*	Trench	8.1		18.5	73.4
Latte Mine	73001*	Block	8.8		33.5	57.7
Supremo Mine	73004*	Block	13.6		36.9	49.5
Latte Oxide West	72147*	-	4.7		17.8	77.5
Latte 80%	72159*	-	2.8		10.5	86.7
Latte 60%	72165*	-	3.8		11.0	85.2
Supremo #1	73007*	-	8.5		16.5	75.1
Supremo #1	73010*	-	8.7		19.2	72.1
Supremo #2	73016*	-	6.8		12.5	80.7
Supremo #2	73019*	-	5.9		11.9	82.1
Supremo 80%	73037*	-	1.1		3.0	95.9

* GLA test results on leach ore residues from the column testing.

4.3.4 Atterberg Limits

Liquid and plastic Atterberg limits were determined for selected samples of the cohesive soils from the clay borrow investigation, and also from the contact zone underlying the leach pad area. The tests were performed in accordance with ASTM D-4318 to aid in classification and correlation and to provide qualitative information concerning engineering parameters of the soils. The results of these tests are summarized in Table 4-3 below.

Table 4-3: Summary of Atterberg Limits for Leach Pad Samples

Hole/Test Pit ID	Sample Number	Sample Depth (m)	Liquid Limit (%)	Plastic Limit (%)	Plasticity Index (%)	Soil Plasticity *	Mod. USCS Class	Natural Moisture Con (%)
SRK-15S-13A	17718	0.9	25		6	Low	CL-ML	16.2
SRK-15S-20	17722	0	23	17	6	Low	CL-ML	73.1
SRK-15S-25	17723	0.6	25	22	3	Low	ML	12.6
SRK-15TP-03	17565	0.5-1	29	23	6	Low	ML	10.5
SRK-15TP-20	17564	0.6	0	29	0	NP	-	20.7
SRK-15TP-43	17560	0.6	0	21	0	NP	-	12.6
Latte 583150	72114*	Trench		24	6			6.6
Latte 583350	72120*	Trench		26	7			6.3
Latte Mine	73001*	Block		27	9			11.4
Supremo Mine	73004*	Block		31	13			11.1
Latte Oxide West	72147*	-		30	13			0.3
Latte 80%	72159*	-		26	8			2.9
Latte 60%	72165*	-		27	11			3.1
Supremo #1	73007*	-		25	6			9.1
Supremo #1	73010*	-						
Supremo #2	73016*	-		25	8			8.9
Supremo #2	73019*	-						
Supremo 80%	73037*	-		33	16			2.5

* GLA test results on leach ore residues from the column testing.

4.3.5 Compaction

Moisture-density relation tests (compaction curves) were conducted on samples in accordance with ASTM D-698 (standard Proctor). These tests were performed to help classify the soils and to aid in preparing permeability and shear strength test samples. Results of the moisture-density relation tests are summarized in Table 4-4.

Table 4-4: Summary of Compaction Tests for Leach Pad

Sample ID	Sample No.	Sample Depth (m)	Corrected Maximum Dry Density (kg/m3)	Corrected Optimum Moisture Content (%)
SRK-15TP-09	17,579	1	2,094	10
SRK-15TP-04	17,562	1	2,007	10
SRK-15TP-34	17,558	1	1,885	11
Latte 583150	72114*	Trench	2,318	6
Latte 583350	72120*	Trench	2,372	5
Latte Mine	73001*	Block	1,887	8
Supremo Mine	73004*	Block	1,988	11
Latte Oxide West	72147*	-	2,283	4
Latte 80%	72159*	-	2,497	3
Latte 60%	72165*	-	2,332	3
Supremo #1	73007*	-	2,347	4
Supremo #2	73016*	-	2,400	4

* GLA test results on leach ore residues from the column testing.

4.3.6 Direct Shear

Two direct shear tests were performed in accordance with ASTM D3080 on intact for sample from SRK-15S-13A, and remolded material for sample from SRK-15TP-17. The remolded sample was compacted to maximum Standard Proctor dry density and optimum moisture content based on the Proctor density and optimum moisture content from sample SRK-15TP-09. The direct shear test results are summarized in Table 4-5.

Table 4-5: Summary of Direct Shear Tests for Leach Pad

Hole/Test Pit ID	Sample No	Sample Depth (m)	Sample Description	*Moisture Content (%)	*Wet Density kg/m^3	Coh. (kPa)	Friction Angle (deg)
SRK-15S-13A*	17718	0.9	SILT & SAND, some gravel, trace clay	16.2	1686.0	12.0	35.0
SRK-15TP-17	17566	0.4	SAND & GRAVEL, silty, trace clay, brown.	9.6	2172.0	77.0	42.0

* Moisture content and density data shown represent the average of the three test points for each direct shear test.

4.3.7 Triaxial Shear Testing

GLA conducted tri-axial shear testing on samples of different leached ore materials from the metallurgical column testing. These samples had been obtained from trenching and drilling, then crushed in the metallurgical laboratory, and leached to determine gold recovery and reagent consumption. The residues were then split and samples taken to the GLA laboratory for geotechnical testing. The test results are included in Appendix B. A summary of the triaxial testing performed is summarized in Table 4-6 below.

Table 4-6: Triaxial Shear Test Results for Ore Materials

Sample Descr.	Sample No.	Initial Sample State		Confining Stress (kPa)	Dev. Stress (kPa)	Pore Press. (kPa)	Strength Parameters		Final Moist. Content %
		Moist. Content %	Dry Dens. kg/m^3				Effect. Stress kPa; ^{0}	Total Stress kPa; ^{0}	
Latte 583150 Trench	72114*	8.4	1802.1	95.8	200.6	55.2	c = 6.89 Ø = 40.2°	c = 27.6 Ø = 22.3°	11.9
				239.2	390.2	135.1			
				487.5	672.2	311.6			
Latte Oxide West	72147*	6.8	1730	95.8	121.3	57.2	c = 0 Ø = 40.6°	c = 6.9 Ø = 20°	12.5
				239.2	273.7	158.6			
				487.5	513.7	350.3			
Supremo #1	73010*	8.9	1699.6	95.8	124.1	68.9	c= 0 Ø = 40.9°	c= 13.8 Ø = 18.9°	13.7
				239.2	254.4	160.6			
				487.5	490.2	360.6			

Table 5-6 converted to SI units (results in Appendix B.2 are in imperial units)

4.3.8 Hydraulic Conductivity

Table 4-7: Summary of Hydraulic Conductivity for Ore Materials

Sample I.D.	Sample No.	Normal Stress		Moisture Content %	Dry Density		Hydraulic Conductivity (cm/sec)
		psi	kPa		pcf	tonnes/m^3	
Latte 583150 Trench	72114*	Initial		2.1	97.6	1.56	
		35	241		98.4	1.58	4.80E-01
		70	483		98.7	1.65	4.10E-01
		140	965		99.1	1.53	3.60E-01
		(Drained)		7.6	99.1	1.59	
Latte Oxide West	72147*	Initial		1.9	94.6	1.52	
		35	241		95.2	1.52	4.90E-01
		70	483		95.5	1.53	4.80E-01
		140	965		95.8	1.53	4.50E-01
		(Drained)		6.9	95.8	1.53	
Supremo #1	73010*	Initial		0.8	102.7	1.65	
		35	241		116.1	1.86	7.00E-02
		70	483		120.8	1.94	6.20E-02
		140	965		127.5	2.04	3.50E-02
		(Drained)		11.9			
Supremo #2	73019*	Initial		1.9	95.4	1.53	
		35	241		105.8	1.69	6.50E-01
		70	483		110	1.76	6.70E-01
		140	965		113.5	1.82	6.90E-01
		(Drained)		9	113.5	1.82	

4.3.9 Liner Interface Shear testing

Liner interface shear testing was performed for use in the stability evaluation of the leach pad. A 2 mm single sided textured LLDPE and a GCL were used, together with leached ore material screened to a P_{100} of 38mm (to be used as the overliner) and a subbase material representative of local soils. The results of this testing are shown in Table 4-8.

.

Table 4-8 Summary of Liner Interface Testing

Geomembrane I.D.	Interface Description	Normal Stress- kN/m^2	Shear Stress kN/m^2		Strength Parameters kN/m^2; Deg ^{0}	
			Peak	Residual	Peak**	Large Deformation*
Cetco Bentomat DN GCL (roll #6448) & GSE 2mm LLDPE: (roll #103194141)	GCL ~ LLDPE; Substrate: 73007, Supremo #1	239.4	148.0	93.4	c =34.0 ϕ = 25^o ϕ_{sec} = 27^o	c =63.7 ϕ = 8^o ϕ_{sec} = 12^o
		478.8	246.6	143.6		
		861.8	431.9	187.2		
Cetco Bentomat DN-9 GCL (roll #5) & GSE 2mm LLDPE: (roll #103194141)	GCL ~ LLDPE; Substrate: 73007, Supremo #1	239.4	133.1	91.9	c =13.9 ϕ = 26^o ϕ_{sec} = 27^o	c =48.8 ϕ = 12^o ϕ_{sec} = 14^o
		478.8	249.5	156.1		
		861.8	440.5	221.7		
GSE NWL-45 GCL: (roll #502204100) & GSE 2mm LLDPE: (roll #103194141)	72114, Latte 583150 Trench ~ LLDPE; Substrate: GCL	239.4	130.7	124.4	c =11.0 ϕ = 27^o ϕ_{sec} = 28^o	c =51.2 ϕ = 20^o ϕ_{sec} = 22^o
		478.8	259.0	243.2		
		861.8	449.1	351.9		
	GCL ~ LLDPE; Substrate: 72114, Latte 583150 Trench	239.4	159.4	70.4	c =91.0 ϕ = 19^o ϕ_{sec} = 24^o	c =67.5 ϕ = 0^o ϕ_{sec} = 5^o
		478.8	282.5	64.2		
		861.8	382.1	70.4		
	73037, Supremo 80% ~ LLDPE; Substrate: GCL	239.4	129.3	123.1	c =3.35 ϕ = 30^o ϕ_{sec} = 30^o	c =19.6 ϕ = 26^o ϕ_{sec} = 27^o
		478.8	478.8	279.6		
		861.8	861.8	435.7		
	GCL ~ LLDPE; Substrate: 73037, Supremo 80%	239.4	147.5	70.4	c =66.6 ϕ = 20^o ϕ_{sec} = 24^o	c =53.15 ϕ = 4^o ϕ_{sec} = 7^o
		478.8	256.2	81.4		
		861.8	381.1	110.1		
GSE NWL-60 GCL: (roll #502237806) & GSE 2mm LLDPE: (roll #103194141)	GCL ~ LLDPE; Substrate: 72114, Latte 583150 Trench	239.4	155.6	74.2	c =22.5 ϕ =30^o ϕ_{sec} = 31^o	c =68.5 ϕ = 5^o ϕ_{sec} = 8^o
		478.8	306.9	128.3		
		861.8	516.1	128.3		
Agru GCL: (roll #G14G131003) & Agru 80mil Microspike LLDPE: (roll #F14A251015)	GCL ~ LLDPE; Substrate: 72114, Latte 583150 Trench	239.4	164.7	85.2	c =25.4 ϕ = 30^o ϕ_{sec} = 31^o	c =94.8 ϕ = 2^o ϕ_{sec} = 7^o
		478.8	292.1	134.1		
		861.8	516.6	108.7		

4.3.10 Liner Puncture Testing

Two liner puncture tests were done using 2 mm thick LLDPE liner to be use on the leach pad. Test No. 1 used GSE 2.0 mm LLDPE SST liner on top of GCL. Leached ore from Latte 583150 Trench (sample 72114) screened to a P_{100} of was used as the overliner material. The subgrade material was from Latte 583150 Trench screened to -19 mm and compacted to 95% of max dry density of 122.4 pcf (1961 kg/m^3) at a moisture content of 10.3%.

Test No. 2 used GSE 2.0 mm LLDPE SST liner on top of GCL. Ore material from Supremo 80% (sample 73037) screened to -38mm was used as the overliner material, and the subgrade material from Supremo 80% screened to -19 mm and compacted to 95% of max dry density of 120.0 pcf (1922 kg/m3) at a moisture content of 10.3%. Leached ore was used under the GLC to simulated the graded natural ground. Which this is an imprecise representation, with the GCL between the geomembrane and this material the puncturing exposure to the geomembrane was from the overlying material, which accurately represented the overliner gravel.

The puncture tests were run for 48 hours with a normal loading rate of 10 psi (68.9 kPa) per minute to 240 psi (1654.7 kPa), an equivalent height of ore loading to 100 meters.

Both tests indicated moderate yielding of the LLDPE liner material and no puncturing observed. The test results are included in Appendix B.1 and in Table 4-9 below.

Table 4-9: Summary of Liner Puncture Testing

Description	Test 1	Test 2
Upper Material:	72114 Latte 583150 Trench (-38 mm) Ore	73037, Supremo 80% (-38 mm) Ore
Geosynthetic:	GSE 2.0mm LLDPE SST (Text)	GSE 2.0mm LLDPE SST
Geosynthetic:	GCL (burnished against BDK)	GCL (burnished against BDK)
Subgrade:	(-19 mm) compacted 72114 latte 583150	(--19 mm) compacted 73037, Supremo 80%
Dry Density (95% $\gamma_{d\text{-}max}$):	19.2 kN/m^3	18.8 kN/m^3
Moisture Content:	10.30%	10.30%
Test Duration	48 Hrs	
Normal Stress	1,655 kPa	
Equivalent Ore Height	100m	
After Test Description	Moderate yielding; No punctures observed or noted	

5 SITE BORROW MATERIALS

Borrow materials for the construction of the heap leach pad and ancillary facilities will predominantly be obtained from on-site sources. Borrow materials that will be required for construction of the heap leach facility will consist of structural fill materials for the heap leach pad and ponds.

Structural fill materials will be obtained during grading operations for the leach pad construction. Additional structural fill material can be sourced from selected overburden stripping of the mine pits which is non-acid generating in nature and is suitable for structural fill.

Overliner construction material for the heap leach pad will be sourced from competent ore material which will be crushed and screened onsite. A small amount of drain gravel (inert) for leak detection sumps will be sourced from local sources.

5.1 TOPSOIL

A thin layer of topsoil covers much of the site and this will be removed during each stage of construction and stockpiled for use during progressive and final closure. While not expressly a borrow source per se, it is expected that during some stages of leach page construction (most likely stages 3 and 4), the topsoil stripped during construction will be directly placed over the older heap areas (e.g., stages 1 & 2) as part of progressive closure. Further, some of the topsoil which is stockpiled will be augmented or supplemented with other organic sources (including chipped wood) to increase the volume and organic content).

5.2 PERMAFROST EXCAVATION

As with the topsoil, some permafrost will be used in progressive and final closure. Some of this material will be stockpiled expressly for that purpose and in ways to encourage thawing and drying before it is needed. This material may also be mixed with topsoil or other organic products to increase the volume and organic content.

5.3 STRUCTURAL FILL

Structural fill material for the heap leach pad and the ponds is expected to be obtained from grading operations within the leach pad area. A modest quantity may also be obtained from the open pit waste rock stripping. Most of the structural fill material will be used in the perimeter access road and the interior berms of the leach pad. In general, the fill required from each stage of leach pad construction will be obtained from within that stage of construction.

5.4 LINER COVER BORROW SOURCE

Overliner material will be select, durable ore from the mine. This material will be crushed on site using either a contractor crushing and screening plant (for pre-production construction) or the production crushing plan with a finer screening stage added (after initial plant commissioning). Ore has been selected for overliner because it has been tested and verified as physically and chemically suitable, it provides no additional reagent consumption and it otherwise compatible with the leaching system. It also avoids the need to develop another borrow source or a separate crushing and screening circuit.

During normal operations, ore will be crushed to 80% passing (P_{80}) 50 mm size. For overliner use, this material will either be crushed finer or subjected to secondary screening to produce a product that is 100% passing 38mm and with not greater than 5% fines. The permeability of the overliner is estimated to be between 3.5 x 10^{-2} cm/sec (Supremo #1) and 6.9 x 10^{-1} cm/sec (Supremo #1). Competent ore material that is resistant to degradation is the preferred liner cover material as it has economic value and reduces the needed heap capacity. This will require pit overburden stripping and mining of the proposed ore to access the overliner material for use in construction.

6 SEISMIC EVALUATION

The Coffee Gold Project is located in west central Yukon Territory, approximately 330 km northwest of Whitehorse. The project will involve the design, construction, operation and closure of a number of facilities associated with the gold mining processing facilities. These will include an open pit, waste dumps, sedimentation control facilities, surface diversion works, a heap leach pad, various ponds and water storage reservoirs, crushers, pipelines, roads, gold recovery plant, and various mine and administrative buildings. All planned facilities will need to be designed to resist the expected seismic (earthquake) loading.

Western Canada, southeast Alaska, and the Aleutian volcanic arc are among the more seismically active areas in North America. Southwestern Yukon Territory is directly affected by this area of elevated seismicity. The Coffee Gold Project site is located in an area of moderate seismicity known as the Tintina Trench that also holds some of the region’s major river systems including the Yukon, Pelly, and Stewart rivers. The northern boundary of the trench is the Tintina fault, a long right lateral strike slip fault responsible both for the formation of the trench and its name. The area to the north of the Tintina fault would seem to be associated with a higher level of historic seismicity than the area within the trench itself. The southern boundary of the trench is the highly active Denali fault, a long right lateral strike slip fault responsible for generating the largest of the earthquakes in close proximity to the site. The area to the southwest of the Denali fault holds some of the most seismically active features in North America, or for that matter, in the world. They include the Fairweather-Queen Charlotte fault and the Aleutian Megathrust zone. The Coffee Gold Project lies very near the geographic center of the trench being approximately 125 km from both the Tintina fault to the northeast and the Denali fault to the southwest.

Most practitioners would suggest that investigations extend for a minimum radius of 150 km from the site. For important projects, a radius of 150 km to 300 km might be more appropriate (the Nuclear Regulatory Commission or NRC requires 200 km). For this project the historic data set was collected for an area within a 1000 km radius. This was done not because damaging impacts are expected from earthquakes originating at a distance of 1000 km, but rather in order to understand more completely the nature of the very complex and highly active sources of seismic energy that begin a mere 125 km southwest of the Coffee Gold Project. Regional seismic source zones were assigned to a map of historic earthquake events for the specific purpose of evaluating potential seismic loads for the Coffee Gold Project site. We have identified six (6) seismic source zones which (proceeding from northeast to southwest) are as follows:

1. Tintina North Zone
2. Inland Alaska Zone

3. Tintina Trench Zone
4. Denali Fault Zone
5. Aleutian Megathrust Zone
6. Fairweather-Queen Charlotte, Transition Fault Zone

The purpose of identifying discrete seismic source zones is to characterize and quantify the nature of the largest earthquake that is likely to occur within the zone. This information can be utilized in either a deterministic seismic hazard analysis (DSHA) or a probabilistic seismic hazard analysis (PSHA). Although different in approach, they probably have more in common than they have differences. Of interest is the largest earthquake that could reasonably be expected to occur 2 within the zone, the mean rate of occurrence or recurrence interval, and the location of the earthquake. Seismic source zones come in two (2) varieties:

1. An Aerial Seismic Source where earthquakes are uniformly distributed throughout the area and assumed to be equally likely to occur anywhere within the area.
2. A Linear Seismic Source where earthquakes occur along a narrow linear band (fault), but are again assumed to be equally likely to occur anywhere along the fault line.

For the DSHA approach, the largest earthquake event that could occur within the zone is estimated and then relocated to the position that most closely approaches the site (either along the fault in a narrow linear source zone or within the area boundaries in an aerial source zone). Site specific ground motion parameters are then estimated by attenuating ground motion between the earthquake location and the site. This seems like a very logical “worst case” type of approach, but can sometimes lead to unrealistically severe design criteria. To address this concern, in the PSHA approach the single input parameter of earthquake magnitude is replaced with a cumulative probability distribution relating earthquake magnitude to recurrence rate and an associated exceedance probability. The simple location parameter or distance to the site is replaced by another cumulative probability distribution that addresses the uncertainty associated with where the earthquake event might occur within the source zone (i.e., the variability in the distance to the site). Finally, there is also uncertainty associated with ground motion attenuation and a random component is applied to the ground motion.

The steps involved in a DSHA analysis are as follows:

1. Using information derived from geologic maps, fault maps, and plots of historic earthquake events, identify discrete seismic source zone polygons.
2. Extract “clipped” data sets lying within each seismic source zone that represent the nature of the seismicity within the zone.
3. Estimate the Maximum Credible Earthquake (MCE) associated with each seismic source.
4. Estimate the closest point of approach to the site of interest for each seismic source zone.
5. Estimate site specific ground motions by attenuating motions over the distance between the earthquake epicenter and the site.

Results of analyses for all seismic source zones are summarized in Table 6-1 and plotted in

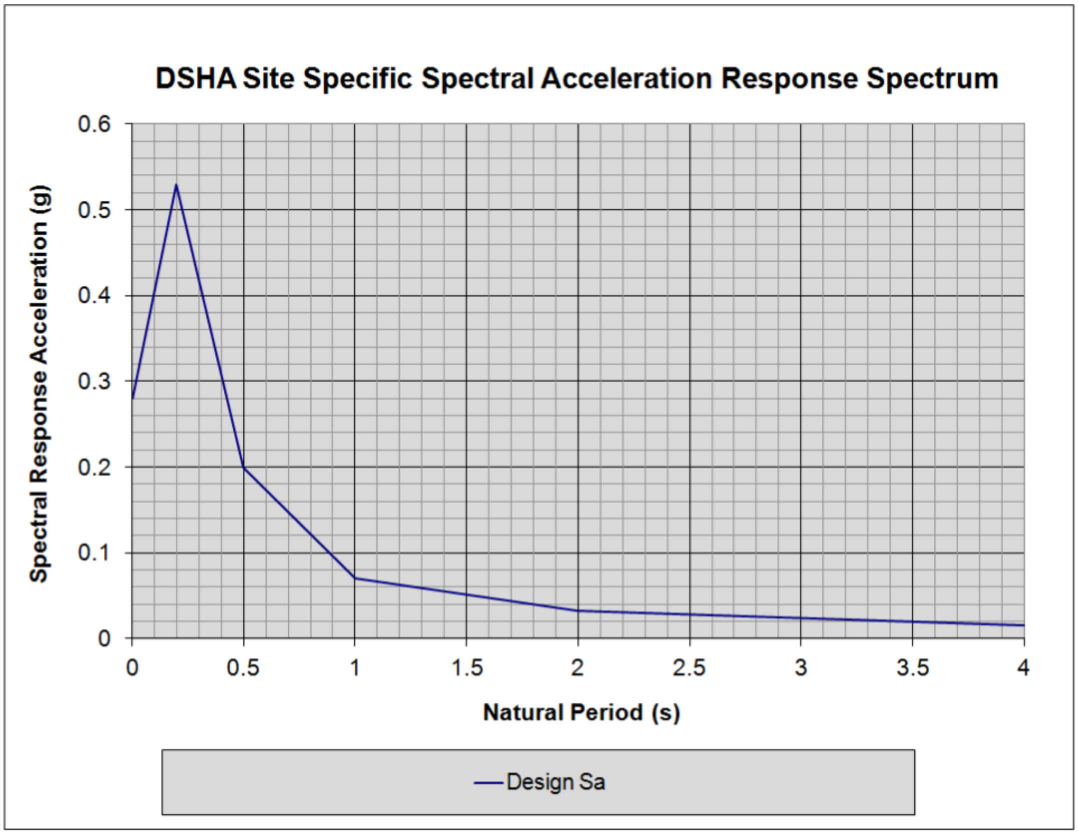
Figure 6-1. Most building codes (including the 2010 National Building Code of Canada (NBCC)) allow for either a site specific deterministic design approach or a probabilistic approach. For the site specific DSHA procedure the estimated spectral acceleration values at the various natural periods as shown in Table are used to develop a design spectral acceleration response spectrum. These design accelerations are then used to determine seismic loads for structural design, which will also vary as a function of occupancy and use.

Table 6-1: Summary of DSHA Results

Seismic Source Zone	Magnitude				Distance (km)			Pseudo-Spectral Acceleration (g)				
	Max Historic	Gutenberg-Richter	Surface Fault Rupture	MCE	Mean	Median	Closest	PGA	0.2 s	0.5 s	1.0 s	2.0 s
Tintina North	6.9	7.2	---	7.2	547	496	118	.026	0.047	0.038	0.022	0.011
Inland Alaska	7.1	7.1	---	7.1	656	633	420	---	---	---	---	---
Tintina Trench at 55 km	4.6	5.0	---	5.0	550	591	55	0.0085	0.0155	0.0065	0.0002	0.0001
Tintina Trench at 1 km	4.6	5.0	---	5.0	---	---	1	0.28	0.53	0.20	0.070	0.020
Denali Fault	7.9	7.3	8.0	8.0	360	350	105	0.06	0.11	0.085	0.050	0.0025
Aleutian Megathrust	9.2	8.3	---	9.2	662	623	364	0.07*	0.153*	---	---	0.092*
Fairweather-Queen Charlotte and Transition Fault	7.9	8.7	8.0	8.7	509	517	267	0.022**	0.036**	0.084**	0.056**	0.032**

Notes: * - Estimated from 2003 Atkinson and Boore equations for ground-motions in megathrust subduction zones
** - Estimated using M8.5 event at 267 km and the Abrahamson and Silva 2013 NGA West 2 attenuation relationship

Figure 6-1: Plot of DSHA Results



The PSHA analysis can be performed using the same seismic source zones and by replacing the maximum credible earthquake with the probability distribution of earthquake events, the site distance with the probability distribution of site distances, and adding a random component to the attenuated spectral acceleration values, then using a Monte Carlo type sampling model to compile a new distribution of spectral accelerations associated with an exceedance probability. However, some developed countries, including the U.S. and Canada, have their own web-based PSHA programs that use regionally based maps of seismic source zones and compute site distances by asking you to enter a specific geographic site location using latitude and longitude. The Canadian PSHA model is located on the Natural Resources of Canada website and was utilized to generate the PSHA results summarized in Table 6-2 and plotted in Figure 6-2: Plot of PSHA Results. The results assume a Site Class C (stiff soil and/or soft rock in the upper 30 m of the subsurface profile) and are consistent with the observed site conditions at the Coffee Gold Project site.

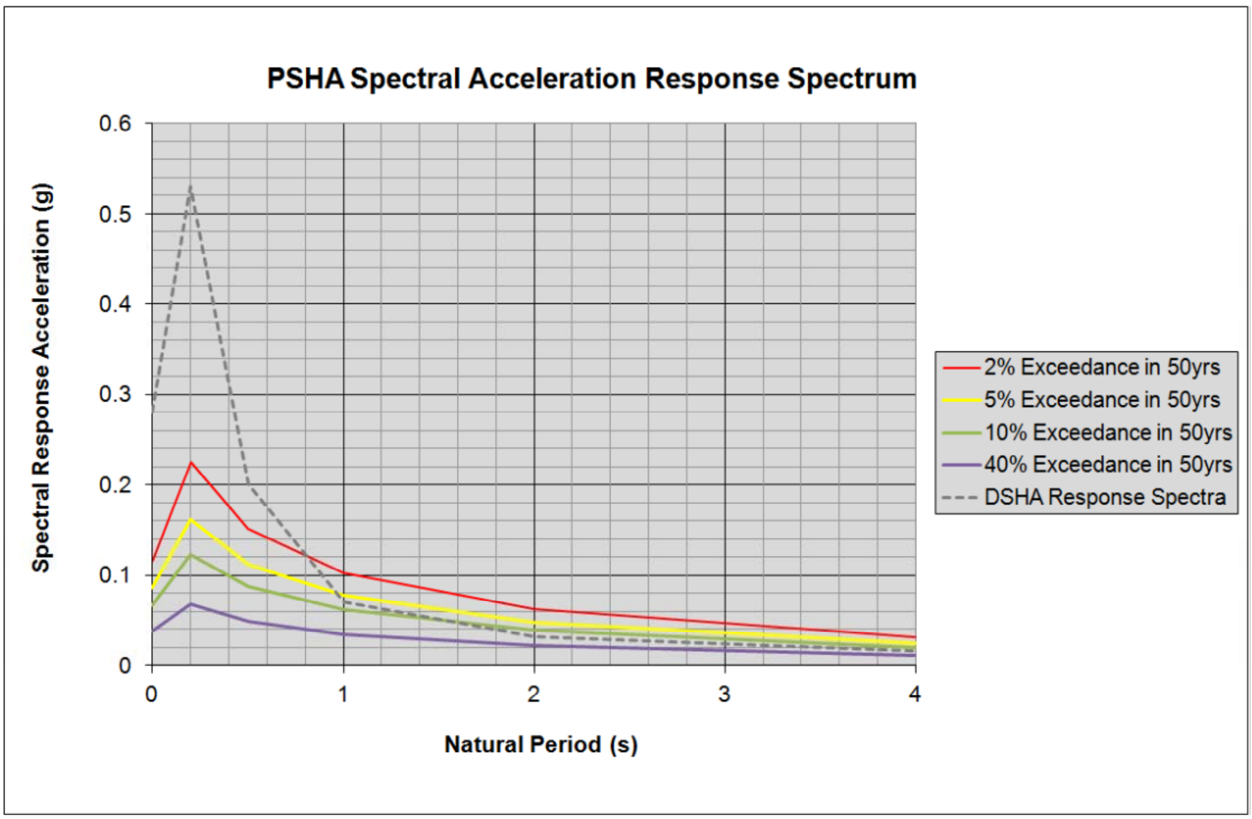
Figure 6-2 plots the DSHA spectral response spectra (shown as a dashed gray line) on the same plot as the PSHA spectral response spectra in order to permit a comparison of the two methods. As expected the design ground motions from the DSHA analysis are similar but more conservative, at least for natural periods less than 0.8 s. The reduced probability of larger earthquakes occurring in very close proximity to the site moderates the distribution of spectral accelerations resulting in reduced short period Sa values from the PSHA. Beyond the natural period of 0.8 s the design Sa is slightly less than both the 2% and 5% exceedance earthquakes (return periods of 2475 yrs and 1000 yrs respectively), implying that these rare, large earthquakes occurring at larger radial distances have a magnitude higher than the MCEs selected for the DSHA analysis.

Table 6-2: Summary of PSHA Results

Annual Exceedance Probability	0.01	0.0021	0.001	0.000404
Recurrence Interval (yrs)	100	475	1000	2475
Probability of Exceedance in 50yrs	40%	10%	5%	2%
Period (s)	Sa (g)	Sa (g)	Sa (g)	Sa (g)
PGA	0.038	0.067	0.086	0.116
0.2	0.068	0.123	0.161	0.225
0.5	0.049	0.087	0.112	0.151
1	0.034	0.062	0.078	0.103
2	0.022	0.039	0.048	0.063
4*	0.011	0.0195	0.024	0.0315

Note: * - Per the 2010 NBCC the 4s Sa = (2s Sa)/2

Figure 6-2: Plot of PSHA Results



For the design of buildings and other structures at the Coffee Gold Project site the PSHA results shown in Table 6-2 and Figure 6-2for the earthquake event having a 2% probability of exceedance in 50 yrs coupled with the criteria and design procedures for estimating seismic loads found in the 2010 NBCC should be applied for structural design. Seismic design in connection with earthen geotechnical structures (earth dams, embankments, excavations, etc.) is usually associated with slope stability analysis. Current practice in slope design incorporates a hierarchy of slope analysis procedures that progress from simple and conservative, to more rigorous but less conservative. The first level in the hierarchy is the pseudostatic analysis.

For the Coffee Gold Project site, the recommended pseudostatic acceleration coefficient for use in pseudostatic stability analysis is 0.06g. The second level in the hierarchy of seismic stability analysis is displacement analysis indicating acceptably small displacements. For critical structures, the last level in the hierarchy of seismic stability analysis tools is a full dynamic analysis, again indicating acceptably small displacements.

7 LEACH PAD THERMAL EVALUATION

Heap leaching in cold climates requires special considerations to prevent freezing of the heap and the leach solution. Factors that impact the solution and in-heap ore temperatures include ambient temperature at the time of stacking, burial of drip irrigation lines, use of heap covers, the temperature of the barren solution as applied to the heap, and the thermal mass of the heap (tonnes under leach, width and depth of heap). Thermodynamic modeling was performed in two stages to simulate expected ore and solution temperatures under different operating strategies. The first stage was used to determine the general sensitivity of the operating temperatures to a range of operating criteria, and the second stage to optimize the specific design parameters. Mean monthly temperatures at the site ranged from a low of -22.5^{o} C to a high of 13.6^{o} C, while *in situ* ore temperatures (in the mine) range from -0.5 to about -1.5^{o} C.

The thermodynamic computer model used for Coffee Creek was originally developed by Geo-Logic Associates to simulate changes in heap ore temperatures over time for a demonstration copper sulfide heap leach (Schrauf, et al, February 2014). The model was shown to closely match measured temperature profiles in the heap over a 30-month period, including the impact of the placement of a geomembrane "thermal" cover over the heap. The key difference between the Coffee Gold and the original model was that the original modeling considered heat input from sulfide oxidation, which does not occur in a cyanide heap leach. There is a small tonnage of ore at Coffee Creek that contains sufficient sulfide minerals to generate additional heat in the heap, but this was conservatively ignored.

The Coffee Gold model used the following key input variables:

- Meteorological data (ambient air temperature, wind speed, solar radiation, ratio of measured vs potential maximum solar radiation, and relative humidity) specified for each time step;
- Surface cover characteristics (solar absorptivity and thermal resistivity) specified for each time step;
- Ore properties (heat capacity, lift thickness, lift area, and ore temperature when stacked) specified for each ore lift; and
- Solution properties (flow rate, temperature addition to barren solution, total dissolved solids) specified for each time step.

Several inherently conservative assumptions were used in the model, which suggests that the forecast operating temperatures are lower than will actually be realized. This matches the experience at Fort Knox (Alaska) and Veladero (Argentine Andes) in that both of these sites have similar climates to the Coffee project and their heaps are operating slightly warmer than the Coffee model forecasts.

The conservative assumptions and simplifications include:

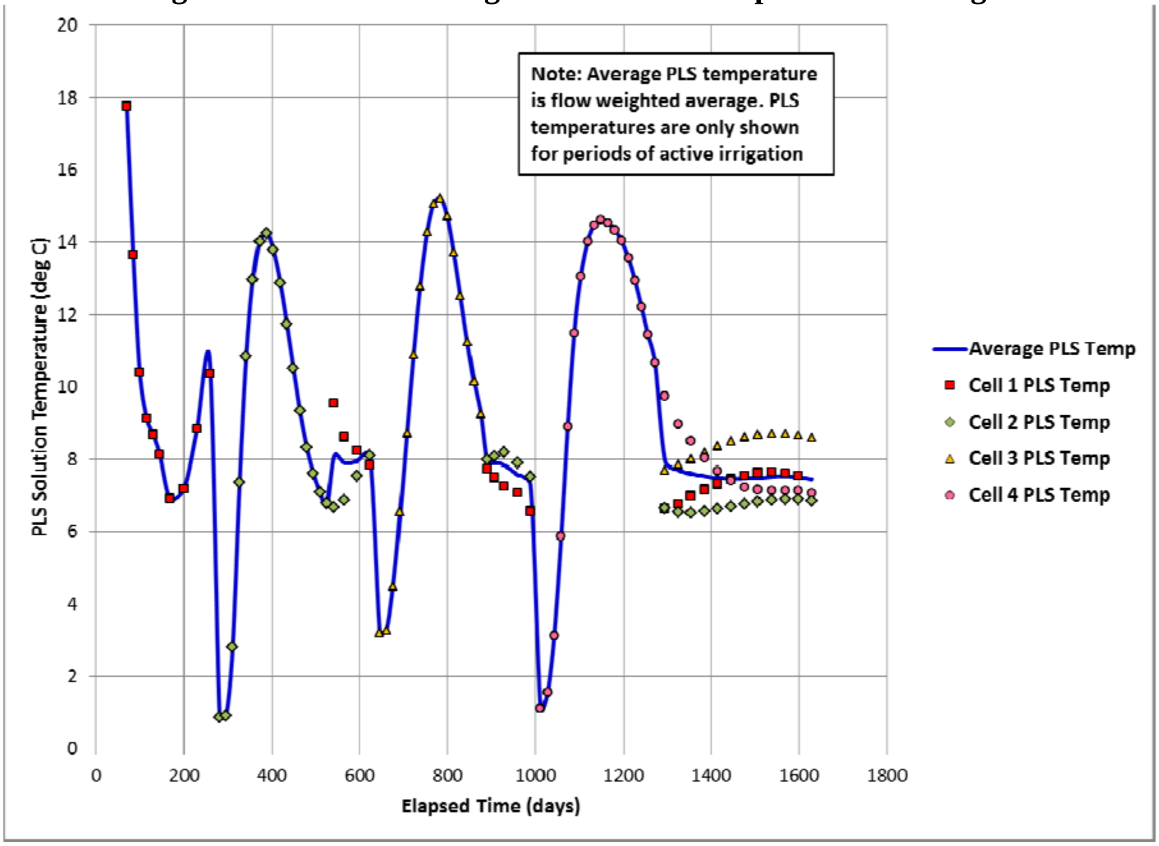
- The temperature of the ore as stacked on the heap equals the average daily ambient temperature. This is conservative because (a) stacking will generally not occur during the coldest hours of the day, (b) operators can elect to not stack ore during the coldest days if solution temperatures are lower than desired, and (c) the model ignored the solar heat gained by the ore during handling;
- The on-site weather station data suggests that, during the critical winter months, the site is

slightly warmer than the data used for the thermodynamic modeling;
- Waste heat from the electrical generation plants will be available from April 1 to December 31 of each year. In the winter, the power demand will be much lower since the crushing circuit will not be operational, and most of the available waste heat will be dedicated to heating buildings. It was estimated that from April 1 to December 31, 4^{o} C to 6^{o} C of heating (before considering line losses) can be applied to the barren solution. This was ignored in the thermodynamic modeling; and,
- The exothermic acid-base reactions and the heat contribution from sulfide mineralization were ignored.

The modeling results were used to optimize the operational parameters and indicated that a combination of drip line burial and heating of the leach solution during the winter months will be required to maintain solution temperatures above freezing during the initial years of stacking. As the size of the heap increased and thermal covers added in Year 3, solution heating will no longer be necessary for a year with an average temperature profile. The following key design parameters were determined from the thermal modeling:
- Drip lines will be buried (or covered with a layer of crushed ore) 1 m deep during the period of November 1 through March 31 each year;
- A spare irrigation area equal to 100% of the required area will be provided in every winter starting in November of Year 1 (sufficient room cannot be provided in Year -1 because of the limited area of the Stage 1 pad and the trade-off between area and ore depth);
- Ore crushing and stacking for 275 days per year, from April 1 to Dec. 31, only;
- Solution heating from November 1 through March 31 of each year, through Year 3, by increasing the barren solution temperature as applied to the heap by 6^{o} C above the pregnant solution temperature. It was further decided to set the boiler capacity at 6^{o} C for a flow of 600 m3/hr and 9^{o} C for a flow of 455 m3/hr;
- Any waste heat from the electrical generation plants that is not otherwise used will be used to additionally heat the barren solution. It is expected that an increase in the barren temperature of up to 6^{o} C can be achieved from April 1 to December 31; and,
- Beginning in Year 3, raincoats will be used during the spring snowmelt and summer rainy seasons to reduce the meteoric water entering the process circuit. Modeling of the Coffee heap as well as the prior modeling and experience at other operations determined that geomembrane covers also act as thermal insulators. In the specific case of Coffee Creek, they increase the pregnant solution temperatures by about 4^{o} C in the winter. Therefore, it was decided to leave the raincoats in place in the winter to take advantage of this benefit, beginning in Year 3.

Figure 7-1 shows the pregnant solution temperature by month for the first 54 months of operations. This modeling period was elected because it takes the heap through leaching and into rinsing of the first two stages of the pad, and it shows warming temperatures at the end of the modeling period. Further, Year 3 (about day 1,000 on the figure) is when thermal covers will begin use. Importantly, by this time the thermal mass of the heap is significant and the resulting internal heap temperatures less directly affected by ambient conditions.

Figure 7-1: Forecast Pregnant Solution Temperatures through Year 3



In parallel with the thermodynamic modeling, an industry practices study was also completed (Sinha, K and Smith, M.E., Sep 2015). This study found that there are or have been about 70 heap leach operations in artic and sub-artic climates, including the high altitude sites in the Southern Andes. Of these, 28 were surveyed in locations ranging from the southern Andes to Alaska, Yukon Territory to Mongolia. The authors used data from direct discussions with operators and engineers, reviews of published literature, and personal experience to compile operating statistics for some 210 Mt per annum (Mtpa) of heap leach production, with leaching rates ranging from under 1 Mtpa to 30 Mtpa. The severity of the winter months was quantified in two ways: the average daily temperature for the coldest month, and the length of the cold season as measured by the number of months per year with an average daily temperature below zero for an average year. Of the sites surveyed, the coldest months average –18.1^{o}C and ranges from –6^{o}C to –31^{o}C, and the length of the cold season ranges from 4 to 10 months. The coldest temperatures are found at the Russian sites, while the longest winters are experienced in the southern Andes. Conventional leach pads are used at the majority of sites (57%). Dynamic heaps (on/off leach pads) and impounding valley leach pads (VLP) are used by 28% and 18% of the sites, respectively (the total exceeds 100% because one site used both conventional dynamic heaps).

Table 7-1 summarizes the statistics from the mines studied.

Table 7-1: Summary of cold-climate heap leach operations

Description	Number of operations	Percentage of total number
Total number of operations included in the study	28	100%
Heap leach pad type:		
Conventional pad (*)	16	57%
Dynamic heap (on/off pad)	8	28%
Impounding valley leach pad	5	18%
Metal mined:		
Gold	24	86%
Copper	3	11%
Nickel	1	4%
Year-round stacking	12	43%
Year-round irrigation	16	57%

* One mine in the study uses both conventional and dynamic pads, hence it is included in both categories.

Key findings from this study, which influenced the design of the Coffee Gold HLP, can be summarized as:

- Ore temperature being close to the average daily temperature when it is stacked, winter ore stacking creates a heat sink in the system. Successful winter stacking in very cold climates requires provisions such as: solution heating, ore temperature monitoring, thermodynamic modeling, and a relatively large thermal mass. Also, large snow and ice inclusions in the heap during winter stacking can create ice dams and promote heap freezing;
- Operational problems during the first one or two winters after commissioning are relatively common, suggesting that additional measures should be implemented until the heap has a larger thermal mass;
- Many sites with similar winter temperatures have operated successfully with solution heating only from waste electrical generation heat;
- Exposed ponds can freeze in the winter and special provisions are required to protect liners and avoid water shortages during stepped-up demand when ore stacking resumes in spring. On the other hand, impounding VLPs are inherently more likely to leak and create compliance issues (Breitenbach, A.J. and Smith, M.E., July 2012);
- Average temperature-based planning (which is generally the case) can require operational adjustments such as additional solution heating, deeper dripper burial, and a shorter stacking season, to accommodate colder-than-average winters. Likewise, heating requirements and the corresponding fuel consumption and costs will be lower during warmer-than-average years;
- Although there are successful impounding VLPs and dynamic heaps in cold climates, the industry seems to prefer conventional pads (57% of surveyed installations). This may be partly for economic reasons as conventional pads are less ex (Breitenbach, A.J. and Smith,

M.E., July 2012)pensive than VLPs (Smith, M.E. and Parra, D., 2014), and partly due to the increased risk of liner leakage and compliance issues inherent with in-heap solution storage (Breitenbach and Smith, 2012);

- For year-round irrigation, insulating and heat tracing or burying all piping including drip lines (at depths dependent on site conditions) may be necessary. Providing spare irrigation area(s) for the coldest sites in case primary drip areas freeze is also a practice recommended by some successful operators. Seasonal irrigation requires pond capacity for the seasonal drainage of the heap and the Coffee modeling indicated no thermodynamic benefit to cessation of irrigation in the winter; and,
- Winter ore stacking strongly affects the operating temperatures and most sites elect to not stack in the coldest months.

8 FACILITY DESIGN CONSIDERATIONS

8.1 OVERVIEW

The heap leach facility consists of a conventional lined leach pad to support a multi-lift, free-draining heap, events and rain water ponds, access roads, solution distribution piping (barren solution to the heap) and heap drainage solution collection piping (pregnant solution and rinse water to the ponds and plant).

Ore will be stacked with trucks in nominal 10-m thick lifts. Barren solution containing cyanide will be irrigated onto the ore using drip irrigation. Pregnant solution will be collected at the base of the heap by the leach pad liner and collection system, which will route the pregnant solution to the process plant for gold recovery and reagent reconditioning. Once an area has been leached for the target time or metal recovery, another 10-m thick lift of ore will be stacked and the process repeated. This will be continued until the heap is stacked to the design elevation of 1350m for the 47 Mt capacity. Prior stages will cap out at lower elevations.

The leach pad will consist of a graded area along the ridgeline to the west of the process plant and mines (Drawing 157801 -G1). The leach pad will be constructed in stages, with each stage large enough to provide ore leaching capacity for 1.5 to 3 years. For each stage, topsoil will be removed and stockpiled for use in reclamation; for stages 3 and 4 the topsoil may be directly placed on the oldest portions of the heap as part of progressive reclamation. Ice-rich soils (often called permafrost), which averages 1.5 to 2.0 m thick across the leach pad and pond areas, will be removed and either disposed of in a waste dump or stockpiled for future conditioning and use with topsoil in closure.

After removal of topsoil and permafrost, the site will be graded by cutting and filling to achieve targeted slopes, elevations and grades. Stability berms in the graded toe area as well as the cell separation berms will be constructed as part of the subgrade preparation. The resulting subgrade will then be prepared to make it suitable for supporting the liner system; this will principally consist of ensuring a fine-grained soil layer is present to eliminate puncture risk. Prior laying of the liner system, a leak detection collection system consisting of a wick drain will be laid on the subgrade directly below the primary solution collection pipes for each cell and will report to the perimeter solution collection trench. The pad area will then be lined with two layers of synthetic liner in a composite manner, consisting of a 2.0mm-thick single-side textured LLDPE geomembrane overlying and in direct contact with a 6mm-thick heavily reinforced geosynthetic clay liner (GCL). A network of drainage pipes and drainage gravel will be placed to protect the liner from damage, to control the maximum hydraulic head over the liner system to well under 1 meter, and to collect the pregnant solution and rinse water.

The ponds will be located between the east end of Stage 1of the leach pad and the process plant (Drawing 157801 -G1). A total of four ponds are planned, with two event ponds, EP-1N and EP-1S, to be constructed as part of the initial, pre-production construction (year -1), a rainwater pond (clean water or rain water) to be constructed in year 3 of operations when raincoats are first put in service, and a third events pond (EP-2) to be constructed in year 8. The initial two event ponds, EP-1N and EP-1S, will be lined with the primary (top) liner being a 2.5 mm thick smooth HDPE (or other suitable material), and the secondary liner being a 1.5 mm HDPE or LLDPE drain liner to function as a leak detection layer between the liners. These two geomembranes will be underlain by a GCL to form a composite bottom system liner. A leak detection sump will be installed in the low corner of the ponds. Operating solution will be stored in tanks located at the process plant. There will be three tanks, one each for pregnant, barren and heap rinse or wash water. The solution tank sizes are included in process plant design report. The event ponds will be rarely used, and EP-2 most likely will never be used, reserving their capacity for large storm or freshet events or other process plant upset conditions. The rainwater pond will be used to store the water diverted away from the heap by the raincoats, for either use as make-up water for leaching operations, or to be tested and then discharged into the water basin or for other site use such as dust control.

8.2 PROCESS

The ore from the open pit transported to the crusher location and crushed to 80% passing 50mm size. The crushed ore will then be transported to the leach pad facility using haul trucks. The ore will be stacked in nominal 10-meter lifts until sufficient area is available for installing the solution drip lines. The minimum area available for irrigation is 45,500 m^2 for current gold production forecasts. For the first year of operation (year -1) this is the area that will be provided. In subsequent years (year 1 to end of mining) there will generally be double this area at the end of each year to allow for the installation of 200% of the required area as a contingency against freezing of the primary irrigation area.

Leaching will consist of the application of pH-buffered sodium cyanide solution to the heaped ore with drip emitters; emitters will be buried at least 1-meter in the winter. A perforated pipe network embedded in the liner cover material will collect the pregnant leachate solution at the base of the heap. The solution collection pipes will transport the solution flows to collection points where the solution is directed into 450-mm diameter CPE (or equivalent) pipe installed within lined solution collection ditches along both the north and south perimeter of leach pad facility. These CPE solution collection pipes then transport the pregnant leachate solution to the pregnant tank at the process plant prior to being processed through the ADR facility. The Liner in the solution collection ditch around the leach pad will serve as secondary containment for the CPE pipes. Where the pipes exit the leach pad, a dual wall pipe will be installed to provide secondary containment.

Precious metals in the pregnant solution will be recovered using a conventional ADR circuit consisting of carbon adsorption, desorption and metal recover by electro-winning. The resulting barren solution will be returned to the barren solution tank. Barren solution will be buffered to the correct cyanide content, and lime to maintain concentrations and pH, and then recirculated to the heap to continue the leaching cycle. The ADR plant will process at an average rate of about 455 m^3/hr of pregnant solution, with peak capacity of 600 m^3/hr.

8.3 HEAP LEACH PAD

8.3.1 Design Criteria

Based on the criteria developed by Kaminak and consultants, as well as the mining regulatory environment in Yukon, the following criteria were used for the feasibility design of the leach pad facility:

- The Stage 1 pad will accommodate approximately 6 Mt of crushed ore at an average stacked density of 1.6 dry tonnes per cubic meter. Ore will be truck stacked in 10-meter lifts to an elevation of 1310 meters.
- A height of 30 meters of stacked ore material, toe to crest, will be stack going into the winter period to maintain thermal capacity of the heap with the assistance of heated barren solution.
- The ultimate pad will accommodate 47 Mt of ore and will be constructed in 4 Stages, expanding in a westerly direction, with each Stage having separate cells for solution collection. Stages 2, 3 and 4 will be constructed to process 19.24, 39.4 and 47 Mt of ore respectively. (Stage 5 would be a pad expansion to accommodate 60 Mt should economics warrant this expansion.).
- 2.5H:1V (horizontal to vertical) overall heap side slopes have been used for the heap to ensure physical stability and facilitation heap closure and capping.
- The ore will be leached at an average solution application rate of 10 liters per square meter per hour (l/m2/hr).
- All pregnant solutions will gravity drain to the pregnant solution tank at the process plant.
- The process plant will have a maximum operating capacity of 600 m3/hour. Normal operating capacity for the process plant will be 455 m3/hr. Barren solution after gold extraction will be returned to a barren tank for recycling to the heap after the necessary addition of cyanide and lime.
- Event ponds are sized to contain the following cumulative volume:
  (a) the runoff from a PMP event;
  (b) full heap drainage during a power outage or pump failure;
  (c) seasonal water accumulation.
- The PMP storage volume is of sufficient volume to contain the runoff from two back-to-back 200-year, 24-hour storm events. Full heap drainage has been estimated by applying the full operating pregnant flow rate of 455 m^3/hr for a 72 hr period without attenuation. In reality, once power or pumping capacity is lost and the irrigation flow ceases, the rate of flow out of the heap will begin to decline rapidly.
- Storm flows from the terrain surrounding the pad and pond, resulting from the 100-year, 24-hour storm, will be diverted around the facility.
- During the operating life of the plant, the leach pad facility will be a closed system. All solutions will be recirculated, treated and used for rinse water, or evaporated. The closed

system maximizes precious metal recovery and provides a resilient environmental protection.

The following considerations were made for the layout of the proposed heap leach pad facility:

- The permafrost zone Active layer will be stripped to acceptable foundation material for all stages of heap leach construction.
- Earthwork construction requirements (cuts and fills) will be minimized based upon grading requirements. The majority of fill material required will be obtained from grading within the leach pad area, or where additional fill is required, from suitable materials from overburden stripping in the open pits and thus not requiring the development of other borrow sources.
- Solution collection ponds and process facilities will be located on the down gradient and east of the of the heap leach pad providing for gravity flow of all solutions.
- The toe grades for the liner for the heap leach facility will be designed to minimize grading and earthwork volumes, and while ensuring long-term physical stability of the heap.

Table 8-1 itemizes the design criteria used and calculated in this feasibility design report.

Table 8-1 Summary Table of Design Criteria for the Heap Leach Pad Facility

ITEM DESCRIPTION	CRITERIA	VALUE	BASIS
PONDS & CHANNELS			
Peak Storm Event: (channel design)	100-year recurrence, 24-hr event duration	79 mm	L
Pond Capacity:	Normal operational volumes Heap drainage Storm event, Probable Maximum Precipitation (PMP) Freshet Maximum seasonal surplus water	Nil 32,760 m3 280 mm Included in PMP Water balance modeling	A C L L C
Pond Liner System EP-1N & EP1-S	Top geosynthetic liner (smooth) Drainage layer Bottom geosynthetic liner (smooth or single-side textured) Composite clay liner Prepared subgrade	2.5 mm HDPE Drainage layer integral with bottom geomembrane 1.5 mm HDPE drain liner GCL Local soil	C C I
EP2	Top geosynthetic liner (smooth or single-side textured) Composite Bottom Liner Prepared subgrade	2.5 mm HDPE GCL Local soil	C
Rain Water Pond	Top geosynthetic liner (smooth or single-side textured) Composite Bottom Liner Prepared subgrade	2.5 mm HDPE GCL Local soil	
HEAP			
Ore Preparation	Crush size, P80 Agglomeration Cement addition	50 mm none none	B B B

Ore Stacking	Stacking method Lift thickness, nominal As-stacked: dry density Moisture content (dry weight basis) Stacking season Stacking rate, nominal	Trucks 10 m 1,600 kg/m3 2.6 – 4.5% Apr 1 - Dec 31 275 days 5.0 Mtpa 18,182 tpd	B B B B C B B
Leaching	Leach cycle, primary Barren solution: Application method, primary Application rate Average flow Maximum flow Minimum irrigation area Surplus irrigation area (starting in year 2)	40 days drip emitters (buried in winter) 10 L/m2/h 455 m3/h 600 m3/h 45,500 m2 45,500 m2	B I B B B C
Heap Slopes	Inter-bench slope angle (repose angle) Overall slope angle, toe-to-crest (not steeper than)	~1.5H:1V 2.5H:1V	I I
Heap Capacity (min)	Stage 1 (min) Stage 2 (min) Ultimate (nominal) Ultimate (w/ expansion capacity)	6.0 Mt 10.0 Mt 47.0 Mt 61.5 Mt	C C B B
Leach Pad Liner System	Overliner Top geosynthetic liner Bottom geosynthetic liner	500mm ore screened to P_{100} = 38mm plus drainage pipes 2.0 mm LLDPE, single-side textured GCL, reinforced	C I

	Leak detection: Prepared subgrade over natural ground or structural fill	Horizontal wick drains Local soil, rock	
Solution Collection System	Primary corrugated, perforated polyethylene pipes designed to control head over primary liner to a maximum of: Secondary corrugated, perforated polyethylene pipes to control solution head on primary liner to a maximum of: Maximum flow under normal operations, percent of pipe capacity: Maximum flow under normal operations plus 100-yr, 24-h event, percent of pipe capacity:	450 mm 150 mm 50% <100%	C C I I
SLOPE STABILITY			
Minimum factors of safety (FOS)	Static FOS Short-term or temporary slopes Long-term or permanent slopes Seismic pseudo-static FOS Peak strength parameters	 1.3 1.5 1.1	I I I I
Seismicity	PGA_{rock} (MCE) PGA for pseudo-static analysis (50% of peak)	0.116g 0.06g	C I

Basis:

A: Assumed
B: Owner specified
C: Calculated as discussed in following sections
I: Industry standard practice
K: Lorax Environmental Consultants

8.3.2 Leach Pad Liner Selection

The selected leach pad liner system consists of five layers, as discussed below (from the top-most layer downward):

An overliner layer will be provided to protect the geomembrane primary liner from mechanical damage during ore stacking as well as weather conditions before the geomembrane is covered with ore. The overliner will also provide drainage of leach solutions and storm water entering the system both through the permeability of the drainage gravel and a network of drainage pipes installed within the overliner. The overliner material will be 500mm thick and consist of select, durable crushed ore screened to a P_{100} of 38mm.

The primary geosynthetic liner will be a robust, 2.0 mm thick LLDPE material with the bottom side textured to provide an intimate bond with the underlying GCL. The two materials used on nearly every heap leach pad in the industry is high density polyethylene (HDPE) and linear low density polyethylene (LLDPE). LLDPE has been chosen for its superior resistance to puncturing; overall durability; and proven performance in heap leach applications. LLDPE also maintains flexibility at lower temperatures and has a larger window of ambient temperatures during which it can be installed. Leaks through geomembranes occur through damage rather than inherent permeability or transmissivity. In other words, these materials are impermeably for most practical considerations as they are manufactured. The highest risk period of their service life is not the static load of the heap under leach, but rather construction stresses and especially those caused by installing the liner and placing the overliner. The installation specifications include performance of electrical leak location surveys at the completion of each stage of installation to give the highest assurance of a leak-free installation.

The GCL layer will be a heavily reinforced material to give it maximum durability during installation and heap loading, and to provide the highest available shear strengths to increase stability of the slopes of the heap. The permeability of the GCL will be less than 5×10^{-9} cm/sec under little or no load (i.e. under the first lift of ore), and this will decrease to less than 1×10^{-10} cm/sec under the ultimate load of the heap (Athanassopoulos, C. and Smith, M.E., 2014), (Athanassopoulos and Meyer, 2014).

The leak detection system for the leach pad will consist of wick drains placed directly underneath the primary collection pipes between the prepared subgrade and the GCL liner in each of cells for the leach pad. These wick drains be extended to, and booted through the perimeter solution collection trench liner system to discharge into the lined solution collection trench 1 meter above the trench bottom. An observation port will be constructed with a removable cap for each cell, on both the north and south solution collection ditches, to enable visual monitoring and sampling of the leak detection ports as necessary. Details of the leak detection system are shown on drawings 157801-LD2.

Following the stripping of the leach pad foundation of permafrost active layer and unsuitable permafrost materials, the subgrade will be prepared to the design grades and elevations. The final subgrade surface to receive the liner system, will be bedding material with 100 percent passing 38 mm sieve size, and compacted to 95% of maximum density (ASTM 698) using a smooth drum roller for the final surface to receive GCL.

8.3.3 Layout and Grading

Layout of the leach pad considered the following critical design parameters:

- 6 Mt heap capacity for the Stage 1 leach pad,
- 47 Mt heap capacity for the ultimate leach pad,
- Adequate slope to allow positive and rapid drainage of the solution to minimize hydraulic head over the liner system,
- A shallow slope at the toe of the heap to ensure both static and dynamic stability,
- A grading plan that minimizes cut and fill quantities and compresses required time for construction for Stage 1.
- Cell separation berms at 100-meter spacing for future solution management.
- Construction of “stability berms" in the graded toe area of the leach pad.

A 100-meter wide graded toe of the leach pad is incorporated in the leach pad design with a maximum 2 percent slope.

Where possible, final subgrade pad elevations will match the natural ground surface after stripping of topsoil and permafrost to minimize earthworks volumes, leach pad foot print (since cut and fill slopes extend the foot print horizontally), total ground disturbance, time require to construct, and other impacts from construction. In some locations relatively large fills are required along the perimeter of the leach pad to provide proper solution drainage and ensure a stable heap slope. The design grades were developed using balanced volumes of cut and fill were possible, within the accuracy of the current design. Each stage also roughly balances, but there will be some movement of material between adjacent stages. In general, the central portion of the leach pad along the ridge line will be excavated and the perimeters filled. Final design will include further optimization of earthworks for Stage 1 and this may shift the cut and fill balance modestly in favor of less overall earthworks. Where local (inside the leach pad) sources of material are inadequate, mine waste will be used for fill.

All areas that receive structural fill (including mine waste) will be tested during construction in order to ensure that materials are placed at a minimum of 95 percent of maximum dry density as determined by ASTM D698 (standard Proctor) or conform to a method specification for compaction for rocky fills.

1-meter high cell separation berms, running in a north/south direction, will be constructed at 100-meter spacing in an east to west direction to provide solution management capabilities for operations and future reclamation (see drawing 157801-SC1).

8.3.4 Volumetrics for Stages and Stacking

The leach pad is designed for a total tonnage capacity of 47 Mt. Four stages (Stage 1 through 4) of leach pad construction are anticipated during the life-of-mine which is approximately 10 years. These stages were sized to accommodate the stacking tonnages and leaching area criteria for the processing the ore material.

The stage 1 heap leach pad was laid out to accommodate a minimum of ±6 Mt of crushed ore material at a stacked dry density of 1.6 dry tonnes/m^3. For year -1, 3.5 Mt of ore is to be processed and loaded on the leach pad and leached. The daily production rate for the initial 3.5 Mt is at a slightly higher daily production rate of 19,445 tpd than the long term rate of 18,182 tpd. The 6 Mt Stage 1 leach pad capacity minimizes the initial leach pad capital costs and allows for additional leach pad construction in year +1 when the ore processing is 5 Mtpa and additional leach capacity is required for ore production, stacking and leaching.

The majority of the earthworks for the Stage 1 leach pad will be completed in year -2 allowing for minor additional earthworks and fine grading to be done in year-1 prior to the installation of the liner during the initial spring and summer during the 2nd quarter of year -1. Once sufficient area of synthetic liner is installed together with the liner cover and solution collection piping, and is tested for compliance and approved, stacking of Stage 1 ore material can commence.

For the future operational expansion of the leach pad, additional stages will be constructed as required. The stages shown on the drawings are primarily based on the area required for solution application and also to obtain a30 meters of the heap height to maintain the target thermal balance. The leach pad stages are shown on drawings 157801-G1 and G2. Table 8-2 shows a summary of Stages 1 through 4 with tonnages.

Table 8-2: Summary of Stages of Leach Pad Construction

Stage	Elevation (m)*	Pad Area (m^2)	Cum Capacity (Mt)	Est. Year of Construction
1	1270	271,289	7.2	-1
2	1300	226,827	19.2	+1
3	1340	256,281	39.4	+3
4	1350 (final)	64,653	47.3	+7

* Elevation when additional pad is needed

The leach pad can be expanded in a westerly direction accommodating over ±61.5 Mt total capacity should the project ore resource be expanded.

Stacking Plan

Stacking ore material on the leach pad is scheduled to start early in the 3rd quarter in year -1, and leaching of the stacked ore will start beginning of the 4th quarter year -1. A stacking plan was developed for the first 4 years (year -1 through year +3) or Stages 1 and 2. Due to the geometry and configuration of the base of the leach pad, stacking of the ore material will have to alternate from the north to south side of the pad to establish suitable leaching areas and to have the ability to concurrently stack the ore material and leach the stacked ore. The stacking plan maximizes the ore capacity for each stage of the pad construction to where the physical stacking of the ore with the haul trucks is limited to a 100-meter width of top surface area.

Table 8-3 summarizes the sequencing of the ore placement till the end of year +3.

Table 8-3: Summary of Stacking Plan for Years -1 to +3

Year	Stage	Lift	Elev (m)	Ore Lift Depth (m)	Top Area	Volume	Cum Tonnage per year	Stacking Rate	T/Cycle	Days to Stack
-1	1	1-1N	1230	10	31,256	212,459	339,934	19,444	777,778	17
-1	1	1-2N	1240	10	41,260	369,374	930,933	19,444	777,778	30
-1	1	1-1S	1230	10	22,630	103,243	1,096,122	19,444	777,778	8
-1	1	1-2S	1240	10	51,927	319,956	1,608,051	19,444	777,778	26
-1	1	1-3N	1251.6	10	75,568	701,240	2,730,035	19,444	777,778	58
-1	1	1-3S	1250	10	54,361	487,235	3,509,611	19,444	777,778	40
+1	1	1-4S	1260	10	44,850	459,573	735,317	18,182	727,273	40
+1	1	1-4N	1260	8.4	90,688	752,142	1,938,744	18,182	727,273	66
+1	1	1-5S	1270	10	45,948	532,017	2,789,971	18,182	727,273	47
+1	1	1-5N	1270	10	54,590	569,644	3,701,402	18,182	727,273	50
+1	2	2-1N	1250	10	16,584	118,231	3,890,571	18,182	727,273	10
+1	2	2-2N	1260	10	43,263	298,579	4,368,298	18,182	727,273	26
+1	2	2-1S	1250	10	19,956	114,195	4,551,010	18,182	727,273	10
+1	2	2-2s	1260	10	33,485	293,865	5,021,194	18,182	727,273	26
+2	2	2-3S	1270	10	66,856	590,695	945,112	18,182	727,273	52
+2	2	2-3N	1270	10	64360	507,070	1,756,424	18,182	727,273	45
+2	1	1-6N	1280	10	46,,651	551,431	2,638,714	18,182	727,273	49
+2	1	1-6S	1280	10	54,996	569,230	3,549,482	18,182	727,273	50
+2	2	2-4N	1250	10	68,415	626,856	4,552,451	18,182	727,273	55
+2	2	2-4S	1260	10	53,131	480,145	5,320,683	18,182	727,273	42
+3	1+2	1290	1290	10	177,829	1,931,463	3,090,341	18,182	727,273	170
+3	1+2	1300	1300	10	129,856	1,436,763	5,389,162	18,182	727,273	126

8.3.5 Solution Collection System

A drainage layer consisting of a thickness of free draining gravel and drainage piping will be placed over the composite liner system to:

- Provide rapid drainage of process solutions from the base of the ore pile,
- Minimize hydraulic (solution) head over the liner system to reduce risk of solution leakage, and to ensure stability of the heap
- Protect the composite liner system from damage during ore stacking.

Industry standard practice is for the drainage layer to be designed to allow no more than 500 mm of solution head over the liner; such a maximum produces an average head of much less than 500 mm and that applies only to the area under active leach. The remainder of the pad will be dry most of the time. The selected overliner is 38-mm crushed and screened ore selected from the most durable areas of the mine, which have been tested to both provide excellent drainage and puncture protection.

The base of the leach pad design includes separation cells designed such that sections of the leach pad can be isolated should a leak be detected. These separation cells will additionally provide ability to separate pregnant solution from rinse water flows for future progressive reclamation. One-meter high cell separation berms will run in a north-south direction across the entire width of the leach pad with an east-west spacing of nominally 100 meters. The liner system will be placed over the cell separation berms, with the primary cell collection pipe (450-mm diameter corrugated perforated polyethylene [PCPE]) placed alongside the upstream side of the berm (see Drawing 157801-SC1) The primary cell collection pipe reports to the perimeter solution collection pipes (pregnant and rinse water pipes) after transitioning to a non-perforated pipe through a containment berm and a valve system to switch to the appropriate solution pipe. Initially solutions will be directed to the pregnant solution pipe and, once rinsing commences, rinse solutions will be switched to the rinse water collection pipe.

The hydraulic design of the collection pipe system was based upon an equation correlating pipe spacing, solution application rate, liner cover material permeability and pipe spacing (van Zyl, D. et al, 1988). The equation is:

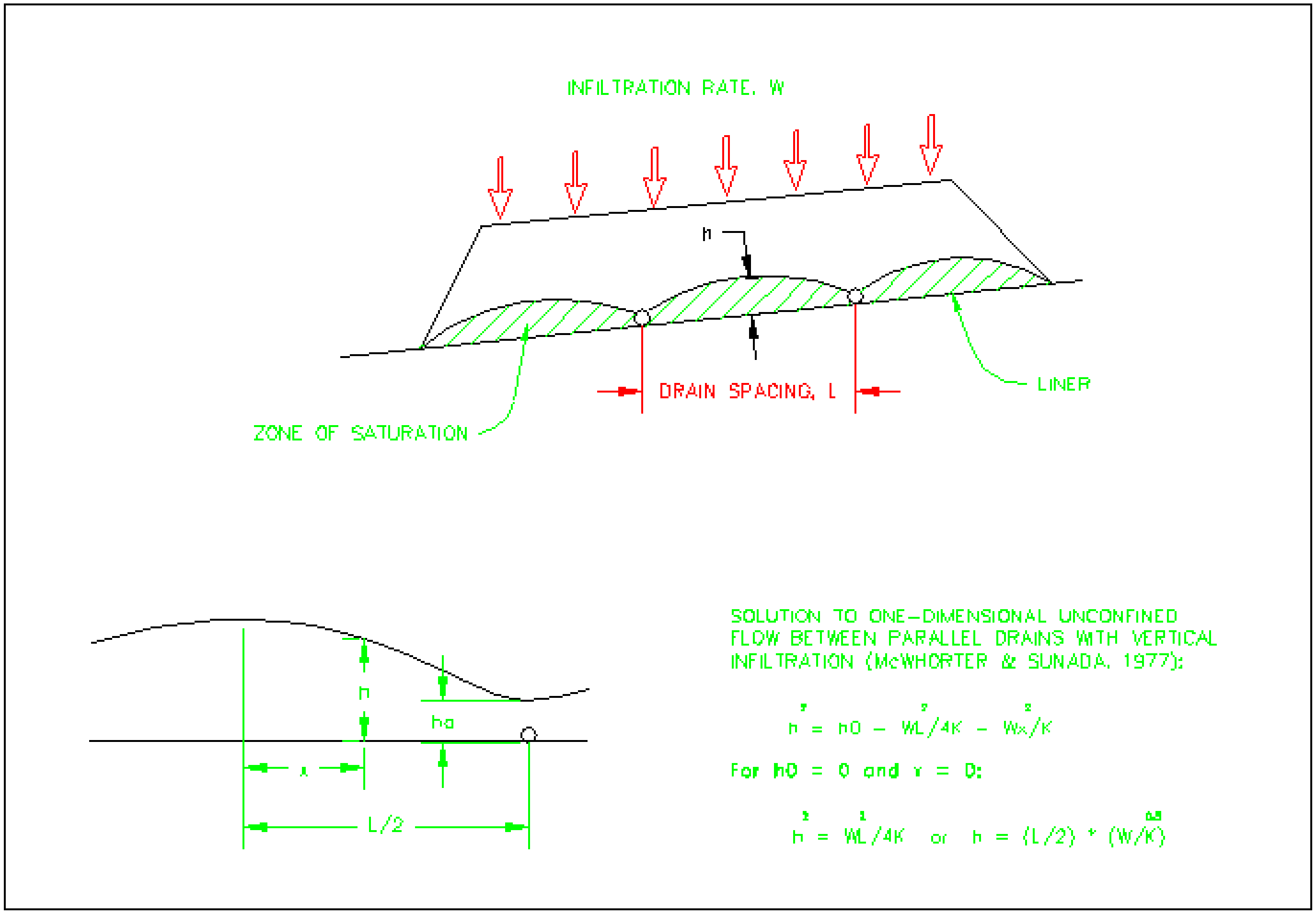
$$h = \frac{L}{2} * \sqrt{W / K}$$

Where,	h (m)	= maximum head across drain spacing
	W (m/sec)	= solution application (irrigation) rate = 2.7778E-6
	K (m/sec)	= permeability of liner cover material = 4.04×10^{-3}
	L (m)	= pipe spacing

The various features of the above equation are shown in Figure 8-1.

The solution application rate, W, was calculated for normal operating flows of 455 m^3/hr to the leach pad and the application rate of 10 liters/m^2/hr. The value for K used in the calculation

Figure 8-1: Pipe Spacing Diagram



was the mean permeability (K=4.04 x 10^{-3} m/sec) from the results obtained from testing of the ore types (see section 5.3.8, Table 4-7). A section showing the various features of the above equation is shown in Figure 8-1. In fact, this is rather conservative since the overliner layer will be constructed of the best quality, higher permeability ore.

In our case, we set the pipe spacing at L = 15 meters which resulted in a head on the liner of h = 0.2m. The equation has a couple of simplifying assumptions, most notably (a) that the head at the pipe is zero and (b) that the pad cross slope is zero. The actual head will be a fraction of the drain pipe diameter, which acts to slightly increase the maximum head. The pad slope is positive, which acts to reduce the maximum head. But the higher than assumed permeability of the overliner reduces the head, likely to less than calculated.

The pipe spacing of 15 m is a practical spacing which allows for aggressive control of head over the liner and the resulting reduction in potential leakage, and for changes in permeability of the liner cover material, either due to break down or material or from migration of fines from the overlying ore material. This pipe spacing can be reconsidered for final design or for future stages as operational experience is gained.

Primary Solution Collection Pipes Sizing (cells)

The primary cell collection pipes, perforated corrugated polyethylene pipe (CPE), were sized for the leach application flow rate (455 m^3/hr) and a storm event flow rate from infiltration equivalent to the solution flow rate, i.e. a total of 910 m^3/hr. A minimum slope of 2 percent and a Manning n=0.012 was used for calculating Depth- Flow Rate in Table 8-4 below.

Table 8-4: Results for 450 mm Diameter Primary Collection Pipe

Depth (m)	Q (m^3/s)	Area (m^2)	Velocity (m/s)	Wp (m)	Yc (m)	Top Width (m)	Energy (m)
0.045	0.009	0.008	1.104	0.290	0.067	0.271	0.1072
0.090	0.039	0.023	1.694	0.419	0.134	0.361	0.2363
0.135	0.086	0.040	2.132	0.522	0.204	0.413	0.3669
0.180	0.147	0.059	2.478	0.617	0.271	0.441	0.4932
0.225	0.220	0.080	2.751	0.709	0.332	0.450	0.6109
0.270	0.294	0.100	2.947	0.799	0.381	0.441	0.713
0.315	0.366	0.119	3.076	0.893	0.412	0.412	0.7976
0.360	0.427	0.136	3.130	0.997	0.427	0.360	0.8596
0.405	0.466	0.151	3.087	1.125	0.433	0.269	0.8909
0.450	0.437	0.159	2.746	1.414	0.430	0.000	0.8345

Secondary Solution Collection Pipes Sizing

In addition to the requirements for the pipe spacing, the area contributing to each secondary collection pipe in the cells is considered. The contributing area of each secondary collection pipe is 100m x 15m spacing, i.e. 1,500 m^2. The maximum flow rate for the solution application rate and infiltration for this contributing area is 15 m^3/hr (0.0042 m^3/ sec). Allowing for storm infiltration, the flow rate is 0.0084 m^3/ sec. The Depth-Flow Capacity of the secondary 150-mm diameter PCPE pipe (Manning's n=0.012) with a minimum 1.0 percent grade indicates a maximum flow rate of 0.018 m^3/sec (maximum flow) for a factor of safety of 2.1. The results for the secondary solution collection pipe sizing are shown in Table 8-5 below.

Table 8-5: Results for 150mm Dia. Secondary Collection Pipe

Depth (m)	Q (m^3/s)	Area (m2)	Velocity (m/s)	Wp (m)	Yc (m)	Top Width (m)	Energy (m)
0.015	0.000	0.001	0.375	0.097	0.018	0.090	0.022
0.030	0.001	0.003	0.576	0.140	0.037	0.120	0.047
0.045	0.003	0.004	0.725	0.174	0.052	0.138	0.072
0.060	0.006	0.007	0.842	0.206	0.070	0.147	0.096
0.075	0.008	0.009	0.935	0.236	0.085	0.150	0.120
0.090	0.011	0.011	1.001	0.266	0.098	0.147	0.141
0.105	0.014	0.013	1.045	0.298	0.110	0.137	0.161
0.120	0.016	0.015	1.064	0.332	0.119	0.120	0.178
0.135	0.018	0.017	1.049	0.375	0.125	0.090	0.191
0.150	0.016	0.018	0.933	0.471	0.119	0.000	0.194

The layouts of the solution collection piping systems for the composite lined pad Stage 1 are presented on Drawing 157801-SC1 and -SC2.

Solution in the 450-mm diameter CPE pipe (Perforated for pregnant solution and non-Perforated for the rinse water) installed in the perimeter solution collection ditches will flow by gravity directly to the pregnant tank at the process plant. The CPE pipe in the perimeter solution collection ditch is considered as the primary containment. Within the leach pad limits secondary and tertiary containment will be provided by a 2-mm thick LLDPE liner and composite GCL liner in the ditch. When the solution collection pipes exit the leach pad the pipe will transition to a buried, dual-walled pipe or a pipe within a pipe to convey solutions to the process plant while maintaining primary and secondary containment.

8.4 EVENT PONDS

8.4.1 Event Ponds

Event Pond sizes were determined by performing modeling the seasonal and multi-year water balance of the heap leach facility (See Appendix G for complete report on Water Balance Modeling). This modeling included consideration of a range of scenarios to allow optimization of the size and timing of each pond. The water balance modeling was performed in three phases, as follows:

- Preliminary modeling assumed no special water management features, which resulted in the need to treat and discharge water from HLP at the rate of 6 liters per sec (l/s) beginning early in the project life and increasing to 60 l/s at the end of ore stacking and for at least 5 years thereafter. This modeling also estimated make-up water demands of up to 32,000 m^3 per month (172,000 m^3 in the first full year of operations);

- Scenario modeling to consider a range of optimizations including managing surplus water in the wet months to be used as make-up water in the dry months, application of raincoats to keep rainwater and snow melt out of the process circuit (with a range of heap coverage and timing), and a sensitivity analysis of the range of as-mined ore moisture contents seen in the metallurgical column testing; and
- A final model using the optimized operating conditions and the final heap and pond configurations.

An economic trade-off study was completed in combination with the second phase of water balance modeling to determine the economically optimum combination of raincoat use (percent of heap area covered by year), pond sizes and water treatment rates. This was coupled with the phase 2 modeling to select the optimum scenario, which became the basis for the design and the final (phase 3) water balance modeling. The second and third phase of modeling also recognized concurrent reclamation of the heap

Key input parameters for the water balance modeling were:

- A synthetic, site-specific daily climate model developed by Lorax, which provided precipitation, maximum and minimum temperatures for the period of October 14, 1986 through September 21, 2014. Lorax also provided a synthetic, site-specific evaporation records for the prior from September 1, 2001 through November 30, 2014;
- The Lorax synthetic record was reordered to provide the dry sequence of years early in the life of the mine to simulate peak make-up water requirements, and the wet sequences of years later in the life to simulate peak surplus water generation. The model was performed on the selected scenario with and without these “embedded” wet and dry cycles as a sensitivity analysis. A 17-year record was constructed to simulate 9 to 12 years of operation and 5 to 8 years of rinsing and transition to closure;
- Ore properties including dry density, as-stacked and drained moisture contents from the metallurgical column testing performed by KCA in Reno, Nevada;
- Leach pad construction beginning in March, Year -1, ore stacking beginning in July, Year -1, and irrigation beginning in September, Year -1;
- Ore stacking to occur July 1 to December 31 in Year -1, and April 1 to December 31 each year after Year -1;
- Irrigation at the average rate of 455 m^3/hr (10 L/m^2/hr) in the Phase 2 modeling (this was varied during Phase 1 and 2 modeling as part of the sensitivity analyses); and,
- Leach pad, heap and pond areas by stage of development (approximate for the second phase of modeling, and aligned with the feasibility design for the third phase).

The results of the modeling were used to aid the selection of the heap, pad and pond design parameters, the timing of construction of the ponds, the timing and extent of the deployment of raincoats, and the timing and amount of water treatment for discharge. The following design criteria resulted from this modeling:

- Raincoats will be used beginning in Year 3 and continuing through rinsing and closure;
- Concurrent closure will begin with heap rinsing in Year 4 and will progress at the rate of 100,000 m^2 annually;
- The combined coverage of raincoats and closed areas will be 40% of the heap area initially and increasing to 80% through Year 8, and 90% through closure;

- Surplus water requiring treatment for discharge into the Latte pit beginning in March, Year 9 (last year of mining) at an average rate of 10 l/s. The Latte pit will be available for water in Year 3 or 4;
- The ponds are sized to contain the maximum seasonal contact water accumulated each year, plus the following additional volumes:
  - Pond freeboard of 500 mm;
  - Full heap drainage to simulate an extended power outage, modeled as 72 hours at the full irrigation rate;
  - Probable maximum precipitation (PMP) applied to the area of the heap less the area of completed closure and raincoats (Lorax memo dated 26 Aug 2015).

The final design includes 4 ponds dedicated to the heap leach area. Three of these are event ponds (called EP-1S, EP-1N and EP-2), and one is the Rain Water pond to store the water flowing off of the raincoats. EP-1S and EP-1N will be built as part of the pre-production construction (Year -1). EP-2 will be constructed in Year 6, when the leach pad has been expanded to a point that EP-1S and EP-1N no longer supply sufficient capacity. The Rain Water pond will be constructed in Year 2 and serves two purposes. First, it will store fresh water during wet months for later use as make-up water in dry months, thereby reducing the net water supply needs. It will also provide a retention basin for routine testing to confirm the pond holds only non-contact water for use on site or for discharge. A summary of the event pond design capacities is shown in Table 8-6 below.

Table 8-6: Event Pond Design Criteria and Containment Capacities

Pond	Design Criteria (m^3) *[1]	To Free Board Elev.	To Pond Crest Elev.	Placed in Service
EP-1N	191,360*[2]	112,349	122,183	Year -1
EP-1S		89,777	97,810	Year -1
EP-2	210,000	222873	240,468	Year +6
Raincoat	47,000	51,925	57,074	Year +3

Note: 1. Design Capacity from Mark Smith memo dated September 2015 utilizing data from the Raincoat Trade-off Study. The capacities include seasonal water accumulation, heap drainage and the 24-hour PMP storm event.
2.This is the combined containment capacity required through year 6.

The criteria developed from the water balance modeling, use of raincoats, and progressive reclamation. Due to the location of the heap leach straddling a ridge, and to accommodate runoff collection from future "raincoats" to be used on the heap leach pad, two ponds need to be constructed for runoff from the Stage 1 pad. These ponds are designate EP-1N and EP-1S. The heap leach pad Event Pond(s) are designed to contain to contain the following:

- All ponds related to heap leach pad runoff and containment utilize a Free Board depth of 0.5 meters;
- Complete drain down from the heap in the event of a barren pumping failure or power outage, simulated as 72-hours at 100% of the design flow rate of 455 m^3/hr;
- Containment to accommodate runoff from 2 each 200-year, 24-hour storm events. Precipitation from the 200-year, 24-hour event is 90mm; and
- Runoff volume from a 24-hour PMP storm event of 280mm (precipitation as evaluated

and recommended from LORAX Memorandum "Probable Maximum Precipitation Estimate- Coffee Creek (Draft)," dated August 26, 2015). The PMP volume is an elective containment criterion for the heap leach facility (not a channel design criteria based on flow rates, and is not currently a regulatory requirement) and was adopted as a conservative approach to solution management. This exceeds industry standards. A common industry and regulatory standard is to contain a volume produced by the 100-yr 24-hr (or twice the 200-yr 24-hour storm events in prescribed circumstances). The PMP criteria used here is equivalent to three 200-yr 24-hr events before considering freeboard.

For event ponds EP-1N and EP-1S, the primary (top) liner will be a 2.5 mm thick smooth HDPE (or other suitable material), and the secondary liner being a 1.5 mm HDPE or LLDPE drain liner to function as a leak detection layer between the liners. These two geomembranes will be underlain by a GCL to form a composite bottom system liner. A leak detection sump will be installed in the low corner of the ponds. Decants from these ponds will be dual walled, heat taped pipes with an internal diameter of 300 mm, and will feed back to the pregnant solution pipe to the process plant. Drain liner is a geomembrane fabricated with a patter on the topside which become drainage channels when installed in contact with an overlying geomembrane. As an alternative to drain liner, a conventional synthetic drainage net can be used.

Event pond EP-2 will have the same primary liner (2.5mm HDPE or other suitable material) underlain by a GCL. No leak detection is envisioned for this event pond as any storm events that are stored in this pond will be the first solutions to be evacuated and used. This pond is also unlikely to every be used because it is sized for the PMP. The decant system for EP-2 will similarly feedback to either the pregnant solution pipe to the process plant or the rinse water pipeline and tank.

Figure 8-2: Volume-Depth Curve for Storm Event Pond EP-1N

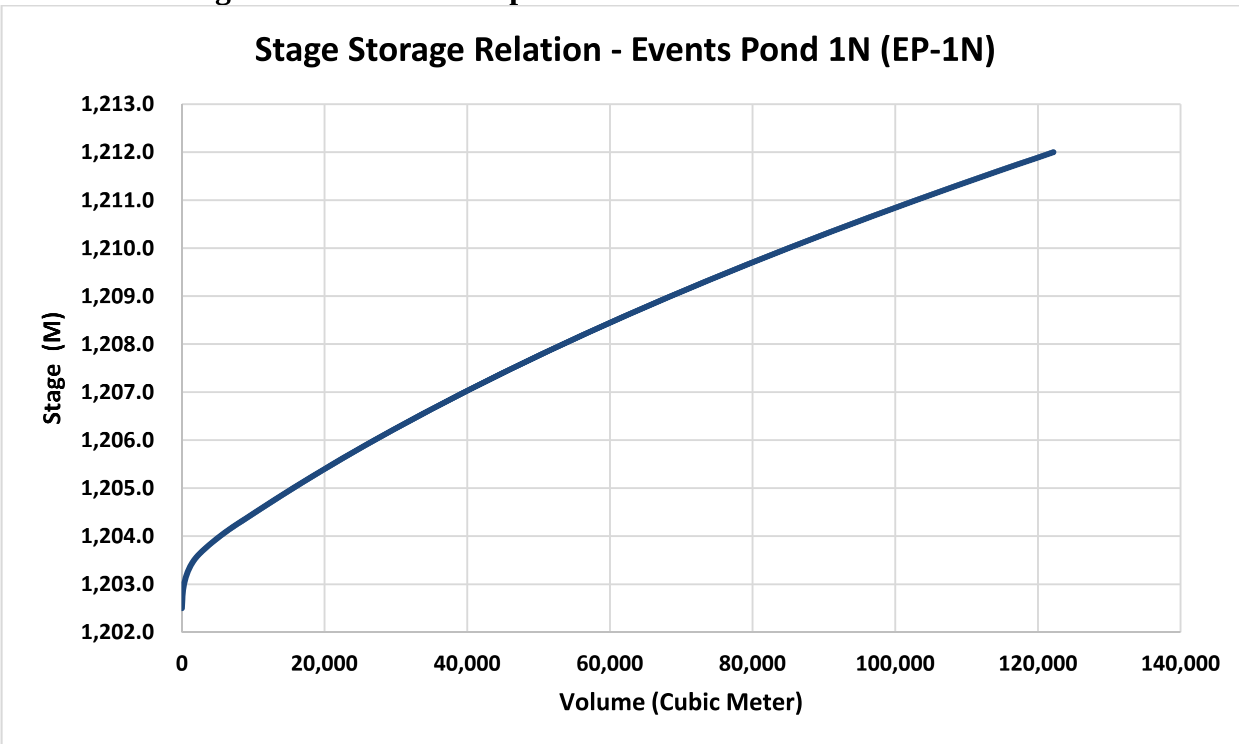


Figure 8-3: Volume-Depth Curve for Event Pond EP-1S

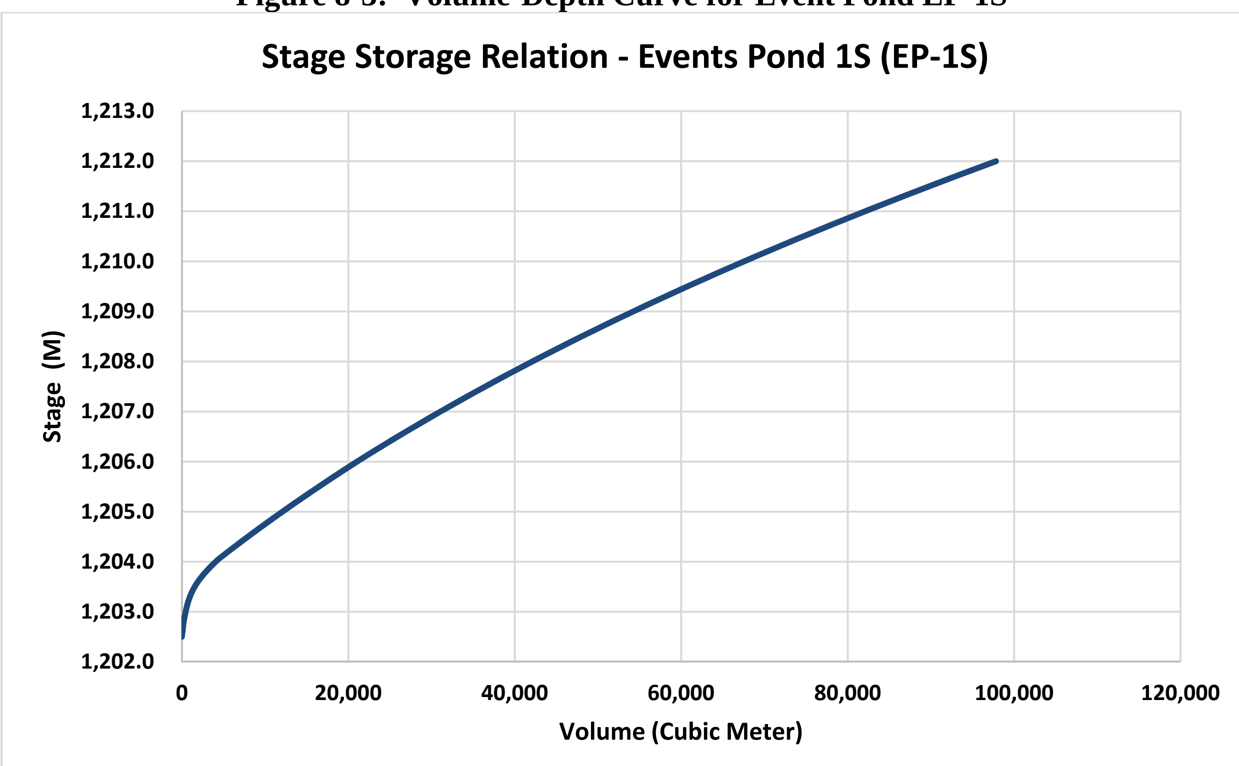
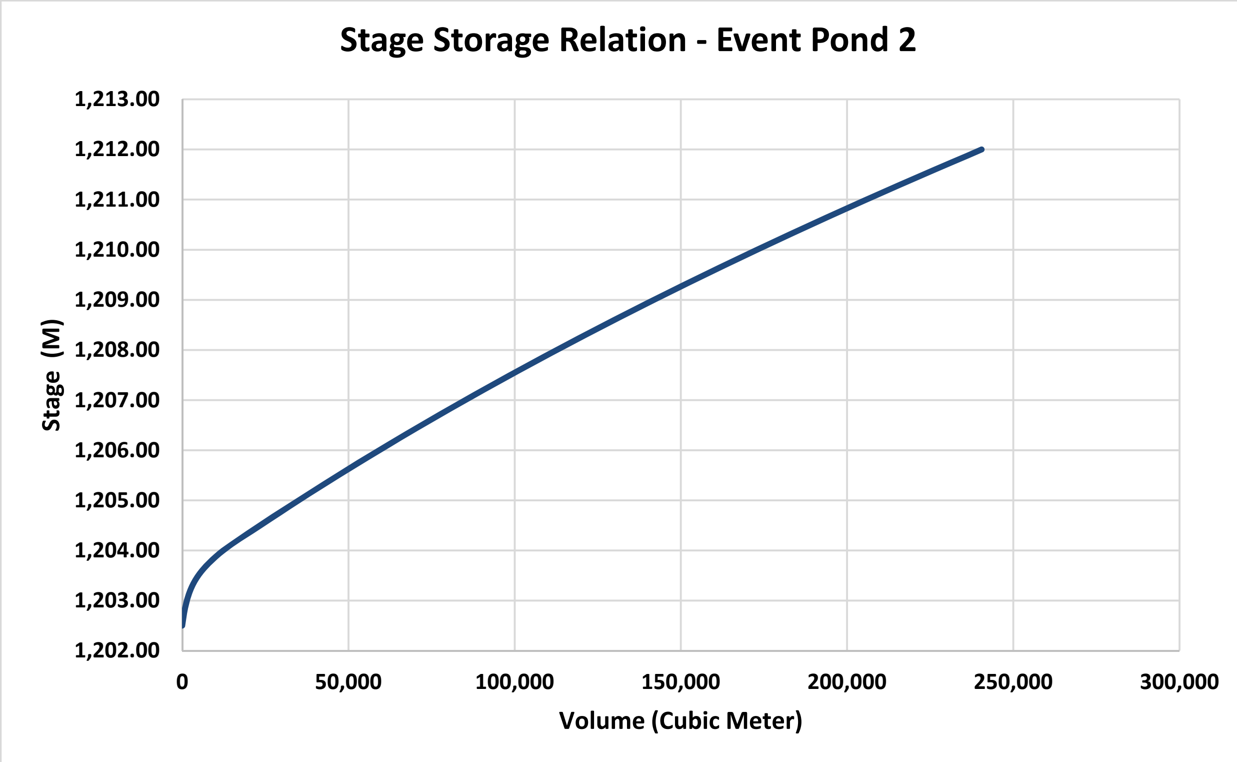


Figure 8-4: Volume-Depth Curve for Event Pond EP-2



8.4.2 Rain Water Pond

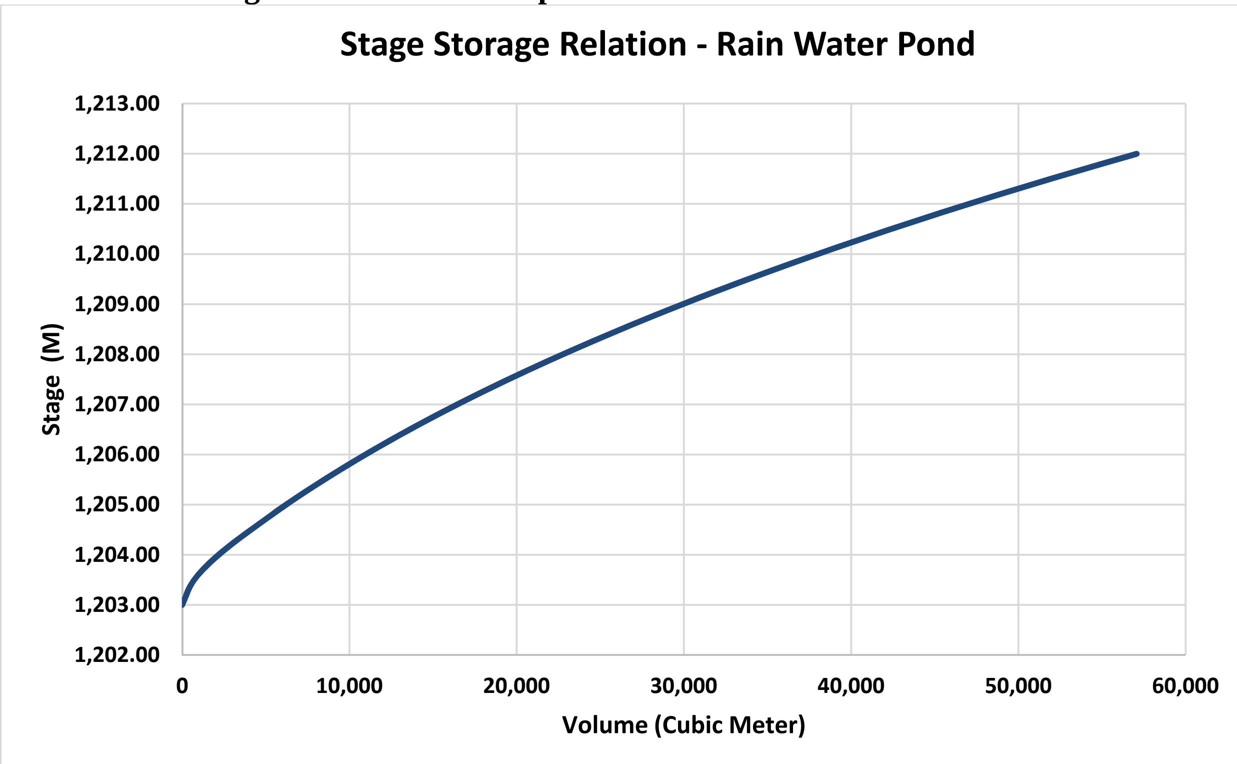
The Rain water pond was designed to contain clean water runoff from snow melt and summer rainfall events from installed "raincoats" on the heap leach spent ore surfaces. The main function of the Rain Water pond is meant to provide make-up water for the heap leach pad and to provide a mechanism for minimizing contact water within the heap leach operation for concurrent closure and reclamation. The Rain Water pond is a holding pond for testing water prior to site use other than the closed heap leach facility, or for discharge after testing for applicable non contamination of the contained water.

The Rain Water pond was used in the water balance modeling to minimize accumulations of solution in the heap leach circuit and to provide make up water for the leaching operations. A complete evaluation on the water balance modeling for the heap leach pad in attached in Appendix G.

The volume-height capacity for the Rain Water pond is shown in Figure 8-5 and location is shown on drawing 157801-G1.

The Rain water pond will have the same liner system as EP-2. No leak detection system has been designed for this pond based on the fact that it is intended to store clean runoff water from the "rain coats." The decant pipeline for the Rain Water pond will similarly be a 300 mm diameter dual walled, heat taped pipeline which can feed the makeup water tank at the process plant, be switched to discharge to the site wide water management system, or direct discharge to the environment pending suitable water quality testing results.

Figure 8-5: Volume-Depth Curve for the Rain Water Pond



The Rain water pond will use the same liner system as for EP-2. No leak detection system has been designed for this pond based on the fact that it is intended to store clean runoff water from the "rain coats." The decant pipeline for the Rain Water pond will similarly be a 300 mm diameter dual walled, heat taped pipeline which can feed the makeup water tank at the process plant, be switched to discharge to the site wide water management system, or direct discharge to the environment pending suitable water quality testing results.

8.4.3 Pond Liner System and Leak Detection System

In order to meet or exceed the current industry standards adopted by mining companies internationally, the two of the event ponds, EP-1N and EP-1S, have been designed using primary(top) and secondary synthetic liners, with a leak detection system placed between the two liners. A GCL liner will be installed below the secondary liner and form a composite liner system. Event pond EP-2 and the rain water pond will have a primary liner overlying a GCL with no leak detection systems.

The primary liners for the ponds will be 2.5-mm thick HDPE geosynthetic liner which has a long established and proven record for non-degradation for exposure to the elements and ultra violet light and should provide protection from ice damage to the primary liner. The secondary liner for EP-1N and EP-1S will be a 1.5 mm thick HDPE or LLDPE geosynthetic drain liner to provide for leak detection between the primary and secondary liners. Underlying the geomembrane GCL liner will be installed to form a composite liner.

A leak detection and recovery sump will be installed for event ponds EP-1N and EP-1S with and access pipe to monitor for leakage, and for extraction of solutions in the event of a leak. Details of the pond liner and leak detection systems are shown on Drawing 157801-P7.

The decision to use HDPE for both the primary and secondary pond liners was based on the following:

- For the Exposed primary liner, HDPE is highly resistant to ultraviolet light and is preferable for ponds and ditches where there is prolonged exposure to sunlight.
- HDPE has a higher tensile strength than LLDPE and would be more resistant to tearing by ice.
- Both HDPE and LLDPE are chemically resistant to the high pH solutions used in the gold heap leaching process and both would be suitable for secondary liners not subject to UV light exposure.
- Further evaluation of the optimum top liner material should be carried out before final design, and that the performance of the as-installed liner in event ponds EP-1N and EP-1S should be re-evaluated prior to final design and installation for EP-2 and Rain Water ponds.

In the event of a leak in the primary liner, the solution will be collected in the leak detection layer and transported by gravity to a leak detection sump in one corner of each pond. The sump will contain a 750 mm thick layer of free-draining gravel surrounding a 200 mm diameter perforated PVC pipe within the sump. The bottom 750 mm of the 200 mm diameter PVC pipe will be perforated and fitted with an end cap, and will extend up the slope from the sump between the primary and secondary liners, and booted through the primary liner to provide an inspection port and for sampling, or evacuating the leak detection sump. In the event of a major leak in the primary liner of the ponds, a pump may be used to evacuate the collected fluids from within the leak detection pipe and sump. The riser pipe will be constructed of solvent-welded 200 mm diameter PVC pipe.

8.5 CONSTRUCTION CONSIDERATIONS

The construction season for the site location in the Yukon Territory is very short. Major grading for the heap leach facility should be done in year -2 with final fine grading being done early in the following year to be able to complete the aggressive construction schedule and complete the leach pad and ponds in a timely manner for ore loading and leaching.

Fill material will be available from overburden stripping of the open pit and from excess cut during construction of the leach pad. Oversize material (500 mm in diameter and larger) will be removed from the fill prior to compaction. Gradational specifications for structural fill is presented in the Technical Specifications attached in Appendix I.

Due to the lack of suitable clay liner material within the project boundary or otherwise within a reasonable haul distance, a geosynthetic clay liner material (GCL) is planned for use in the composite liner systems for both the leach pad and the ponds. A GCL consists of two non-woven, or one non-woven and one woven geotextile materials, bonded together with sodium bentonite powder between the textiles by heat bonding the geotextile threads from top layer to the bottom layer after needle punching. There are a number of advantages to using the geosynthetic clay liners [adapted from (Athanassopoulos, C. and Smith, M.E., 2014)]:

- Installation of a GCL in comparison to that of a natural clay is a GCL can much more rapidly installed providing a suitable bedding surface is prepared ahead on installation. A natural clay will require continual moisture conditioning to prevent the surface from cracking or freezing until the liner is deployed. the bedding material for the deployment of a GCL only requires that the surface is smooth.
- The GCL only requires overlapping and applying 0,5 kg of bentonite per 0.3 meters between the overlapping seams, no welding of the seams is required. The placement of the GCL is rapid resulting in an accelerated construction schedule given the short construction season in the Yukon Territory.
- For a GCL with a permeability under load of 1 x 10^{-10} cm/sec, the hydraulic performance is equivalent to greater than 10 meters of compacted clay with a permeability of 1 x 10^{-6} cm/sec (Meyer and Athanassopoulos, 2014) and outperforms a composite liner consisting of a geomembrane in contact with a 1,200-mm thick compacted clay liner with a permeability of 1 x 10^{-7} cm/sec ((Nguyen, T.B., Stark, T.D., and Choi, H., 2012)).
- GCLs provide significant puncturing protection, reducing the likelihood, size and frequency of any defects occurring in the geomembrane. This further reduces the actual leakage from GM/GCL composite liner systems.
- The geosynthetic clay liner is self-healing in that it swells on contact with water or solutions, thus making cracking or a puncture a mute issue. Further, the high swell nature of sodium bentonite means that the GCL can actually plug (completely or partially) defects that may occur in the geomembrane.
- The quality of the contact between the geomembrane and the GCL will typically be more intimate than that with a compacted clay liner, and this reduces seepage (all other things being equal).
- Because GCLs are a manufactured product, they are much more consistent and reliable in performance than compacted clay liners.
- The performance of compacted clay liners can be significantly degraded by even a few cycles of freezing and thawing, whereas GCLs are substantially more resistant to cycling freezing and thawing (Kraus, J.F and Benson, C.H., 1994).

8.5.1 Construction Quality Assurance

A third party construction quality assurance (CQA) contractor should be assigned to monitor and provide adequate quality assurance testing and inspection during construction to ensure strict adherence to the project Technical Specifications. The Technical Specifications, presented in Appendix I, define the field and laboratory testing programs that will be implemented during construction of this project.

Included in CQA program will be an electrical leak location (ELL) program to detect and allow repair of any defects in the geomembranes before placed in service. Experience has shown that ELL surveys reduce the actual leakage from in-service liners by an order of magnitude or more (Beck, A, 2014; Thiel et al, 2005).

All test results and locations should be summarized in an "as-built" report. A copy of the "as-built report" should be forwarded to the appropriate agency as proof and for final approval that the facility has been completed per the design and specifications.

9 GEOTECHNICAL DESIGN

9.1 CONSTRUCTION MATERIAL SELECTION

Borrow sources for structural fill and other material to be used in the construction of the Coffee Leach Pad and Pond facilities will be from the sources indicated in Section 6, and will be tested prior to placement during construction.

Regardless of borrow sources selected, adequate quality assurance testing and inspection will be provided during construction to ensure that the project specifications are adhered to. The Technical Specifications presented in Appendix I outline the quality assurance-testing program that will be implemented for this project. All test results and locations will be summarized in the "as-built" report, which will be forwarded to the appropriate agency after construction is completed for approval to operate.

9.1.1 Structural Fill Material

Structural fill material will be obtained from within the confines of the Stage 1 leach pad facility during cut and fill grading. Supplemental fill material required will be obtained from open pit stripping operations or from imported fill from local approved borrow areas. The structural fill material will be placed and compacted to 95 percent of maximum dry density as determined by ASTM D-698 (standard Proctor) at or near the optimum moisture content or in the case for coarse material where ASTM-698 cannot be used, a method specification will be used as described in the technical specifications. Structural fill for the Phase1 leach facility shall comply with the technical specifications as defined in Appendix I.

9.1.2 Liner Cover Material

Liner cover material above the composite liner system for the Stage 1 pad facility will be obtained from screening crushed ore material from the open pits. If ore material is not available, crushing and screening of suitable overburden material will be used. The material will be crushed to 80% passing 50-mm size and will have less than 10 percent fines passing the 200-mesh size. Permeability values for this material are conservatively estimated to be 7×10^{-2} cm/sec, which is acceptable for liner cover material. The required specifications for this material are provided in Appendix I, Technical Specifications.

9.1.3 Leach Pad Liner System

The synthetic liner system for the leach pad will consist of composite liner system consisting of a single-sided textured 2-mm thick LLDPE liner underlain by a geocomposite liner (GCL). This composite liner system will be installed for all areas of the leach pad and for all Stages 1 through 4. The 2 mm thick liner and type of material was chosen for the following reasons:

- 2 mm thick LLDPE will be at less risk to puncturing under high stacking loads
- LLDPE liner has a better puncture resistance than HDPE under high loading pressures
- LLDPE has a higher interface friction strength than HDPE

The specifications for the LLDPE liner are included in Appendix I.

9.2 STABILITY

The Coffee leach pad facility will be constructed on moderate sloping natural topography. The surficial soils are relatively shallow and are ice rich materials extending to a depth of approximately 2 meters with an organic topsoil layer of approximately 0.3 meters thick. The underlying weathered granite bedrock transitions to competent bedrock material quickly with little or no ice rich soils evident in the soils logs. The granite bedrock outcrops on the eastern and western perimeter of the leach pad facility. The leach pad area has been graded to minimize cuts in this granite bedrock environment with the cut material to be used for structural fill where appropriate in the construction of the leach pad facility. Grading will mainly occur in the perimeter toe area of the leach pad facility to ensure that stability (static and dynamic[pseudo-static]), will be achieved for peak and post-peak (large displacement) shear strength. "Stability berms" are to be constructed in the perimeter toe area of the leach pad to enhance the stability of the heap. Initial estimates of the length of toe grade and slope were made to achieve suitable factors of safety for the heap for an overall stacked height of 80 meters. The geometry and configuration of the pad and toe slope is shown on stability summary figures for the leach pad and pond deemed as the critical sections, are included in Appendix E.

9.2.1 Geotechnical Conditions

The leach pad site surface is characterized by at, or near surface, granite bedrock that is slightly to moderately weathered, jointed, and with low increasing to moderate RQD values (visual observation of core log CFD0597). This granite bedrock material is uniformly distributed across the leach pad and pond sites. Hole CFB013 located on the northeastern side of the 47 Mt leach pad did encounter a massive near surface andesite dyke from 1.52 to 10.67 meter depth.

All organic topsoil and permafrost materials will be removed from the leach and pond foundations prior to cut and fill grading operations. The final subgrade for the leach pad will either be weathered bedrock or bedrock material free of ice rich soils.

Regional faulting is discussed in detail in section 3.3. There are no identified active faults or faults system for the leach pad area. If faults do exist in the vicinity of the leach pad, they will probably be inactive and predate any Holcene faulting (recent and active faults).

9.2.2 Liquefaction

As described above, the subsurface soils consist primarily of weathered granite bedrock or bedrock. these materials are should not be susceptible to liquefaction.

Ore material stacked on the leach pad is generally not saturated when being leached, typically about 50% saturated. This ore is shown to be relatively permeable and able to dissipate pore water pressure readily. If the ore were saturated or very near to saturation during seismic events, it would be moderately susceptible to liquefaction. Given the low seismic risk of the site, and practical experience indicating that ore material being irrigated does not approach saturation, there is a very minimal risk of liquefaction for the stacked ore material.

9.2.3 Seismic Coefficient Evaluation

Pseudostatic analysis is a very conservative procedure used as the first step in most seismic stability analyses. It is not a dynamic analysis procedure and does not directly account for dynamic/vibratory loading (i.e., the periodicity or cyclic character of the loads and the short duration of loading). Rather, the procedure models seismic impacts by applying a uniform horizontal static force to slices in a conventional limit equilibrium analysis. If the factor of safety using the pseudostatic procedure is found to exceed the design criteria (usually a Factor of Safety of 1.0 to 1.15 depending on the nature of the facility) then is it safe to assume that no further detailed dynamic analysis is necessary. The level of conservatism obtained using the procedure was investigated by Dr. H. Bolton Seed (Seed, 1979). Seed's evaluation showed that, for non-liquefiable soils, the procedure is extremely conservative and the selection of a pseudostatic acceleration coefficient should be based on the expected magnitude of design earthquake. Dr. Seed recommended the use of a pseudostatic acceleration coefficient of 0.1g for moderate earthquakes up to a moment magnitude of 7.2. His work showed that a pseudostatic acceleration coefficient of 0.15g was conservative even for near field earthquakes with moment magnitudes up to 8.5. The pseudostatic acceleration coefficient is *not* a ground acceleration (despite the acceleration like reference to a percentage of gravity) but is merely a *coefficient*. In the absence of the direct application of Seed's recommendations, it is common practice to estimate a coefficient using a site-specific ground acceleration reduced by some factor (0.4 to 0.8 the ground acceleration for example). This practice is commonly considered to produce a conservative (high) estimate of the appropriate acceleration coefficient. For a maximum credible earthquake for a site of up to a magnitude of 8.5, a pseudostatic acceleration coefficient of 0.15g could be used applying Seed's criteria.

Review of the seismic evaluation for the Coffee project (See Appendix C) indicated that for most of the structures anticipated to be constructed at the site, use of the PSHA results for the maximum considered earthquake having a 2% probability of exceedance in 50 yrs is recommended. This maximum considered earthquake (MCE) has a peak ground acceleration (PGA) of 0.116g and the recommended pseudostatic was 0.06g. This pseudostatic acceleration of 0.06g has been used in the stability analysis for critical heap leach pad and pond sections.

9.2.4 Material Shear Strength

The following shear strengths have been used in the stability analysis and are similar materials tested for heap leach facilities elsewhere. The material shear strengths used in the stability analysis are also summarized in tabular form on the stability figures.

9.2.4.1 <u>Crushed Ore Material</u>

The strengths used for the nominal P_{80} 50-mm crushed ore material were estimated from experience and from literature documenting soil type. The dry unit weight for the crushed ore material is approximately 1.6 tonnes/cubic meter as stacked on the leach facility, and will have moisture content of about 10 percent by weight. A moist unit weight of 17.92 kN/m^3 was used in the analysis. A frictional angle of 38° and cohesion of 0 kN/m^2 was used for the stacked ore material in the stability analysis. This is a conservative number based on the triaxial testing performed on 3 different ore material which indicate a frictional angle ranging from 40.2^o to 40.9^o and cohesions from 6.9 to 27.6 kN/m^2.

9.2.4.2 Liner Cover Material

Liner cover material will be obtained by screening the crushed ore material. The following testing was conducted using screened ore material:

- Triaxial testing was performed on Supremo #1 and Supremo 80%
- Hydraulic testing on Latte 72114, latte oxide west, Supremo #1 and Supremo #2
- LSDS testing used screened ore material from Supremo #1, Supremo 80% and Latte 583150
- Puncture testing on Latte 583150 and Supremo 80% screened to 100% passing 38mm

The testing indicated excellent hydraulic conductivity, and triaxial shear strength characteristics. The liner puncture test results for the screened material indicated satisfactory performance. Screened ore material is suitable for liner cover material and should be 100% passing the 38mm sieve with less than 10% fines (0.075 mm).

Due to the coarse nature of the minus P_{100} 38-mm size with less than 10 percent fines (<0.075 mm), the frictional strength should be slightly higher than the crushed ore material. The strength of the cover material has been assumed to be the same as the crushed and stacked ore material but with a higher moist unit weight of 21.1 kN/m^3 due to this layer being almost saturated for most of the leaching cycle.

9.2.4.3 GCL and Interface Friction

This interface liner testing was performed using 2.0mm single-sided textured LLDPE liner supplied by GSE and AGRU using standard and micro-spike texturing respectively, GCL products, liner cover and subgrade materials. These interface liner friction tests are all summarized in section 4.3 in Table 4.3.9. The interface shear strength used in the stability analysis was an average of the results obtained from 2 tests using Bentomat GCL material (one was “DN” and the second “DN-9”), GSE 2.0 mm LLDPE single sided textured material with the textured side against the GCL, with the cover material taken from the Supremo #1 ore sample (post laboratory column leaching to represent an aged state) which was also used for the subgrade material.

The average of the peak friction angle from these test results is 25.5^o with and averaged peak cohesion value of 24 kN/m^2. The moist unit weight for the liner interface used in the stability analysis was 16 kN/m^3. For final design interface friction tests should be performed using actual materials for construction of the liner system of the heap leach pad; especially important is the GCL and textured geomembrane.

For estimating the "post peak" or large displacement shear strength values, the shear stress/ normal stress plot for large displacement strengths were averaged for the two test used above. The failure envelop was extrapolated for the normal stresses above those used as a maximum in the test apparatus (861.8 kN/m^2) to represent the ultimate build out of the heap. The extrapolation used a constant rate of change from previous values to establish the value for the maximum "post peak" shear stress (234.1 kN/m^2) for the estimated maximum normal stress imposed by the stacked ore material estimate of 1574 kN/m^2 (approximately 90 m of ore vertically over the liner, which is slightly greater than the design of 80 m).

9.2.4.4 <u>Bedrock</u>

In evaluating the shear strength for the granite bedrock material, the most recently developed "*Geological Strength Index*" which is an evolution of the *Empirical Rock Strength* approach was used to estimate the granite shear strength properties. Limited data is available for evaluation of the granite bedrock material strength.

From Table 6.3 in. *Slope Stability in Surface Mining*, SME *Slope Stability in Surface Mining* (William A. Hustrulid, Michael K. McCarter, Dirk J. A. van Zyl, 2000), using Fair (smooth, moderately weathered and altered surfaces) and Very Blocky Structure, a geological strength Index GSI = range of 35 to 55. Using a GSI= 45 and a uniaxial strength of 100 mPa for the granite material, from Figure 6.3 results in an estimated Mohr-coulomb friction angle = 40° and a cohesion = 9200 kPa.

A friction angle = 40° and a cohesion = 10,000 kPa were used for the stability analysis. None of the critical failure surfaces entered the bedrock and thus the analyses are not sensitive to these values.

9.2.5 Stability Analysis and Factors of Safety

Four sections, two on the north and two on the south perimeter, were located for the leach pad stability evaluation deemed critical and representative of the worst case scenarios. Similarly, four sections were located for the event ponds, one each for event ponds EP-1N and EP-1S, and two for event pond EP-2. The stability analyses included the use of internal stability berms to enhance the factors of safety and to limit the extent of the grading of the stability toe area. This approach to improving heap stability is well established in the industry and has an excellent performance history (see, for example, (Breitenbach, A.J. and Athanassopoulous, C., 2013)). The locations of the stability sections are included in Appendix E.

SLIDE 6 is a computer slope stability model which uses a limit equilibrium method of slices for determining the factor of safety (FOS).

Both static and pseudo-static stability analyses were performed for the identified sections to produce theoretical failure surfaces resulting from a maximum stacking height for each of the identified sections. For the leach pad and heap, circular and block failures modes were considered along each section for both static and pseudo-static analyses. For the pond embankments only circular failures were considered. The leach pad analysis used the estimated angle of repose of stacked ore material of 1.5H:1V (33.7^{o}), an ore lift height of 10 m, and a 10 m wide bench between lifts, producing an average overall heap slope from crest to toe of 2.5H:1V. Models were run using both peak and large displacement (post peak) shear strength parameters.

Both Spencer and Morgenstein Price methodologies were used. The Morgenstein Price methodology indicated the lower factor of safety in almost all cases analyzed and hence those have been reported herein. The figures for the individual stability analysis output are included in Appendix E.1 through E.10. The factors of safety are summarized in Table 9-1.

Table 9-1: Summary of Factors of Safety for Heap Leach Pad and Ponds

Pad Stage 1 - 1310m Elevation						
	Block Search FOS				**Circular Search FOS**	
	Static		**Pseudostatic**		**Static**	**Pseudostatic**
	Post Peak	Peak	Post Peak	Peak		
Stage 1-N - 1st Berm	1.78	1.95	1.51	1.78		
Stage 1-N - Stability Toe Area	1.82	2.22	1.54	1.90		
Stage 1-N - Ultimate (Over Berms)	1.30	1.89	1.15	1.63	1.17	1.03
- Through Toe Berms	1.39	1.88	1.18	1.61		
Stage 1-S - 1st Berm	1.77	1.92	1.50	1.75		
Stage 1-S - Stability Toe Area	1.79	2.21	1.51	1.89		
Stage 1-N - Ultimate (Over Berms)	1.25	1.85	1.18	1.59	1.17	1.03
- Through Toe Berms	1.36	1.85	1.20	1.58		
Pad Stage 3 - 1350m Elevation (47 Mt Capacity)						
Stage 3-N - 1st Berm	2.00	2.35	1.68	2.02		
Stage 3-N - Stability Toe Area	1.80	2.37	1.50	2.01		
Stage 3-N - Ultimate (Over Berms)	1.29	1.95	1.13	1.65	1.17	1.03
- Through Toe Berms	1.37	1.96	1.14	1.65		
Stage 3-S - 1st Berm	1.99	2.37	1.68	2.02		
Stage 3-S - Stab Toe Area	1.81	2.33	1.54	1.97		
Stage 3-S - Ultimate	1.32	1.95	1.12	1.64	1.17	1.03
- Through Toe Berms	1.37	1.94	1.13	1.64		
Pad Stage 3/4 - 1370m Elevation (61.5 Mt Ultimate Capacity)						
Stage 3/4-N - 1st Berm	1.83	2.21	1.70	1.89		
Stage 3/4-N - Stability Toe Area	1.80	2.28	1.50	1.94		
Stage 3/4-N - Ultimate (Over Berms)	1.36	2.02	1.15	1.70	1.21	1.07
- Through Toe Berms	1.42	2.01	1.27	1.69		
Stage 3/4-S - 1st Berm	1.78	1.93	1.51	1.74		
Stage 3/4-S - Stability Toe Area	1.78	2.18	1.49	1.86		
Stage 3/4-S - Ultimate	1.30	1.90	1.12	1.61	1.17	1.03
- Through Toe Berms	1.34	1.90	1.12	1.61		
Maximum Pond Sections						
Pond EP1 N	-	-	-	-	1.56	1.35
Pond EP1 S	-	-	-	-	1.56	1.35
Pond EP2 NE (Maximum Sect)	-	-	-	-	1.61	1.39
Pond EP2 S (Maximum Sect)	-	-	-	-	1.59	1.37

For the leach pad and ore heap, the BLOCK failure mode (i.e. sliding along the liner interface) was found to be the most critical mode of failure for both static and pseudo-static analyses, which is normally the case for leach pads. All the other sections analyzed indicated acceptable factors of safety for the heap leach pad facility for both the circular and block modes of failure.

The circular factors of safety noted in Table 9-1 above reflect a tendency for failure to approach that of an infinite slope value when failure is limited to the heap material in the absence of a weak foundation layer underneath the pad. This is also common for heaps with ore having little or no cohesion. Any raveled material will be contained within the leach pad.

It is highly recommended that for final design a finite element analysis be performed to allow optimization of the size and location of the stability berms, and to verify he stress states at and near these berms.

9.3 SETTLEMENT

No formal settlement analysis has been performed for the heap leach pad and pond facilities as these areas will all be stripped of all topsoil and ice-rich soils to firm, non-ice rich ground (an average depth of 2 m). Weathered bedrock or competent bedrock material exists below this stripped horizon. Where weathered bedrock material is encountered after the surficial soils are stripped, this weathered bedrock material rapidly transitions to competent bedrock material within a couple of meters. Stacking of heaped ore material on the leach pad area will essential be on competent bedrock material and only the outer fill berms might be on weathered bedrock material. Settlements on the bedrock areas will be minimal and should not pose a risk for rupturing of the liner system of the heap leach pad. The ore itself will settle after it is stacked but this has no effect on any infrastructure. Most of the heap settlement for competent ore, as is the case at Coffee, occurs with initial wetting (first leach cycle) and will be on the order of 1 or 3 percent of the ore depth.

10 HYDROLOGICAL DESIGN

10.1 RUNON DIVERSION AROUND THE PHASE 1 FACILITY

Runon to the heap leach facility is generally not an issue since the heap facility is located on a ridgeline with overland flow directed to the north or south of the facility. Runoff from the leach facilities not having contact with process solutions such as construction sites, fill areas draining away from the leach pad facility etc., will be directed to and be included in the site wide drainage facilities for sediment control and diversion. This is addressed in a separate report by SRK.

Contact surface runoff solutions within the confines of the heap leach facilities require to be contained and managed. Runoff will be collected in the perimeter channel where it will be directed to the diversion channel to the event ponds.

The heap leach facility will be constructed on a ridge running east-west. Heap leach solutions, contact surface water on the heaped ore and "clean water" will drain to the applicable north or south side of the heap facility. The heap facility is essentially divided into two drainages which has necessitated the construction of the north and south solution collection ditches. Similarly, when raincoats are installed in year +3, provision needs to be made to direction the "clean water" to the Rain Water pond as well as directing contact water to the e vent ponds. The north and south diversion channels to the event pond EP-1N and EP-1S, will be constructed as part of stage 1 (year -1), and have been sized to accommodate both contact water and future "clean water" diversion (beginning year +3) to the respective ponds by building a dividing berm within the channel, and isolating the clean water ditch section with a liner over the dividing berm.

10.2 LEACH PAD SURFACE RUNOFF CHANNEL - CONTACT WATER

Surface runoff water on active process areas of the leach pad require to be fully contained within the solution system for the leach pad. Runoff from the heap slopes require to be directed to the event ponds. Standard international practice is to use the runoff associated with the 100-year, 24-hour storm event. The precipitation for the 100-year, 24-hour event for the Coffee site at 1300 m elevation is 79 mm.

The total leach pad area of 819,050 m^2 was used as a worst case scenario in the evaluation of runoff for contact runoff from the heap (assuming no reclamation or raincoats). Runoff will occur to both the north and the south sides of the heap facility. For purposes of runoff estimation, half the pad area, 409,525 m^2 (41 Ha), was used in the evaluation in each of the evaluations for the north and south sides of the leach pad facility. The ore material was assigned a runoff coefficient CN = 75 for the runoff evaluation. The peak flow was evaluated using HYDROFLOW computer program.

The results shown in Table 10-1 indicated a peak flow rate of 1.764 m^3/ sec to be used for the diversion channel design. The results of the runoff evaluation are included in Appendix H.

Table 10-1 : Summary Results for Contact Water Peak Flow Rate Evaluation

Runoff Area	Peak Flow Rate (m^3/sec)	Volume Runoff (m^3)
North HLP	1.681	10,722
South HLP	1.764	10,764

For the channel design, Manning's equation was used and computed using HYDROFLOW:

$$Q = \frac{1.0}{n} (R_h)^{2/3} (S)^{1/2} A \qquad \text{(Metric unit version)}$$

where:
Q = flow rate (m^3/s)
n = Manning roughness coefficient
Rh = hydraulic radius (m)
S = bedslope (m/m)
A = cross-sectional area of flow (m2)

The results for the for the channel design are included in Appendix H.

10.3 RAIN WATER RUNOFF DITCH - NON-CONTACT WATER

Year +3 will see the use of raincoats as a water management tool in conjunction with concurrent reclamation to minimize contact water volumes. Raincoat usage will be coordinated with progressive reclamation and closure; a combination of raincoats and progressive reclamation will cover up to 90% of the leach pad area by the end of the mining life. The total area covered by the rain coats and the reclaimed pad area is 737,145 m^2 (73.1 ha), and again this is divided in half for the north and south runoff, 37.9 ha. The 100-year, 24-hour storm event was used for evaluating non-contact runoff for diversion to the Rain Water pond.

For the area with 10% cover with raincoats, 4.1 hectares (ha), a runoff coefficient CN = 99 was used, and for the 80% reclaimed area with a cover, the runoff coefficient used was the same as that used in the water balance, CN = 95. Similar gradient and slope lengths were used as to the contact water in section 10.2 above. HYDROFLOW was used to compute the peak flow runoff for use in the Chanel diversion design.

The results shown in Table 10-2 indicated a peak flow rate of 6.437 m^3/sec to be used for the diversion channel design. The results of the runoff evaluation are included in Appendix H.

Table 10-2 : Summary Results for Non Contact Water Peak Flow Rate Evaluation

Runoff Area	Peak Flow Rate (m^3/sec)	Volume Runoff (m^3)
North HLP	6.214	24,154
South HLP	6.437	23,967

11 WATER BALANCE

The Coffee Gold Project is located in west central Yukon Territory, Canada approximately 330 kilometers (km) northwest of Whitehorse. The project will involve the design and construction of a number of facilities associated with the proposed gold mining operation at the site. Facilities may include an open pit, a heap leach pad, various ponds and water storage reservoirs, crushers, pipelines, roads, and various mine and administrative buildings. Annual precipitation at the elevation of the proposed heap leach facility is approximately 485 mm, occurring mainly in the summer and fall months, from May to November. Runoff is characterized by a substantial snowmelt period that typically peaks in the month of May. The project involves the construction and operation of a heap leaching facility (HLF) for the extraction of disseminated gold from a low-grade ore. The heap leaching process involves the management of a large volume of weak cyanide solution, and considerations of water balance are of considerable importance to the successful operation of the facility. An operational water balance model has been developed for the proposed HLF at the Project site. The model provides output to evaluate meteoric (weather) impacts on the facility design and to predict the fresh water demand during operations and subsequent post mining fresh water circulation.

The water balance model for a heap leach pad operation is essentially a water budget that tracks all of the water entering and leaving the lined containment system. Sources of water entering the system include pore water delivered with the ore, precipitation falling as rain or snow, and any fresh water (makeup water) added to the system from outside the lined limits of the pad. System losses are a bit more complicated and include three basic categories of loss.

- Evaporative losses (evaporation from ponds and wetted surfaces)
- Losses due to surface tension (wetting of the ore during operations)
- Extraction losses (water removed by either draining or pumping water out of the system)

There are two (2) different classifications of water balance model that can be used to evaluate heap leach pad performance and makeup water requirements. A deterministic model uses a chain of single valued input parameters to produce a series of single valued results. The weather data (which is the primary input) is often derived from some portion of an existing historic record or may consist of a synthetic record generated using the statistical summaries of the historic record. The potential range of variability can only be evaluated in a general sense over the full time history of the model. The second type of model is a stochastic model, where the single valued input parameters are replaced with probability distributions derived from the computed statistics of the observations A Monte Carlo procedure is then used to propagate the uncertainty through the model by sampling all of the input parameter distributions and compiling output distributions for all the results of interest. In this way results are also probability distributions that permit exceedance probabilities to be associated with each event or outcome. Stochastic model results can be very useful in setting system design criteria and quantifying risk. Deterministic models are more useful for design purposes while stochastic models that quantify and manage risk are more pertinent during permitting efforts. Therefore, this design report will address only deterministic model results at this time.

The deterministic model uses the site synthetic precipitation record, number of days of precipitation, temperature and the synthetic evaporation time history for the same time period to track system storage and makeup water demand on a monthly basis, to compute a single value for all variables and results for each month in the record. Precipitation was studied by Lorax Environmental and utilized multiple sources of data including the site specific Coffee Creek climate station, 21 other published climate stations within a distance of 30 km to 225 km of the site (Coffee Gold Project – Meteorology Baseline, April, 2015).

Results of the deterministic modeling are as follows. In general, outside makeup water demand only appears in year 1 and 2 after the initial water recruited for startup in year -1 and early year 1 is used up in charging the system with water and wetting the ore after stacking begins in mid-year -1. However, due to a moderate level of precipitation, cool temperatures with a low level of evaporation, and the accumulation of moisture in an annual snowpack, the system tends to accumulate water over time. Measures directed at limiting the buildup of water (raincoat covers) are required by year 3 to help exclude meteoric water from the system. Pumping to treatment is likely to be required by year 9 to control the volume of water stored in Event Pond 1 (event ponds EP-1N and EP-1S are combined in the model and are represented as Event Pond 1). Reclamation also helps to shed clean runoff to the environment and limit the accumulation of water.

The water balance model covers the period of leach pad operation that includes startup and Stage 1 through Stage 4. Modeling also includes up to two (2) years of post mining leaching and continued rinsing. Construction is assumed to start in March or April of 2018 (year -2) with ore placement beginning in July of 2019 (year -1) and assumed to continue through November of 2028 (year 9). Operations effectively end with the termination of gold production in late 2030 (year 11). Upon completion of active leaching operations, solution management will be required until such time as the closure cover is established and clean runoff is diverted off the facility. Once the solution draindown rate falls to a level that can be safely and passively contained in the post-closure Event Pond(s), active solution management can cease (i.e., no pumping). The water balance model does not model these post-closure conditions, and therefore results are presented only through the end of 2032.

Construction/Operations Stages for the HLF are as follows:

- Stage 1 – Startup through a heap volume of about 7.2 Mt on a lined footprint of 271,289 m^2 – July of year -1 through October of year 1
- Stage 2 – Expansion of the HLF lined footprint to 498,116 m^2 and continued stacking to a heap volume of 19.2 Mt – November of year 1 through March of year 4
- Stage 3 – Expansion of the HLF lined footprint to 754,397 m^2 and continued stacking to a heap volume of 39.4 Mt – April of year 4 through May of year 8
- Stage 4 – Expansion of the HLF lined footprint to 819,050 m^2 and the termination of mining and ore production at a maximum heap volume of 47.3 Mt – June of year 8 through November of year 9
- No additional ore stacking but continued irrigation of the ore stack for gold production, rinsing, and the beginning of closure, and reclamation – December of year 9 through December of year 13

Results show that normal operating volumes in Event Pond 1 increases over time as the lined footprint of the HLF increases and the system recruits more and more meteoric water. Event Pond 1 levels peak during Stage 3 as the lined footprint grows to nearly 3 times what it was in Stage 1. Use of raincoats begin in year 3 and the resulting clean runoff is either held for use as makeup water or ultimately released to the environment. Clean water discharge from reclaimed areas (expected to begin in year 4) increases as the % of the HLF covered increases. Reclamation reduces the amount of raincoat coverage required. Outside Makeup water demand is highest during startup and Stage 2 and decreases over time as water accumulates in the system.

The results are tabulated in the water balance report in included in Appendix G.

12 ENVIRONMENTAL MONITORING

12.1 SOLUTION PONDS

The solution ponds have been designed with a leak detection/collection layer between the primary and secondary synthetic liners. In the event of a leak in the primary liner, the solution will be collected in the leak detection layer and transported by gravity to a sump in one corner of ponds EP-1N and EP-1S, as shown on Drawing 157801-P1 and -P2 respectively. The typical sump design is shown on Drawing 157801-P7. The sumps will include solution pond observation ports for monitoring each sump location, and the pumps to evacuate any solution will have totalizing flow meters to allow monitoring of the flow over time.

12.2 LEACH PAD

The leach pad has been designed with individual cells for each stage of construction. Each of these cells has a leak detection pipe that discharges into the perimeter solution trench with an observation port to visually monitor and if needs be, to sample and measure flow from the individual ports to determine compliance with the permit regulations.

Any flows from the leak detection pipes will be contained in a lined solution collection trench and will collected by the perforated CPE pregnant pipeline and assimilated back into the contained system. Details of leak detection are shown on Drawing 157801-LD2.

12.3 MONITORING WELLS

A series of monitoring wells will be installed up and down-gradient of the leach pad and mine operations whose primary function will be to initially establish baseline water quality for the site, and thereafter to monitor ground water quality during operations and post closure of the mine and leach pad. The monitoring wells will provide information on any degradation of groundwater flow from the mine and leaching operations. Individual processes such as the leach pad and the ponds will be monitored separately to mitigate contaminant migration and these individual process leak detection systems will form the primary defense against contaminant migration from the process areas.

13 USE OF THIS REPORT

This report was prepared for the exclusive use of Kaminak, it's staff, and consultants for specific application in design of portions of this project at the site described in this report. However, this report, conclusions and interpretations should not be construed as a warranty of the subsurface conditions.

The findings, recommendations and design are based on results of the field explorations and laboratory tests, combined with an interpolation of soil conditions between exploration locations and our understanding of the project as stated in this report. If project details change or additional data becomes available, MINES should be notified so recommendations and design can be verified or modified.

In the event of changes, the conclusions, design, or recommendations contained in this report shall not be considered valid unless the changes are received and conclusions of this report are modified or verified in writing.

14 REFERENCES

U.S. Army Corps of Engineer. (1982). *Engineering and Design Stability for Earth and Rockfill Dams.*

Abrahamson and Silva. (2013). Update of the AS08 ground-motion prediction equations based on the NGA-West2 data set. *Pacific Earthquake Engineering Research Center*.

Allan et al. (2013).

Allan, M.M., Mortensen, J.K., Hart, C.J.R., Bailey, L.A., Sánchez, M.G., Ciolkiewicz, W., McKenzie, G.G., Creaser, R.A. (2013). Magmatic and metallogenic framework of west-central Yukon and eastern Alaska. In: Tectonics, Terranes, Metallogeny and Discovery in the northern circum-Pacific region, M. Colpron, T. Bissig, B. Rusk, and J. Thompson (eds.). *Society of Economic Geologists, Special Publication 17*, p. 111-168.

Athanassopoulos and Meyer. (2014). GCLs in Heap Leach Pads: State of the Art practice. *Proceedings of Geosynthetics Mining Solutions 2014.* Vancouver Canada.

Athanassopoulos, C. and Smith, M.E. (2014, Nov). Design Considerations for GCLs in leach pad liner systems. *Geosynthetics magazine*.

Atkinson & Boore. (2003). Empirical ground-motion relations for subduction-zone earthquakes and their application to Cascadia and other regions. *Bulletin of the Seismological Society of America, Vol. 93*, p. 158-164.

Beck, A. (2014). The financial benefits of electrical-leak location to the mining industry, . *Proc. of Geosynthetics Mining Solutions.* Vancouver, BC.

Bennett et al. (2010).

Beranek, L.P., and Mortensen, J.K. (2011). The timing and provenance record of the Late Permian Klondike orogeny in northwestern Canada and arc-continent collision along western North America. *Tectonics, Vol. 30*, 1-23.

Berman, R. R. (2007). Permian to Cretaceous polymetamorphic evolution of the Stewart River region, Yukon-Tanana terrane, Yukon, Canada: P-T evolution linked with in situ SHRIMP monazite geochronology. *Journal of Metamorphic Geology, Vol. 25*, p. 803-827.

Breitenbach, A.J. and Athanassopoulous, C. (2013). Improving the Stability of High Fill Load Structures Built on Low-Strength Geosynthetic Interfaces. *Proc. Geosynthetics.* Long Beach, CA, Apr 1-4, 2013.

Breitenbach, A.J. and Smith, M.E. (July 2012). Design Considerations for Impounding Valley Leach Pads. *Mining Engineering, Vol. 64*(7), pp. 49-49.

Buitenhuis, E. (2014). The Latte Gold Zone, Kaminak's Coffee Gold Project, Yukon, Canada: Geology, Geochemistry, and Metallogeny. *M.Sc. Thesis*. Department of Earth Science, The University of Western Ontario, London, ON Canada.

Colpron, M., Mortensen, J.K., Gehrels, G.E., and Villeneuve, M. (2006). Basement complex, Carboniferous magmatism and Paleozoic deformation in Yukon-Tanana terrane of central Yukon: Field, geochemical and geochronological constraints from Glenlyon map area. (M. C. Nelson, Ed.) *Geological Association of Canada, Special Paper 45*, p. 131-151.

Colpron, M., Nelson, J., and Murphy, D.C. (2007). Northern Cordilleran terranes and their interactions through time. *GSA Today, Vol. 17*, p. 4.

Colpron, M., Nelson, J.L. and Murphy, D.C., M. (2006). A tectonostratigraphic framework for the pericratonic terranes of the northern Cordillera. In: Paleozoic Evolution and

Metallogeny of Pericratonic Terranes at the Ancient Pacific Margin of North America, M. Colpron and J.L. Nelson (eds.). *Geological Association of Canada, Special Paper 45*, p. 1-23.

Douglas, T. L. (2002). Geochronologic and termobarometric constraints on the metamorphic history of the Fairbanks Mining District, western Yukon-Tanana terrane, Alaska. *Canadian Journal of Earth Sciences, Vol. 39*, p. 1107-1126.

Dusel-Bacon, C., Lanphere, M.A., Sharp, W.D., Layer, P.W., and Hansen, V.L. (2002). Mesozoic thermal history and timing of structural events for the Yukon-Tanana Upland, east-central Alaska: 40Ar/39Ar data from metamorphic and plutonic rocks. *Canadian Journal of Earth Sciences, Vol. 39*, p. 1013-1051.

E. Hoek & J.W. Bray. (1981). Rock Slope Engineering. In E. H. Bray, *Rock Slope Engineering* (3rd ed.). IMM.

Erdmer, P. G. (1998). Paleozoic and Mesozoic highpressure metamorphism at the margin of ancestral North America in central Yukon. *Geological Society of America Bulletin, Vol. 110*, p. 615-629.

Gabrielse H. and Yorath, C.J., 1991. (1991). Tectonic synthesis, Chapter 18. In: Geology of the Cordilleran Orogen in Canada (Gabrielse H, Yorath CJ eds.). *Vol. 4*, p. 677–705.

Johnston, S. (1999). .Large-scale coast-parallel displacements in the Cordillera: a granitic resolution to a paleomagnetic dilemma. *Journal of Structural Geology, Vol. 21*, p. 1103-1108.

Kraus, J.F and Benson, C.H. (1994). Effect of freeze-thaw on the hydraulic conductivity of barrier material: Laboratory and field evaluation. *US EPA Environmental Geotechnics, 95-5*.

Lorax Environmental Consultants. (2016). *Hydro-meteorology Summary Report.* Kaminak Coffee Prtoject Feasibility Study, Appendix J1.

Lorax Environmental Consultants. (2016). *Memorandum: Extreme Precipitation depths and Snowmelt.* Kaminak Coffee Project Feasibility Study, Appendix J.1-1.

Lorax Environmental Consultants. (2016). *Regional Ground Water Assessment.* Kaminak Coffee Project Feasibility Study, Appendix J.2-2.

MacKenzie, D. a. (2010). Structural controls on hydrothermal gold mineralization in the White River area, Yukon. In: Yukon Exploration and Geology 2009, K.E. MacFarlane, L.H. Weston and L.R. Blackburn (eds.). *Yukon Geological Survey*, p. 253-263.

MacKenzie, D.J. and Craw, D. (2012). Contrasting structural settings of mafic and ultramafic rocks in the Yukon-Tanana terrane. In: Yukon Exploration and Geology 2011, K.E. MacFarlane and. (K. M. Sack, Ed.) *Yukon Exploration and Geology*, p. 115-127.

MacKenzie, D.J., Craw, D. and Mortensen, J. (2008). Structural controls on orogenic gold mineralization in the Klondike goldfield, Canada. *Mineralium Deposita, Vol. 43*, p. 435-448.

McCausland, P. S. (2006). Assembly of the northern Cordillera: New paleomagnetic evidence for coherent, moderate Jurassic to Eocene motion of the Intermontane belt and Yukon-Tanana terranes. (R. E. J.W. haggart, Ed.) *Special Paper 46*, 147-170.

McKenzie, G. A. (2013). Mid-Cretaceous orogenic gold and molybdenite mineralization in the Independence Creek area, Dawson Range, parts of NTS 115J/13 and 14. (M. N. K.E. MacFarlane, Ed.) *Yukon Exploration and Geology 2012*, 79-97.

Mortensen J.K., 1. (1992). Pre-mid-Mesozoic tectonic evolution of the Yukon-Tanana Terrane, Yukon and Alaska. *Tectonics, Vol.11*, 836–853.

Nguyen, T.B., Stark, T.D., and Choi, H. (2012). Comparison of four composite landfill liner systems considering leakage rate and mass flux. Retrieved from http://tstark.net/wp-content/uploads/2012/10/CP103.pdf

Piercey, S. a. (2009). Composition and provenance of the Snowcap assemblage, basement to the Yukon-Tanana terrane, northern Cordillera: Implications for Cordilleran crustal growth. *Geosphere, Vol. 5*, p. 439-464.

S2 of Berman et al. (2007).

Sánchez, M. A. (2013). Structural Control of Mineralization Recognized by Magnetite-Destructive Faults of the Western Yukon and Eastern Alaska Cordilleran Hinterland (Poster). *Society of Economic Geologist (SEG).* Whistler BC.

Schrauf, et al. (February 2014). Evaluation of the Effectiveness of a Thermal Cover for Obtaining Elevated Ore Temperatures to Facilitate Thermophylic Heap Leaching of Copper Sulfide Ores. Salt Lake City, Utah: Proc of Society of Mining Engineers (SME) annual meeting.

Seed, H. (1979). Considerations in Earthquake-Resistant Design of Earth and Rockfill Dams. *Geotechnique, 29*(3), 215.283.

Selby, D. C. (2002). Absolute timing of sulphide and gold mineralization: A comparison of Re-Os molybdenite and Ar-Ar mic methods from the Tintina Gold Belt, Alaska. *Geology, Vol. 30*, p. 791-794.

Sinha, K and Smith, M.E. (Sep 2015). Cold Climate Heap Leaching. *Heap Leach Solutions.* Reno.

Smith, M.E. and Parra, D. (2014). Leach Pad Cost benchmarking. *Heap Leach Solutions.* Lima Peru: Infomine.

Steffen Robertson & Kirsten. (2016). *2015 Geotechnical Field Investigation for Kaminak Coffee Gold Project,.* Denver USA.

Thiel, R., Beck, A. and Smith, M.E. (2005). The Value of Geoelectric Leak Detection Services for the Mining Industry. *Geofrontiers Conference Proceedings.* USA, 2005.

U.S. Army Corps of Engineers. (1982). *Engineering and Design Stability for Earth and Rockfill Dams.*

U.S. Department of the Interior. (1980). *Earth Manual.* Water Resources Technical Publication.

van Zyl, D., Hutchinson, I., and Kiel, J. (1988). Introduction to Evaluation , Design and Operation of Precious Metal Heap Leaching Projects. Cushing-Malloy, Inc.

Wainwright, A.J., Simmons, A.T., Finnigan, C.S., Smith, T.R., and Carpenter, R.L., 2011. (2011). Geology of new gold discoveries in the Coffee Creek area, White Gold District, west-central Yukon. (L. W. K.E. MacFarlane, Ed.) *Yukon Exploration and Geology 2010*, p. 233-247.

William A. Hustrulid, Michael K. McCarter, Dirk J. A. van Zyl. (2000). Slope Stability in Surface Mining. SME.

COFFEE GOLD PROJECT

HEAP LEACH FACILITY

CONCEPTUAL DESIGN FOR 67mt

GOLD CORP INC.

LIST OF DRAWINGS

REVISION	DRAWING No.	TITLE
A	T-1	TITLE SHEET
A	G-1	GENERAL HEAP LEACH FACILITIES LAYOUT
A	G-2	HEAP STAGE 1 TO 4 CONFIGURATIONS
A	G-2A	HEAP STAGE 5 CONFIGURATION AND CROSS SECTION
A	G-3	HEAP LEACH PAD SECTIONS
A	G-4	HEAP LEACH PAD SECTIONS
A	L-1	STAGE 1 HEAP LEACH FACILITY LAYOUT AND GRADING PLAN
A	L-2	STAGE 1 HEAP LEACH PAD NORTH AND SOUTH STABILITY TOE SECTIONS
A	SC-1	STAGE 1 SOLUTION COLLECTION PLAN
A	SC-2	STAGE 1 SOLUTION COLLECTION PLAN TYPICAL CELL OUTLET
A	SC-3	STAGE 1 SOLUTION COLLECTION NORTH AND SOUTH OUTLET SECTIONS
A	LD-1	STAGE 1 LEAK DETECTION PLAN
A	LD-2	STAGE 1 LEAK DETECTION PLAN TYPICAL CELL OUTLET
A	P-1	NORTH EVENT POND (EP-1N) LAYOUT AND GRADING PLAN
A	P-2	SOUTH EVENT POND (EP-1S) LAYOUT AND GRADING PLAN
A	P-3	POND FACILITIES LAYOUT AND GRADING PLAN
A	P-4	RAIN WATER POND LAYOUT AND GRADING PLAN
A	P-5	POND SECTIONS
A	P-6	CHANNEL SECTIONS
A	P-7	POND AND CHANNEL SECTIONS AND DETAILS
A	P-8	EVENT POND 1 (EP-1N) OVERFLOW PIPE SECTION AND DETAILS

PREPARED FOR

GOLDCORP

GOLD CORP INC.
SUITE 3400 - 666 BURRARD SREET
VANCOUVER, B.C. CANADA V6C 2X8

PREPARED BY

THE MINES GROUP INC.

1325 AIRMOTIVE WAY #175U
RENO, NEVADA 89502
775-322-7622 (PH)

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	DESCRIPTION	DATE	BY
A	ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

TITLE SHEET

CONSULTANT PROJECT No.	CONSULTANT DWG No.		
178201			
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		T-1	A

C:\COMMON\A1-TUPRAG.DWT

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

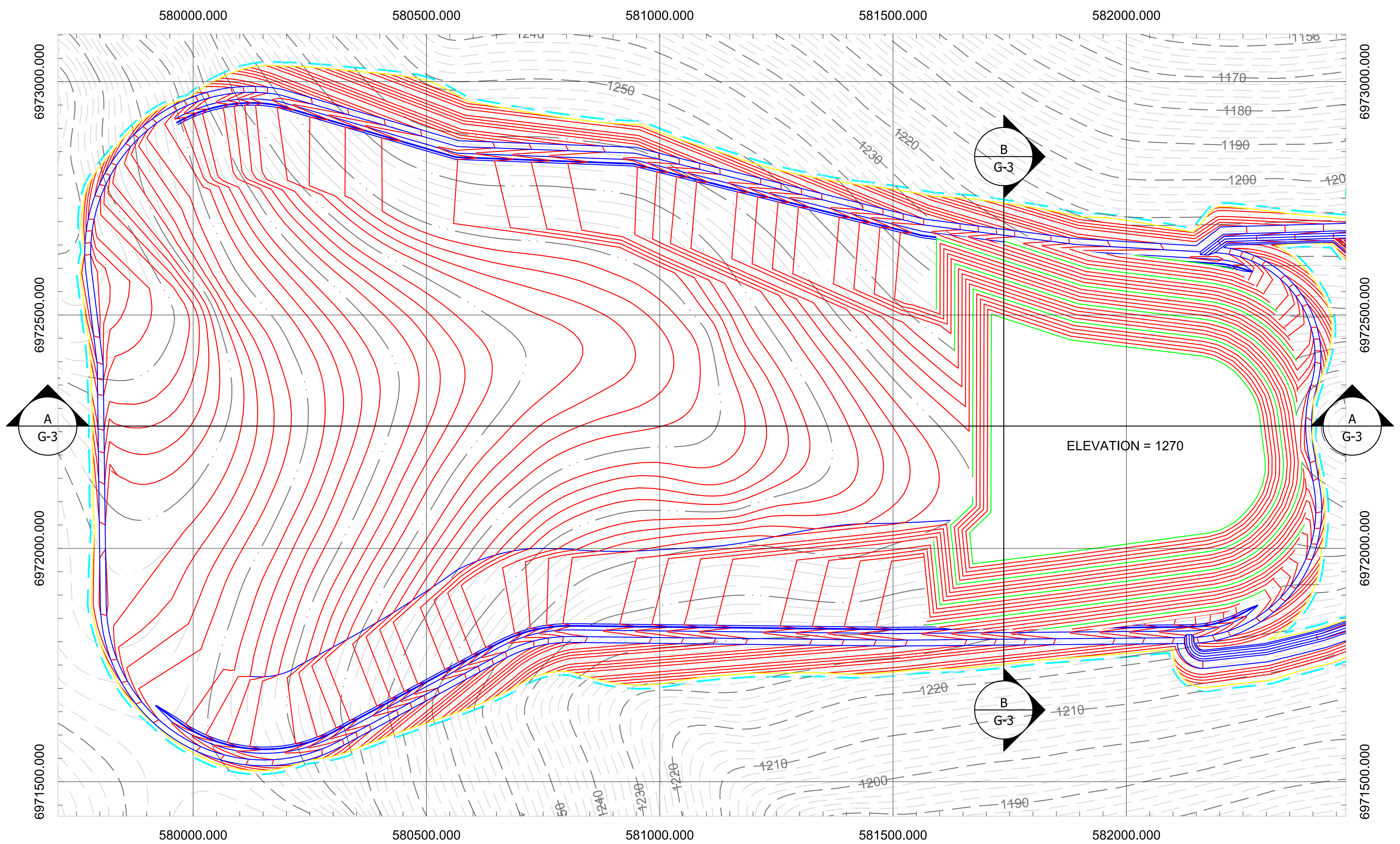
GOLDCORP

THE MINES GROUP INC.

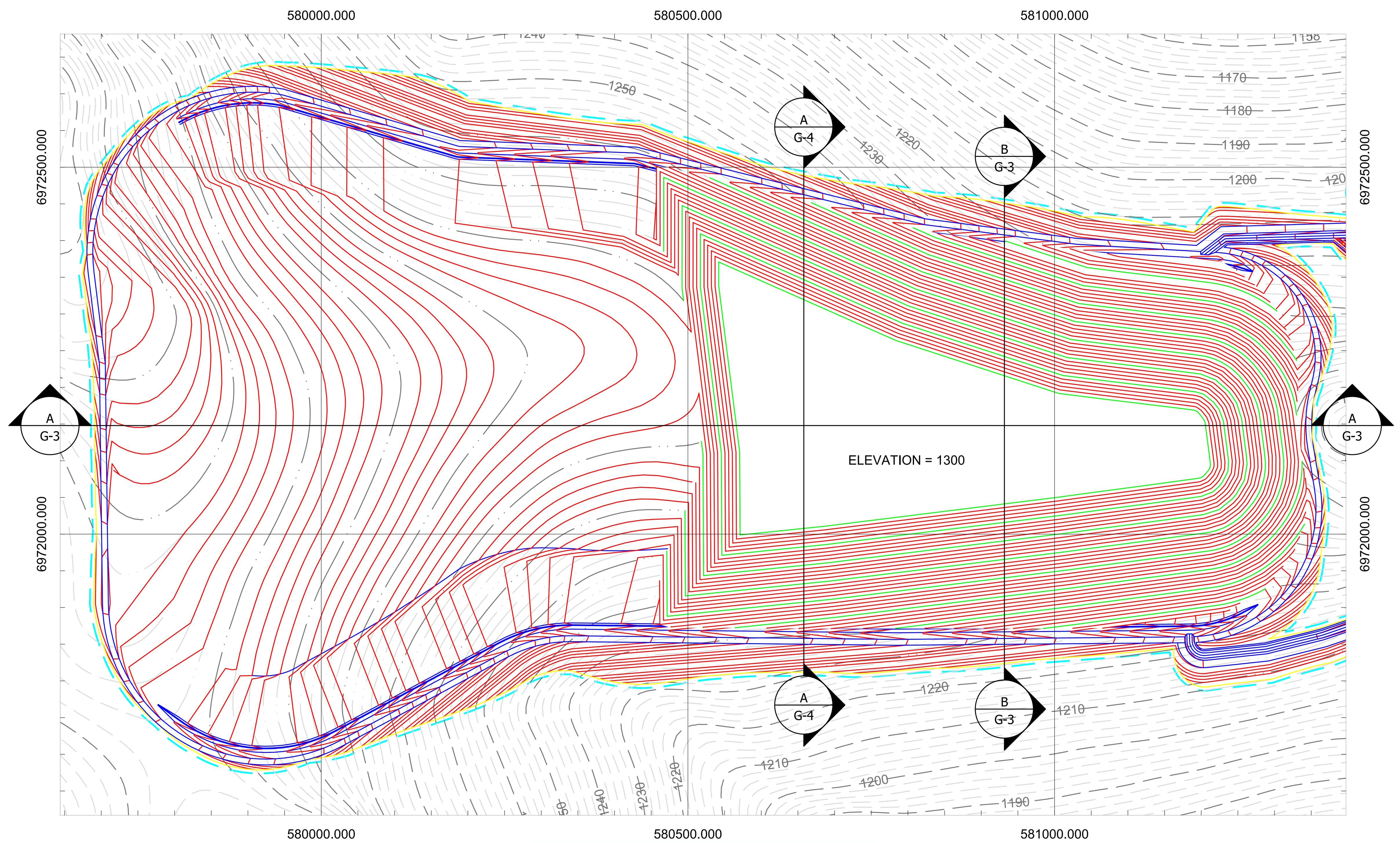
COFFEE GOLD PROJECT

GENERAL HEAP LEACH FACILITIES LAYOUT

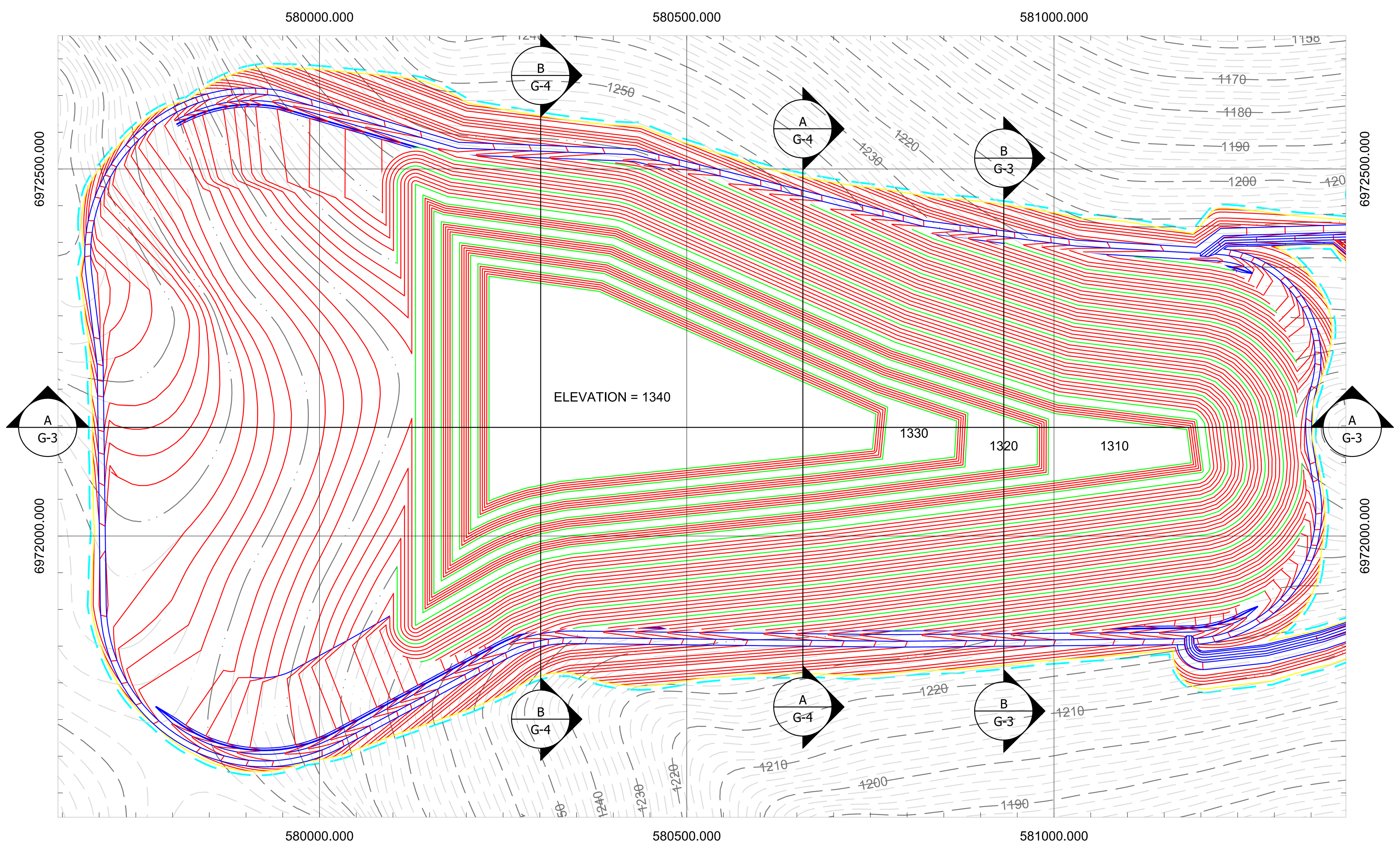
CONSULTANT PROJECT No.		CONSULTANT DWG. No.	
178201			
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		G-1	A



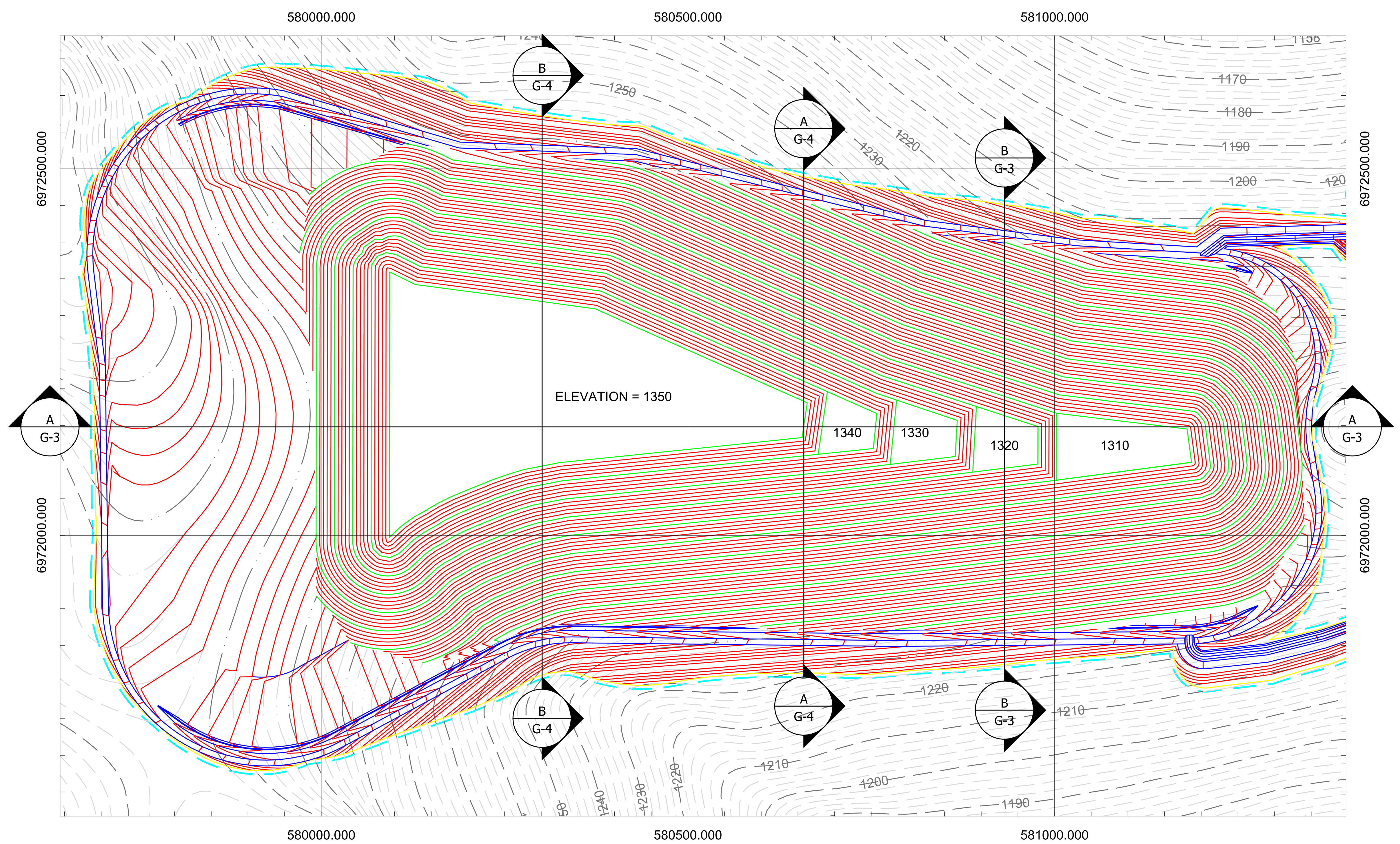
STAGE 1 HEAP LAYOUT
(7.21 MILLION TONNES)



STAGE 1-2 HEAP LAYOUT
(19.24 MILLION TONNES)



STAGE 3 HEAP LAYOUT
(39.4 MILLION TONNES)



STAGE 4 HEAP LAYOUT
(47 MILLION TONNES)

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT | PRO.MAN. | PRO.ENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

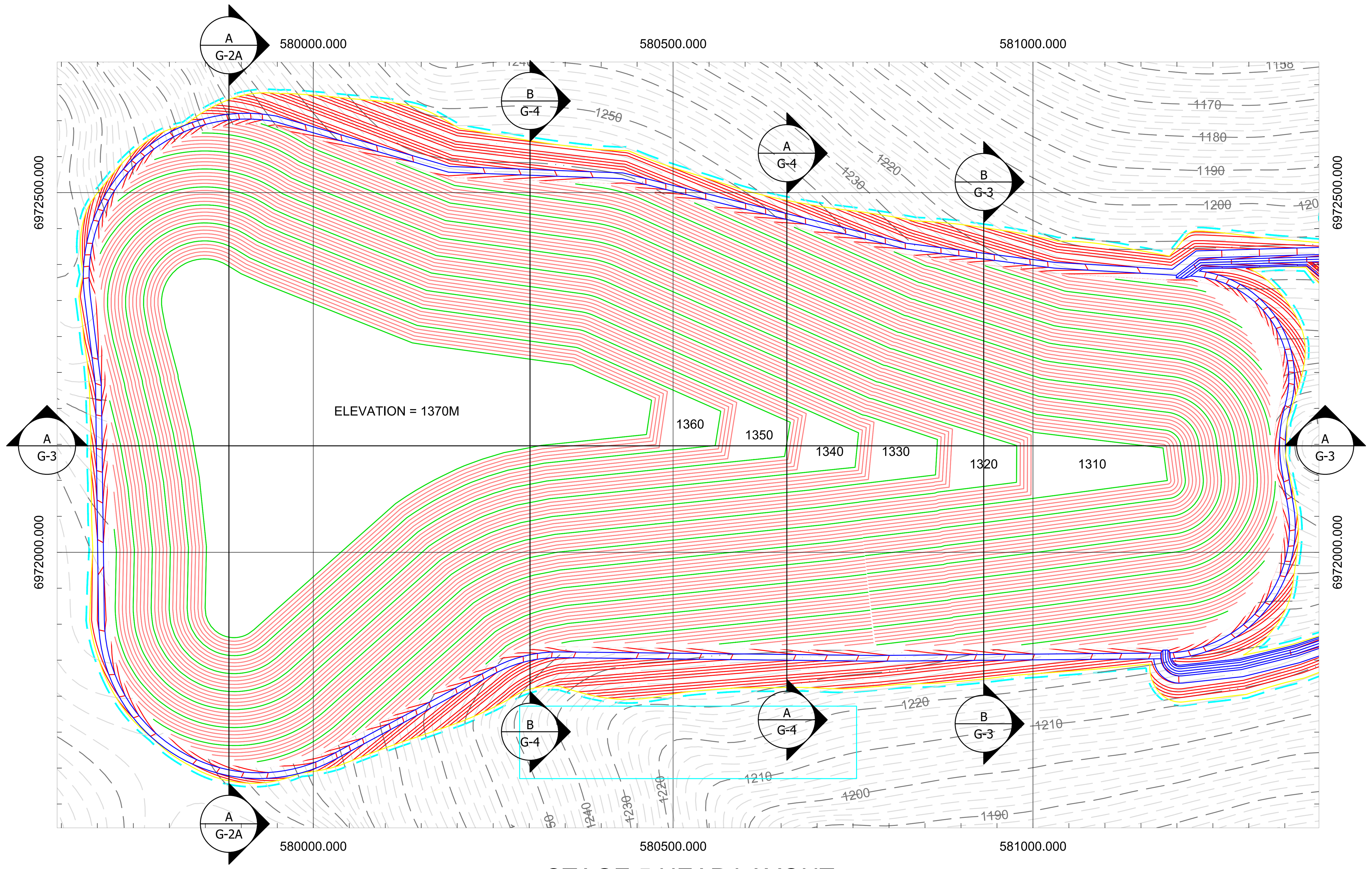
THE MINES GROUP INC.

COFFEE GOLD PROJECT

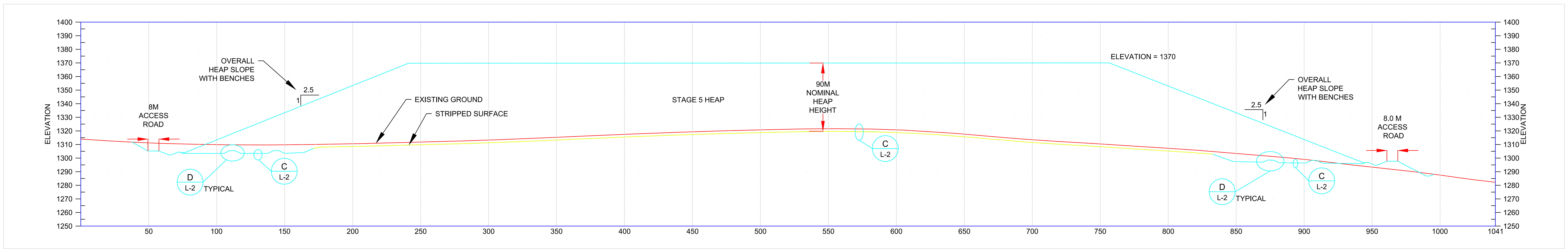
HEAP STAGE 1 TO 4 CONFIGURATIONS

CONSULTANT PROJECT No.	CONSULTANT DWG No.		
178201			
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		G-2	A

C:\COMMON\A1-TUPRAG.DWT



STAGE 5 HEAP LAYOUT
(67 MILLION TONNES)



NORTH–SOUTH HEAP LEACH PAD SECTION (A)

SCALE 1:1500

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

		DATE
DISCIPLINE:	GENERAL	
SCALE:	NONE	
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

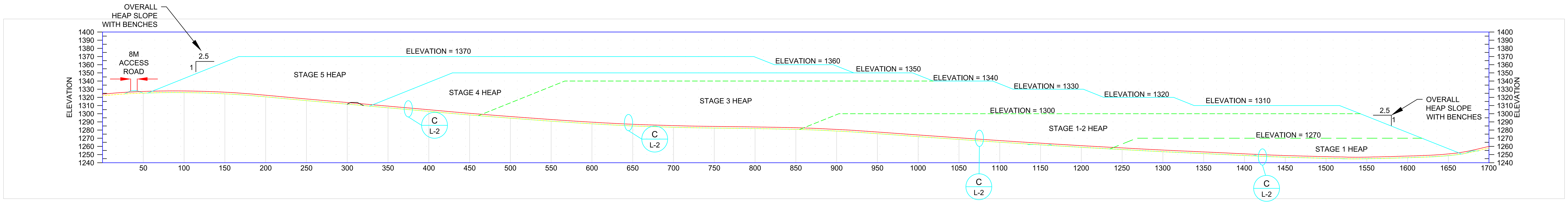
THE MINES GROUP INC.

COFFEE GOLD PROJECT

HEAP STAGE 5 CONFIGURATION AND CROSS SECTION

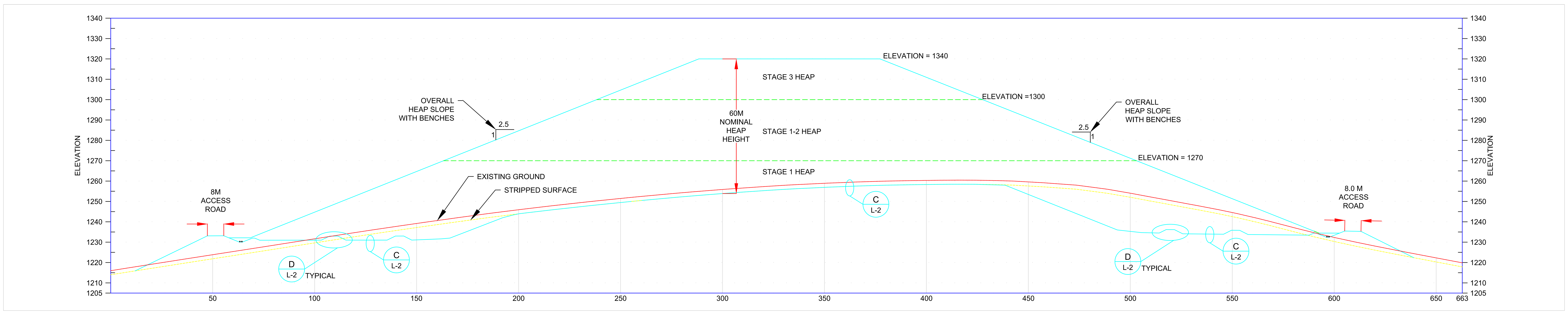
PROJECT NUMBER	DRAWING NUMBER	REV.
178201	G-2A	A

C:\COMMON\A1-TUPRAG.DWT



WEST – EAST LEACH PAD SECTION A

SCALE 1:2500



NORTH–SOUTH HEAP LEACH PAD SECTION B

SCALE 1:1000

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

HEAP LEACH PAD SECTIONS

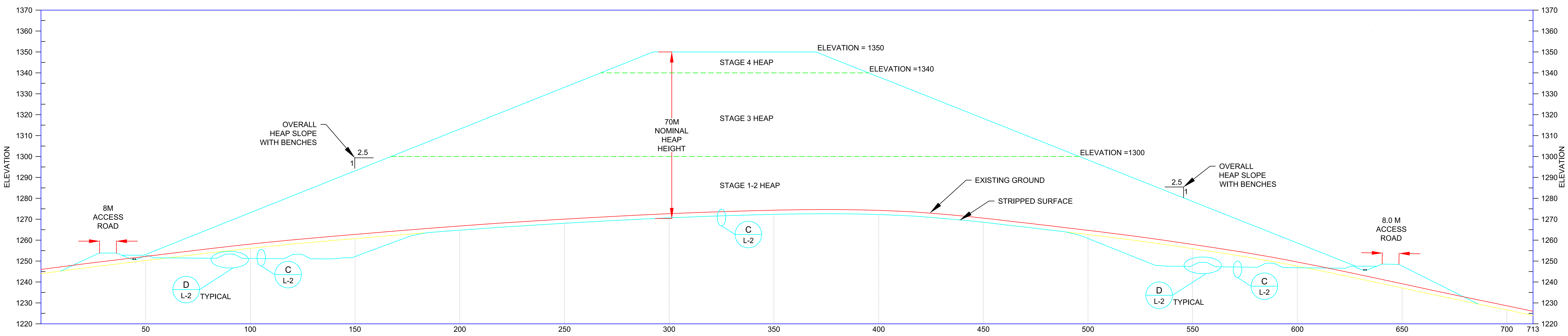
CONSULTANT PROJECT No.		CONSULTANT DWG. No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	178201	G-3	A

C:\COMMON\A1-TUPRAG.DWT

APPENDIX 2-C

Engineering Studies

Part 2 of 2



NORTH–SOUTH HEAP LEACH PAD SECTION A

SCALE 1:1000

NORTH–SOUTH HEAP LEACH PAD SECTION B

SCALE 1:1000

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	DESCRIPTION	DATE	BY
A	ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

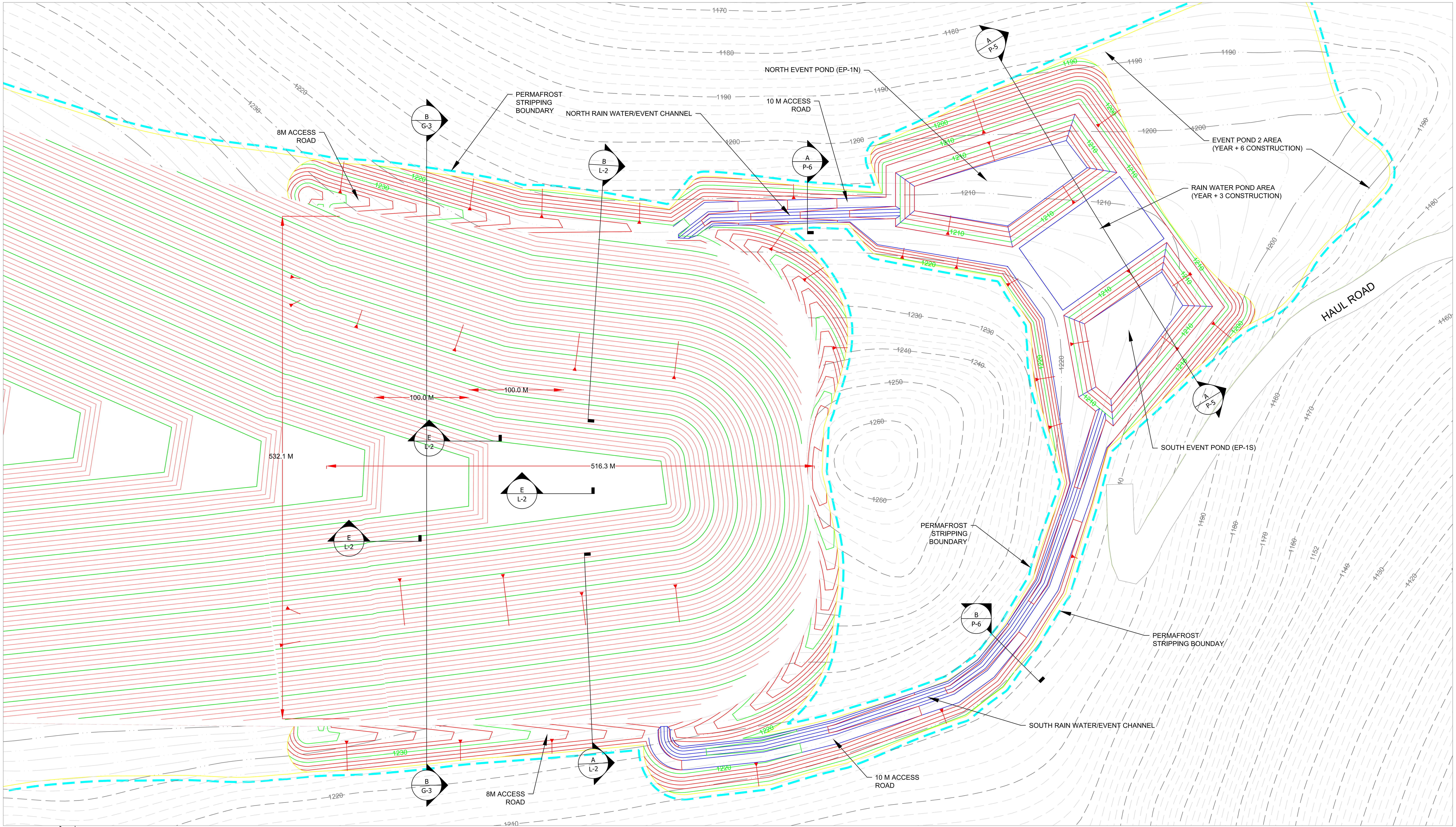
THE MINES GROUP INC.

COFFEE GOLD PROJECT

HEAP LEACH PAD SECTIONS

CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		G-4	A

C:\COMMON\A1–TUPRAG.DWT



(METERS)

60 30 0 60

1:2000

CONTOUR LEGEND

- EXISTING GROUND MINOR CONTOUR
- EXISTING GROUND MAJOR CONTOUR
- STRIPPED EXISTING GROUND* MINOR CONTOUR
- STRIPPED EXISTING GROUND* MAJOR CONTOUR
- FINISHED GRADE MINOR CONTOUR
- FINISHED GRADE MAJOR CONTOUR

NOTE: *2M DEPTH OF PERMAFROST EXCAVATION FROM EXISTING GROUND ELEVATION

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	DESCRIPTION	DATE	BY
A	ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

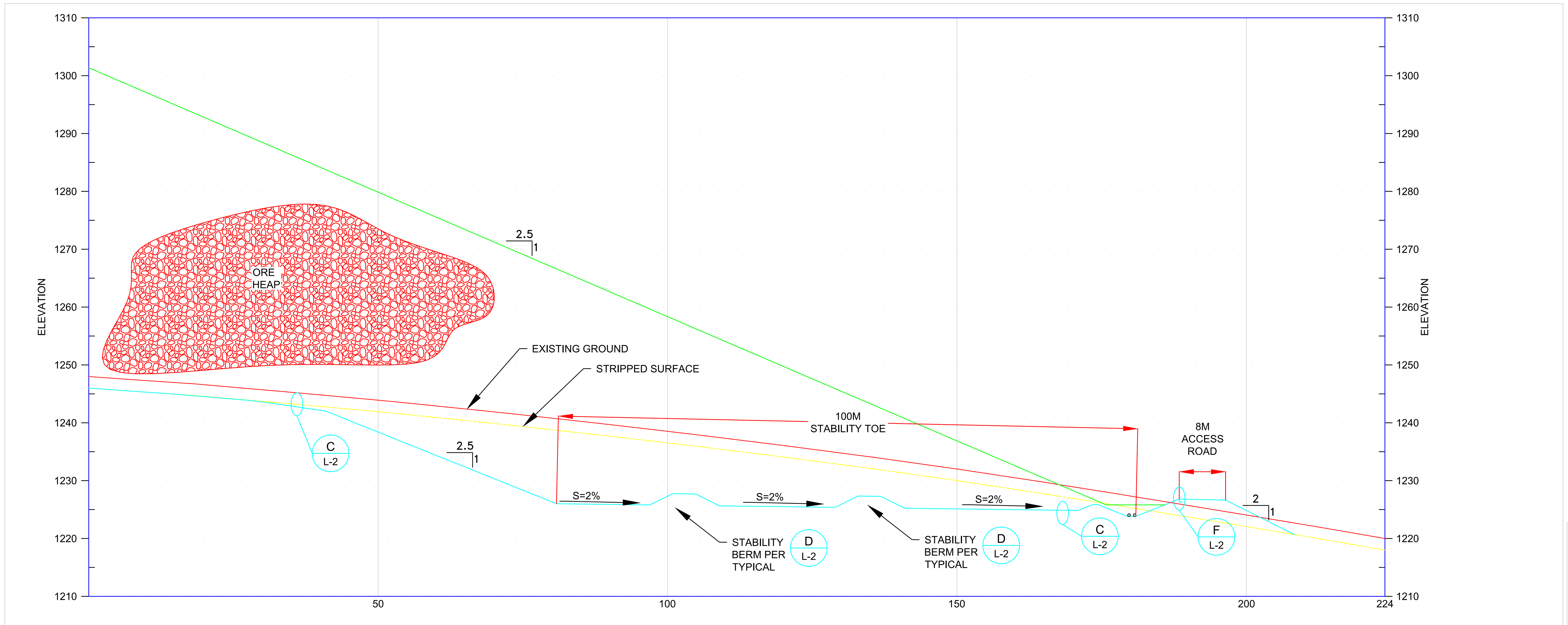
GOLDCORP

THE MINES GROUP INC.

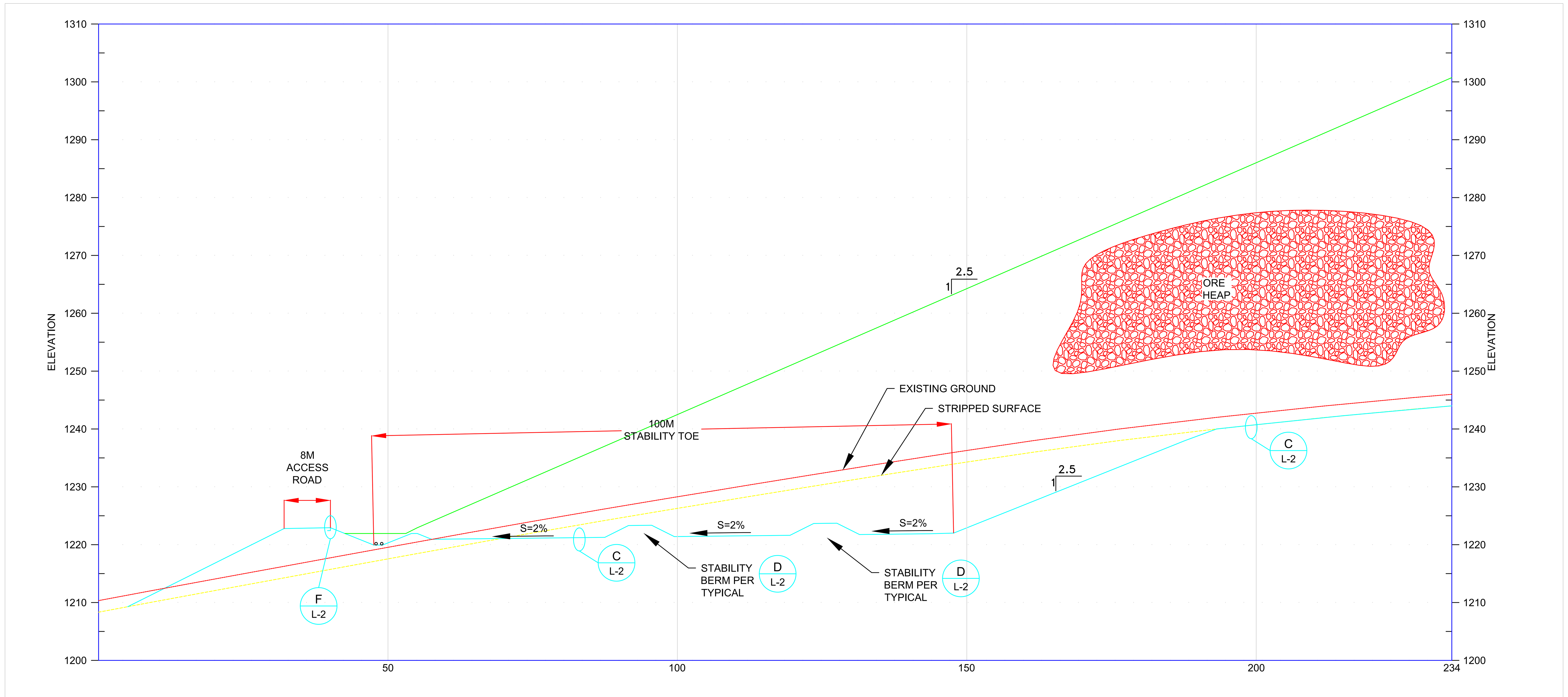
COFFEE GOLD PROJECT

STAGE 1 HEAP LEACH FACILITY LAYOUT AND GRADING PLAN

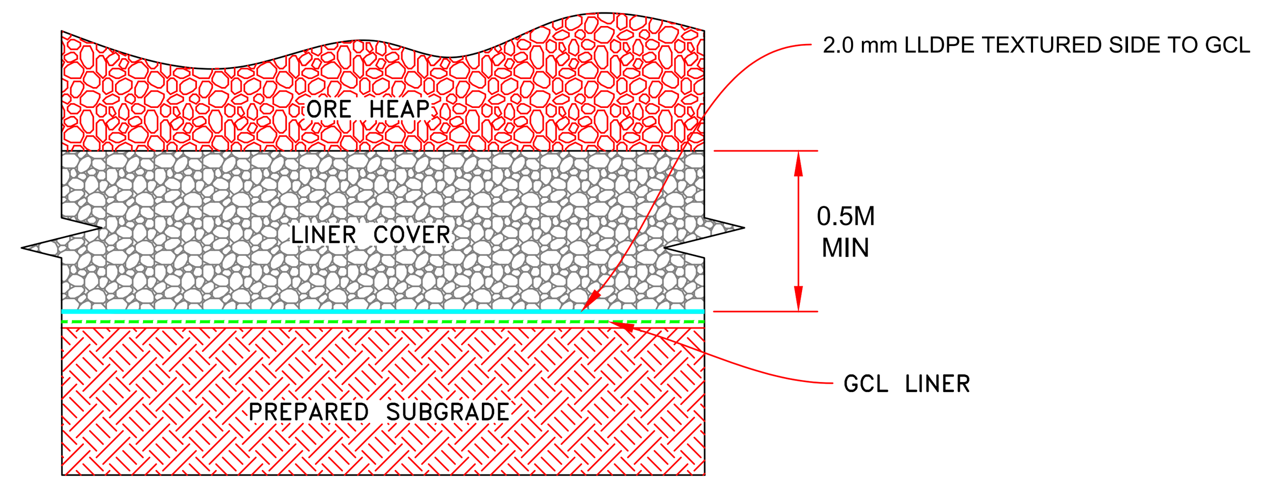
CONSULTANT PROJECT No.	CONSULTANT DWG. No.		
178201			
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		L-1	A



SOUTH HEAP LEACH PAD STABILITY TOE SECTION (TYPICAL) A
SCALE 1:500

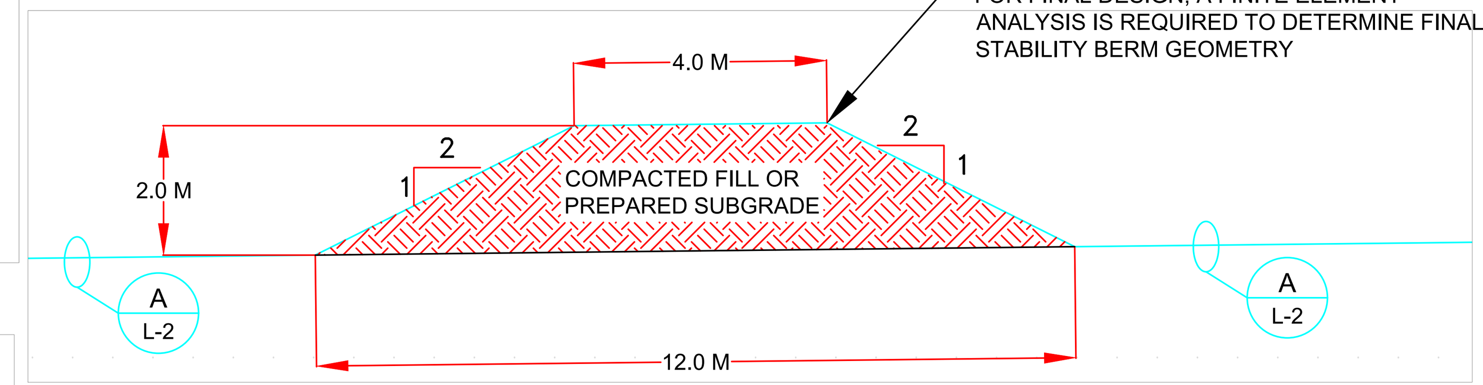


NORTH HEAP LEACH PAD STABILITY TOE SECTION (TYPICAL) B
SCALE 1:500

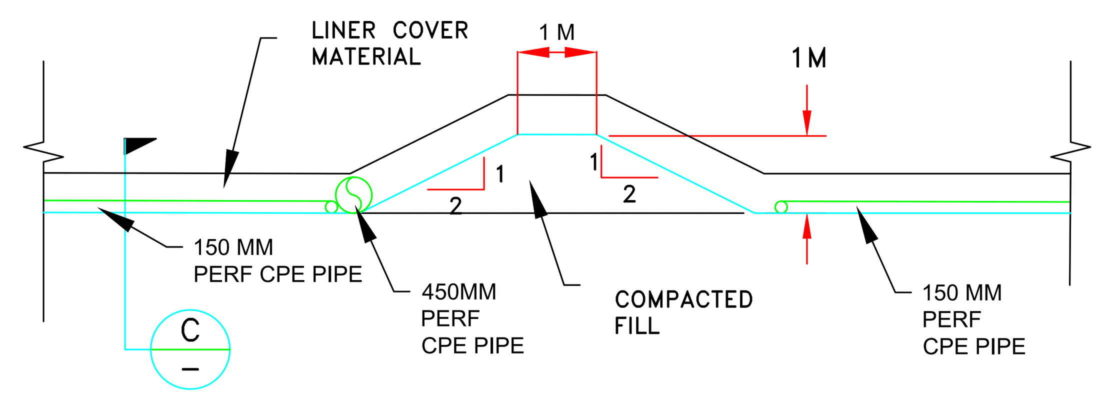


TYPICAL LEACH PAD LINER SECTION C
NTS

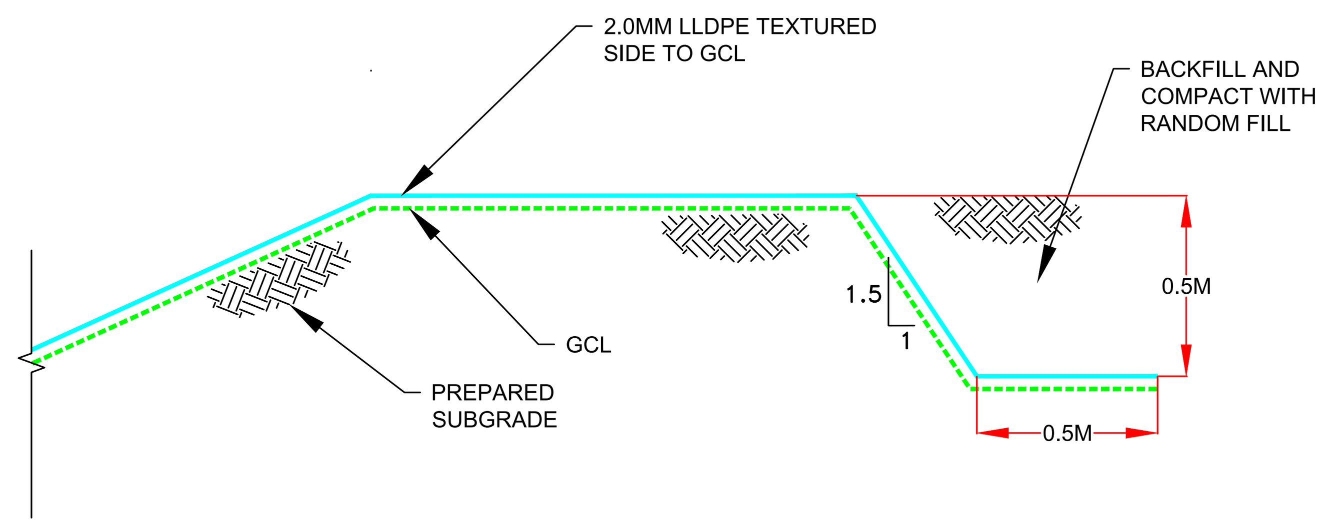
NOTE:
FOR FINAL DESIGN, A FINITE ELEMENT ANALYSIS IS REQUIRED TO DETERMINE FINAL STABILITY BERM GEOMETRY



TYPICAL STABILITY BERM SECTION D
SCALE 1:100



TYPICAL CELL SEPARATION BERM E
SCALE 1:80



LEACH PAD LINER ANCHOR DETAIL F
NTS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT | PRO.MAN. | PRO.ENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

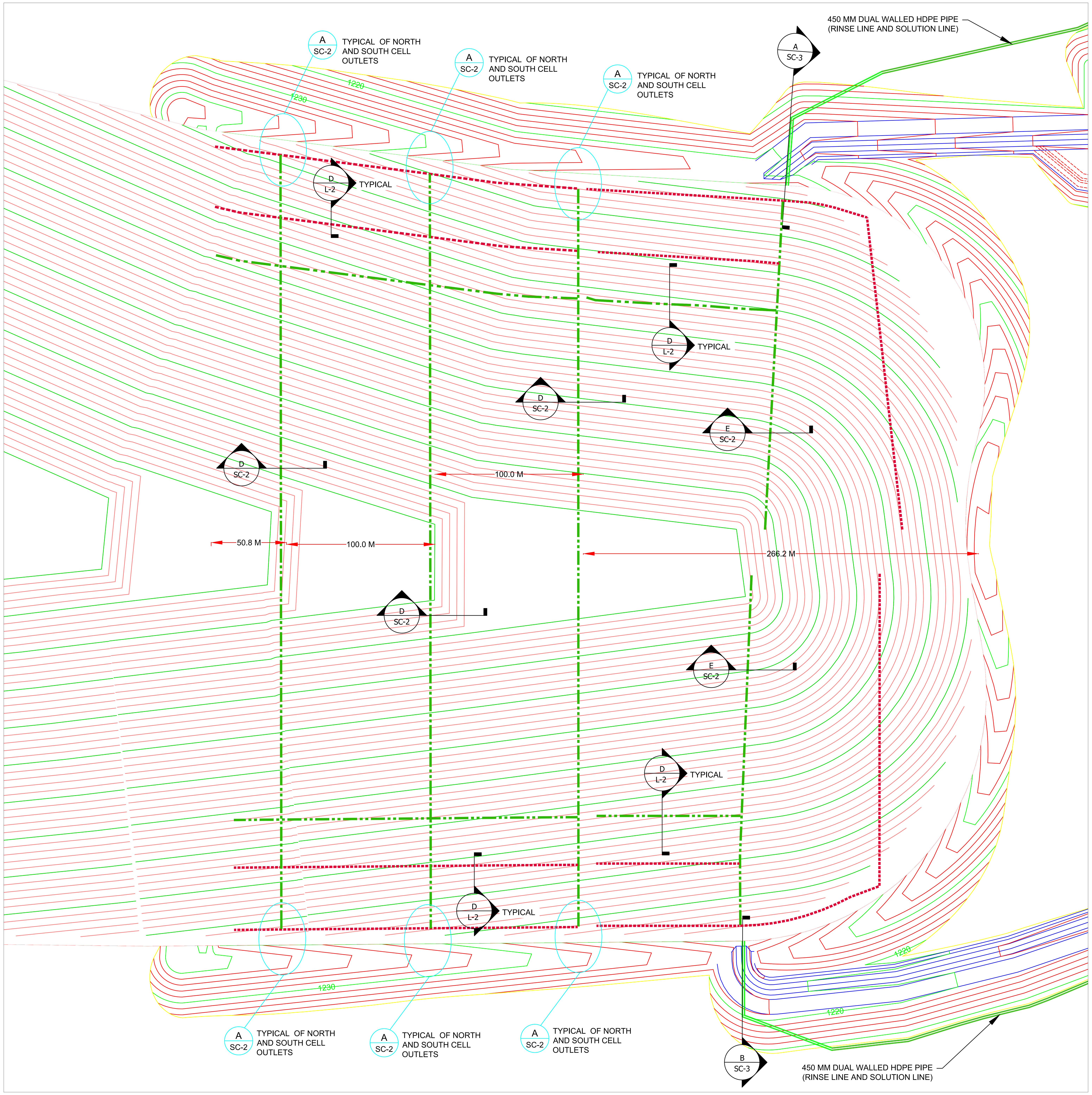
THE MINES GROUP INC.

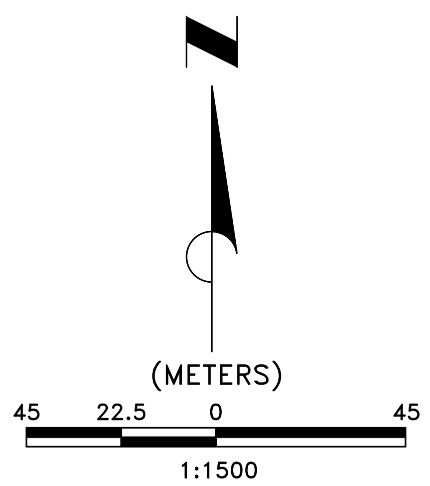
COFFEE GOLD PROJECT

STAGE 1 HEAP LEACH PAD
NORTH AND SOUTH STABILITY TOE SECTIONS

CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		L-2	A

C:\COMMON\A1-TUPRAG.DWT





SOLUTION COLLECTION PIPE PLAN LEGEND

(pink line)	150 MM PERFORATED CPE SOLUTION COLLECTION PIPE (PLACED 15 M ON CENTER @45 DEGREES)
(red dotted line)	150 MM PERFORATED CPE SOLUTION COLLECTION PIPE
(green dash-dot line)	450 MM PERFORATED CPE SOLUTION COLLECTION PIPE

DWG. NO.	REFERENCE DRAWINGS
–	–

CLIENT	PRO.MAN.	PRO.ENG.	PROCESS	ELECTR.	INSTR.	PIPING	MECH.	STRUCT.	SERVICES	ARCH.	LAYOUT	REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
												A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

STAGE 1 SOLUTION COLLECTION PLAN

CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		SC-1	A

C:\COMMON\A1-TUPRAG.DWT

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

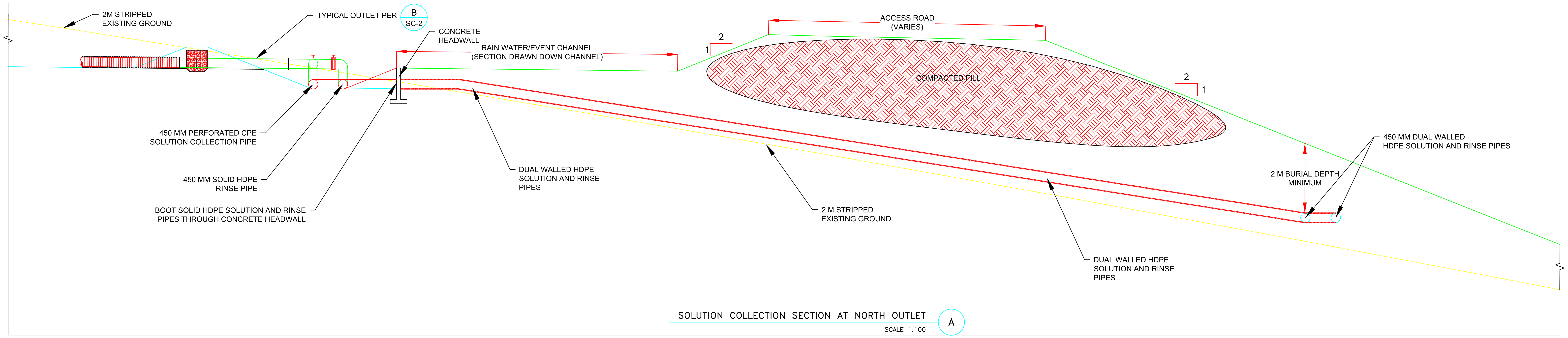
GOLDCORP

THE MINES GROUP INC.

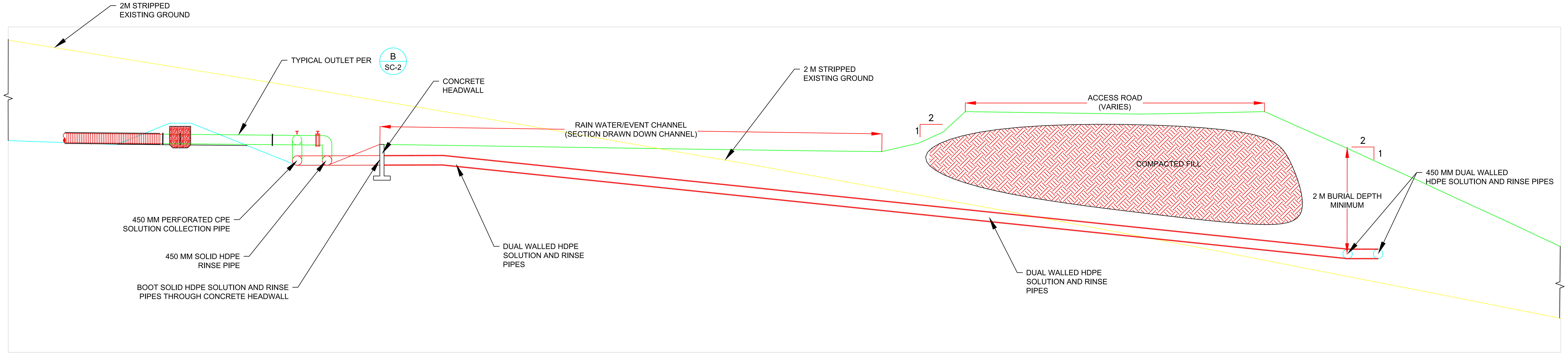
COFFEE GOLD PROJECT

STAGE 1 SOLUTION COLLECTION PLAN
TYPICAL CELL OUTLET

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	178201	SC-2	A



SOLUTION COLLECTION SECTION AT NORTH OUTLET (A)
SCALE 1:100



SOLUTION COLLECTION SECTION AT SOUTH OUTLET (B)
SCALE 1:100

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

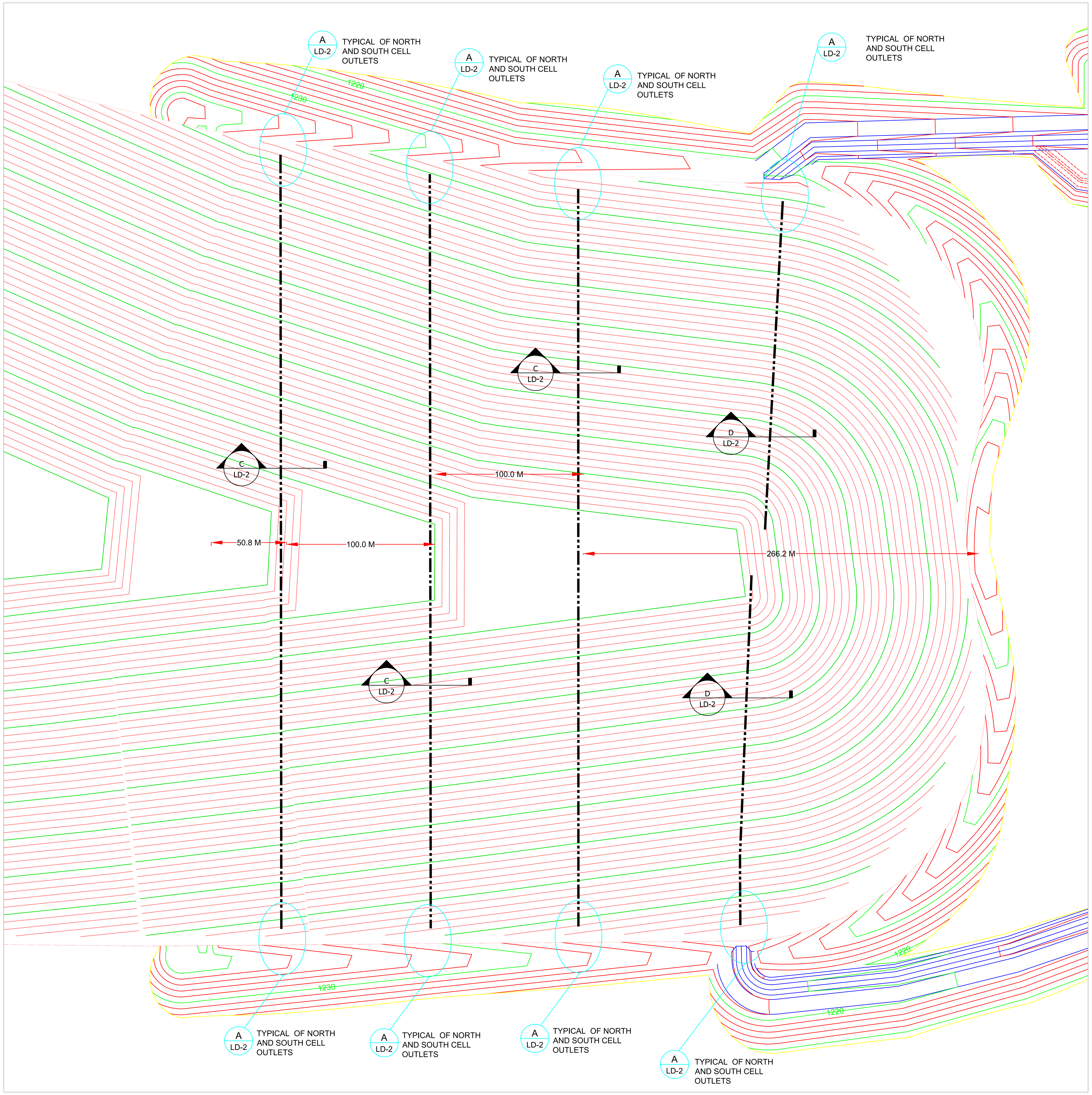
THE MINES GROUP INC.

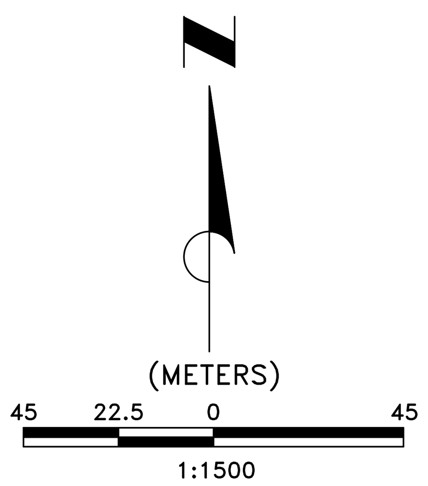
COFFEE GOLD PROJECT

STAGE 1 SOLUTION COLLECTION NORTH AND SOUTH OUTLET SECTIONS

CONSULTANT PROJECT No.		CONSULTANT DWG. No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	178201	SC-3	A

C:\COMMON\A1–TUPRAG.DWT





LEAK DETECTION PLAN LEGEND

100 MM STRIP DRAIN

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT PRO.MAN. PRO.ENG. PROCESS ELECTR. INSTR. PIPING MECH. STRUCT. SERVICES ARCH. LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

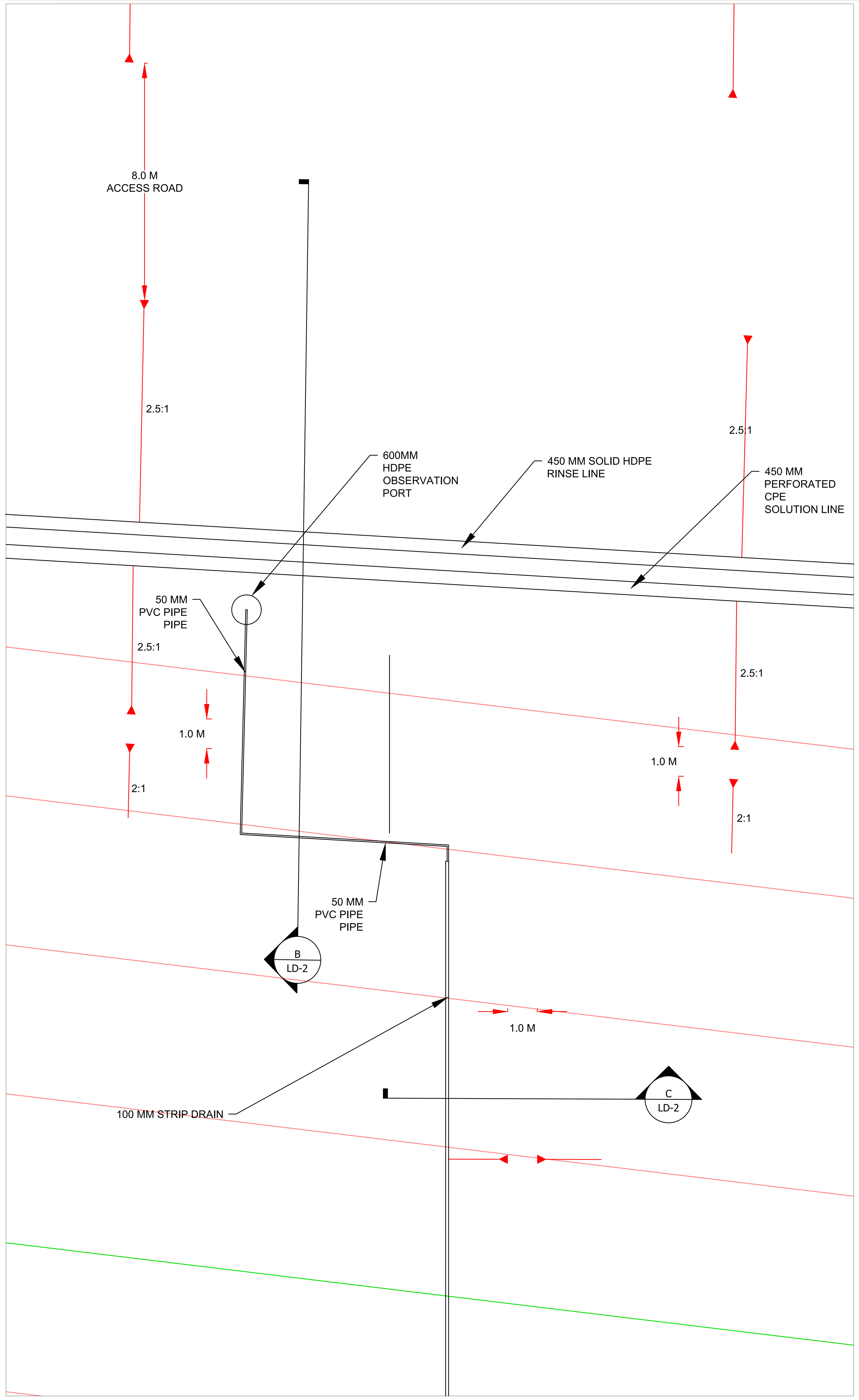
THE MINES GROUP INC.

COFFEE GOLD PROJECT

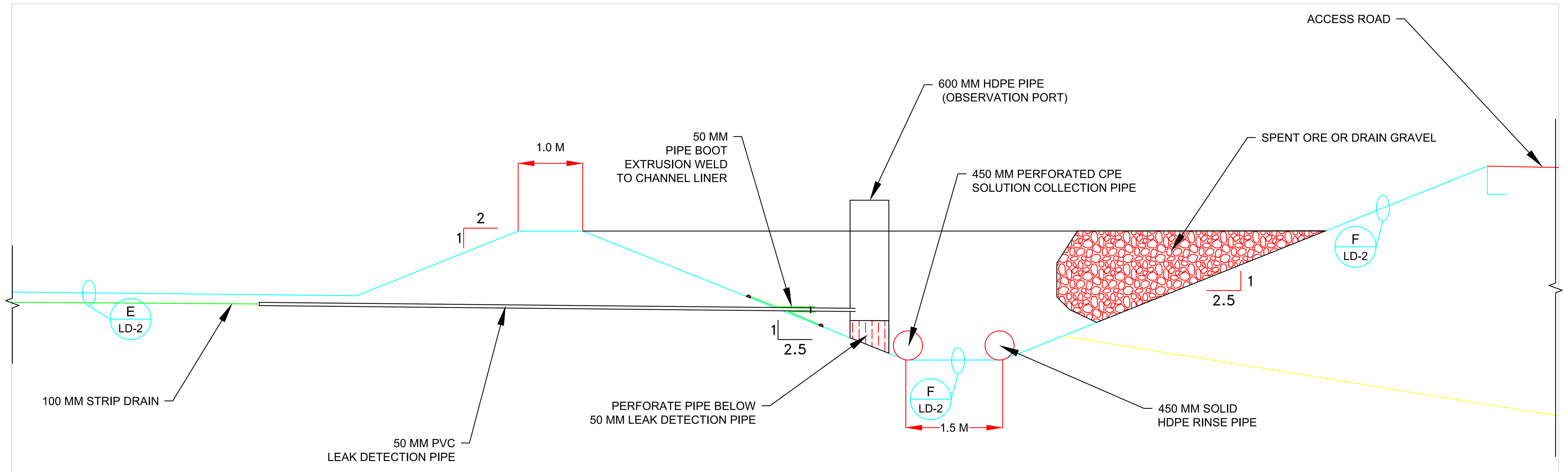
STAGE 1 LEAK DETECTION PLAN

CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		LD-1	A

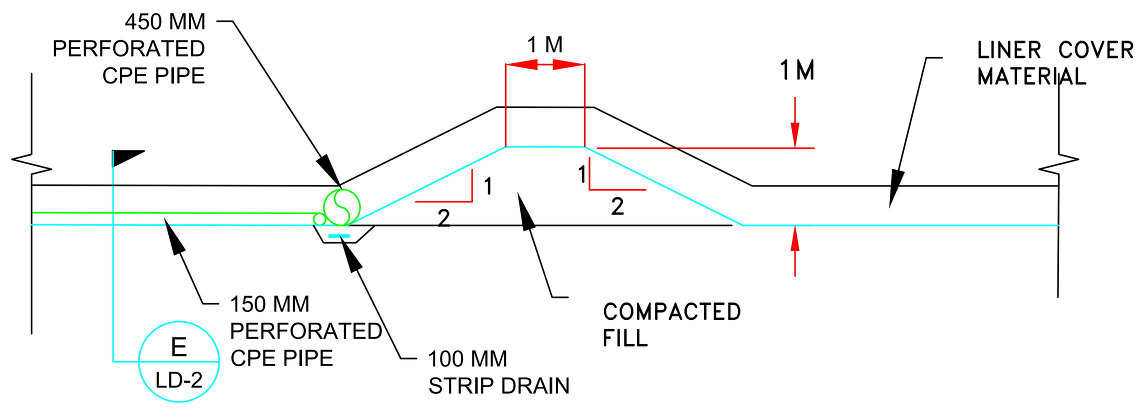
C:\COMMON\A1-TUPRAG.DWT



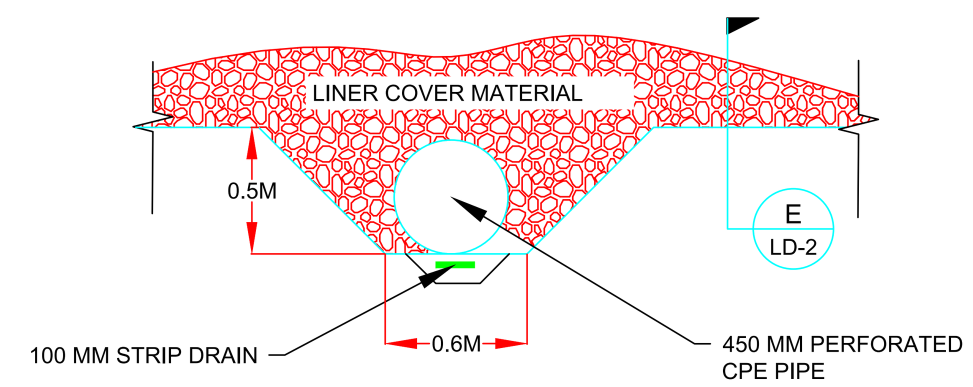
LEAK DETECTION PLAN AT CELL OUTLET
(TYPICAL NORTH AND SOUTH)
A
SCALE 1:100



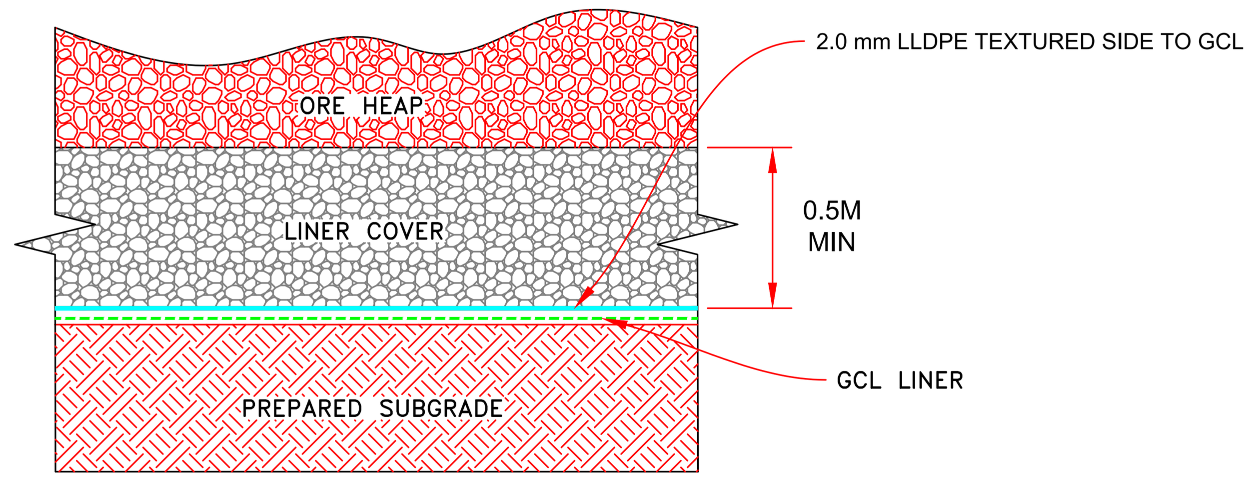
LEAK DETECTION SECTION AT CELL OUTLET
(TYPICAL NORTH AND SOUTH)
B
SCALE 1:50



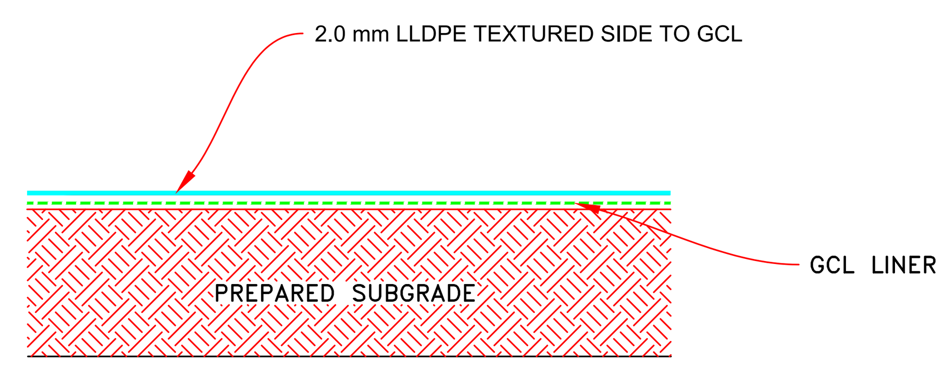
TYPICAL CELL SEPARATION BERM
C
SCALE 1:80



LEAK DETECTION SECTION
D
SCALE: NTS



TYPICAL LEACH PAD LINER SECTION
E
NTS



TYPICAL LEACH PAD SOLUTION COLLECTION
CHANNEL LINER SECTION
F
NTS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

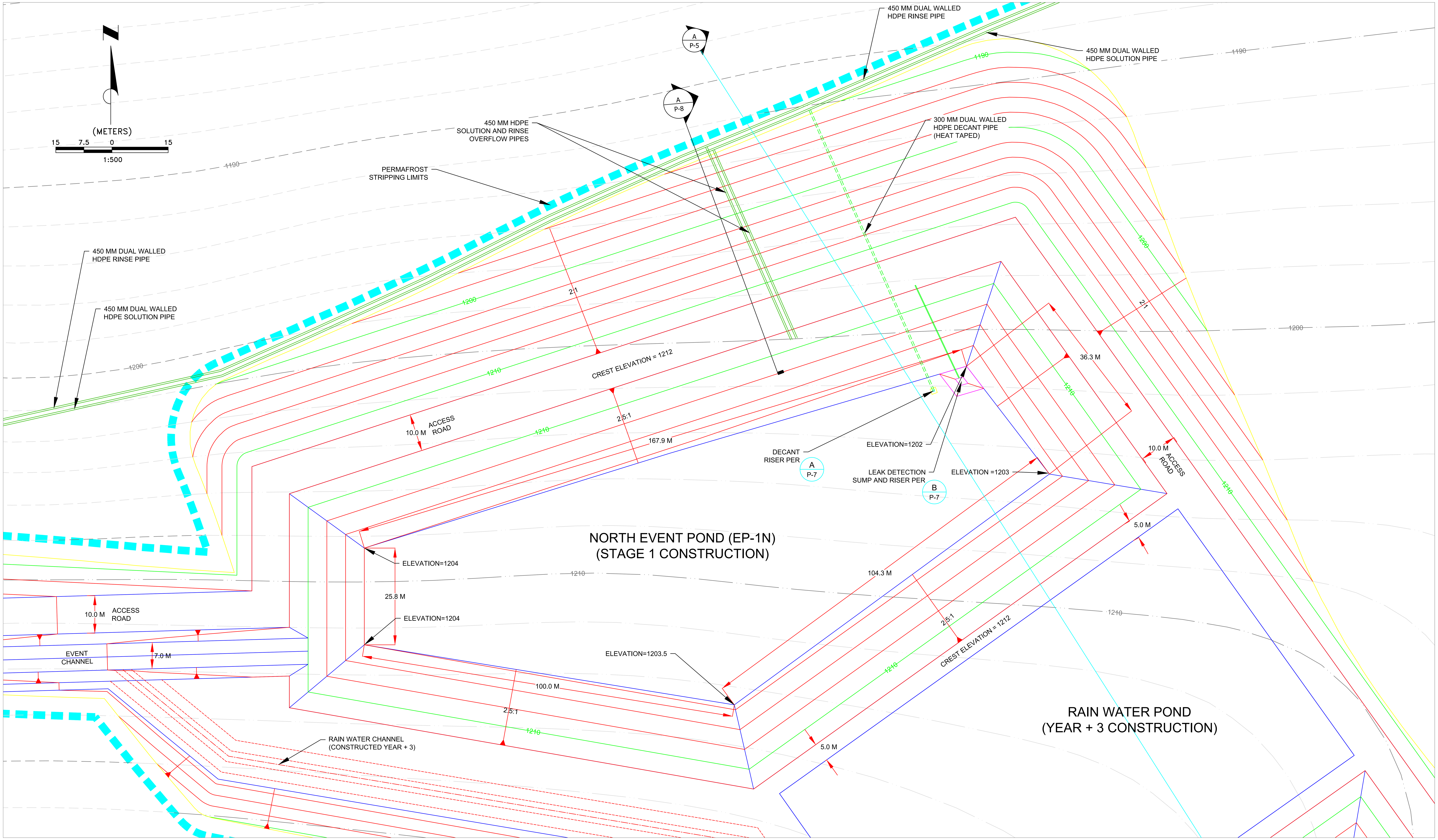
GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

STAGE 1 LEAK DETECTION PLAN
TYPICAL CELL OUTLET

CONSULTANT PROJECT No.	PROJECT NUMBER	CONSULTANT DWG. No.	DRAWING NUMBER	REV.
	178201		LD-2	A



CONTOUR LEGEND

- EXISTING GROUND MINOR CONTOUR
- EXISTING GROUND MAJOR CONTOUR
- STRIPPED EXISTING GROUND* MINOR CONTOUR
- STRIPPED EXISTING GROUND* MAJOR CONTOUR
- FINISHED GRADE MINOR CONTOUR
- FINISHED GRADE MAJOR CONTOUR

NOTE: *2M DEPTH OF PERMAFROST EXCAVATION FROM EXISTING GROUND ELEVATION

NOTE:
3 M HIGH CHAIN LINK FENCE TO BE CONSTRUCTED AROUND ALL PONDS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT / PRO.MAN. / PRO.ENG. / PROCESS / ELECTR. / INSTR. / PIPING / MECH. / STRUCT. / SERVICES / ARCH. / LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

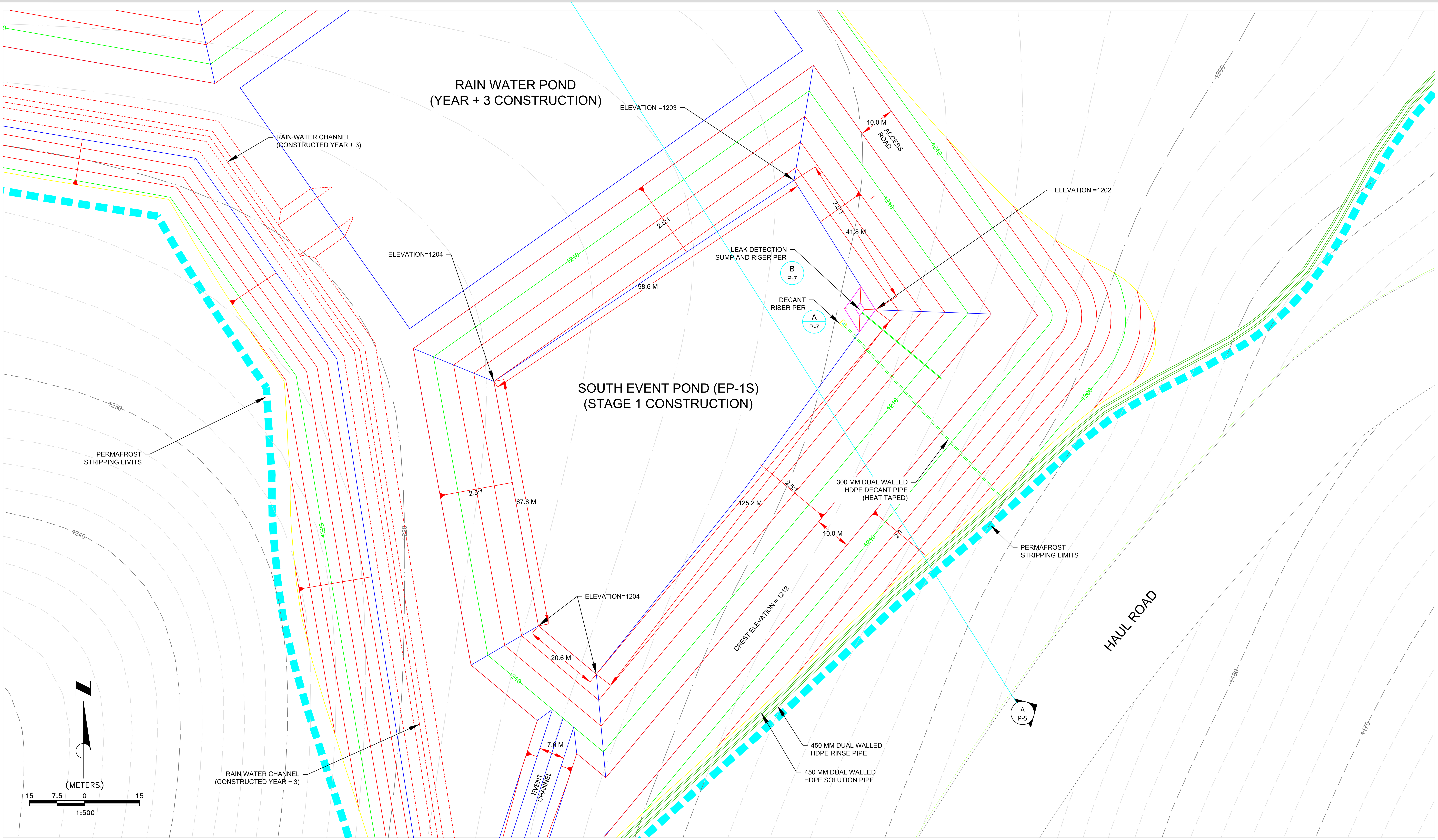
THE MINES GROUP INC.

COFFEE GOLD PROJECT

NORTH EVENT POND (EP-1N) LAYOUT AND GRADING PLAN

CONSULTANT PROJECT No.	CONSULTANT DWG. No.		
	178201		
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P-1	A

C:\COMMON\A1-TUPRAG.DWT



CONTOUR LEGEND

- EXISTING GROUND MINOR CONTOUR
- EXISTING GROUND MAJOR CONTOUR
- STRIPPED EXISTING GROUND* MINOR CONTOUR
- STRIPPED EXISTING GROUND* MAJOR CONTOUR
- FINISHED GRADE MINOR CONTOUR
- FINISHED GRADE MAJOR CONTOUR

NOTE: *2M DEPTH OF PERMAFROST EXCAVATION FROM EXISTING GROUND ELEVATION

NOTE:
3 M HIGH CHAIN LINK FENCE TO BE CONSTRUCTED AROUND ALL PONDS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT | PRO.MAN. | PRO.ENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

SOUTH EVENT POND (EP-1S) LAYOUT AND GRADING PLAN

CONSULTANT PROJECT No.		CONSULTANT DWG. No.	
	178201		
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P-2	A

C:\COMMON\A1-TUPRAG.DWT

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
	A	ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	1:1000	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

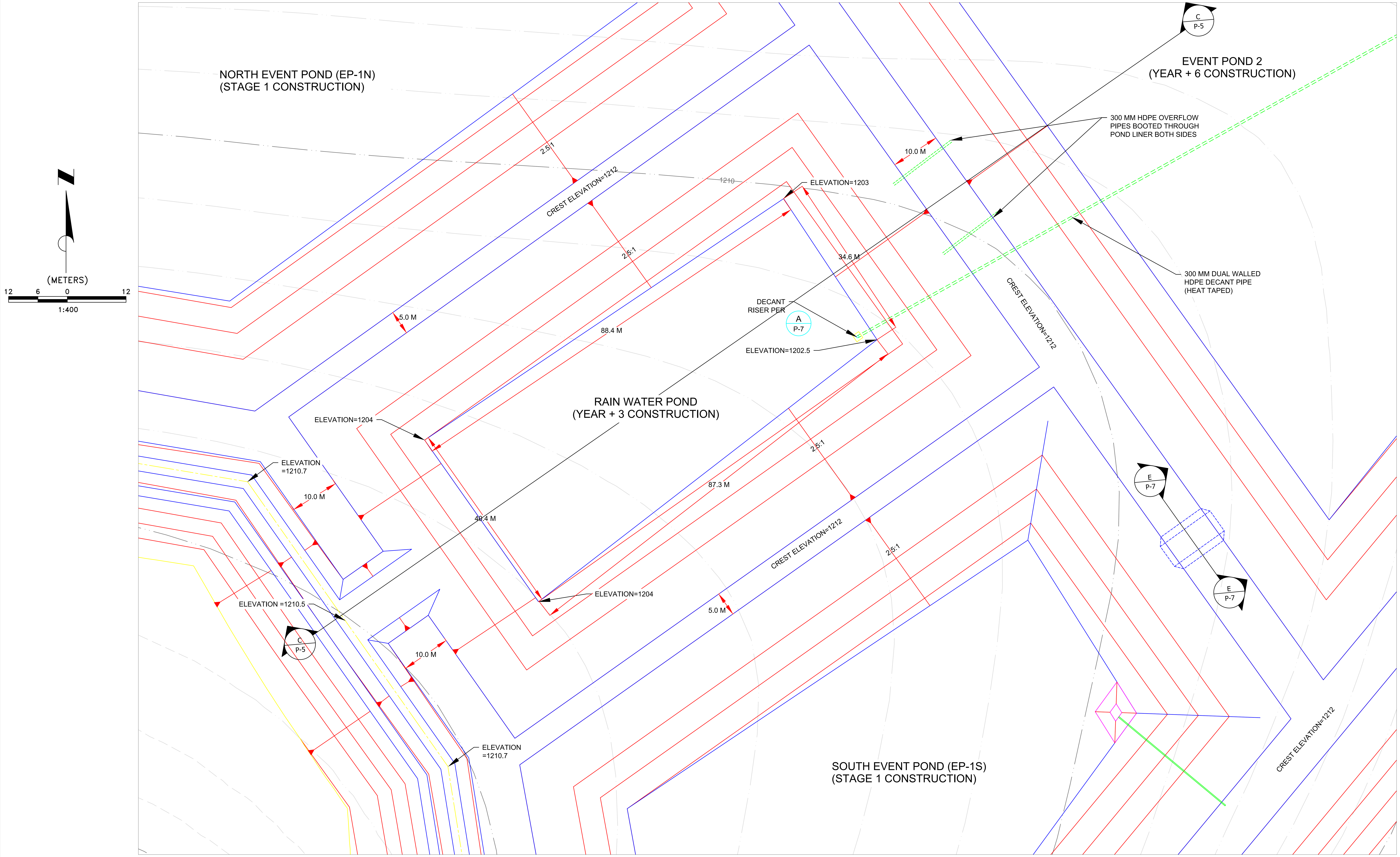
GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

POND FACILITIES LAYOUT AND GRADING PLAN

CONSULTANT PROJECT No.	CONSULTANT DWG. No.		
178201			
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P-3	A



CONTOUR LEGEND

EXISTING GROUND MINOR CONTOUR
EXISTING GROUND MAJOR CONTOUR
STRIPPED EXISTING GROUND* MINOR CONTOUR
STRIPPED EXISTING GROUND* MAJOR CONTOUR
FINISHED GRADE MINOR CONTOUR
FINISHED GRADE MAJOR CONTOUR

NOTE: *2M DEPTH OF PERMAFROST EXCAVATION FROM EXISTING GROUND ELEVATION

NOTE:
3 M HIGH CHAIN LINK FENCE TO BE CONSTRUCTED AROUND ALL PONDS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	1:400	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

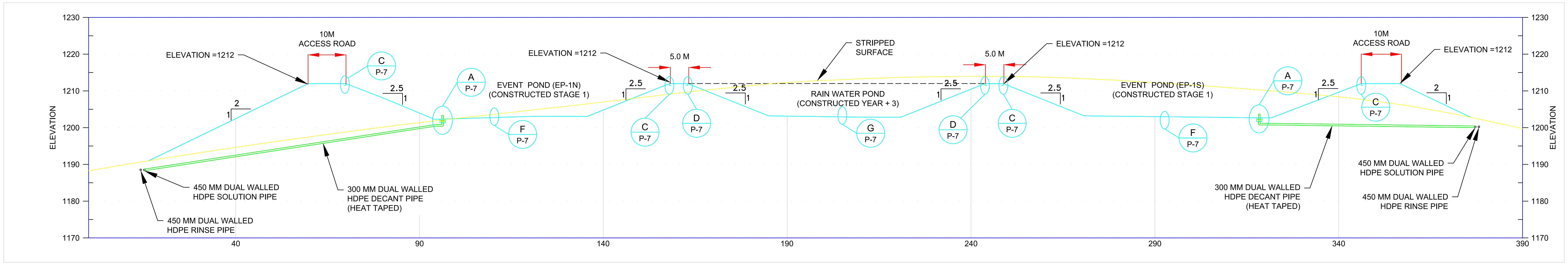
GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

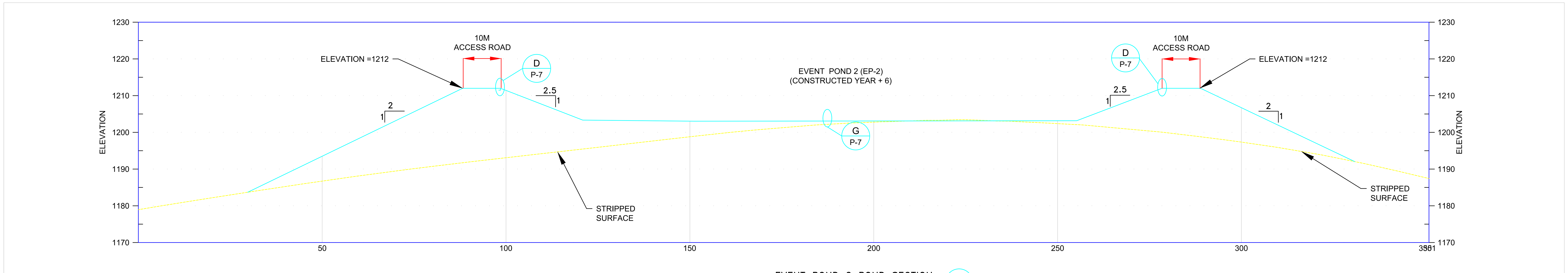
RAIN WATER POND
LAYOUT AND GRADING PLAN

CONSULTANT PROJECT No.	CONSULTANT DWG. No.		
	178201		
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P–4	A



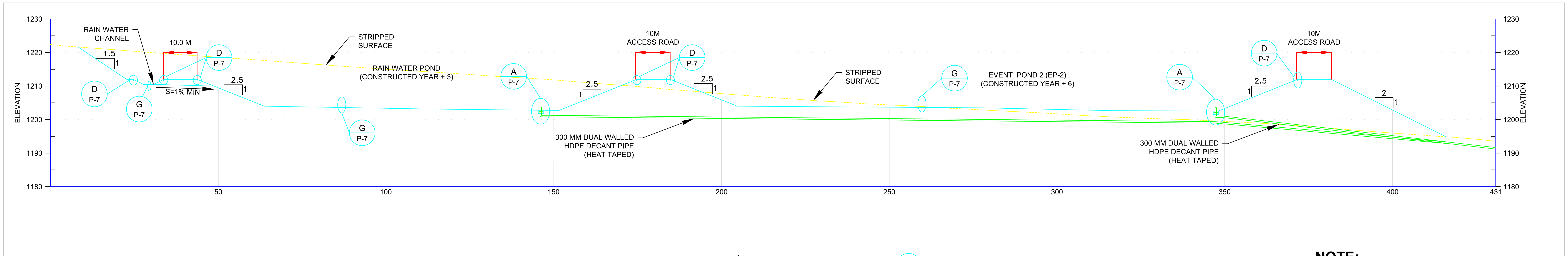
EVENT POND 1/ RAIN WATER POND SECTION A

SCALE 1:600



EVENT POND 2 POND SECTION B

SCALE 1:600



RAIN WATER/EVENT POND 2 POND SECTION C

SCALE 1:600

NOTE:

3 M HIGH CHAIN LINK FENCE TO BE CONSTRUCTED AROUND ALL PONDS

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

CLIENT / PRO.MAN. / PRO.ENG. / PROCESS / ELECTR. / INSTR. / PIPING / MECH. / STRUCT. / SERVICES / ARCH. / LAYOUT

DISCIPLINE:	GENERAL	
SCALE:	AS SHOWN	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

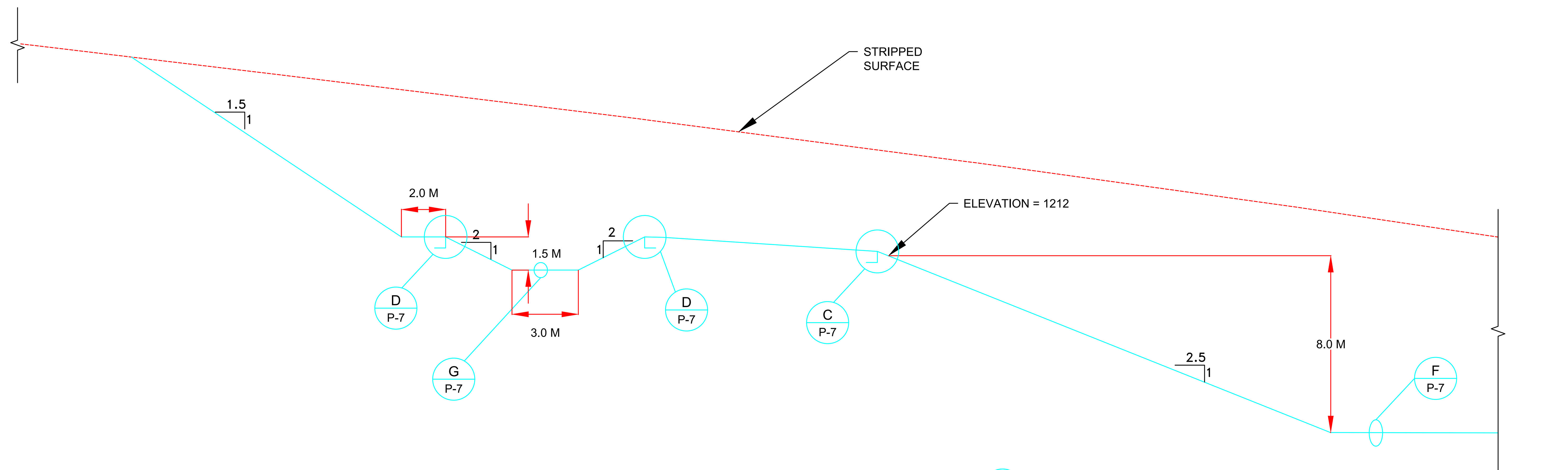
THE MINES GROUP INC.

COFFEE GOLD PROJECT

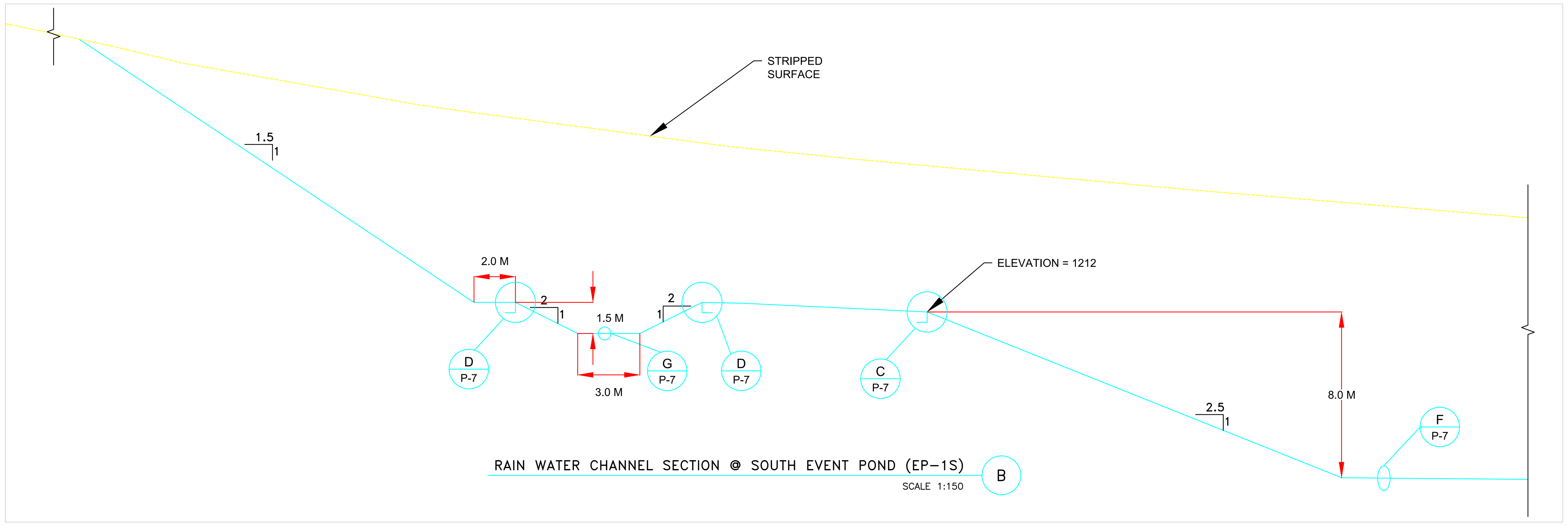
PONDS SECTIONS

CONSULTANT PROJECT No.	178201	CONSULTANT DWG. No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P–5	A

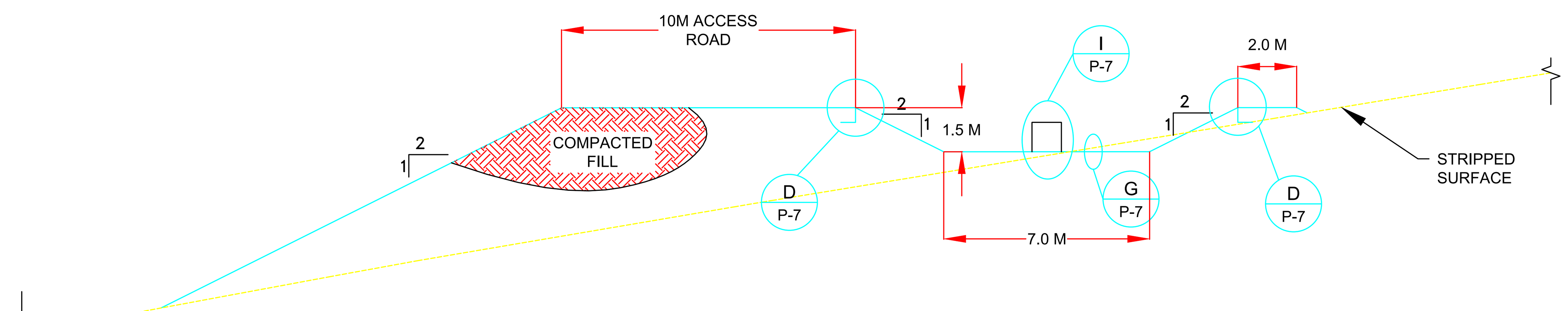
C:\COMMON\A1-TUPRAG.DWT



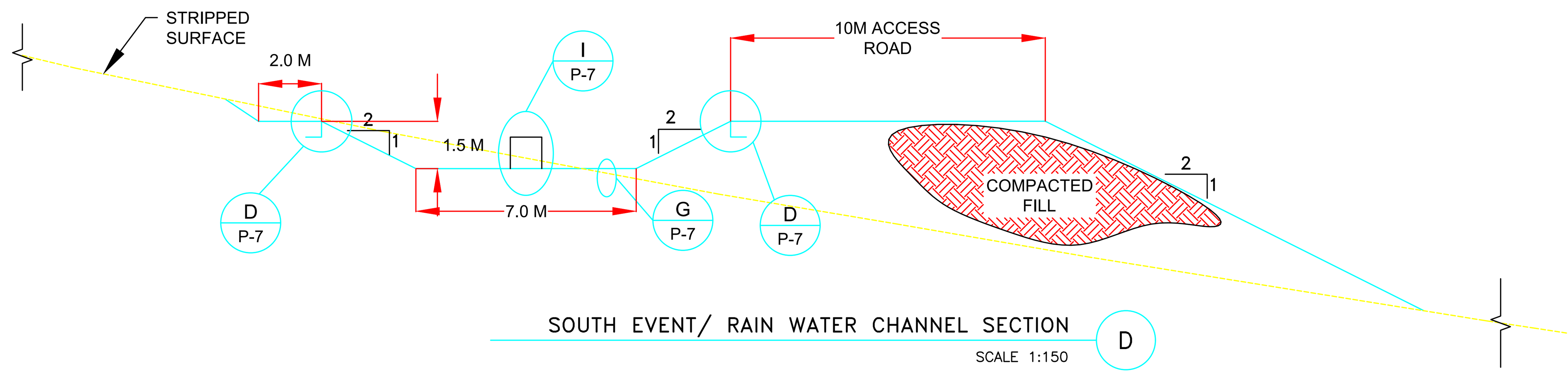
RAIN WATER CHANNEL SECTION @ NORTH EVENT POND (EP-1N) A
SCALE 1:150



RAIN WATER CHANNEL SECTION @ SOUTH EVENT POND (EP-1S) B
SCALE 1:150



NORTH EVENT /RAIN WATER CHANNEL SECTION C
SCALE 1:150



SOUTH EVENT/ RAIN WATER CHANNEL SECTION D
SCALE 1:150

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

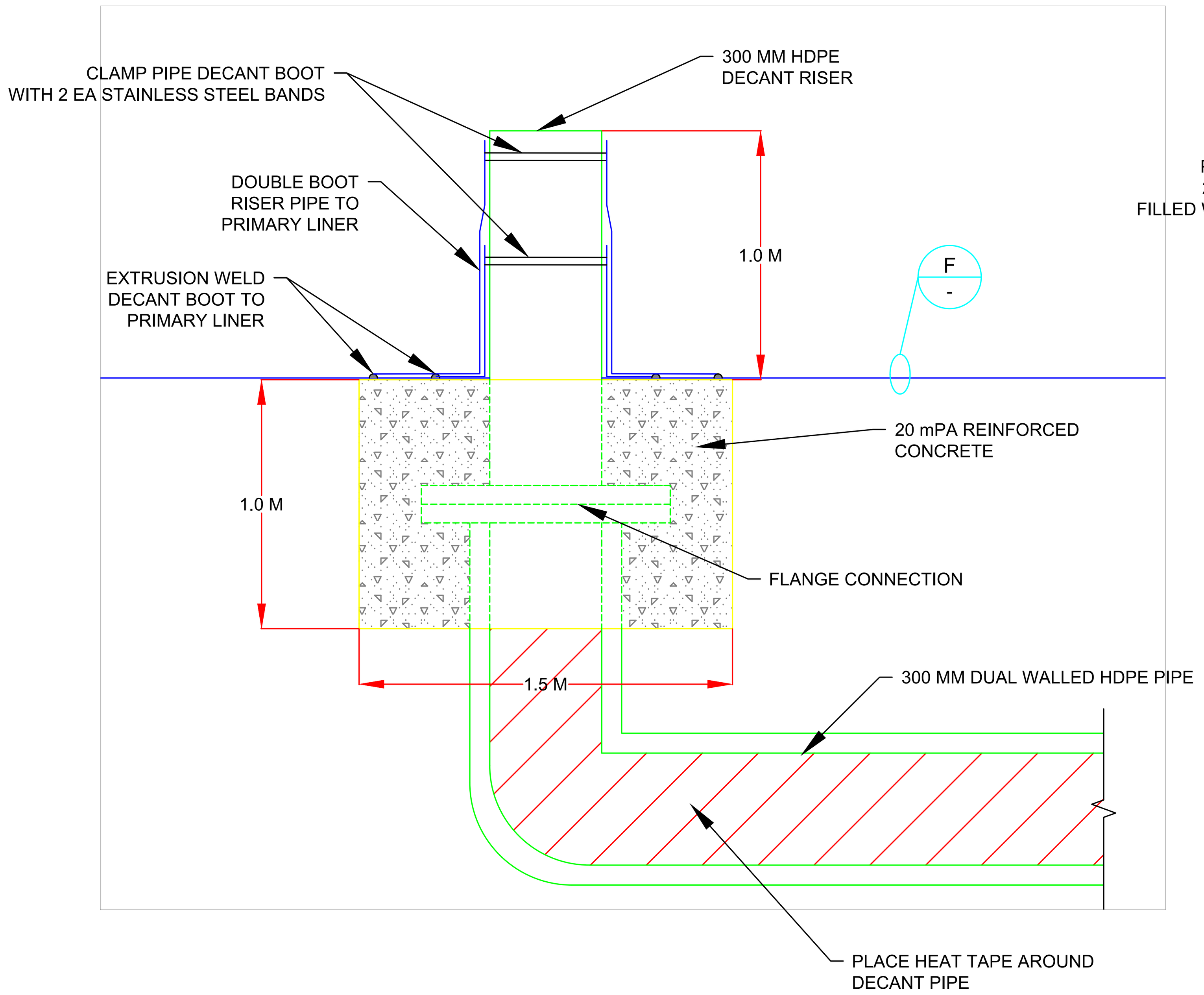
COFFEE GOLD PROJECT

RAIN WATER CHANNEL SECTIONS

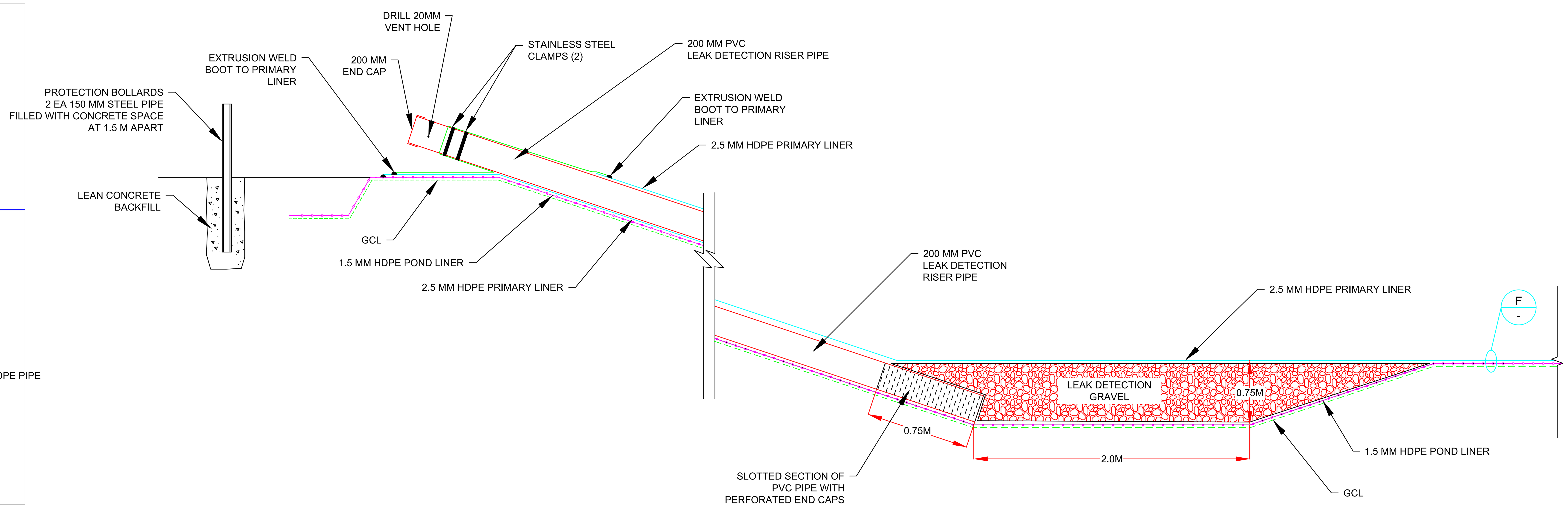
CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P-6	A

C:\COMMON\A1-TUPRAG.DWT

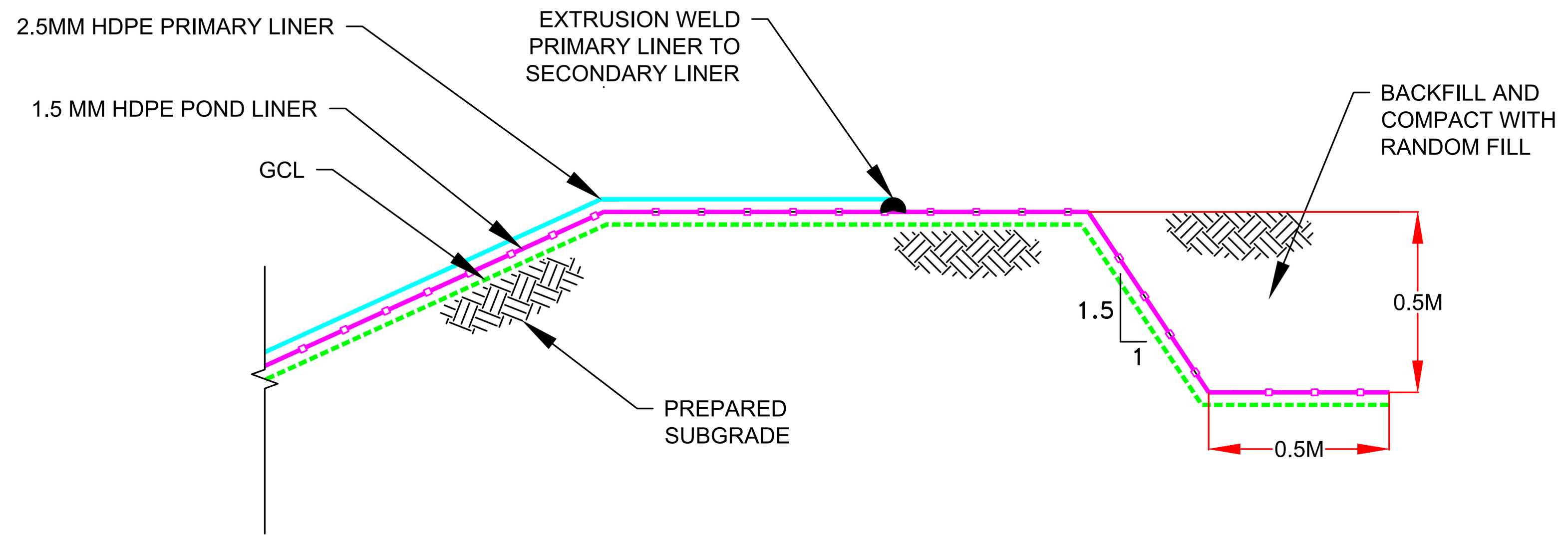
13:38 170217



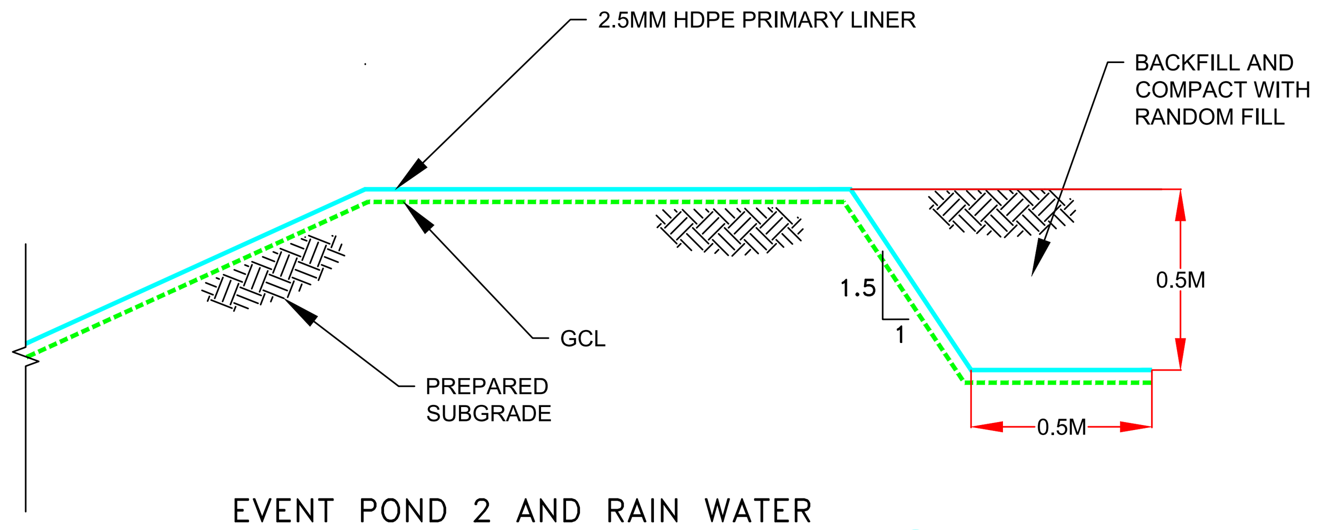
POND DECANT RISER DETAIL A
NTS



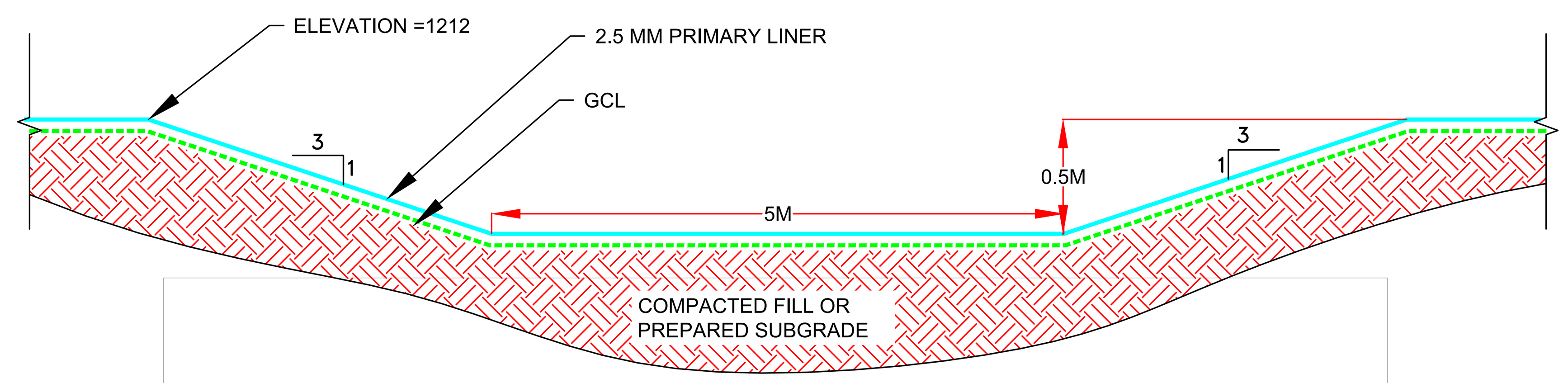
POND DECANT RISER DETAIL B
NTS



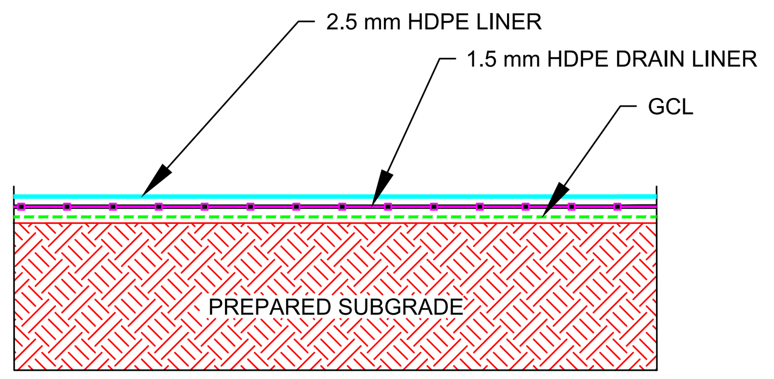
EVENT POND 1 LINER ANCHOR DETAIL C
NTS



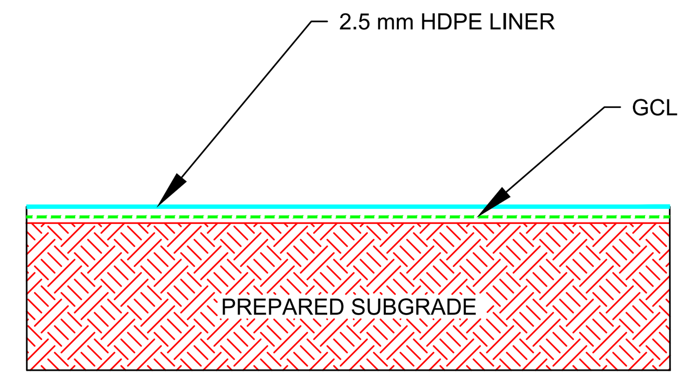
EVENT POND 2 AND RAIN WATER CHANNEL LINER ANCHOR DETAIL D
NTS



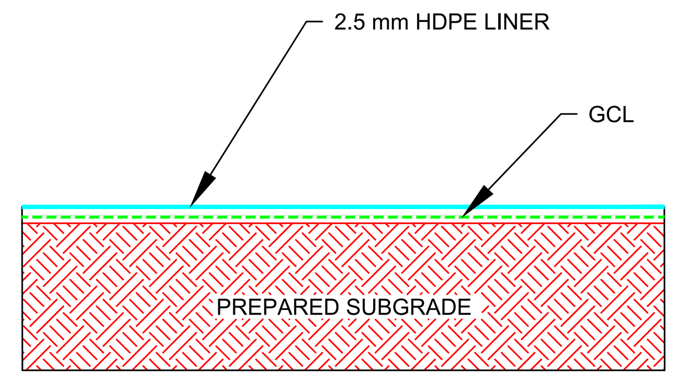
POND SPILL WAY SECTION E
NTS



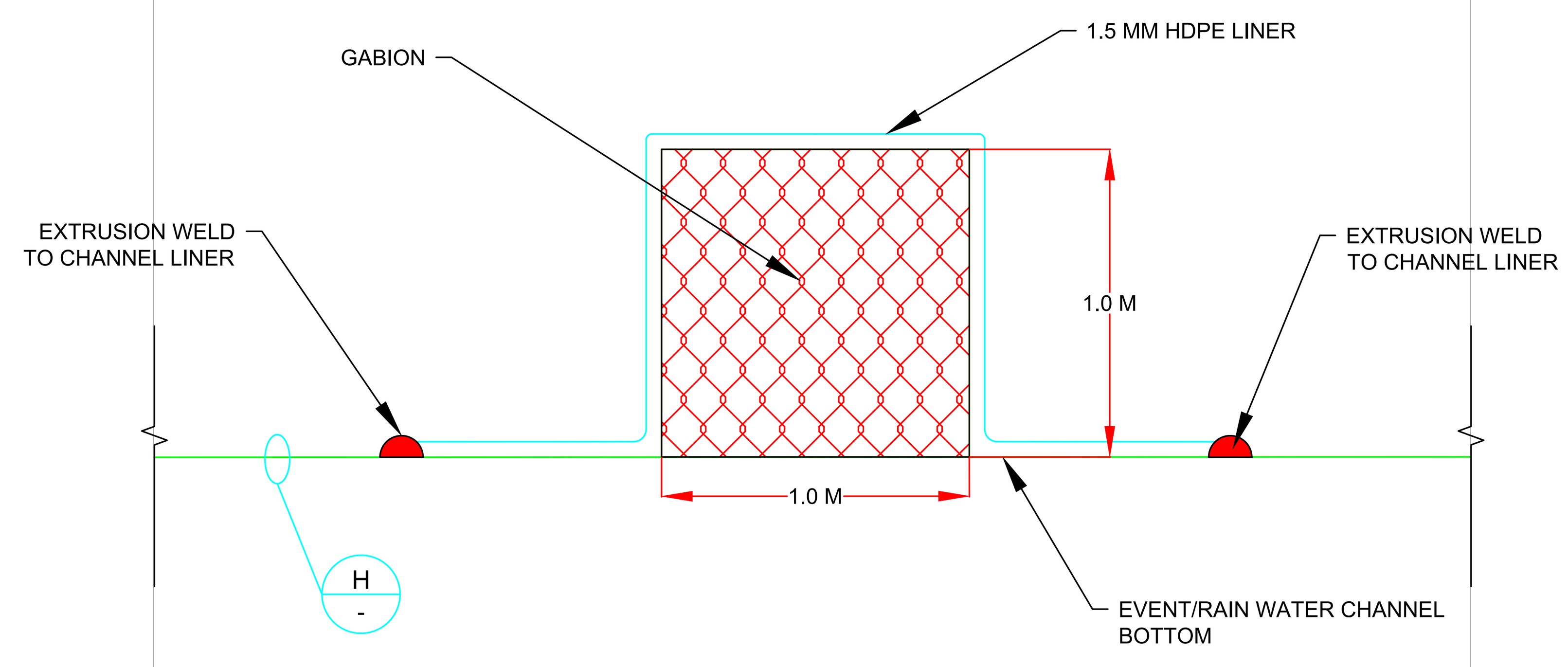
TYPICAL EVENT POND 1 LINER SECTION F
NTS



TYPICAL RAIN WATER POND/EVENT POND 2 LINER SECTION G
NTS



TYPICAL EVENT/RAIN WATER CHANNEL LINER SECTION H
NTS



TYPICAL EVENT/RAIN WATER CHANNEL DIVIDER DETAIL I
NTS

DWG. NO.	REFERENCE DRAWINGS
-	-

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2-17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

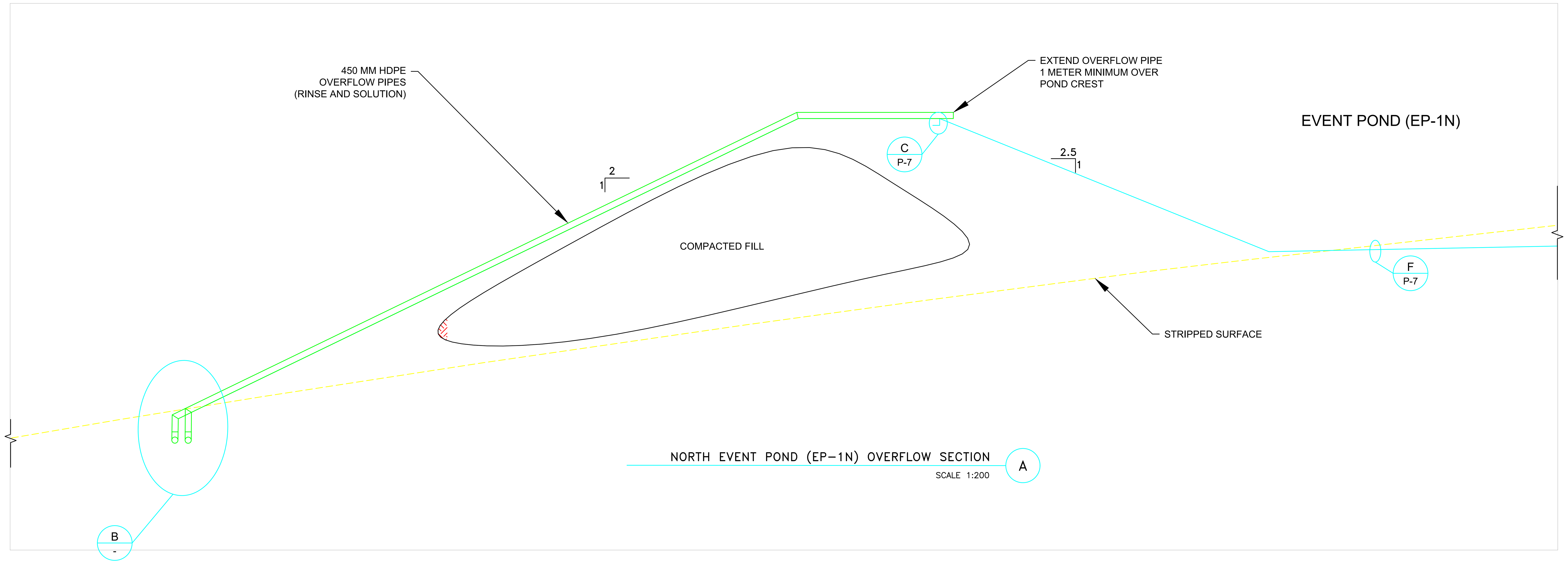
THE MINES GROUP INC.

COFFEE GOLD PROJECT

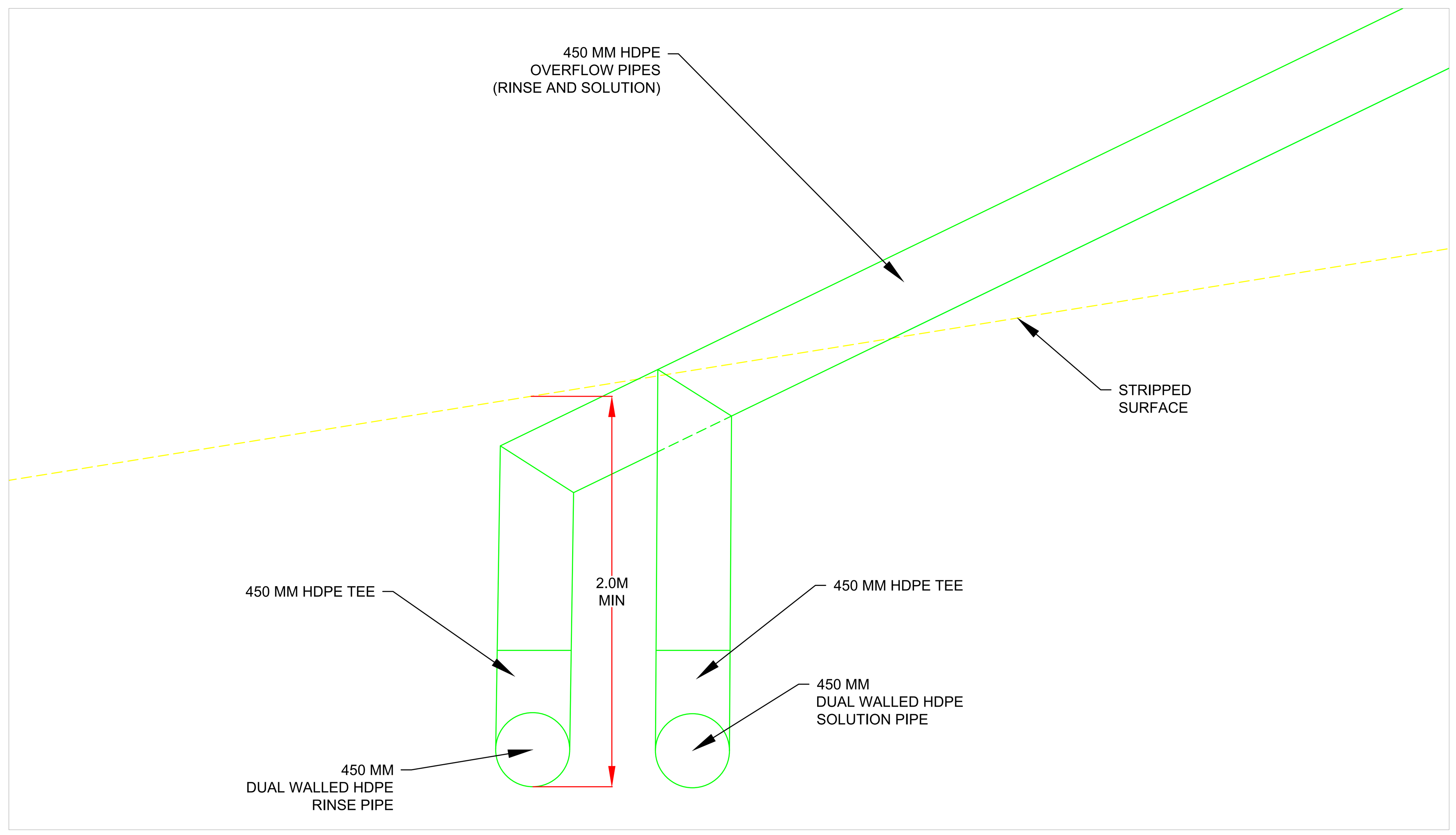
POND AND CHANNEL SECTIONS AND DETAILS

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	178201	P-7	A

C:\COMMON\A1-TUPRAC.DWT



NORTH EVENT POND (EP−1N) OVERFLOW SECTION (A)
SCALE 1:200



NORTH EVENT POND (EP−1N) OVERFLOW PIPE DETAIL (B)
SCALE 1:25

DWG. NO.	REFERENCE DRAWINGS
–	–

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
A		ISSUED FOR CLIENT REVIEW	2–17	AC

DISCIPLINE:	GENERAL	
SCALE:	NONE	DATE
DESIGNED BY:	AEWC	170217
DRAWN BY:	AEWC	170217
CHECKED BY:	AEWC	170217
APPROVED BY:	AEWC	170217

GOLDCORP

THE MINES GROUP INC.

COFFEE GOLD PROJECT

EVENT POND 1 (EP-1N) OVERFLOW PIPE
SECTION AND DETAILS

CONSULTANT PROJECT No.	178201	CONSULTANT DWG No.	
FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
		P-8	A

C:\COMMON\A1–TUPRAG.DWT

APPENDIX 2-C-2

The Mines Group Inc.

Memorandum (dated 6/5/2016)

Draindown Modeling for Coffee Project Heap Leach Facility

Memorandum

To: Mark Smith

From: Ken Myers

Date: 6/5//2016

Re.: Draindown modeling for Coffee Project Heap Leach facility

---

This report will briefly describe the development of a draindown water balance model of the proposed heap leach facility at the Kaminak gold coffee project in the Yukon Territory, Canada. Hour draindown model template was modified to accommodate the specific needs and characteristics of the coffee heap leach facility including progressive reclamation during operations, the use of raincoats, proposed staging of construction, and the proposed schedule for closure and reclamation. The remainder of this report will describe the development of the model, provide initial results, and explore the sensitivity of some assumptions.

Model Development

The draindown water balance model is designed to track the dewatering of the ore stack during the reclamation and closure portion of operations and the final reclamation and closure period following the end of operations. The model contains two separate components tracking changes in volumetric water content of the ore. The first component tracks the water content in the column of ore beneath the active area under leach. The ore in this column is assumed to be maintained at the elevated active operating water content as long as active irrigation and leaching is occurring at the facility. The flow associated with the draindown of solution is assumed to be unsaturated flow at unit gradient and in equilibrium with the associated unsaturated hydraulic conductivity. Wetting fronts associated with meteoric water (rainfall and snowmelt) are assumed to redistribute themselves within the column and to pass through the column as a pressure wave over the duration of the time step adding to the draindown at the base of the column. It is assumed that 30% of the meteoric water remains in the active leach column elevating the water content, and 70% passes through as a pressure wave. This redistribution of moisture elevating the water content of the ore in the leach column is assumed to persist only for one weekly timestep, then adds to the draindown volume dissipating entirely during the next timestep. Once active operations cease, the water content in the ore beneath the active area under leach begins to drop. The model recognizes that the excess solution within the column cannot immediately drain out and report to the ponds, as this would result in over whelming the ponds, filling them and spilling to the environment in a matter of just one or two time steps. Therefore, solution reporting to the ponds continues to be pumped and applied to the active area under leach. However, no makeup water is applied and any losses are permitted to accumulate and reduce the total water volume within the system. Any water pumped to treatment is also removed from the system and reflected as a reduction in the pumping rate and the total volume of solution applied during the next time step. As the volume of solution applied decreases, the water content within the leach column is recomputed and a new unsaturated hydraulic conductivity associated with the new water content is also calculated. As this unsaturated hydraulic conductivity falls, the draindown from the leach column for each

Reno Office:
1325 Airmotive Way
Reno, NV 89502
Phone: 775-322-7622
Fax: 775-322-2660
E-Mail: MINESGrp@aol.com

West Reno Office:
1835 Daniel Webster Drive
Reno, NV 89509
Phone: 775-329-3383
Fax: 775-329-2923
E-Mail: Email Address REDACTED

time step also falls. The volume of solution applied to the area under leach is the smaller of the normal operating solution volume or the solution volume that has accumulated in the pond system over the time step. This pumping and solution application continues with each time step until such time as the water content within the leach column approaches the water content in the unirrigated portion of the ore stack.

The unirrigated portion of the ore stack is assumed to exist at a water content equal to the specific retention until such time as either reclamation or the application of raincoats begins. Meteoric water is again assumed to pass through the ore stack as a pressure wave. Once again, about 30% of the meteoric water volume is assumed to remain within the ore stack elevating its water content and about 70% of the meteoric water volume is assumed to pass through the ore stack as a storm surge, adding to the draindown volume for the time step. As the reclaimed area or the raincoat coverage area increases, the volume of infiltrating meteoric water decreases. With each time step the mean water content of the unirrigated ore decreases until the unsaturated hydraulic conductivity of the ore is in equilibrium with the volume of meteoric water entering the ore.

Once the active leach column has dewatered to the point where the mean water content within the column is equal to the mean water content within the unirrigated portion of the ore, then both areas are assumed to continue dewatering at the same rate influenced only by meteoric water as all pumping and solution application will be assumed to have stopped.

The rate at which the active leach column dewaters is largely controlled by the rate at which water is removed from the system by pumping to treatment. Different rates can be applied to the period prior to the dewatering of the leach column and the period after dewatering of the leach column. Pumping to treatment is assumed to occur only during the months of April through September. During the other months of the year (October through March), water in the ponds is assumed to exist as ice and not be available for pumping.

Model Results

Model results representing currently proposed operating conditions in the Coffee Project heap leach facility are shown in Figure 1 though Figure 7. The timeline begins (elapsed time (ET) = 0) in April of operational year five (5) shortly before concurrent reclamation and use of raincoat covers is expected to begin. The constant relocation of the active area under leach over the entire area of the ore stack is assumed to maintain the water content in the unirrigated portion of the ore at or near the specific retention level. However, once significant portions of the ore stack have completed the leaching process permitting concurrent reclamation to begin, it is assumed that the unirrigated portion of the ore will begin to dewater to water contents below the specific retention. In the ore column below the active area under leach water contents are assumed to be maintained at the elevated active leach water content until the end of operations and ore stacking, assumed to occur by the end of operational year 13 (ET = 104 months or 8.67 yrs). Flow rates for water diverted to treatment are assumed to be at levels associated with the management of pond levels and system water volumes during operations. This is assumed to be 2 liters per second (l/s) in April and 4 l/s for May through September during operations and 5 l/s during April and 10 l/s for May through September immediately after operations. The schedule for progressive reclamation and raincoat cover application were imported into the draindown model from the operational model.

Figure 1 shows a plot of the total draindown in cubic meters per month (m3/month). Operations ceased at ET = 104 months. With a maximum pump to treatment rate of 12 l/s and the last snowmelt event occurring more than 6 months earlier, the ponds should be close to empty at the time operations cease. So pumping rates returning solution to the leach pad are able to fall quickly because water contents and draindown rates are able to fall quickly allowing pumping to stop at about ET = 135 months or only about 2.6 yrs after the end of operations. The 5 l/s (April) to 10 l/s (May through September) schedule results in the removal of 144,540 m^3 annually. This represents about 49% of the entire water volume stored within the leach column. Water contents drop rapidly after pumping stops, and for all practical purposes the dewatering of the leach column is complete by ET = 170 months or approximately 5.5 years after the end of operations. The rapid drop in water content is shown in Figure 5. The water volume stored in ponds is shown in Figure 6. The early portion of the timeline is simply a map of water stored in ponds over the period of normal leach operations. The pond levels shown from ET = 104 months on reflect the mean pond levels expected during and after draindown.

Figure 1 – Total Draindown in M^3/Month

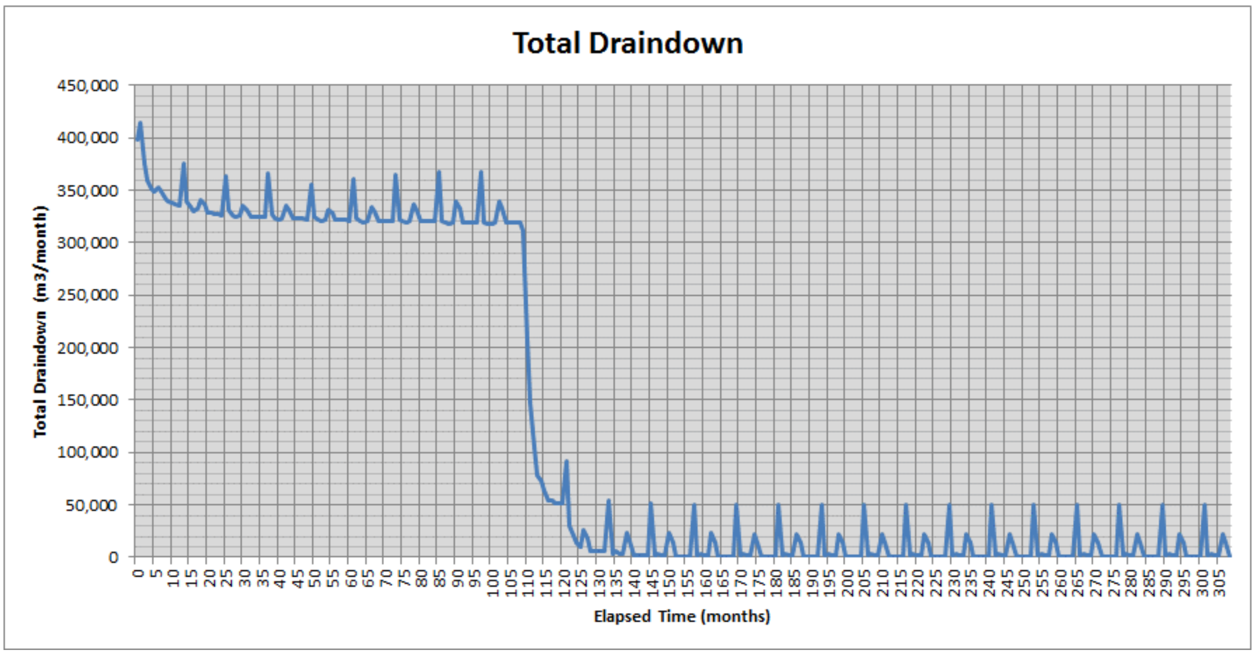


Figure 2 – Draindown Flow Rate in Liters/Second

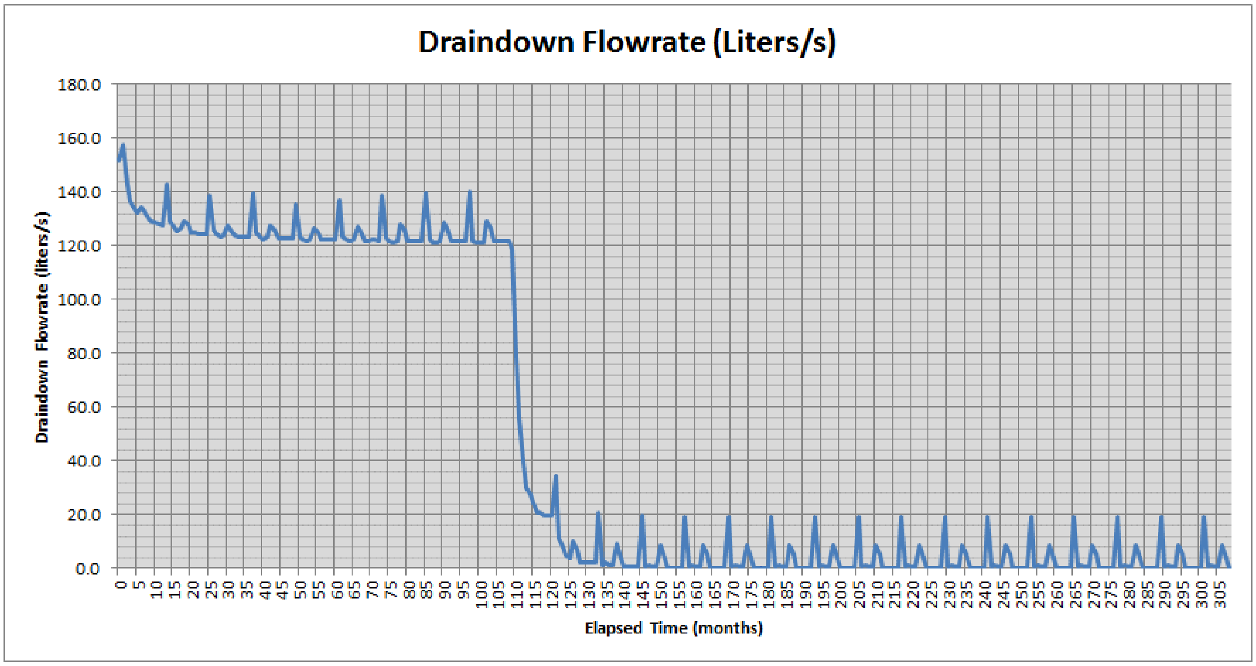


Figure 3 – Draindown Flow Rate in M^3/Hour

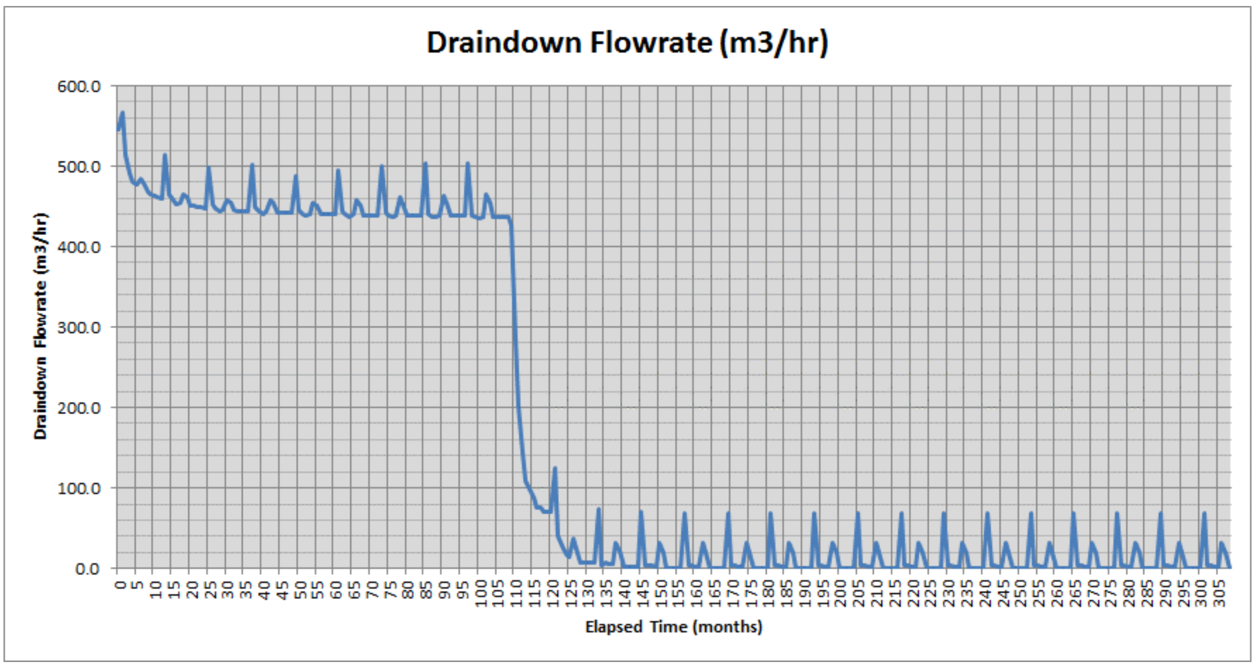


Figure 4 – Total Water Volume in the System

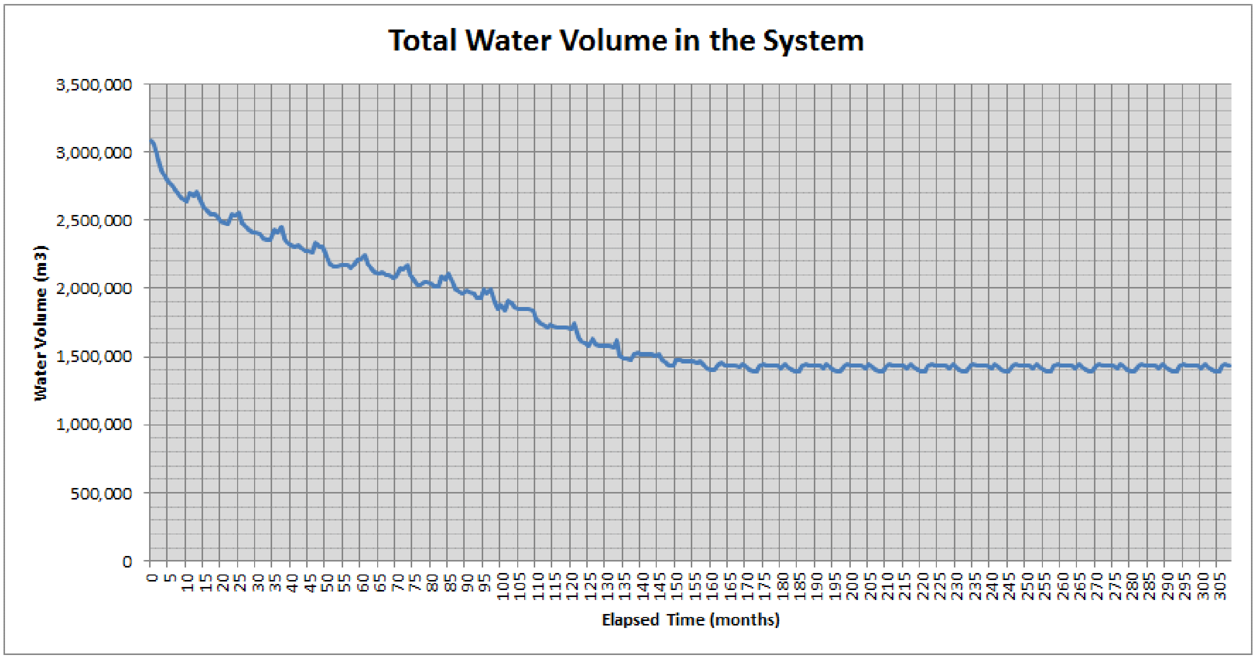


Figure 5 – Changes in Volumetric Water Content (%)

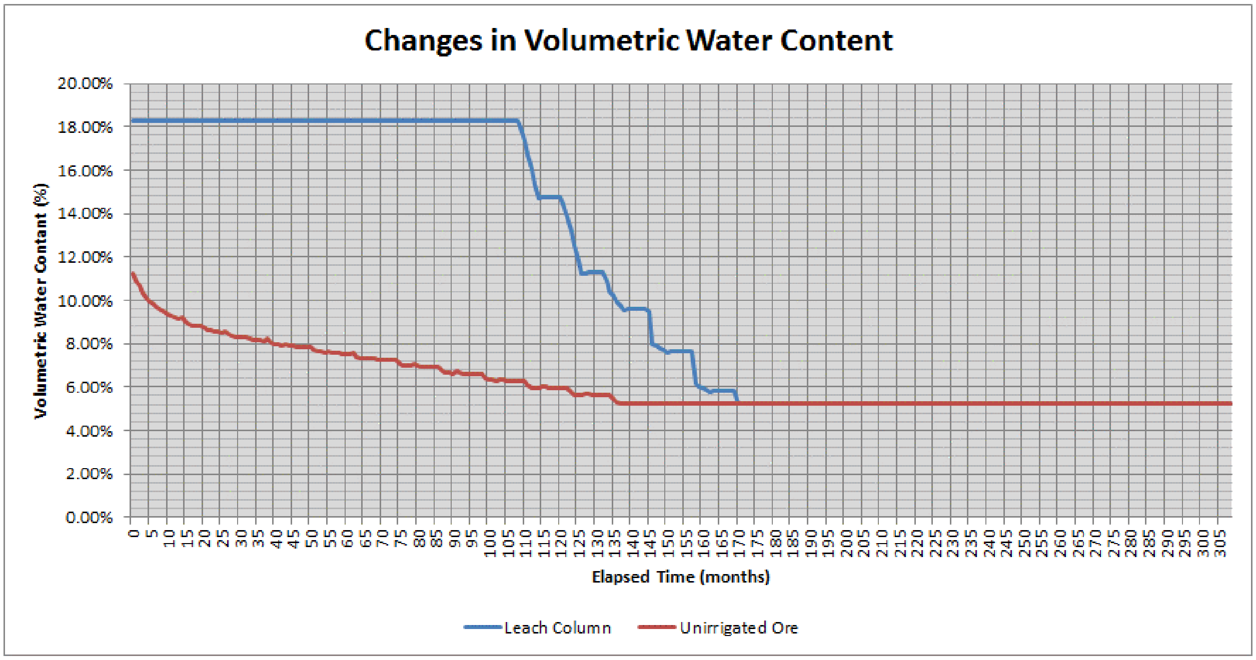


Figure 6 –Water Stored in Ponds (M^3)

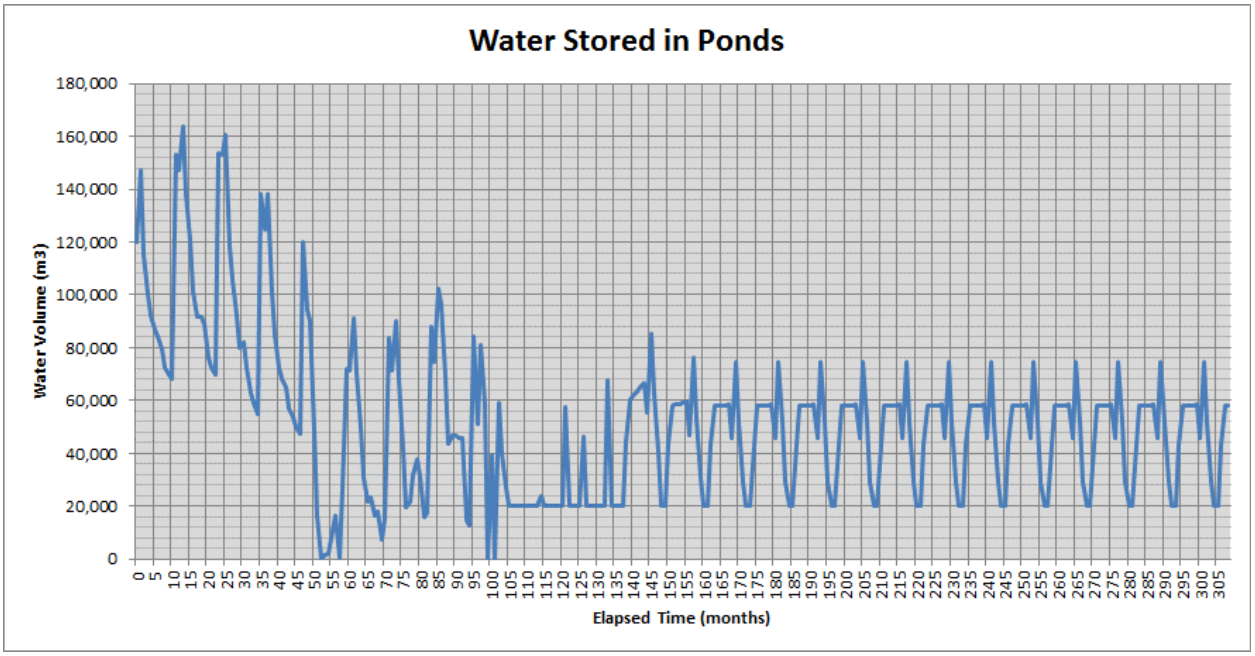
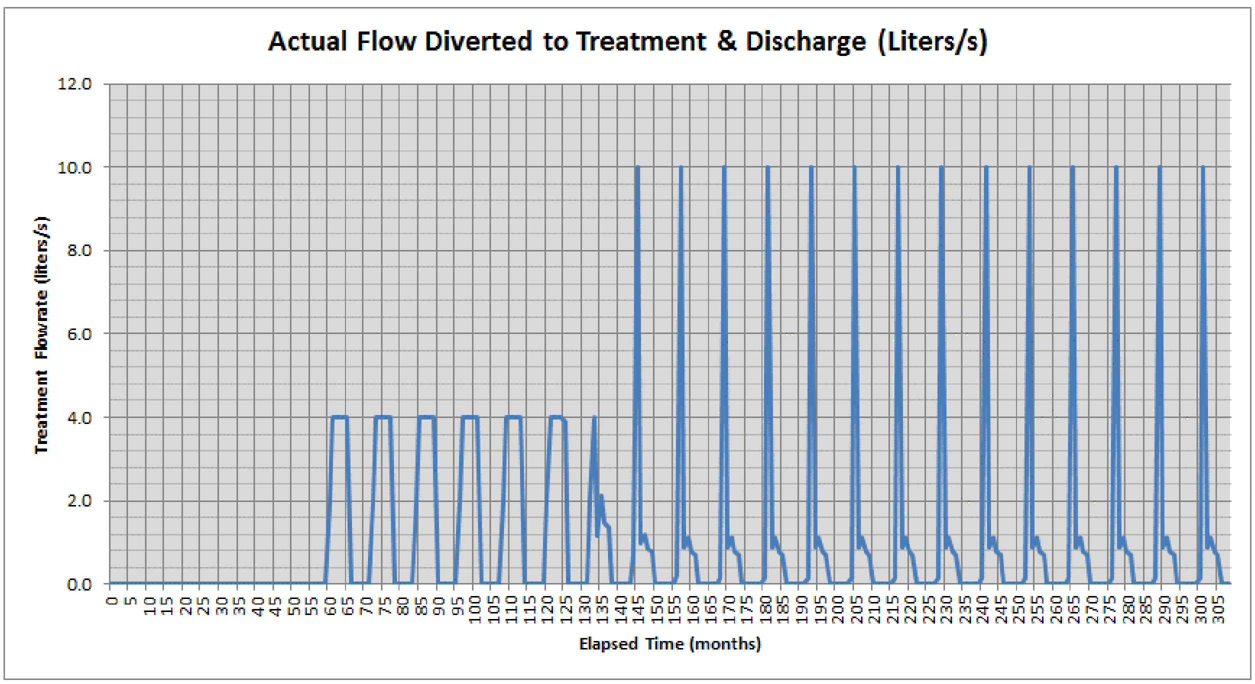


Figure 7 - Actual Flow Diverted to Treatment (Liters/Second)



It should be noted that the model utilizes mean estimates of precipitation, snowmelt, runoff, infiltration, and so on. Spikes in meteoric water and seasonal variability would be managed within the pond system with surges resulting in increases in pond storage that would be reduced over time by the mean pumping to treatment rates. To minimize the variability in pond storage one should consider selecting a long term mean pumping to treatment rate somewhat above the absolute minimum mean rate required to prevent accumulation over the long term (which is about 3.1 l/s under the current set of assumptions).

As regards sensitivity to cover performance and reclamation coverage, a CN of 91 was used in the model to represent final reclamation cover performance. This CN results in a mean annual infiltration estimate on the order of 22% of mean annual precipitation. Decreasing the CN to 90 would reduce the mean annual infiltration to just under 25% of mean annual precipitation.

The intent of this report is to provide the reader with a feel for the potential range of impacts associated with varying model assumptions. It is expected that a series of "what if" scenarios of reasonable and realistic assumptions can be provided and the model used to examine the complex interactions of varying assumptions to optimize post operation performance of the leach pad facility.

We hope this has provided you with the information you required. Should you have any questions or concerns, please do not hesitate to call at Phone Number REDACTED

Signatu ards,

Signature REDACTED

Kenneth L Myers, P.E.

Memorandum

To: James Scott

From: Kenneth Myers

Date: March 22, 2017

Re.: Comparison of draindown performance for heap capacities ranging from 47.3 to 60.4 million tonnes

---

This memorandum will compare and contrast differences in the draindown performance of the proposed Heap Leach Facility (HLF) for the Coffee Project in Yukon Territory, Canada at two different maximum ore capacities (47.3 million tonnes (MT) and 60.4 MT).

Design criteria that are common to both scenarios:

- Ore production rate remains the same for each scenario at 18,265 tonnes per day for 9 months out of each year (5 MT per year).
- Application rate is 10 liters/s/m^2 for both scenarios with a pumping rate of 455,000 liters per hour producing an area under leach of 45,500 m^2.
- Rinsing is assumed to begin in June of year 4 at an application rate of 10 liters/s/m^2 for both scenarios with a pumping rate of 118,000 liters per hour.
- Pumping to treatment begins in March of year 9 at a rate of 2 liters/s (5184 m^3 per month) for April and 4 liters/s (about 10,513 m^3 per month) for the months of May through September.
- Ore characteristics include an estimated delivered gravimetric water content of 4.5%, a specific retention of 6.5%, and an operating water content of 10.6% on an estimated stacked ore density of 1.6 tonnes per m^3.

The timeline for draindown modeling begins (elapsed time (ET) = 0) in April of operational year five (5) shortly before concurrent reclamation and use of raincoat covers is expected to begin. The constant relocation of the active area under leach over the entire area of the ore stack is assumed to maintain the water content in the unirrigated portion of the ore at or near the specific retention level. Once ore stacking stops and the main leach pumping rate is no longer being supported by outside makeup water, the active leach column also begins to dewater to water contents below the specific retention, starting from the higher operating water content. This is assumed to occur after all gold production has ceased and the rinsing operation has completed which for the 47.3 MT scenario is assumed to happen about December of year 13 (ET = 104 months or 8.67 years) and for the 60.4 MT scenario about March of year 15 (ET = 121 months or 10.08 yrs). Flow rates for water diverted to treatment are assumed to be at levels associated with the management of pond levels and system water volumes during operations. This is assumed to be 2 liters per second (l/s) in April and 4 l/s for May through September during operations for both scenarios. For the 60.4 MT scenario, the post operations pumping to treatment rate is 5 l/s during April and 11 l/s for May through September (this increased slightly from the earlier 10 l/s for the 47.3 MT scenario due to the larger lined footprint). The schedule for progressive reclamation and raincoat cover application were imported into the draindown model from the operational model.

Before dewatering of the leach column can end it must catch up to the water content of the unirrigated ore. The leach column is assumed to be dewatered when the mean water content in the leach column equals the mean water content in the unirrigated ore. All dewatering is assumed to end when the mean water content of the ore reaches some arbitrary low value (in our case the estimated permanent wilting point). From then on, water moving through the pad cycles in response to whatever meteoric water it receives.

Figure 1 through Figure 6 show expected mean monthly draindown rates in various units (m^3/month, l/s, and m^3/hr) over the elapsed time in months. The timing of the initiation of the draindown of the leach column varies as a result of the extended duration of operations for the 60.4 MT scenario. However, the increase in the post operations pumping to treatment rate results in an acceleration of the time required to dewater the leach column (Figure 9 and Figure 10).

Figure 1 – 47.3 MT Scenario

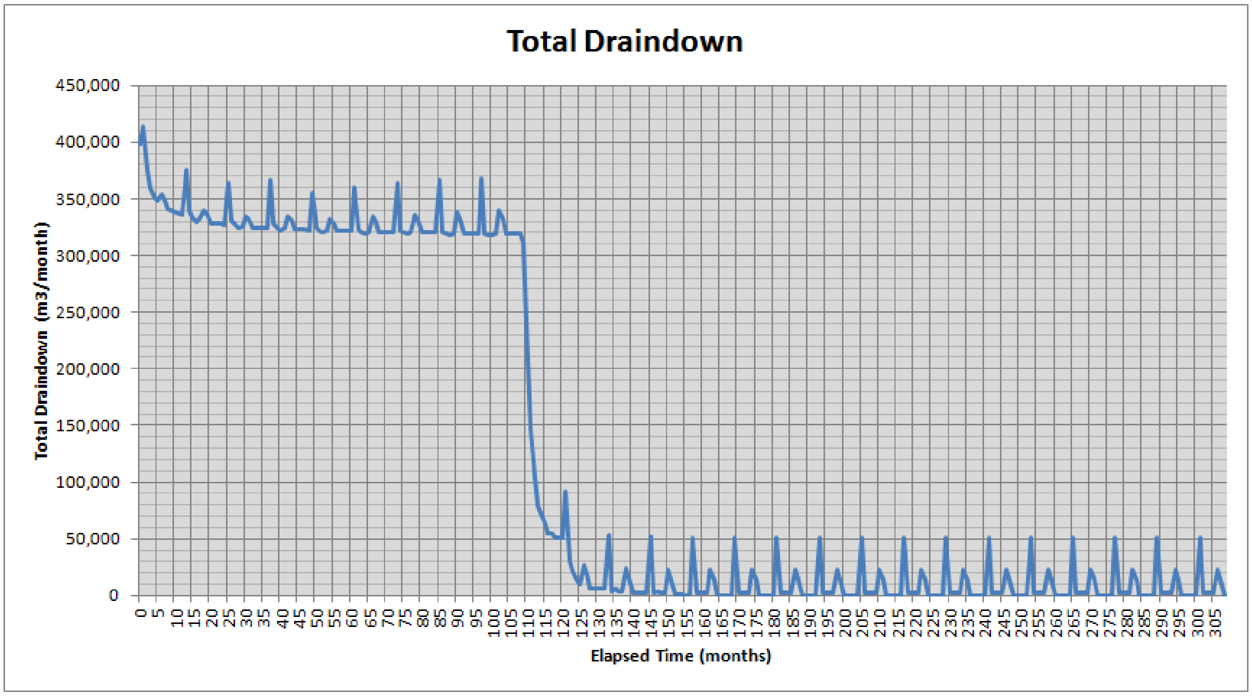


Figure 2 – 60.4 MT Scenario

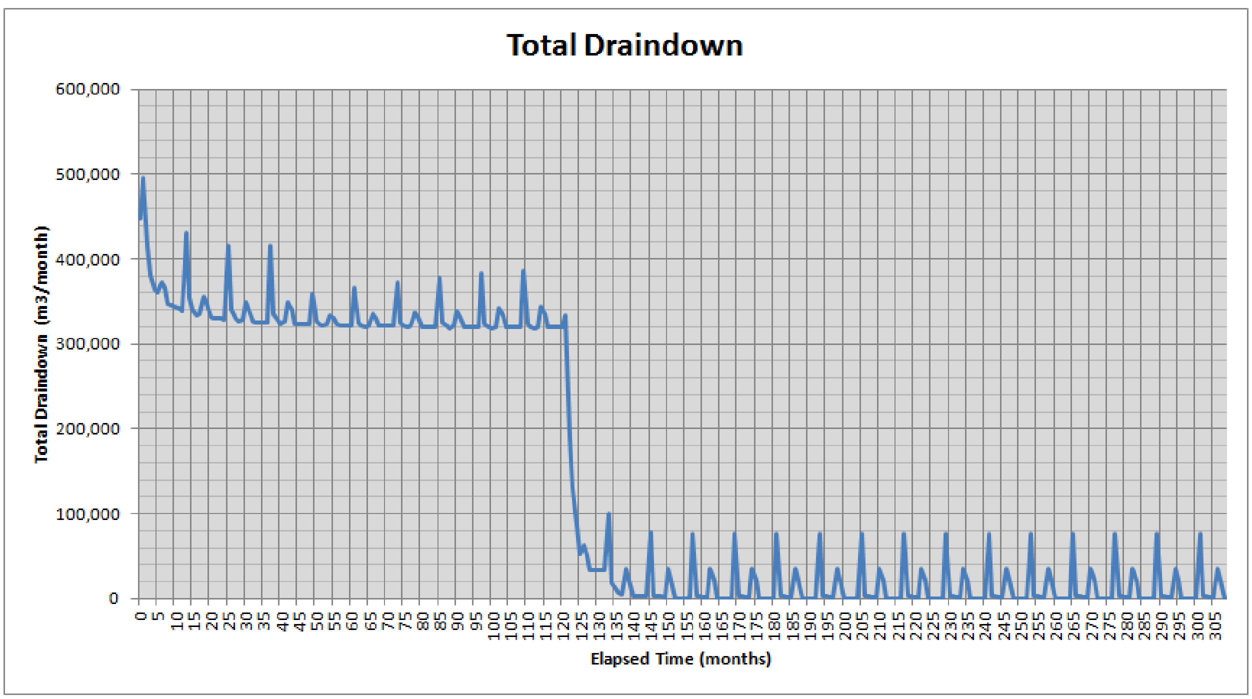


Figure 3 – 47.3 MT Scenario

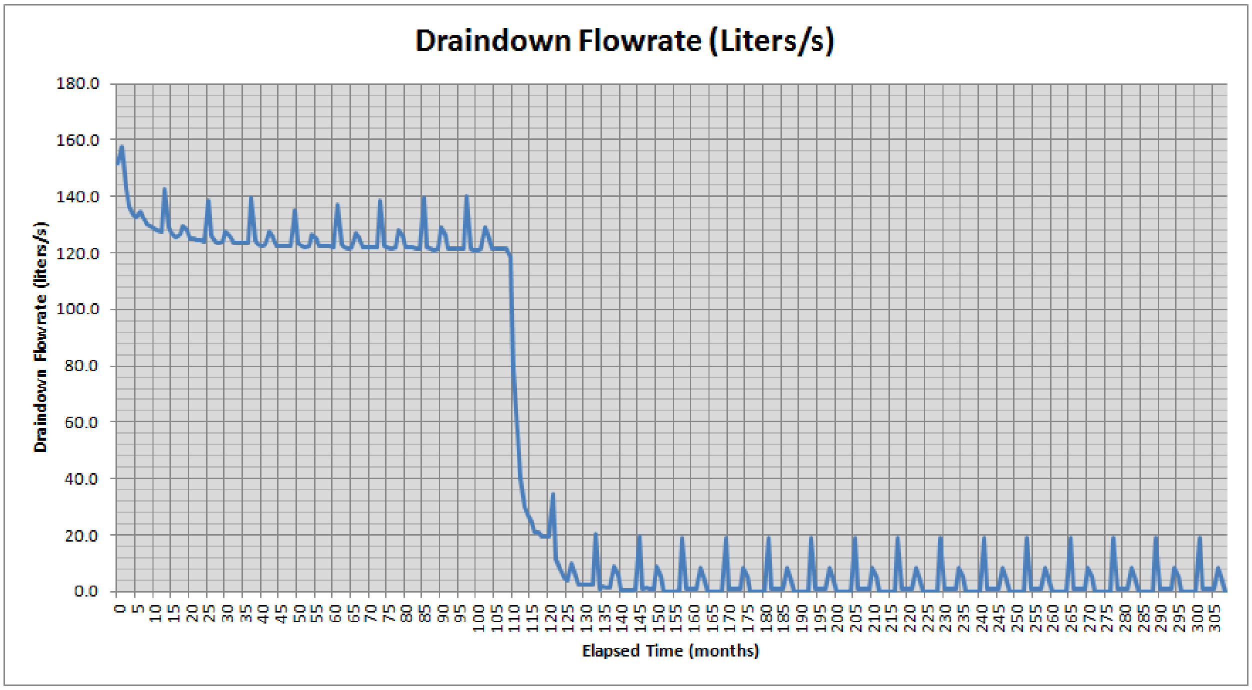


Figure 4 – 60.4 MT Scenario

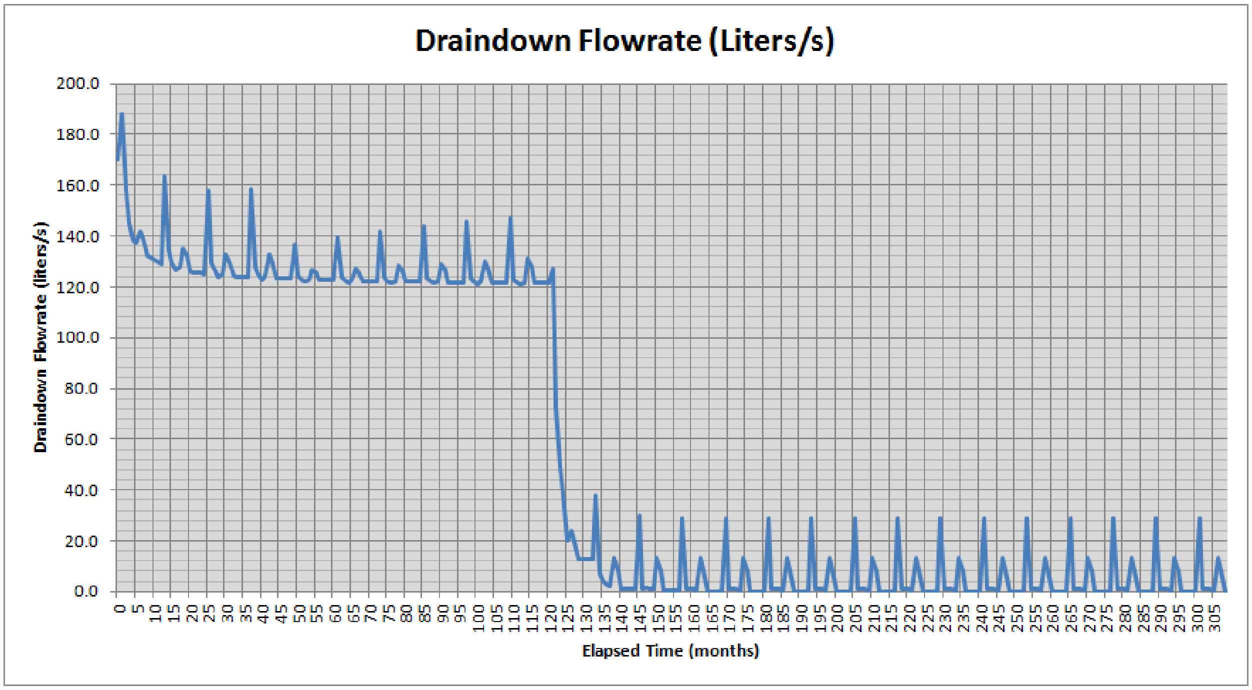


Figure 5 – 47.3 MT Scenario

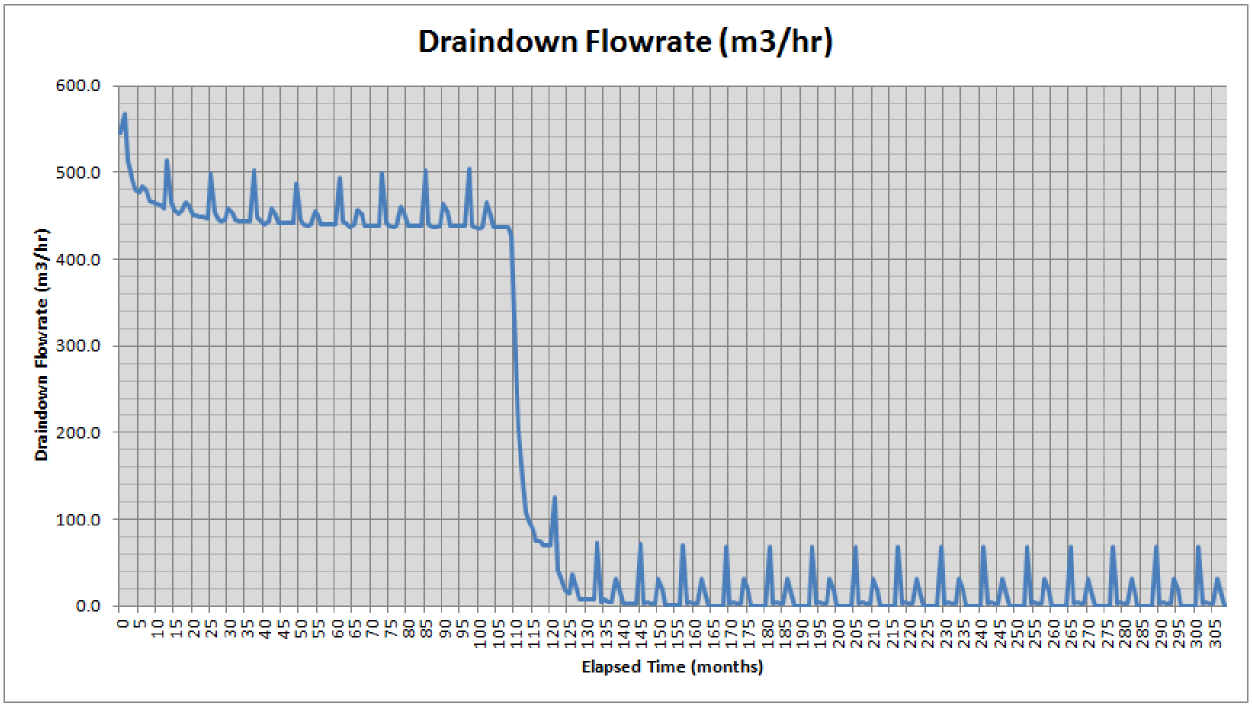


Figure 6 – 60.4 MT Scenario

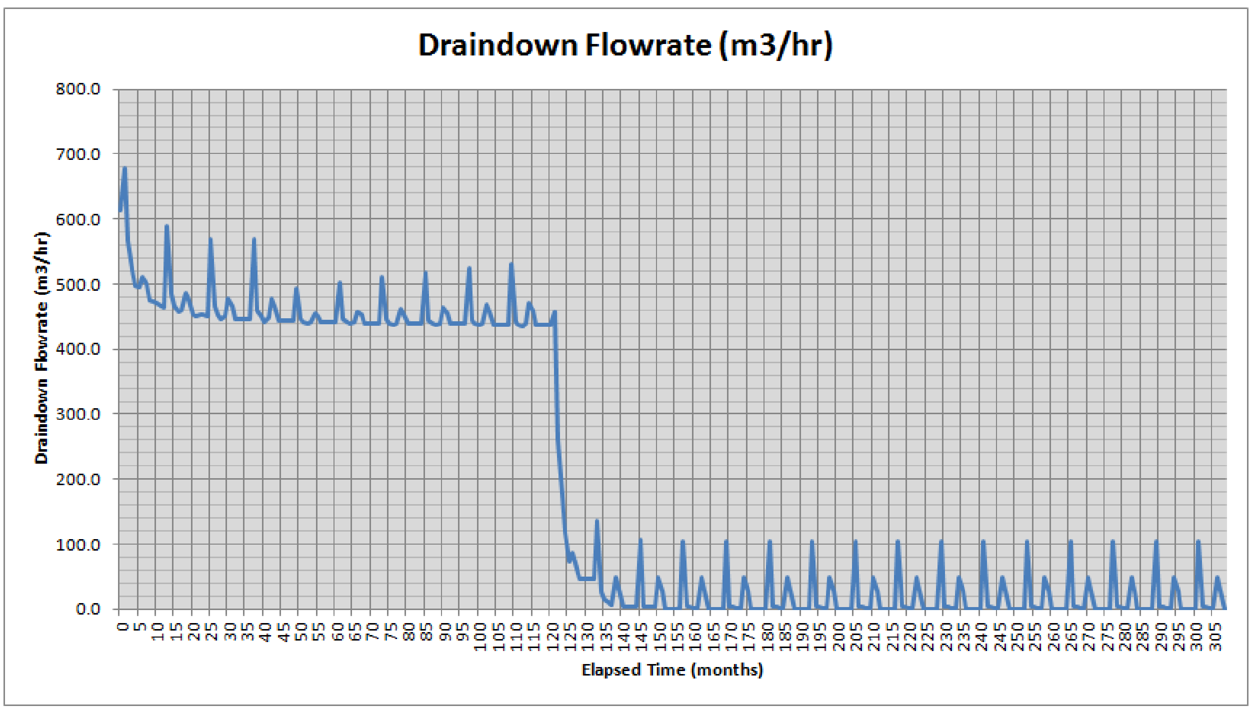


Figure 7 shows an undrainable volume on the order of 1.4 million m^3 for the 47.3 MT scenario while Figure 8 shows an undrainable volume on the order of 1.8 million m^3 for the 60.4 MT scenario.

Figure 7 – 47.3 MT Scenario

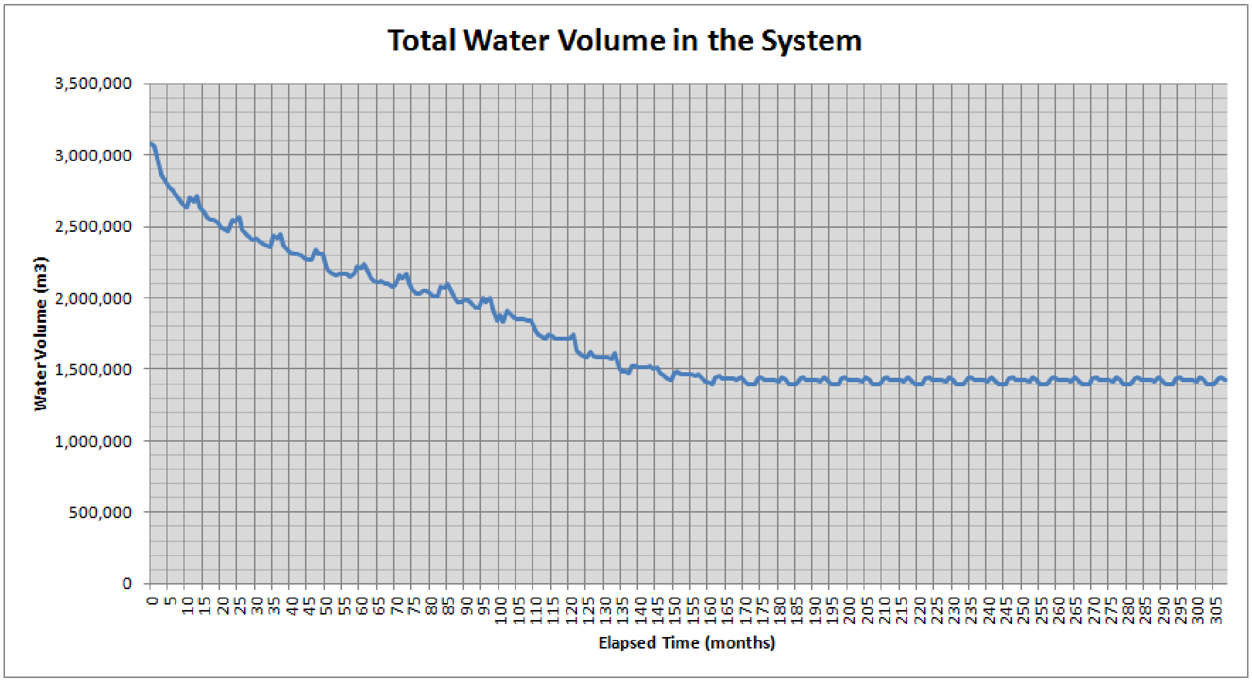


Figure 8 – 60.4 MT Scenario

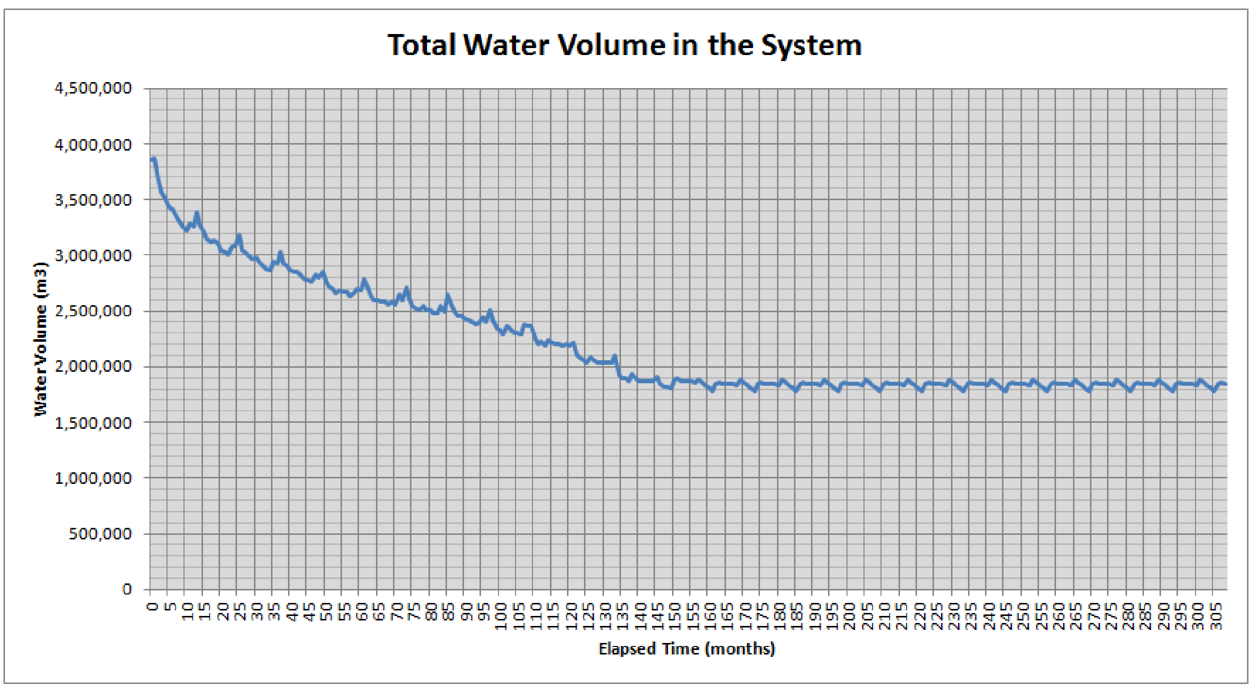


Figure 9 – 47.3 MT Scenario

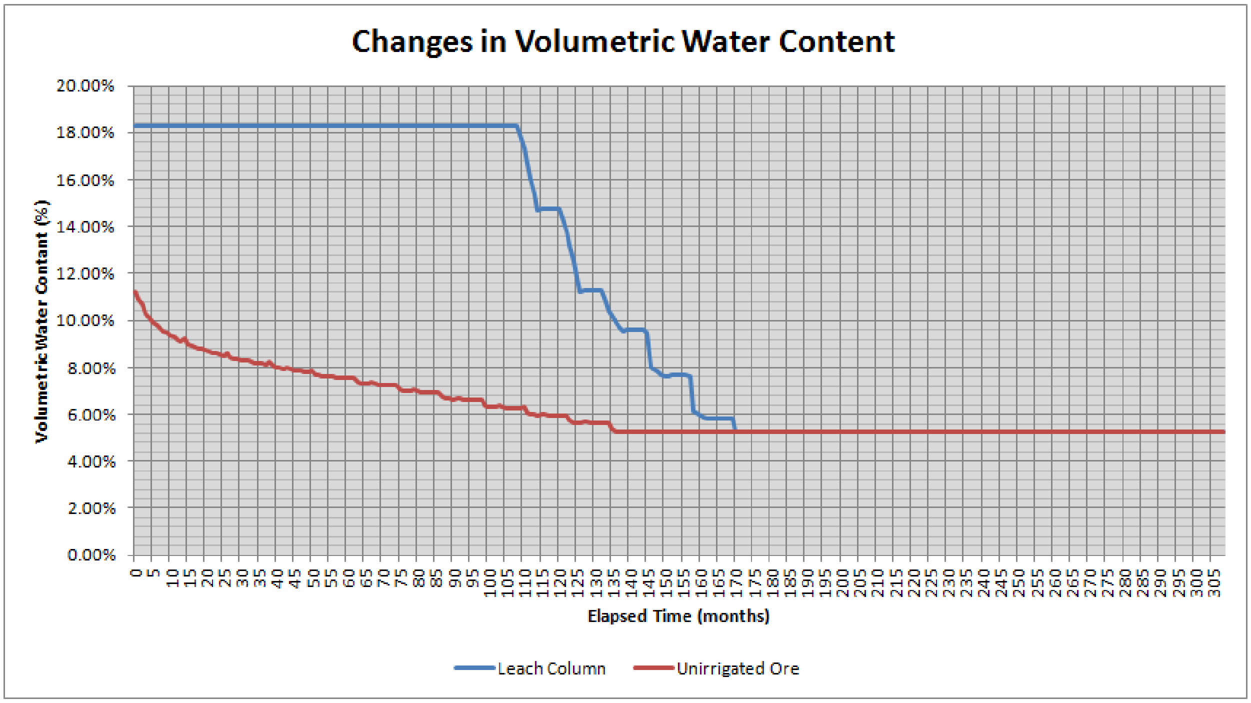
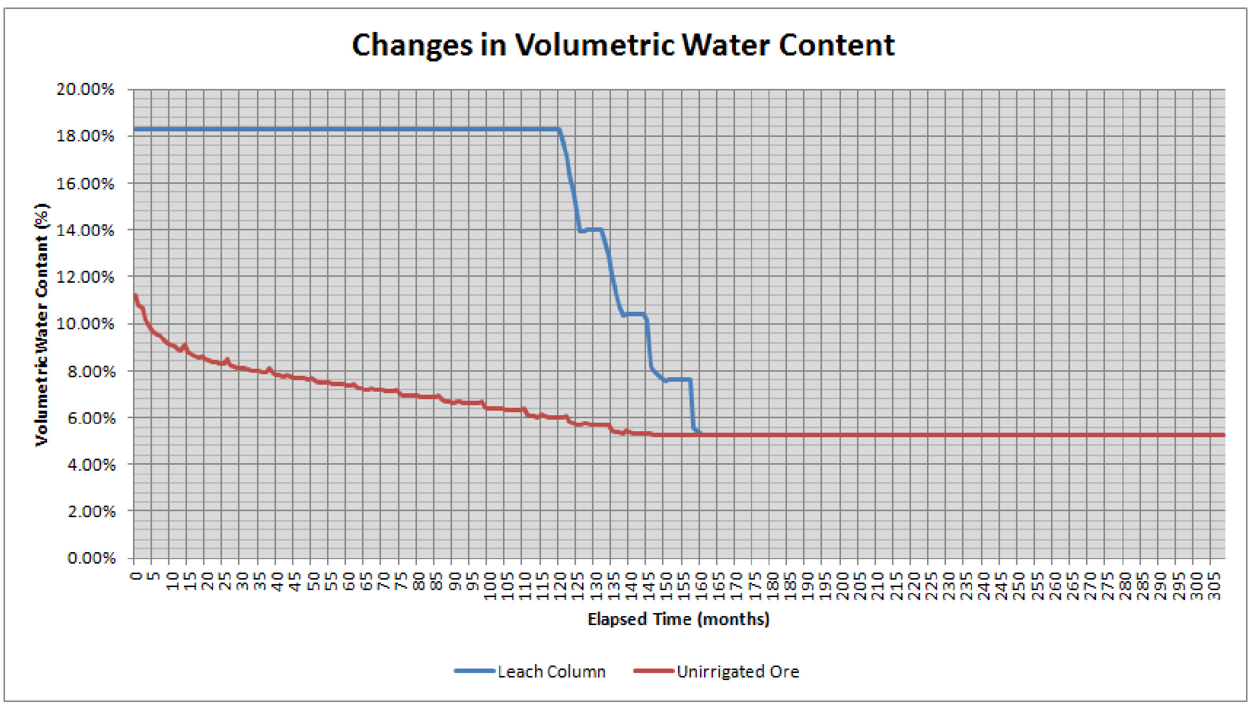


Figure 10 – 60.4 MT Scenario



The larger lined footprint of the 60.4 MT scenario results in an increase in the average maximum annual peak pond storage level from about 75,000 m^3 (Figure 11) to about 120,000 m^3 (Figure 12).

Figure 11 – 47.3 MT Scenario

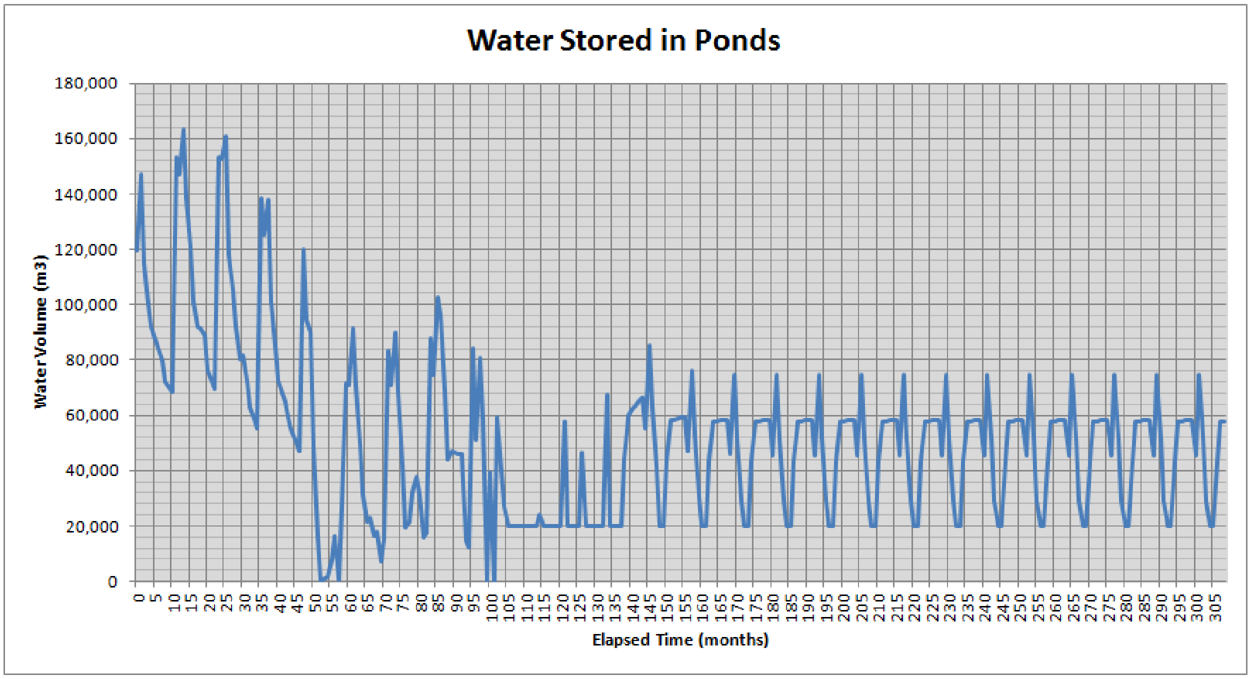
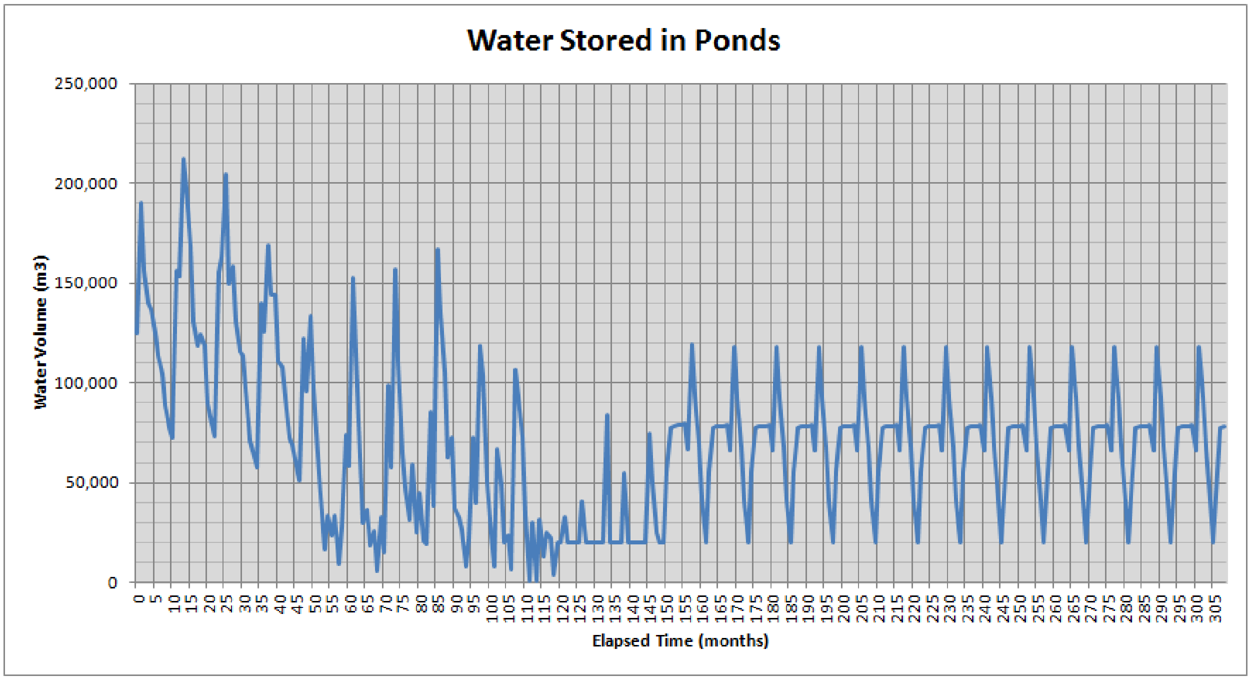


Figure 12 – 60.4 MT Scenario



To prevent accumulation of water in the pond system long term, the peak pumping to treatment rate for the

60.4 MT scenario must be increased from 10 l/s (Figure 13) to 11 l/s (Figure 14). The peak pumping rate can only be applied when there is sufficient water present to sustain it. If there is not sufficient water present, then the mean rate will be that required to empty the pond.

Figure 13 – 47.3 MT Scenario

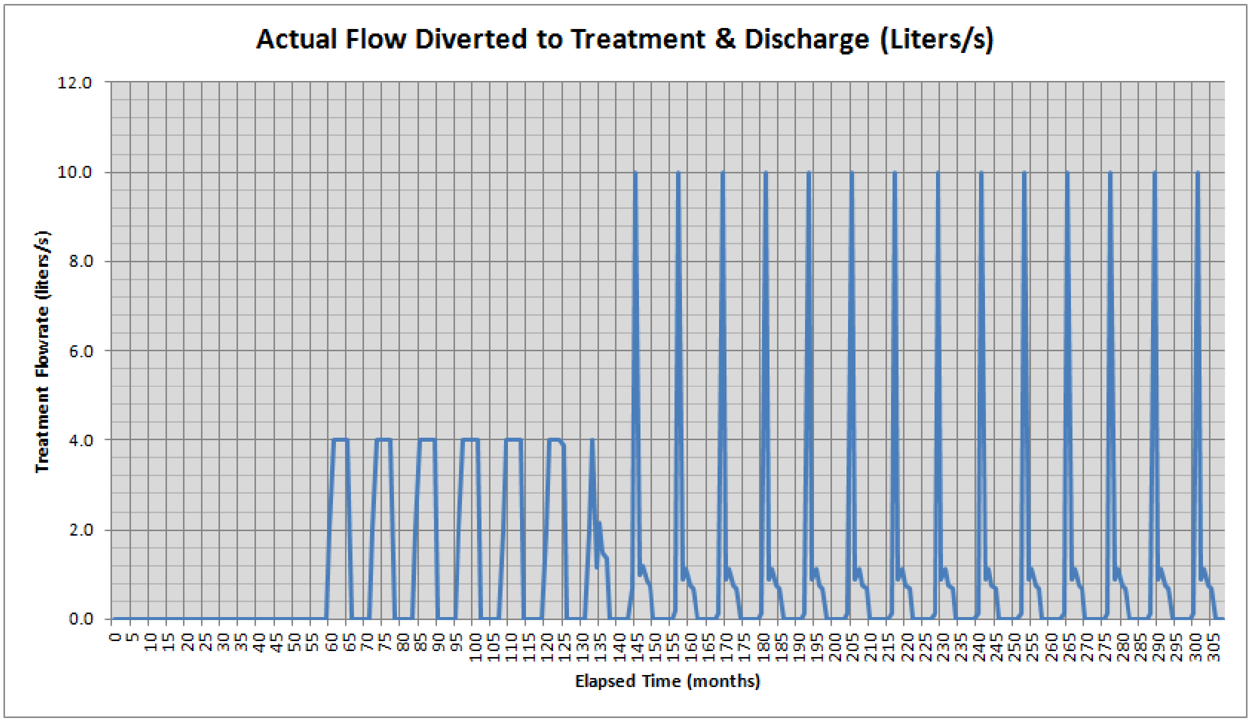
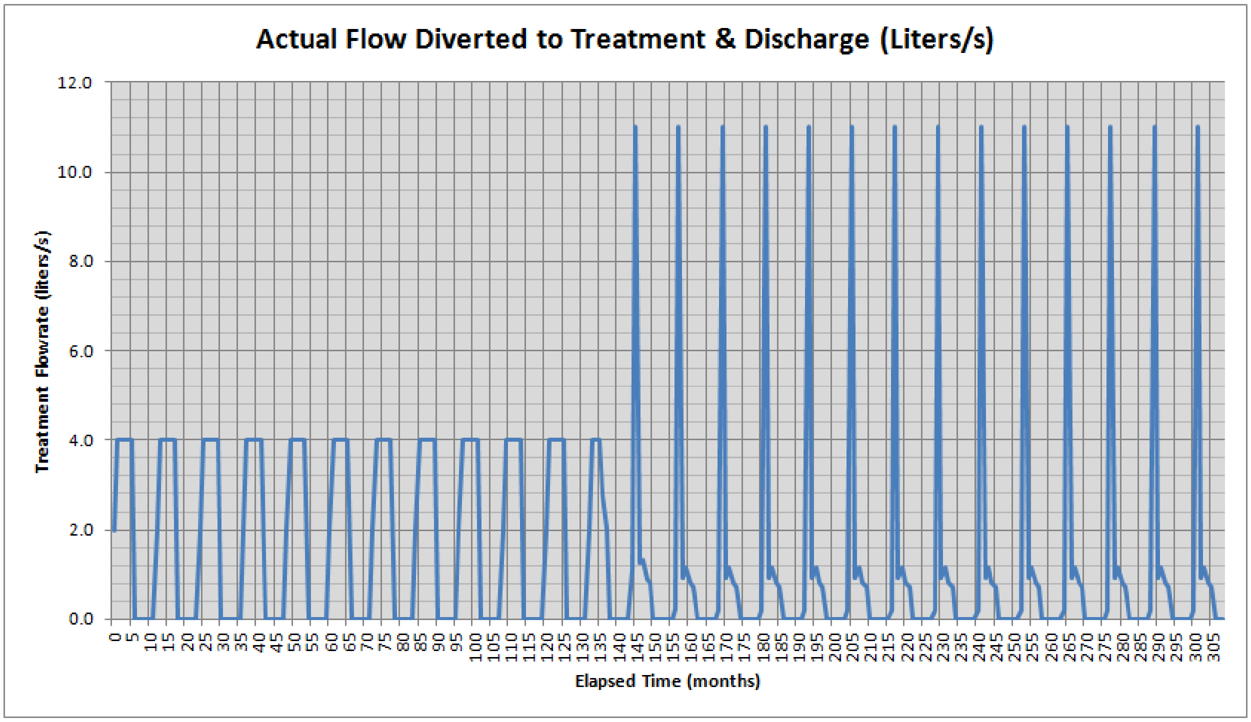


Figure 14 – 60.4 MT Scenario



It should be noted that the model utilizes mean estimates of precipitation, snowmelt, runoff, infiltration, and so on. Spikes in meteoric water and seasonal variability would be managed within the pond system with surges resulting in increases in pond storage that would be reduced over time by the maximum pumping to treatment rates. For the 60.4 MT scenario pumping rate schedule consisting of 5 l/s for April and 11 l/s for May through September, the treatment system would be capable of evacuating up to 155,370 m^3 of water from the system per year.

Please call if you need additional information or have any questions.

Regards,
The MINES Group, Inc.
Signature REDACTED

Kenneth L. Myers

Memorandum

To: James Scott

From: Kenneth Myers

Date: March 21, 2017

Re.: Comparison of water balance performance for heap capacities ranging from 47.3 to 60.4 million tonnes

---

This memorandum will compare and contrast differences in the performance of the proposed Heap Leach Facility (HLF) for the Coffee Project in Yukon Territory, Canada at two different maximum ore capacities (47.3 million tonnes (MT) and 60.4 MT).

Design criteria that are common to both scenarios:

- Ore production rate remains the same for each scenario at 18,265 tonnes per day for 9 months out of each year (5 MT per year).
- Application rate is 10 liters/s/m^2 for both scenarios with a pumping rate of 455,000 liters per hour producing an area under leach of 45,500 m^2.
- Rinsing is assumed to begin in June of year 4 at an application rate of 10 liters/s/m^2 for both scenarios with a pumping rate of 118,000 liters per hour.
- Pumping to treatment begins in March of year 9 at a maximum rate of 4 liters/s (10,513 m^3 per month).
- Ore characteristics include an estimated delivered gravimetric water content of 4.5%, a specific retention of 6.5%, and an operating water content of 10.6% on an estimated stacked ore density of 1.6 tonnes per m^3.

The 47.3 MT scenario continues ore stacking into November of year 9. Given that the ore production rate, solution application rate, and solution pumping rate are the same in both scenarios, performance is virtually identical up through October of year 9. Ore stacking continues in the 60.4 MT scenario into July of year 12, and in order to accommodate the additional volume, additional lined footprint is required. In October of year 9, the lined footprint is increased from 819,050 m2 to 1,090,407 m2 (see Figure 1 and Figure 2). The leaching of the additional volume of ore extends ore wetting losses and makeup water demand into July of year 12 (See Figure 3 and Figure 4). However, makeup water demand continues to be satisfied using fresh water runoff from raincoat areas and concurrent reclamation areas (see Figure 5 through Figure 8).

Reno Office:
1325 Airmotive Way
Reno, NV 89502
Phone: 775-322-7622
Fax: 775-322-2660
E-Mail: MINESGrp@aol.com

West Reno Office:
1835 Daniel Webster Drive
Reno, NV 89509
Phone: 775-329-3383
Fax: 775-329-2923
E-Mail: Email Address REDACTED

Figure 1 – 47.3 MT Scenario

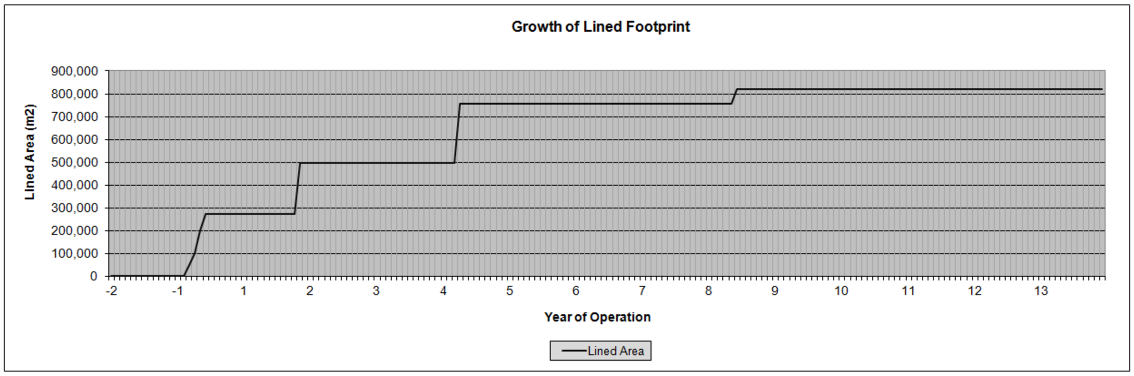


Figure 2 – 60.4 MT Scenario

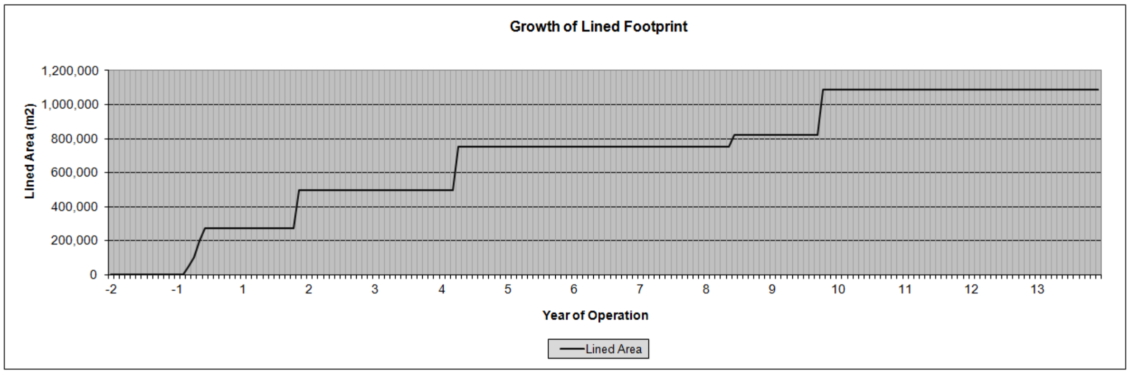


Figure 3 – 47.3 MT Scenario

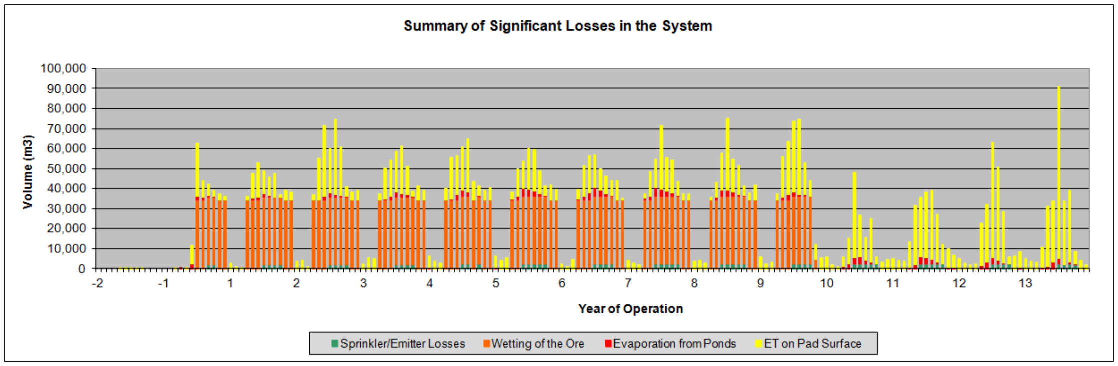


Figure 4 – 60.4 MT Scenario

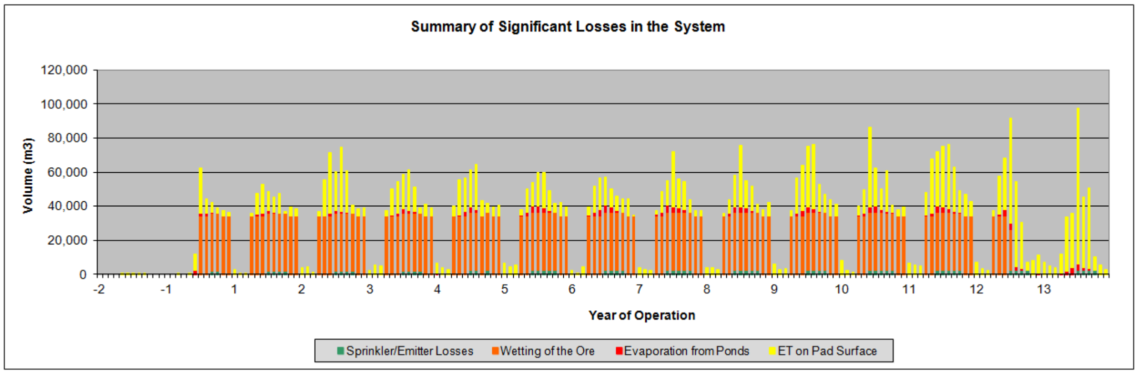


Figure 5 – 47.3 MT Scenario

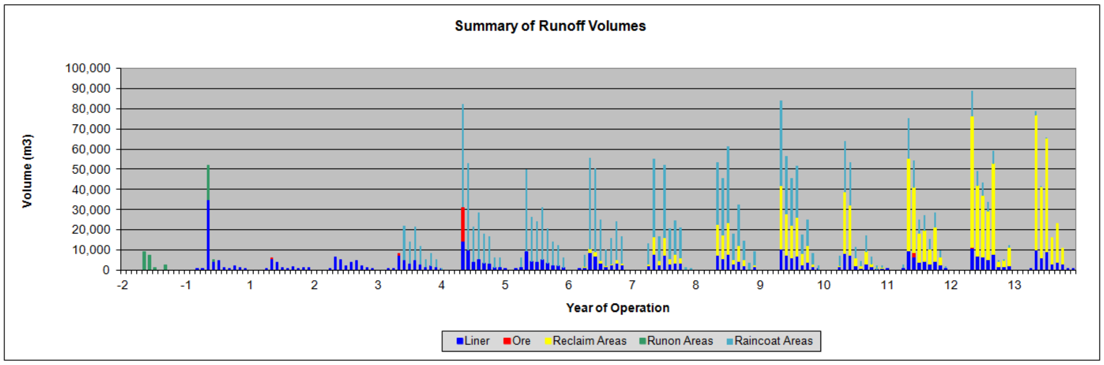


Figure 6 – 60.4 MT Scenario

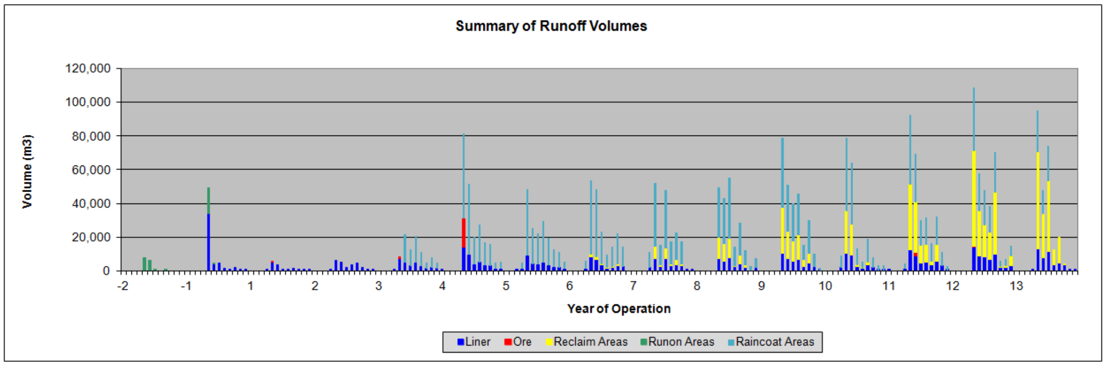


Figure 7 – 47.3 MT Scenario

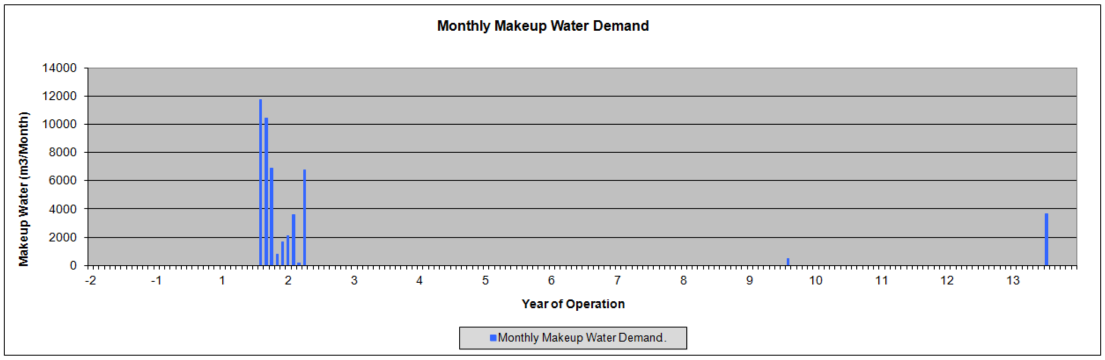
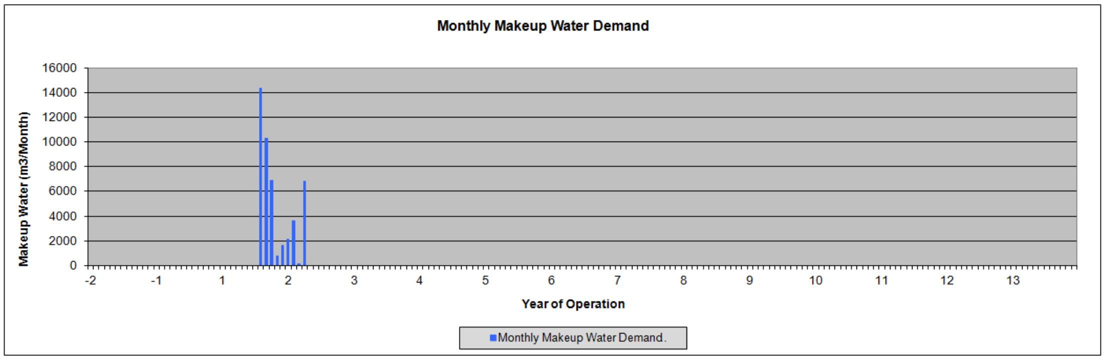


Figure 8 – 60.4 MT Scenario



Given the increase in lined footprint for the 60.4 MT scenario, the HLF system does accumulate more meteoric water. However, continuation of the pumping to treatment during operations at a rate of 4 liters/s prevents excessive buildup of solution within the pond system. The maximum volume of seasonally accumulated water in the pond system is essentially the same at 194,091 m^3 for the 47.3 MT scenario and 212,209 m^3 for the 60.4 MT scenario (see Figure 9 and Figure 10). However, the increase in lined footprint with no increase in pumping to treatment rate results in a change in the peak seasonal pond volume accumulation (after initiation of treatment) from +/- 100,000 m^3 for the 47.3 MT scenario to +/- 150,000 m^3 for the 60.4 MT scenario.

Figure 9 – 47.3 MT Scenario

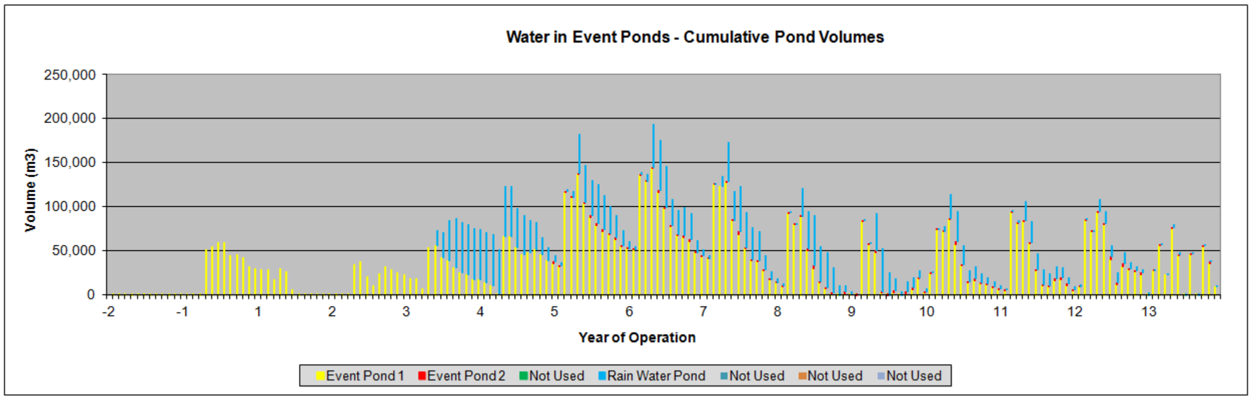
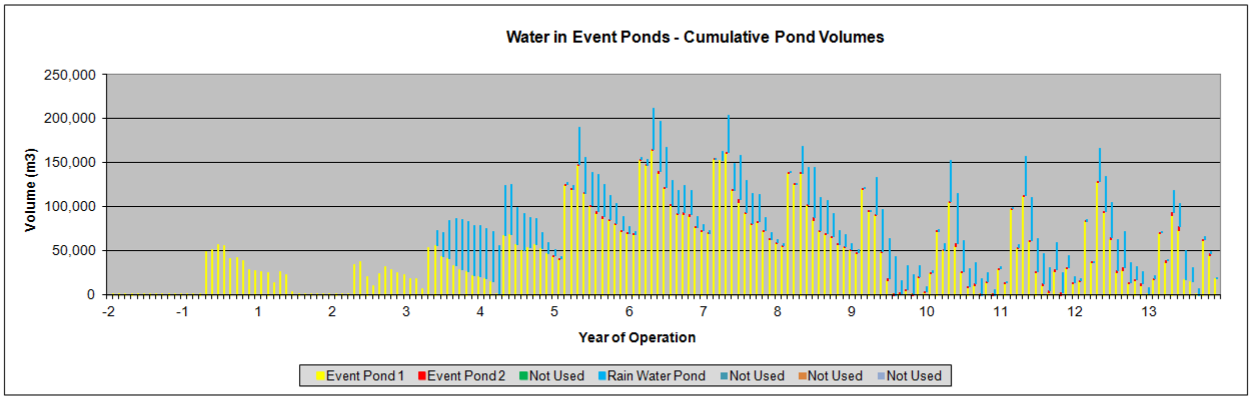


Figure 10 – 60.4 MT Scenario



During the final month of ore stacking for the 47.3 MT scenario (November of year 9), the total volume of water stored within the ore stack prior to draindown is 3,225,248 m^3. During the final month of ore stacking for the 60.4 MT scenario (July of year 12), the total volume of water stored within the ore stack prior to draindown is 4,070,765 m^3.

Please call if you need additional information or have any questions.

Regards,
The MINES Group, Inc.

Signature REDACTED

Kenneth L. Myers