REPORT

Nalunaq Gold Mine

Hydrological and Hydrogeological Study

Submitted to:

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Distribution List

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1.0 INTRODUCTION

1.1 Context

Nalunaq A/S ("the Company") has engaged Golder Associates (UK) Ltd ("Golder") to provide technical support for at its Nalunaq Gold Mine ("the Project") in southern Greenland. This report comprises the principal deliverable related to the Project's water resources, flood risk and water management aspects of the work undertaken by Golder.

In this report are presented the current conceptual understanding of the surface water and groundwater regime in the vicinity of the Nalunaq Mine; an assessment of flood risk; and a proposed water management plan and associated infrastructure, including a high-level water balance for the development. The approximate location of the mine is shown in Figure 1.

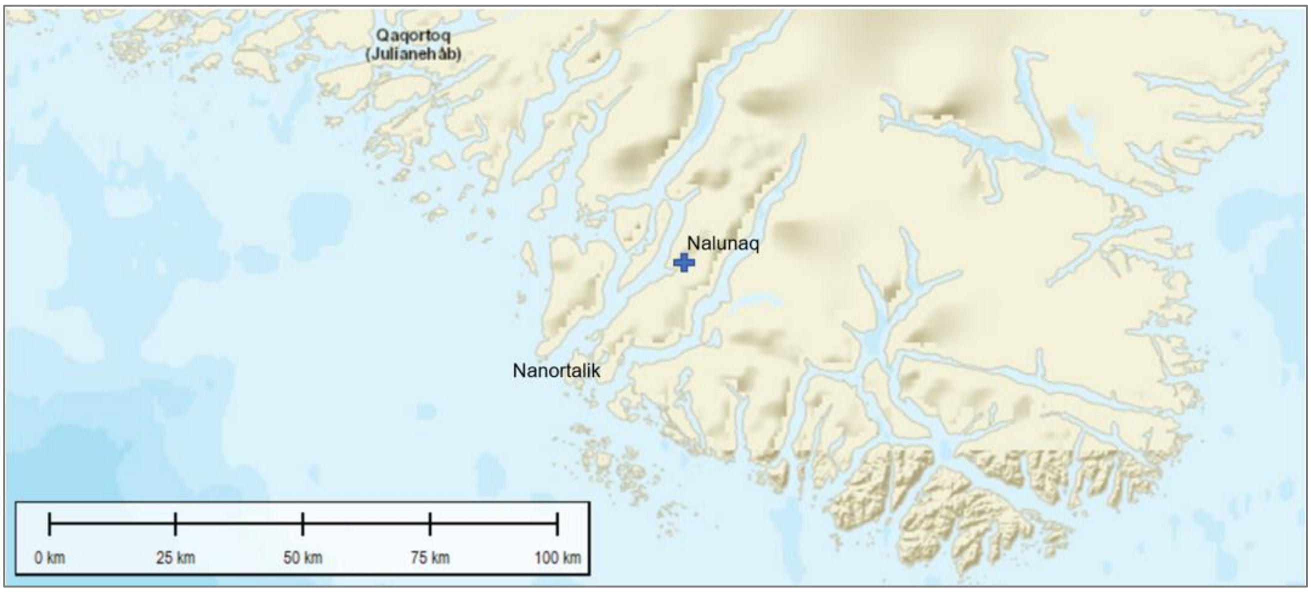

Figure 1: Approximate location of Nalunaq Mine, Greenland

1.2 Background

Following discovery in the early 1990s and development and operation by Crew Gold Corporation ("Crew Gold"), development was continued by Angus & Ross plc and Angel Mining (Gold) A/S, between 2004 and 2013. Subsequently additional exploration work has been undertaken in the Project area. It is understood that Nalunaq A/S, who is a wholly owned subsidiary of AEX Gold Inc., is aiming to restart mining operations in 2021.

Golder was engaged by the previous owners to provide support on the project with regards to tailings disposal, geotechnical engineering, underground rock mechanics and water management between 2002 and 2009. The key reports prepared by Golder at the time are as follows:

- Review of Surface Tailings Options – 2002 – Kvaerner Engineering & Construction UK Ltd;
- Geotechnical Review – 2003 – McIntosh Engineering on behalf of Crew Developments;
- Waste Management and Mineral Processing – 2009 - Angus Ross PLC;
- Geotechnical Assessment of Proposed Mineral Processing Chamber – 2009 - Angus Ross PLC;
- Geotechnical Assessment of Proposed Mineral Processing – 2009 – Angel Mining (Gold) A/S;
- Conceptual plug design for the Nalunaq Mine. Draft Technical Memorandum dated 10 July 2009; and

- Site Visit – 2009 - Angel Mining (Gold) A/S.

2.0 SITE CONCEPTUAL MODEL

2.1 Environmental Setting

The Project is situated in a mountainous periglacial area in southern Greenland on the northern side of the Kirkespirdalen (Kirkespir Valley) approximately 35 kilometres (km) to the northeast of the town of Nanortalik in the Municipality of Kujalleq (60º21’N 44º50’W) (Figure 1). Kirkespirdalen in which the mine is situated is typical of a glacially eroded valley with steep sides into which feed a number of previously glaciated cirques. A lake is situated in the upper reaches of the valley that is drained by the Kirkespir River to the Sarqå Fjord. To the south of where the Kirkespir River enters the fjord the proposed mine camp area is located on a raised beach. An unsurfaced road, approximately nine km long, connects a jetty with the camp area and onwards up the valley to the process plant, Dry Stack Tailings Storage Facility (DTSF) and mine. It is understood from the Company personnel that generally the fjord does not freeze during the winter and navigation by boat to the jetty is possible for most of the year. The approximate layout of the proposed mine as of November 2020 is shown in Figure 2.

Figure 2: Nalunaq Mine, Kirkespirdalen, Greenland

The valley may be broadly divided into two areas: a lower section below the Repeater Station (Figure 2) where the river descends approximately 70 metres (m) over a distance of approximately 500 m via a series of small rapids; and an upper section east of the Repeater Station where the proposed DTSF and process plant are situated in the braided channel of the Kirkespir River.

The topography in the area is mountainous with steep slopes (Figure 3) reaching from sea level to elevations of approximately 1,500 m above sea level (masl) that show the geomorphological influence of recent glaciation. The Kirkespirdalen is a typical glacial valley that is infilled with fluvioglacial deposits, with talus slopes above and with several hanging valleys and glacial cirques along its length from which a number of rock glaciers emerge (Figure 4). The deposit is situated at approximately 215 masl which also marks the approximate upper limit of vegetation cover (Kvaerner, 2002). To benchmark information provided prior to the proposals and to collect additional data, a site visit was conducted by Golder staff in October 2020.

Figure 3: View from the Repeater Station down valley towards the Sarqå Fjord

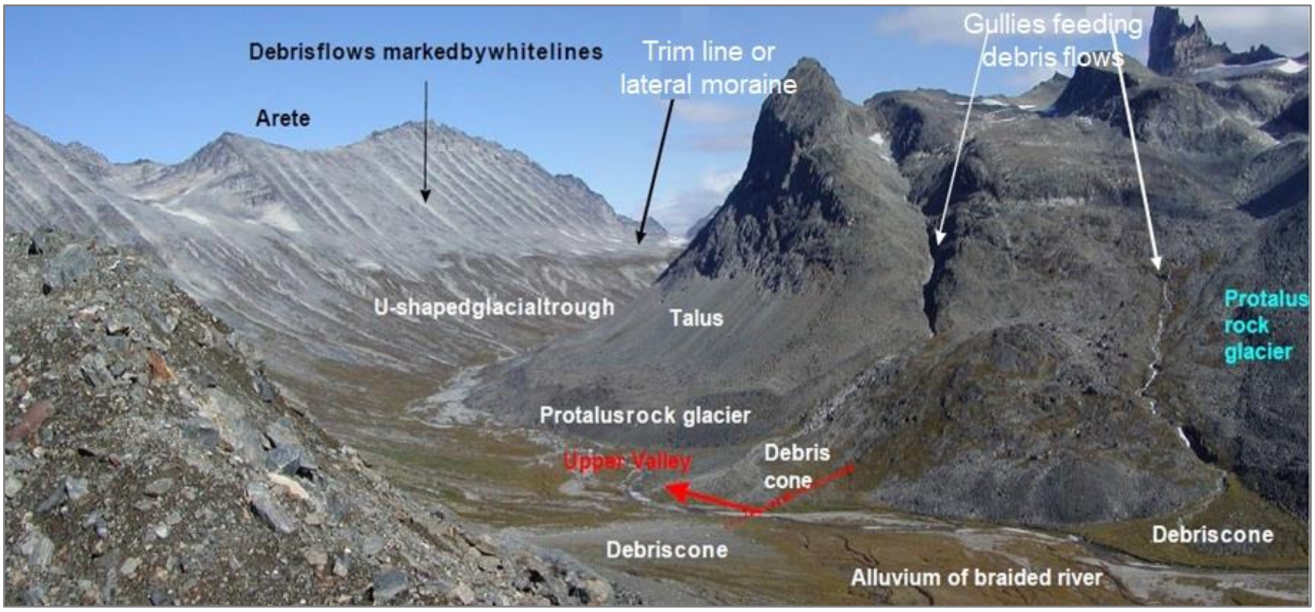

Figure 4: Geomorphological features of the upper Kirkespirdalen viewed from the 300 Level portal

2.2 Site Infrastructure and Layout

A visit to the Project site ('the site') was undertaken by Golder in October 2020 to benchmark the available information and undertake a surface geotechnical and underground rock mechanics investigation. The results of the visit are presented in Golder 2020a; the results of the surface geotechnical investigation are presented in

Golder 2021a; and the results of the underground rock mechanics investigation are presented in Golder 2020b and Golder 2020c.

The layout of current (2020) project infrastructure at the site is presented in Figure 2, with a current view from the 300 Level portal in

Figure 5. The project infrastructure currently comprises the following:

- **Jetty** – The jetty is situated to the west of the 2020 temporary camp and is used for access to and from the site by boat.
- **Roads** – A 9 km long gravel road connects the mine to the jetty for mine access. The road crosses a river by a bridge near the jetty and the Kirkespir River by a bridge constructed of reinforced containers.
- **Camp** – A temporary camp with accommodation for approximately 20 persons is situated approximately 1 km to the east of the jetty.
- **Mine** – Access to the mine is currently via the 300 Level portal (Figure 6) which also connects to the surface at the partially blocked 350, 400, 450 and 600 Level portals.

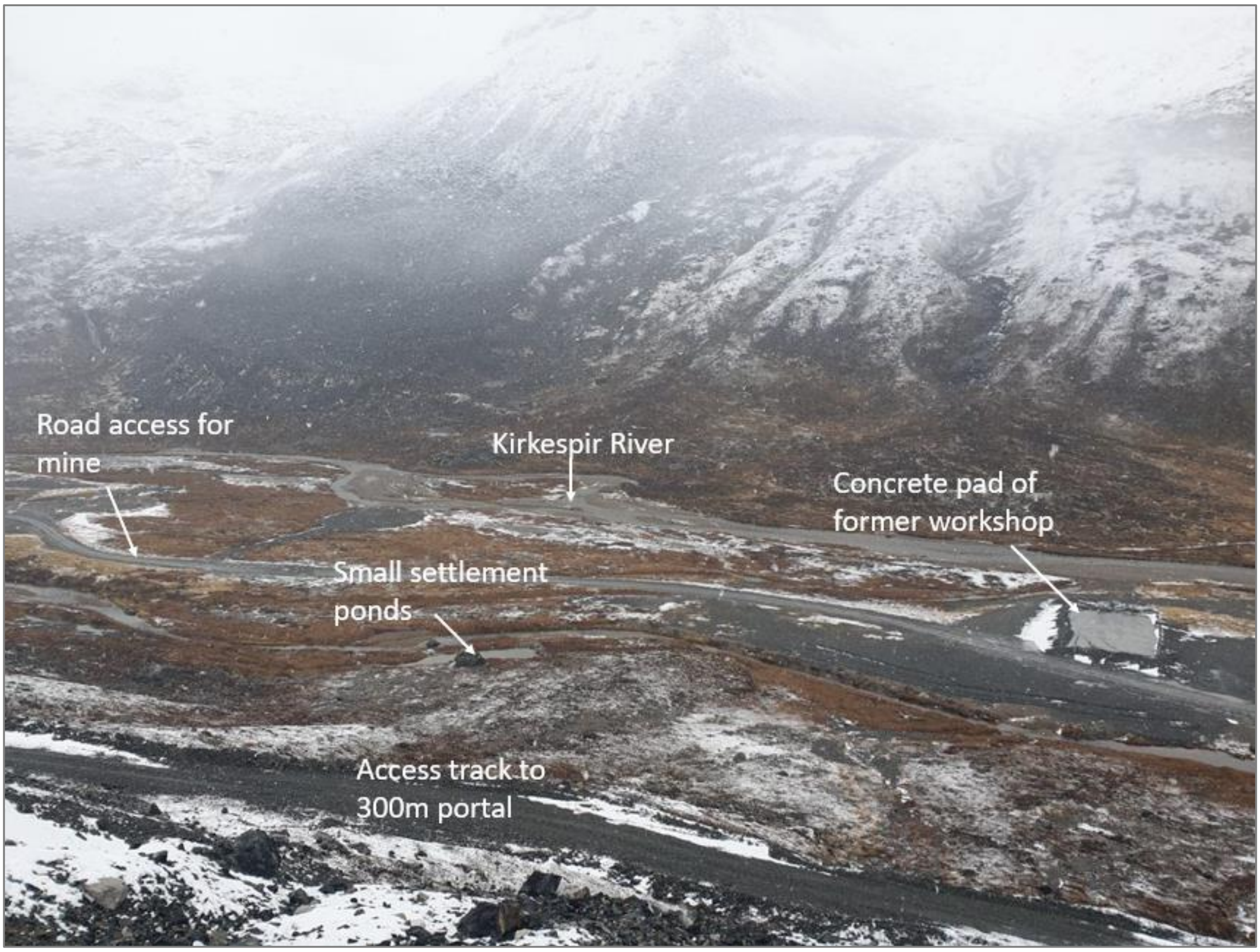

Figure 5: View from the 300 Level portal, looking towards the southeast

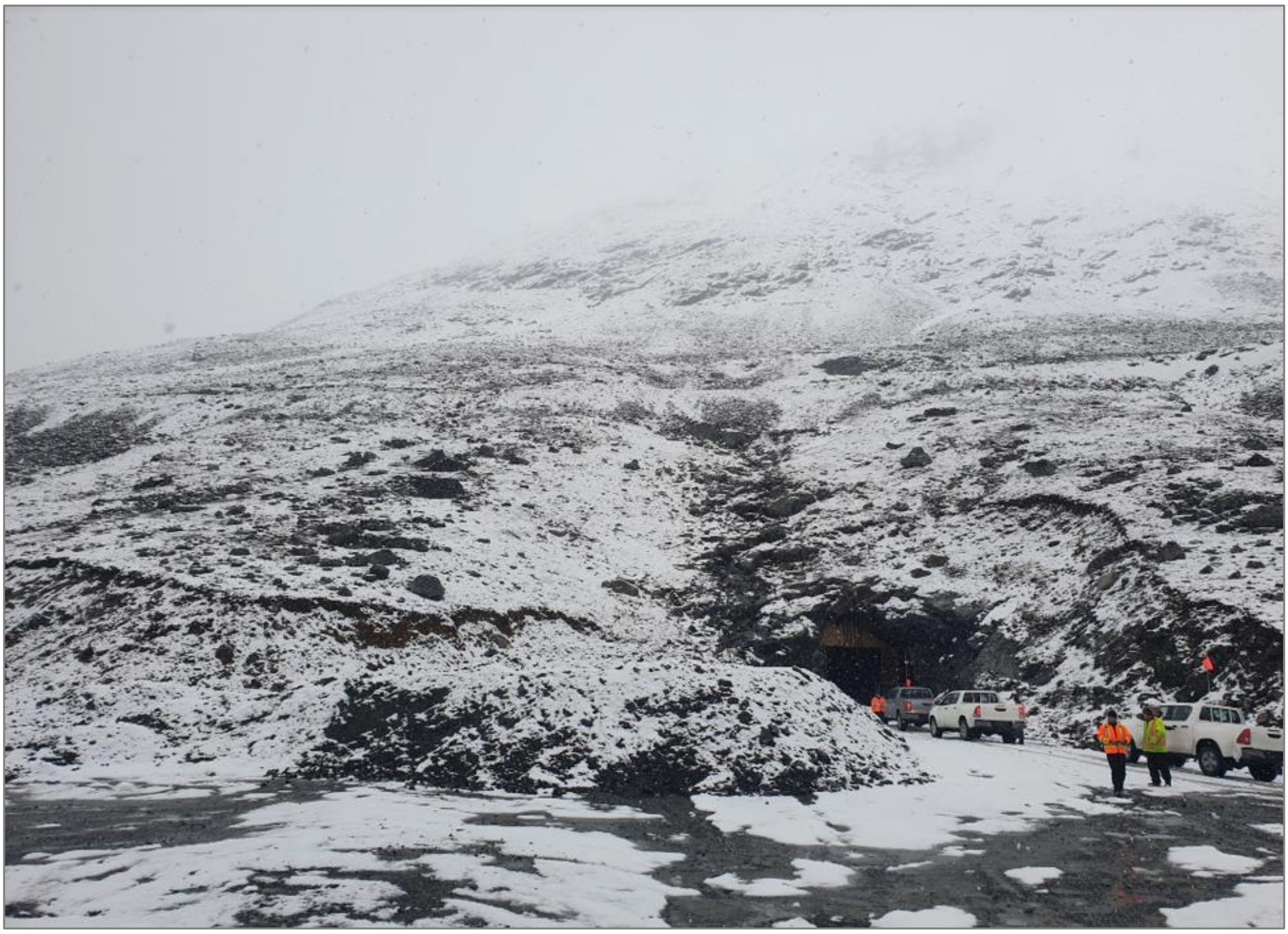

Figure 6: Entrance to the 300 Level portal

2.3 Meteorology and Climate

A summary of the key meteorological and climatological conditions for the site is presented in the following sections. Further analysis of the available meteorological data and climatological setting is presented in Golder 2021b (APPENDIX A).

2.3.1 Climate Setting

The site location has a tundra climate with strong oceanic and polar influences (SRK Consulting, 2002). Precipitation (including both rainfall and snowfall) is moderate with an annual average cumulative depth of approximately 602 millimetres (mm). Snow cover is relatively limited within southern Greenland, with an annual average cumulative snowpack depth of 194 mm although extremes have been observed (SRK Consulting, 2002). Temperatures show relatively little variation between seasons. July is the hottest month with a mean temperature of 10.7 degrees Celsius (°C) and February is the coldest month with a mean temperature of -7.9 °C.

2.3.2 Regional Climate Stations

There is no onsite meteorological station at the site, with only short climate datasets available during which local data capture (e.g. rainfall) has been carried out as part of specific site-based study. These datasets are considered too short a record to be sufficient for hydrological analysis. As such, daily precipitation and temperature data from two stations (Nanortalik Heliport and Narsarsuaq) were sourced from NOAA (2020) and Tutiempo (2020), respectively. The location of these stations relative to the site are shown in Figure 7 and Table 1. As less than five years of daily precipitation data was available for Nanortalik station this record was dismissed in favour of Narsarsuaq, which has a longer and more complete dataset (1973 to 2003). For consistency the same record was used for temperature.

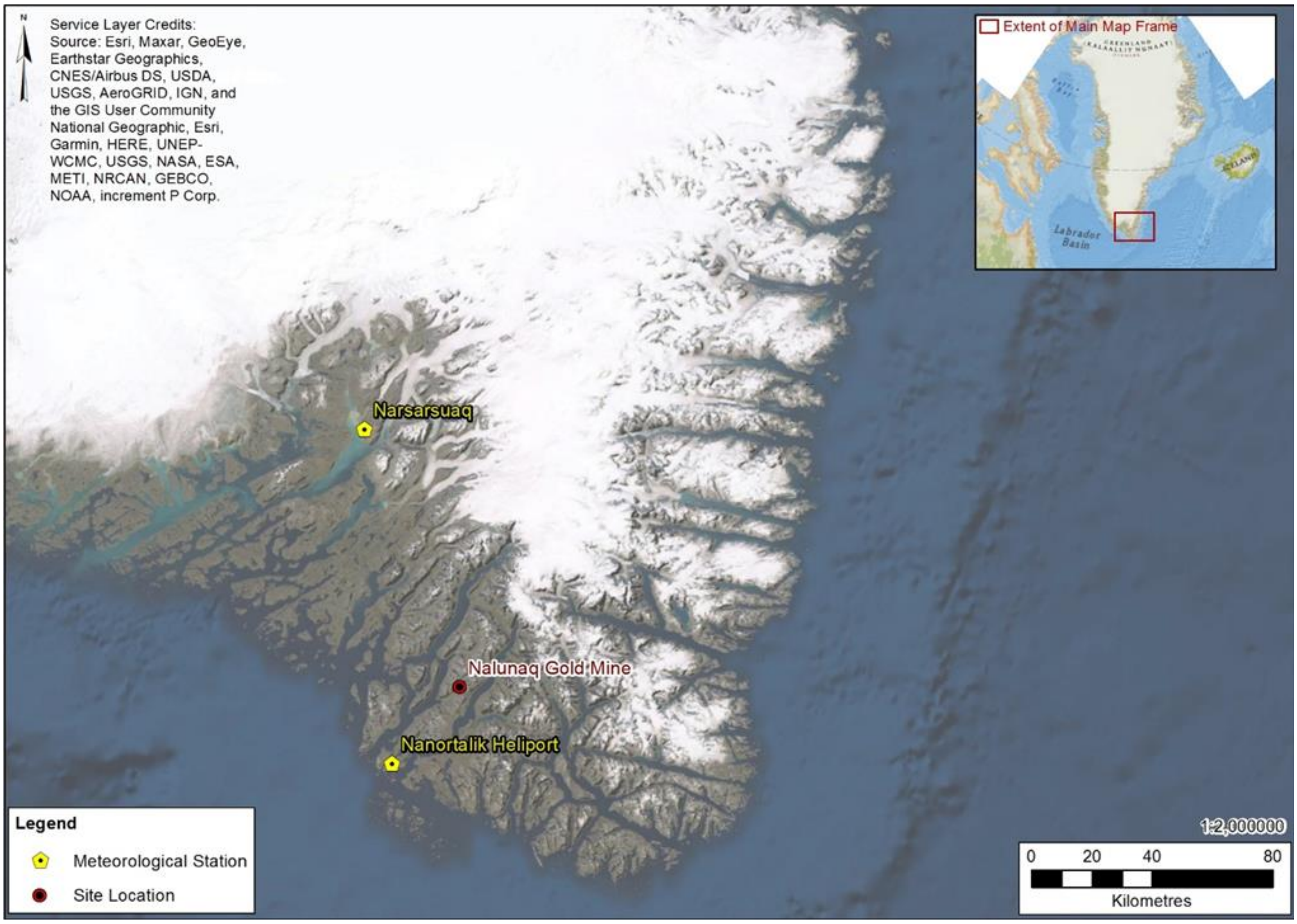

Figure 7: Location of Climate Stations Relative to the Site

Table 1: Climate Station Details

Station Name	Latitude Longitude	Distance from Mine (km)	Elevation (m)	Record	Data Type	Portion of Record Complete
Nanortalik Heliport	60.13°N -45.23°E	35 (SE)	5	01/01/1980 02/11/1985	Daily Precipitation	92.5%
				01/01/2014 10/07/2020	Hourly Average Air Temperature	89.7%
Narsarsuaq	61.13°N -45.41°E	91 (NNE)	34	01/01/1973 31/12/2003	Daily Precipitation	98.8%
					Daily Average Temperature	99.5%

2.3.3 Precipitation

Total precipitation depths (i.e. including both rainfall and snowmelt) were available for the Narsarsuaq station. To estimate rainfall and snowfall values, potential snowfall depths were derived using the degree-day method (Maidment, 1993). For the purpose of the calculations a, base daily average air temperature of 0 °C was assumed between April and October, while a base daily average air temperature of 2.5 °C was assumed between November and March. Any daily recorded precipitation which occurred on days with recorded daily air temperatures that exceeded the "base" temperature was assumed to report to the site as rainfall[1]. The assignment of these base temperatures reflects lower air temperatures required to trigger snowmelt between April and October, as opposed to other times of the year. This is due in part to energy available from the sun, as well as other factors, such as warmer rainfall and higher ground temperatures. A melt factor of 0.9 millimetres per degree Celsius per day (mm/°C/d) was also applied, which accounts for the accelerating effect of rainfall on the melting of the snowpack (and hence rate of snowmelt).

Annual total precipitation averaged 601.8 mm of which it was estimated approximately 68% was to occur as rainfall and the remainder as snowfall. The wettest month was September, with an average monthly total precipitation depth of 73.8 mm, and the driest month was March, with an average monthly total precipitation depth of 35.6 mm. Measurable snowfall occurred from October to April, with rainfall occurring predominantly in the summer months.

Precipitation, rainfall and snowfall depths for Narsarsuaq are provided in Table 2 and Figure 8.

Table 2: Average Monthly Precipitation at Narsarsuaq Station (1973 to 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)	44.0	37.7	35.6	45.6	35.8	57.4	58.2	64.6	73.8	57.6	47.6	43.9	601.8
Rainfall (mm)	3.2	7.5	2.4	33.5	35.0	57.4	58.2	64.6	73.1	50.4	16.2	6.4	407.8
Snowfall (mm) (1)	40.7	30.3	33.3	12.2	0.8	0.0	0.0	0.0	0.6	7.2	31.4	37.5	194.0

NOTES: (1) As water equivalent.

[1] Note a higher base temperature is used in winter to reflect the fact that melt does not start immediately once temperatures exceed 0 °C, as it is also influenced by relative humidity and the short-wave vs long-wave radiation balance.

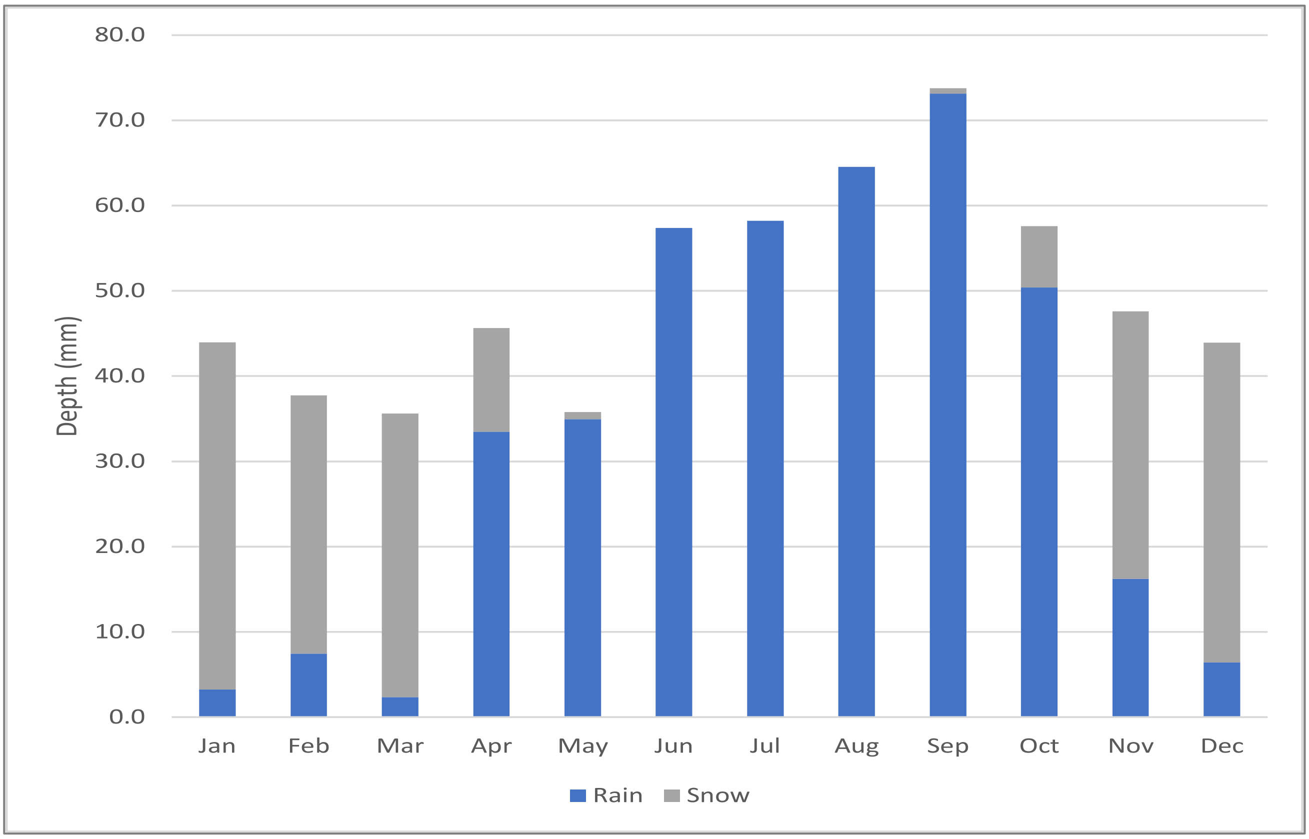

Figure 8: Average Monthly Rainfall and Snowfall at Narsarsuaq Station (1973 – 2003)

2.3.4 Temperature

Average temperature data recorded at the Narsarsuaq Station between 1973 and 2003 are presented in Table 3, including the mean (average) minimum, mean maximum and mean daily temperatures for the 30-year period of record. The mean annual temperature during this period was 0.9 ˚C. Temperatures were highest from April to October, and lowest from November to March (mean temperatures did not exceed 0 °C). July was the hottest month with a mean maximum temperature of 20.3 °C. February was the coldest month, with a mean minimum temperature of -24.0 °C. The highest temperature recorded in the 30-year record was 25.0 °C (02/04/1998) and the lowest was -39.8 °C (23/01/1984).

Table 3: Average Temperature at Narsarsuaq Station (1973 to 2003)

Parameter	Temperature (°C)												
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Mean (Average) Maximum Daily Temperature	8.4	7.0	8.6	12.3	16.2	19.2	20.3	19.2	16.6	12.8	11.4	9.1	13.4
Mean (Average) Daily Temperature	-7.5	-7.8	-6.0	0.3	5.4	8.9	10.7	9.4	5.8	0.8	-3.5	-6.0	0.9

Parameter	Temperature (°C)												
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Mean (Average) Minimum Daily Temperature	-23.3	-24.0	-21.1	-13.1	-4.3	1.3	3.4	2.3	-3.1	-9.8	-17.9	-20.7	-10.9

2.3.5 Evaporation

As set out in Golder 2020e, potential evapotranspiration (PET) at the Narsarsuaq station between 1973 and 2003 was calculated from the temperature dataset using the Thornthwaite method (Thornthwaite, 1948). The average monthly and annual PET is presented in Table 4. The average annual evapotranspiration over the 30-year period of record was 465.2 mm. The calculated potential evapotranspiration rates were highest from June to August (over 95 mm of evaporation occurred in each month) and were lowest from November to March, with little to no evaporation in these months.

Table 4: Average Potential Evapotranspiration at Narsarsuaq Station (1973 to 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Potential Evapotranspiration (PET) (mm)	0.1	0.5	0.0	14.9	64.4	100.6	118.0	96.3	56.3	12.2	1.5	0.4	465.2

2.4 Geology

2.4.1 Regional Geology

The mine is situated in the basement rocks of southern Greenland. According to Dominy *et al.* (2006) Nalunaq is situated within the Ketilidian Mobile Belt, which is related to the accretion of a Palaeoproterozoic continental margin against the Archaean Core of southern Greenland. Dominy *et al.* (2006) report that the site lies in the Psammite Zone, a supracrustal succession of psammites with pelites and interstratified mafic volcanic rocks. Gold mineralisation at Nalunaq is hosted by a meta-volcanic unit composed of basaltic pillow lavas and pyroclastics intruded by dolerite sills. The volcanic rocks are reported (Dominy *et al.*, 2006) to be metamorphosed to amphibolites and the area is intruded by late- and post-tectonic granitoid plutons. It is also reported by Dominy *et al.* (2006) that at Nalunaq granitoid rocks surround three sides of the meta-volcanic mass hosting the vein mineralisation. A regional geological map is presented at Figure 9.

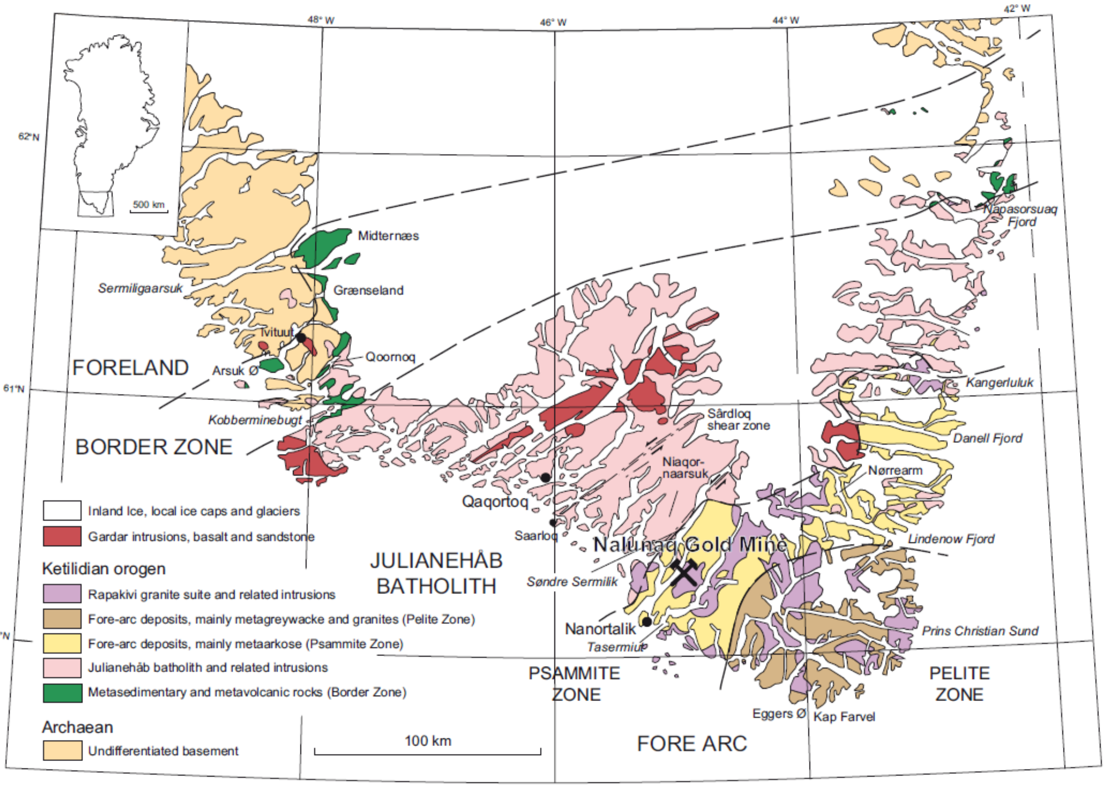

Figure 9: Geological map of southern Greenland with the location of the Nalunaq Mine (from Secher *et al*., 2008)

2.4.2 Local Geology

On the Nanortalik peninsula metabasic rocks have been found in Ippatit, Nalunaq and Lake- 410 (Figure 10). These three areas have been interpreted, by Petersen *et al.* (1997) as separate parts of the Nanortalik Nappe where tholeiitic basalt flows and doleritic sills have been thrust over metasediments and intruded by later granites and several generations of late aplite and pegmatite dykes. The local geology consists mainly of fine-grained amphibolites and coarse-grained dolerite (Figure 11). The stratigraphy has been assigned into the structural footwall ("FW") and structural hanging wall ("HW") with respect to the main mineralised vein (Nalunaq Main Vein, "MV"). Between the granite of the deep footwall and the amphibolite and dolerite of the shallow footwall, silicified and pyrite-impregnated siltstones with intercalations of graphitic beds and altered fine-grained siltstones are present. The gold mineralised quartz vein is located at or close to the contact of fine-grained amphibolite and coarse-grained dolerite. A geological map of the area in the vicinity of the mine is presented at Figure 12.

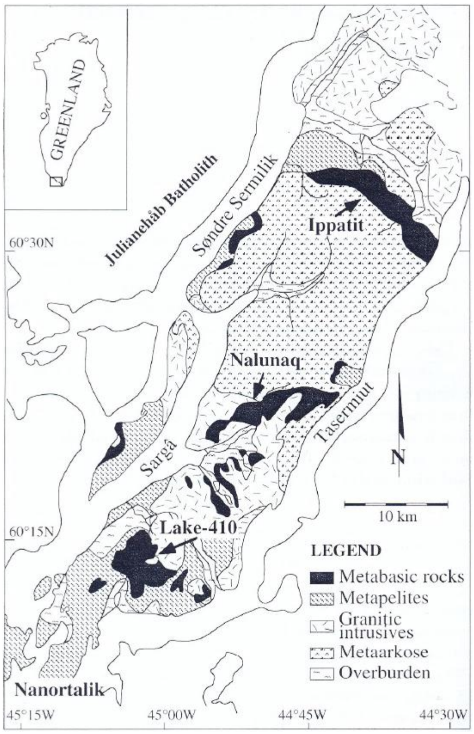

Figure 10: Geological map of Nanortalik peninsula. (from Petersen *et al*., 1997)

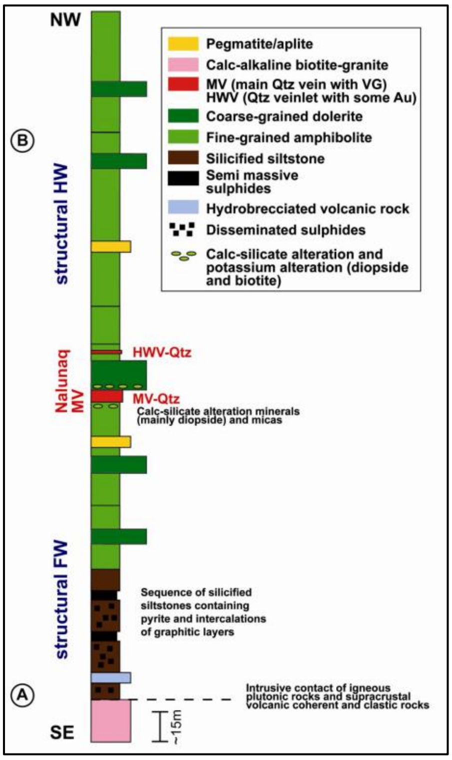

Figure 11: Simplified stratigraphic column of Nalunaq (Schlatter and Olsen, 2011)

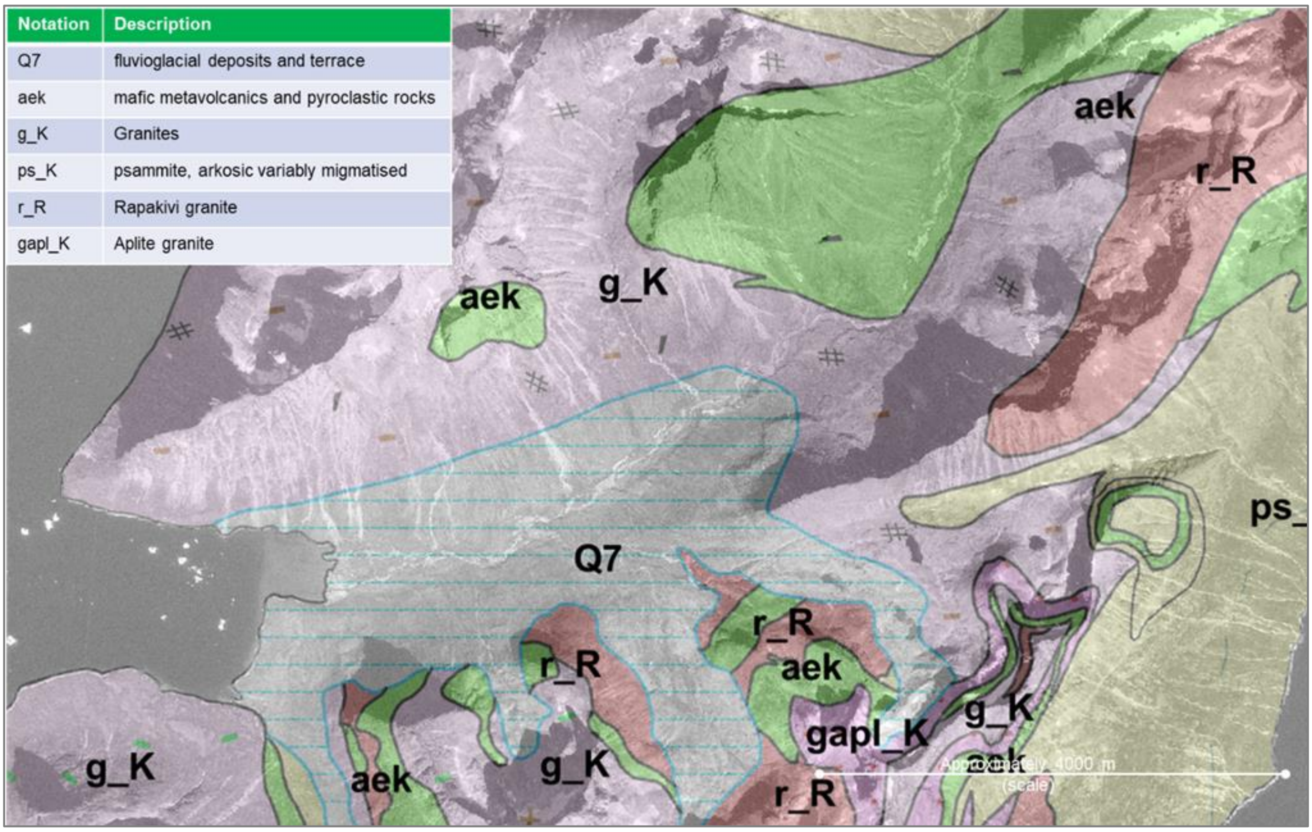

Figure 12: Geological map of Kirkespirdalen and the area in the vicinity of the Nalunaq Mine (GEUS, 2019)

2.4.3 Nalunaq Deposit

Nalunaq is a high-grade narrow vein gold deposit hosted in a package of metabasic rocks including metadolerites and fine grained amphibolites (Kvaerner, 2002). The Nalunaq Main Vein is exposed on two faces of Nalunaq Mountain (Figure 13). The vein is subparallel to the foliation and to the regional thrust/ shear planes, occurring about 100 m above the thrust-base (Petersen *et al*., 1997). On a local scale, the vein occurs along the contact between a medium grained metadolerite and fine-grained amphibolite in the footwall. The ore horizon is a calc-silicate zone with a discontinuous central filling of sheeted quartz veins often made up of slightly off-set flat quartz lenses which onlap laterally to yield swelling ore shoots connected to others by quartz-calc-silicate seams. Intensive calc-silicate altered amphibolites occur in discrete bands elsewhere in the series, particularly below the Main Vein, and may represent internal shear zones with enhanced fluid flow (Petersen *et al*., 1997).

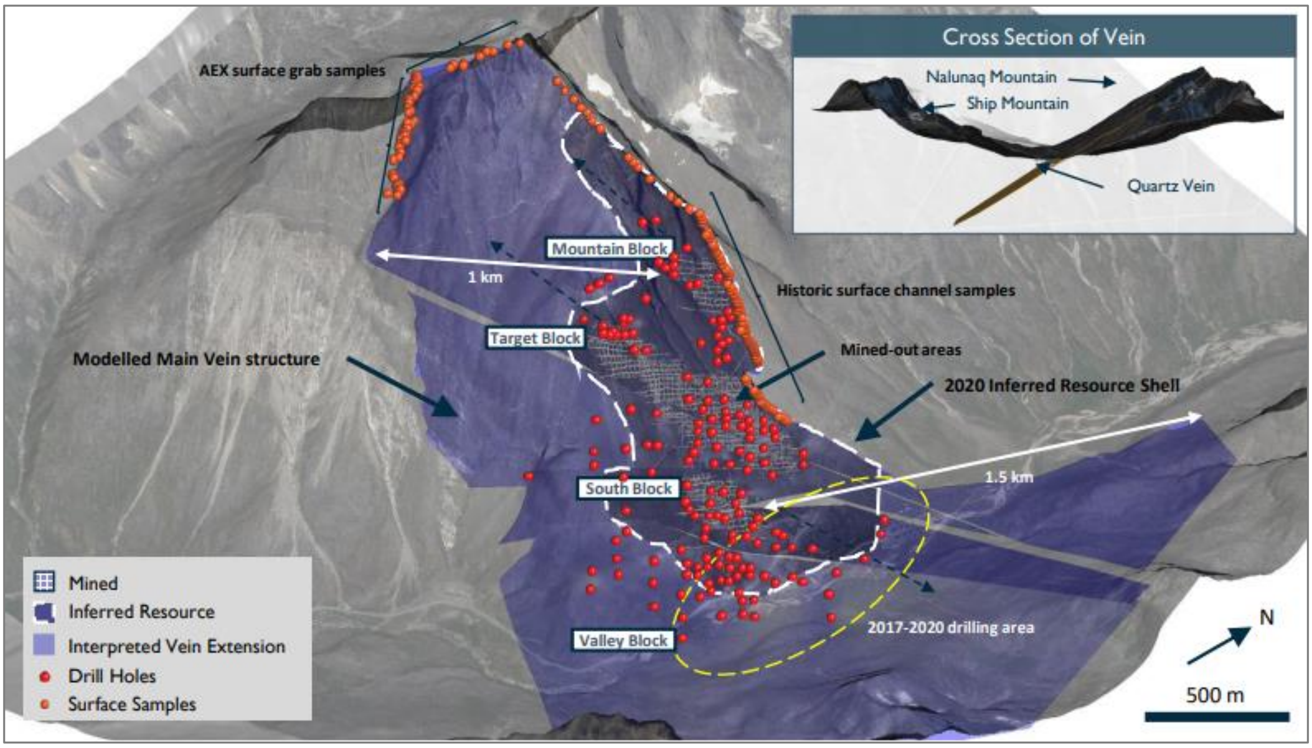

Figure 13: Nalunaq Mountain from the southeast (AEX, 2020)

2.4.4 Structural Blocks

The Nalunaq deposit is divided into four main structural blocks based on their post-mineralisation faulting. From southeast to northwest these are Valley Block, South Block, Target Block and Mountain Block (Figure 13). South Block and Target Block are separated by the Pegmatite Fault between, which displays normal fault movement causing approximately 80 m of vertical offset of South Block relative to Target Block, and dextral displacement of approximately 85 m (SRK, 2016). The main orebody lies on the downthrown side of the Pegmatite Fault (Golder, 2020c).

Two further faults crosscut the orebody, the shallow dipping Your Fault and the more steeply dipping Clay Fault. Both faults occurred post-mineralisation and typically show less than 5 m of displacement (Golder, 2020c). The immediate zone around the Clay Fault is described (Golder, 2020c) as being highly disturbed whilst the ground leading up to it and beyond does not appear to be any more heavily fractured than surrounding areas.

2.5 Hydrology

The Kirkespirdalen consists of a main valley surrounded by steep mountains up to 1,575 masl. The Kirkespir River flows 14 km along the valley from a lake in the upper reaches (0.3 km^2) at 747 masl to the Sarqå Fjord. Approximately twenty tributaries from a few smaller side valleys feed water to the main river along its course, creating a total catchment area of 67.9 km^2 (Figure 14).

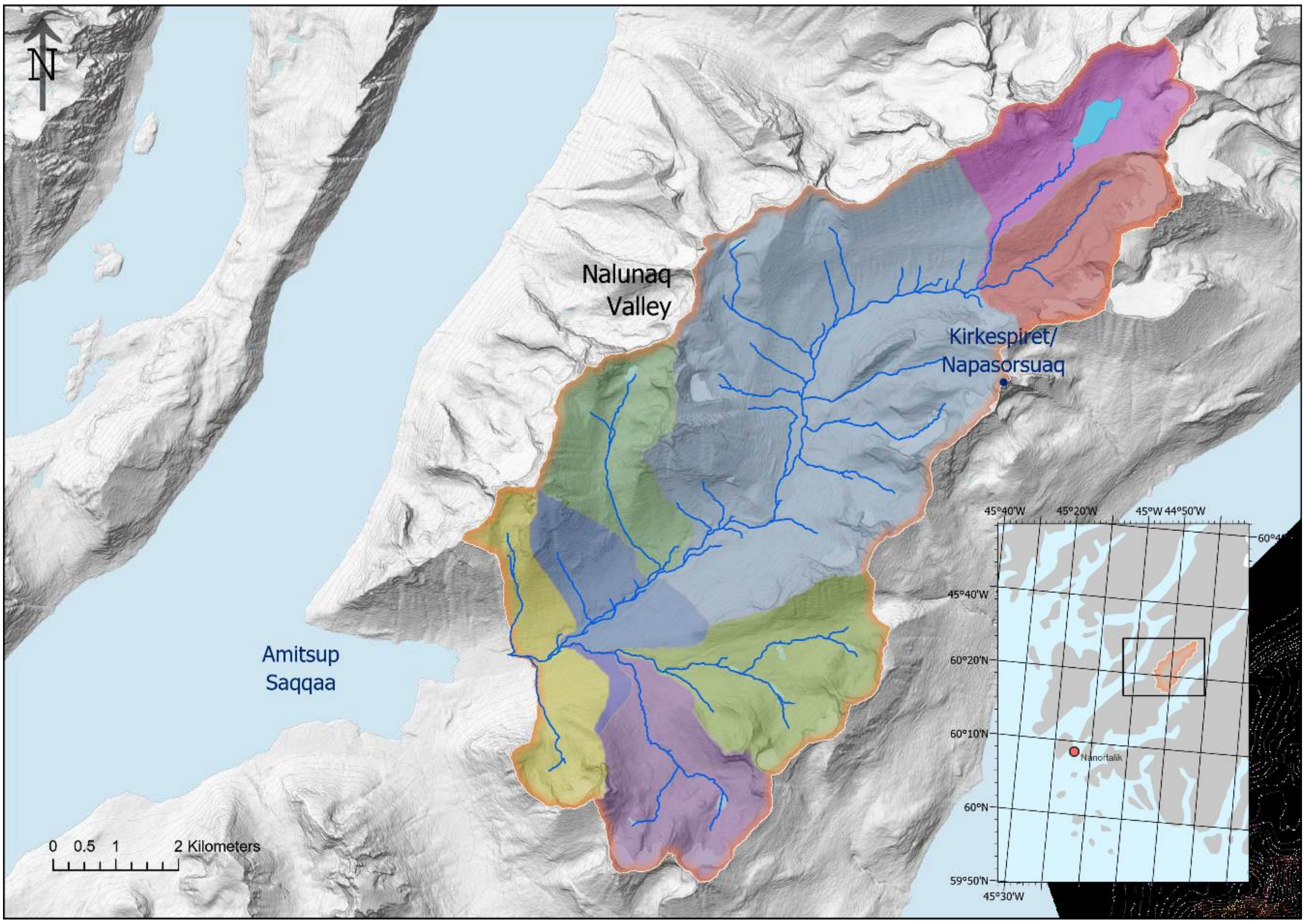

Figure 14: Nalunaq Valley Watershed. Catchment defined with orange border and stream network with blue lines together with sub-basins (areas in different colour) (from Asiaq, 2019).

Asiaq Greenland Survey conducted a desktop water resources study on behalf of Nalunaq A/S in 2019 (Asiaq, 2019). Due to the lack of direct measurements of the water resource in Nalunaq Valley an average of four estimates of the specific mean annual water resource were calculated by Asiaq (2019) via estimates from secondary sources. Two of the estimates were obtained from the modelled water resource (runoff) in Kirkespirdalen from a regional climate model HIRHAM (a regional climate model run by the Arctic and Climate Research section at the Danish Meteorological Institute) and two further estimates were obtained using data from a site located approximately 15 km to the southwest of Nalunaq valley (Asiaq, 2019). By combining the specific mean annual water resource estimates with the upstream area runoff Asiaq (2019) that the mean annual flows along the Kirkespir River increased from approximately 10,000,000 m^3/year upstream of the mine to 46,000,000 m^3/year at the mouth of the river (Figure 15).

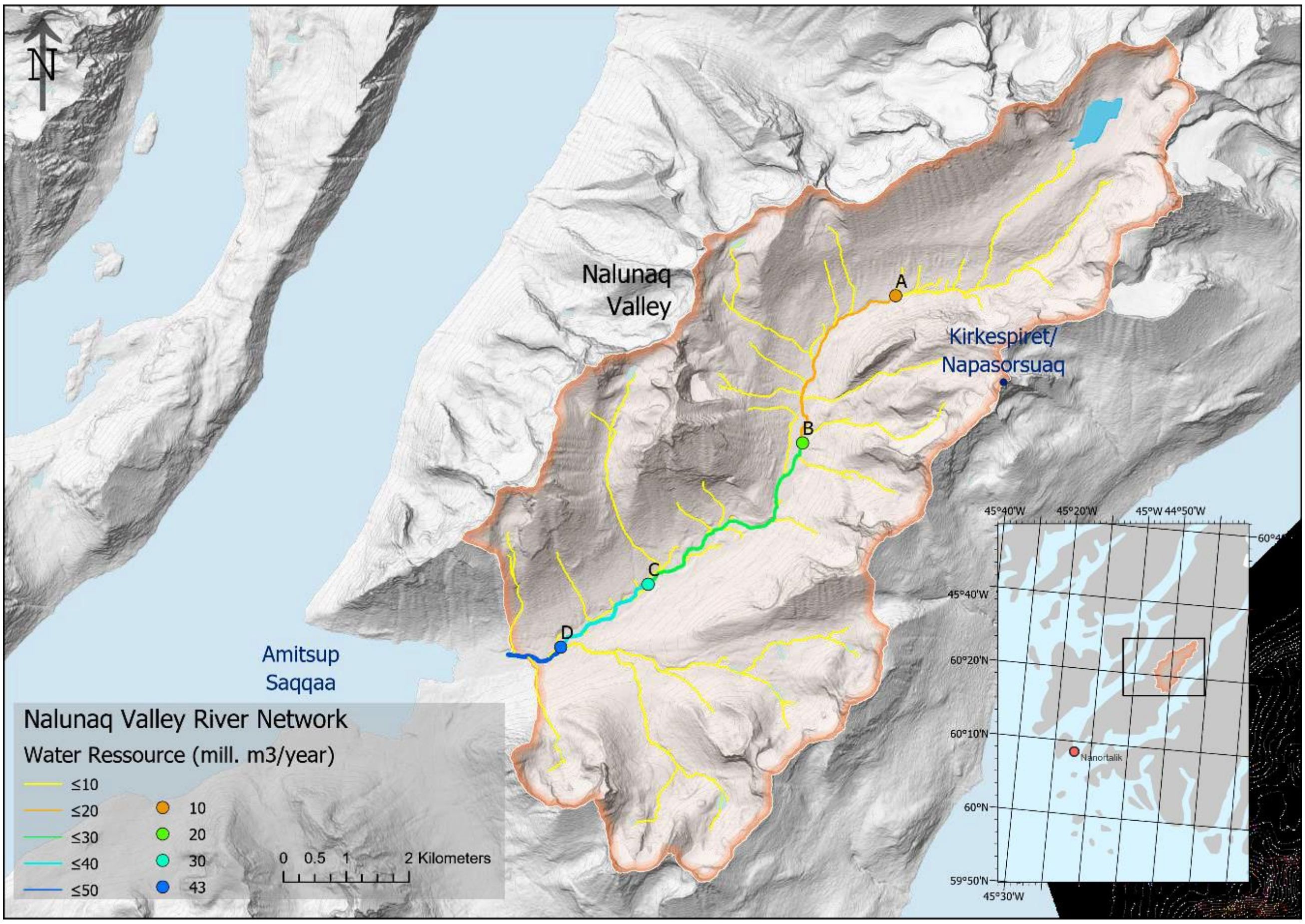

Figure 15: Mean annual water resource along the river network in Nalunaq Valley. Water resource data is extracted for four example points A, B, C and D along the river (from Asiaq, 2019).

Considerable year-to-year variation in river flows is indicated depending on the natural variation in the weather (Asiaq, 2019). Asiaq (2019) considered that the annual flows are likely to vary between 50% of the mean annual value (dry years) and 150% of the mean annual value (wet years).

2.6 Hydrogeology

2.6.1 Site Observations

A site visit conducted by Golder staff in October 2020 (Golder, 2020a) confirmed that the site may be divided into the following four hydrogeological domains:

- Fractured bedrock;
- Talus and debris flow deposits;
- Fluvio-glacial deposits; and,
- Raised beach deposits.

During the site visit a number of trial pits (TP) were excavated in the vicinity of the proposed DTSF, process plant and proposed camp location (Golder, 2020a). The position of the trial pits is presented in Figure 16 and Figure 17. As set out in Golder 2020a, vertical slotted pipes were installed in five trial pits (TP01, TP03, TP04, TP05 and TP06) to facilitate ongoing groundwater level monitoring. It is noted that due to the large slot size it may be necessary to bail silt and sand from the monitoring facilities prior to future groundwater level monitoring.

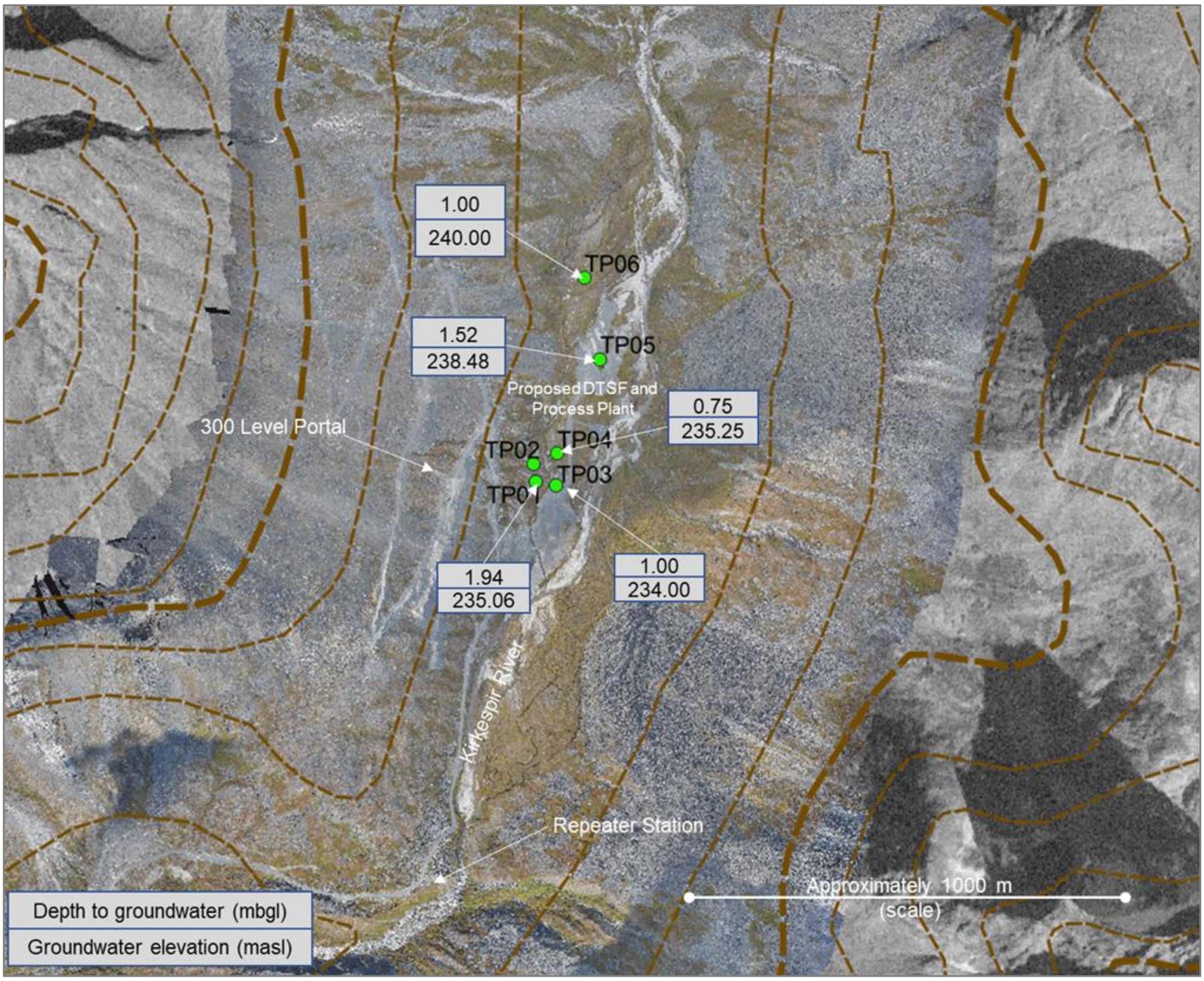

Figure 16: Trial pit locations in the vicinity of the proposed DTSF and process plant (boxes show depth (m) to and elevation (masl) of groundwater on 5 October 2020.

Based on the results of the trial pitting, previous reporting (Golder, 2002) and visual observations it is considered that the DTSF and process plant area are underlain by fluvio-glacial deposits, comprising sand and gravel with some silt (Figure 18) interbedded at the valley sides with talus and debris flow deposits (Figure 19). Valley slopes are covered with talus from exposed rocks above and talus-derived rock glaciers are present on the opposite valley slope to the east of the mine portal (Figure 20). Gullies associated with debris flows extend out onto the valley floor.

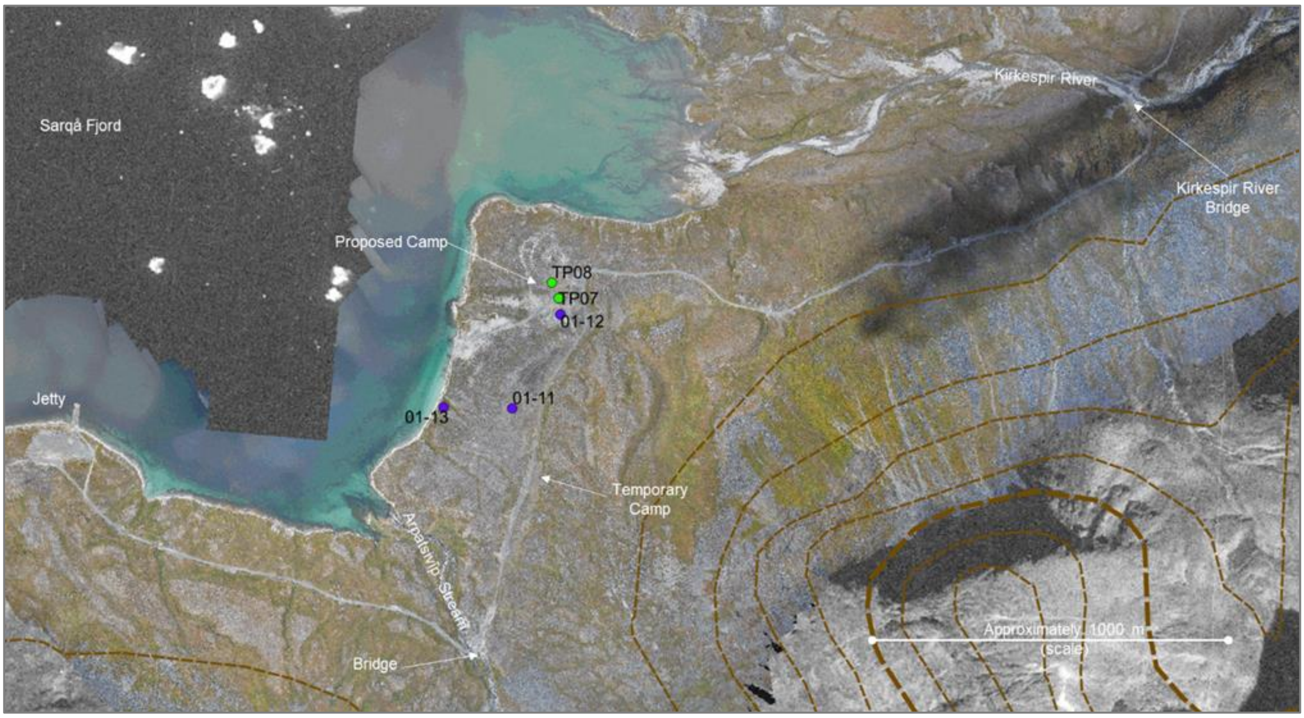

Figure 17: Approximate location of trial pits 7 and 8 in the vicinity of the proposed camp

Figure 18: Trial pits TP-01 to TP-04 in excavated in fluvioglacial deposits in the vicinity of the proposed DTSF and process plant

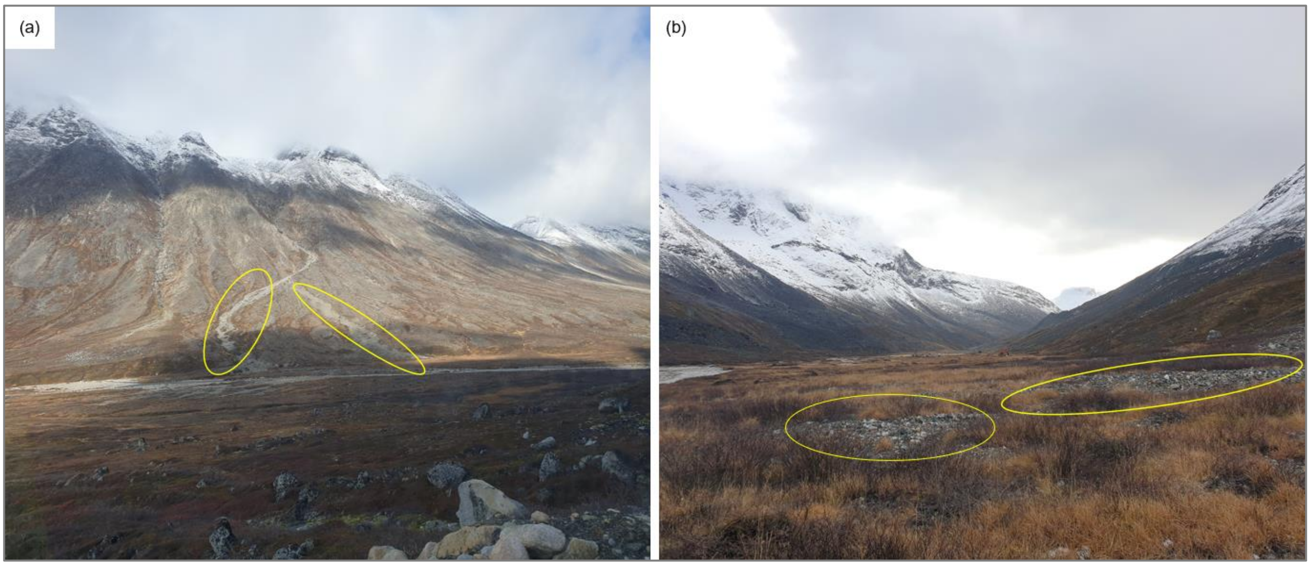

Figure 19: Debris flow deposits

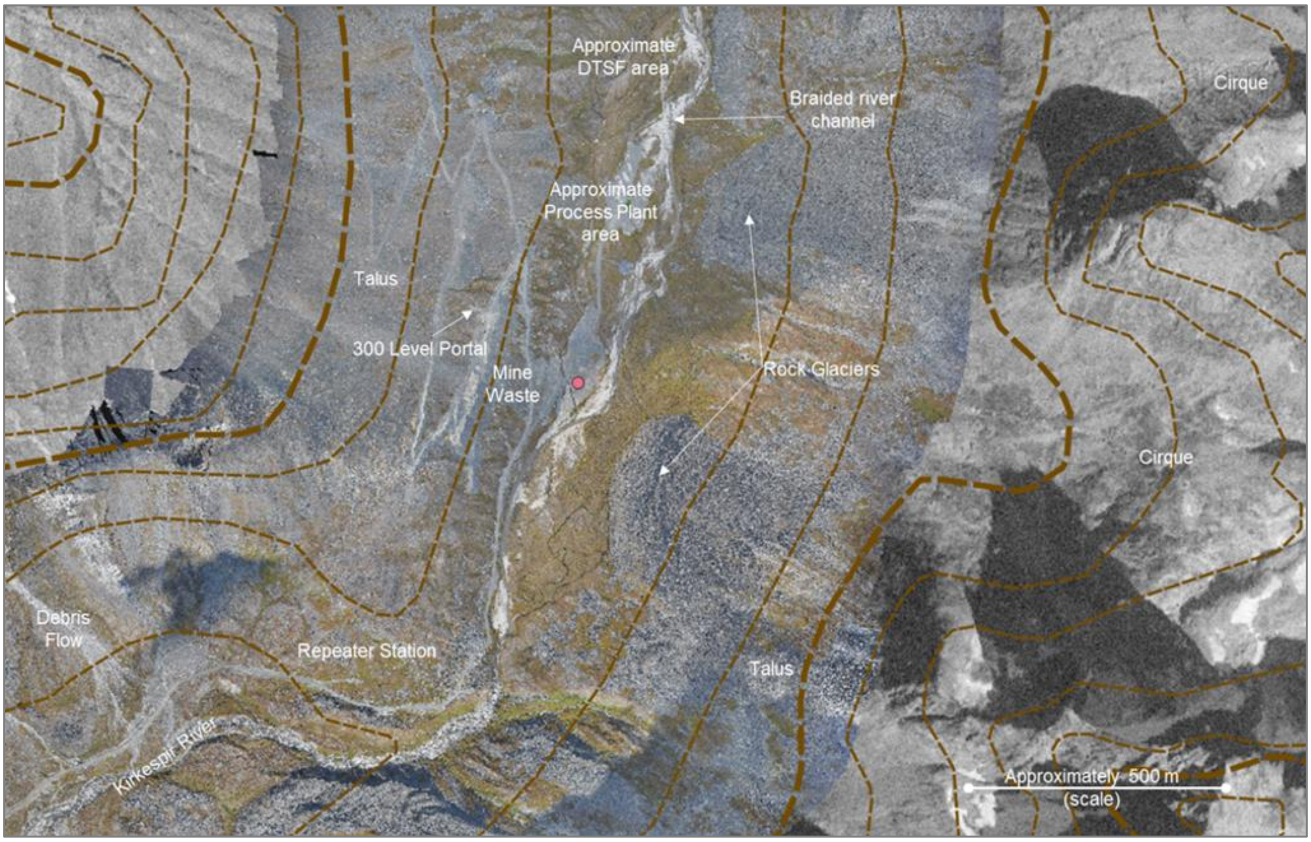

Figure 20: Principal geomorphic features

During the site visit (Golder, 2020a) a walk-through inspection of the underground working was undertaken to investigate groundwater inflows to the mine. This resulted in the following principal observations:

- Groundwater inflow was limited to discrete fractures;
- Higher elevation areas of the mine had less inflow than the lower areas at the time of the site visit;
- In South Block the mine was flooded up to the 270 Level Figure 21a);

- In Target Block the mine was flooded up to the 350 Level (Figure 21b) due to tailings storage behind a bulkhead on the 300 Level; and
- There was evidence that significant water flows occur at times based upon areas of scour and sediment deposition on drives (Figure 21c).

Based on the field observations it is considered that the bedrock forms a fractured groundwater bearing unit with some compartmentalisation resulting from the differing interconnectivity of the fracture systems and that recharge from rainfall and snowmelt enters the mine through the fractured bedrock and flows down through the mine via the drives and stopes, to a natural rest water level at approximately the 270 to 300 Level. It is considered likely that groundwater flows from the bedrock and discharges into Kirkespirdalen through the overlying talus and fluvioglacial sediments; ultimately discharging to the Kirkespir River.

Tailings are deposited behind a bulkhead on the 300 Level of Target Block and water ponded on top of the tailings, as is observed on the 350 Level of Target Block (Figure 22). It has not been possible to determine the depth to the tailings, due to the high concentration of suspended solids in the water.

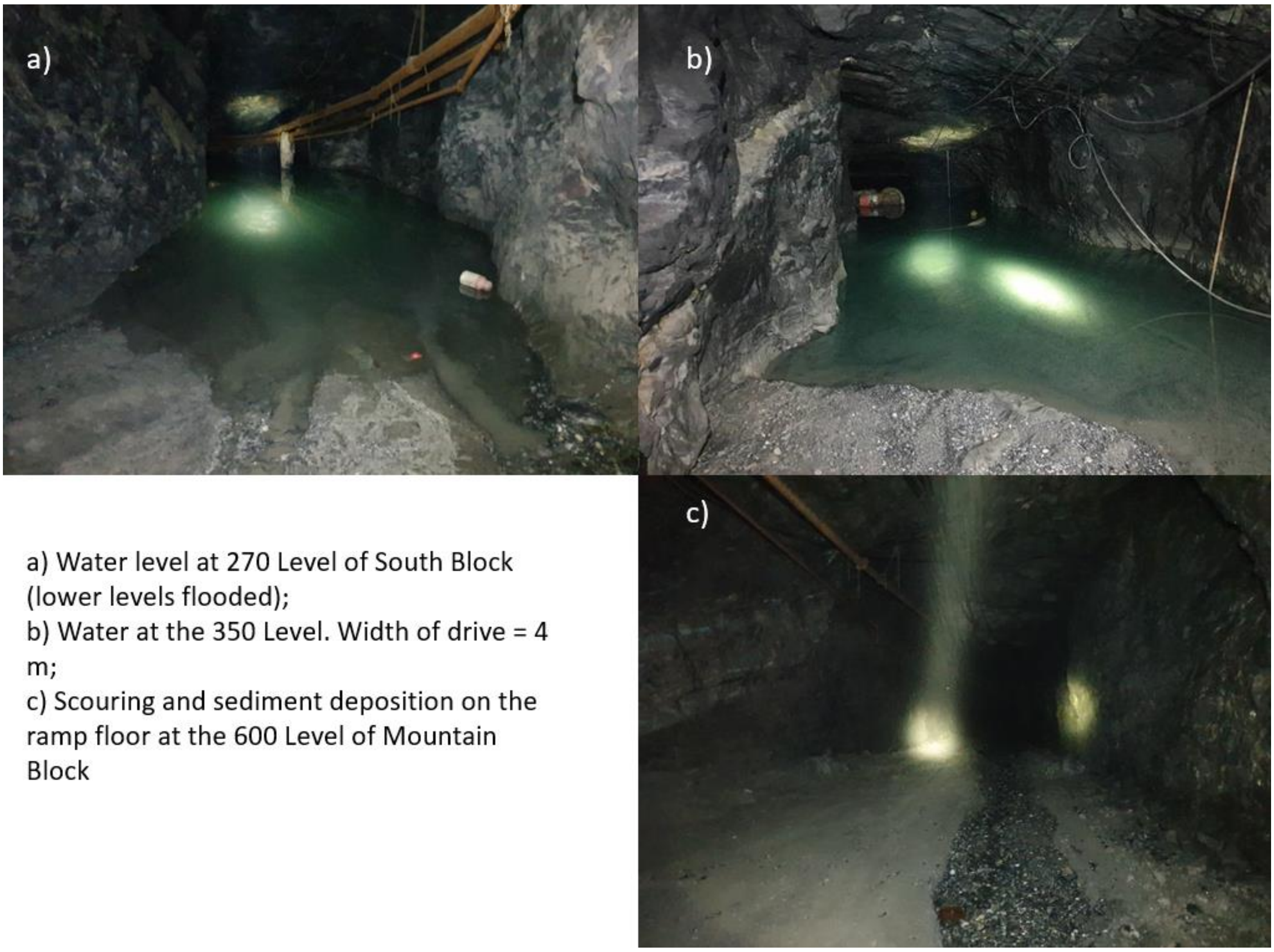

Figure 21: Flooded levels and sediment deposition within the mine (Golder, 2020a)

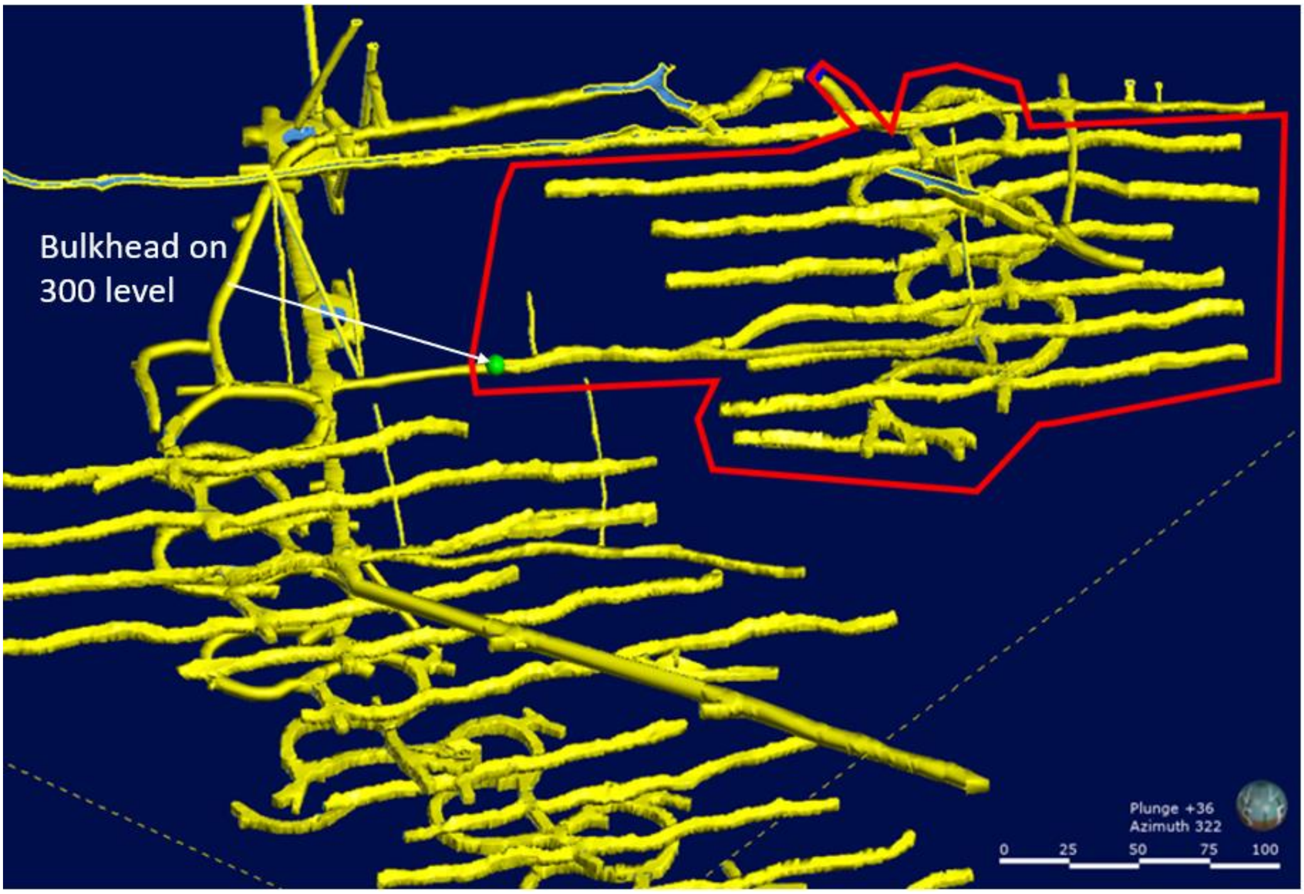

Figure 22: Oblique view of the bulkhead storage area (highlighted in red). Bulkhead marked in green, location of current water level in blue.

2.6.2 Hydrogeological Properties

It is considered that, based on site observations of the nature of the bedrock, the bulk hydraulic conductivity is likely to be in the order of 1 x 10^{-7} m/s to 1 x 10^{-10} m/s (from Domenico and Schwartz, 1988). During previous mining operations it was reported that average outflows from the mine's 300 Level portal were approximately 50 m^3/hour (Angel Mining, 2009). Scoping calculations (Golder, 2021c), presented at APPENDIX B, based on a water balance and a range of hydraulic conductivities return flows of this order of magnitude and are consistent with a bulk hydraulic conductivity in the range assumed.

Based on the results of particle size distribution (PSD) analysis work reported by Golder (2002a, 2002b) hydraulic conductivity and porosity values have been calculated for the fluvioglacial deposits, the results of which are presented at APPENDIX C. The hydraulic conductivity in the area of the proposed DTSF and process plant was estimated to be approximately 2.45 x 10^{-4} m/s, with a porosity of approximately 27%.

2.6.3 Groundwater Levels

A number of boreholes were drilled in the valley floor during previous investigations (Golder 2002a, 2002b). The coordinates of these boreholes is presented together with water level data and location coordinates in Table 5 and Figure 23. Groundwater depth and the position of trial pits excavated in 2020 are presented in Table 6 and Figure 16.

Table 5: Groundwater depths and elevations for boreholes installed in 2002 (N/A = not applicable)

Borehole ID	Easting	Northing	Ground Surface Elevation (masl)	Borehole Depth (mbgl)	Groundwater depth (mbgl)	Elevation of water table (masl)
2002 Boreholes						
BH01-01	509454.9	6691110.1	235.5	8.1	0.4	235.1
BH01-02	509455.7	6691500	240.1	8.5	0.8	239.3
BH01-03	509366.7	6691502	245	4.6	2.4	242.6
BH01-04	509346.1	6691301	248.1	4.9	2.4	245.7
BH01-05	509349.5	669110.4	239.1	7.3	1.5	237.6
BH01-06	508399.7	6689580	132.2	27.4	0.35	131.8
BH01-07	507725	6689325	96.6	22.6	0.9	95.6
BH01-08	507950	6689729.9	133.6	9.2	Dry	N/A
BH01-09	507410.1	6689385	119.8	6.6	3.1	115.9
BH01-10	503833.3	6686470.5	6.73	3.2	Dry	N/A
BH01-11	504031.8	6686486.3	30.63	15.6	10.3	20.3
BH01-12	504144.1	6686766.9	29.95	8.5	Dry	N/A
BH01-13	503833.3	6686470.5	6.7	10.7	4.1	2.6
BH01-14	506715.7	6688157.6	58.1	16.2	4.1	54
BH01-15	506942	6688247.7	74.3	9.2	6.3	68

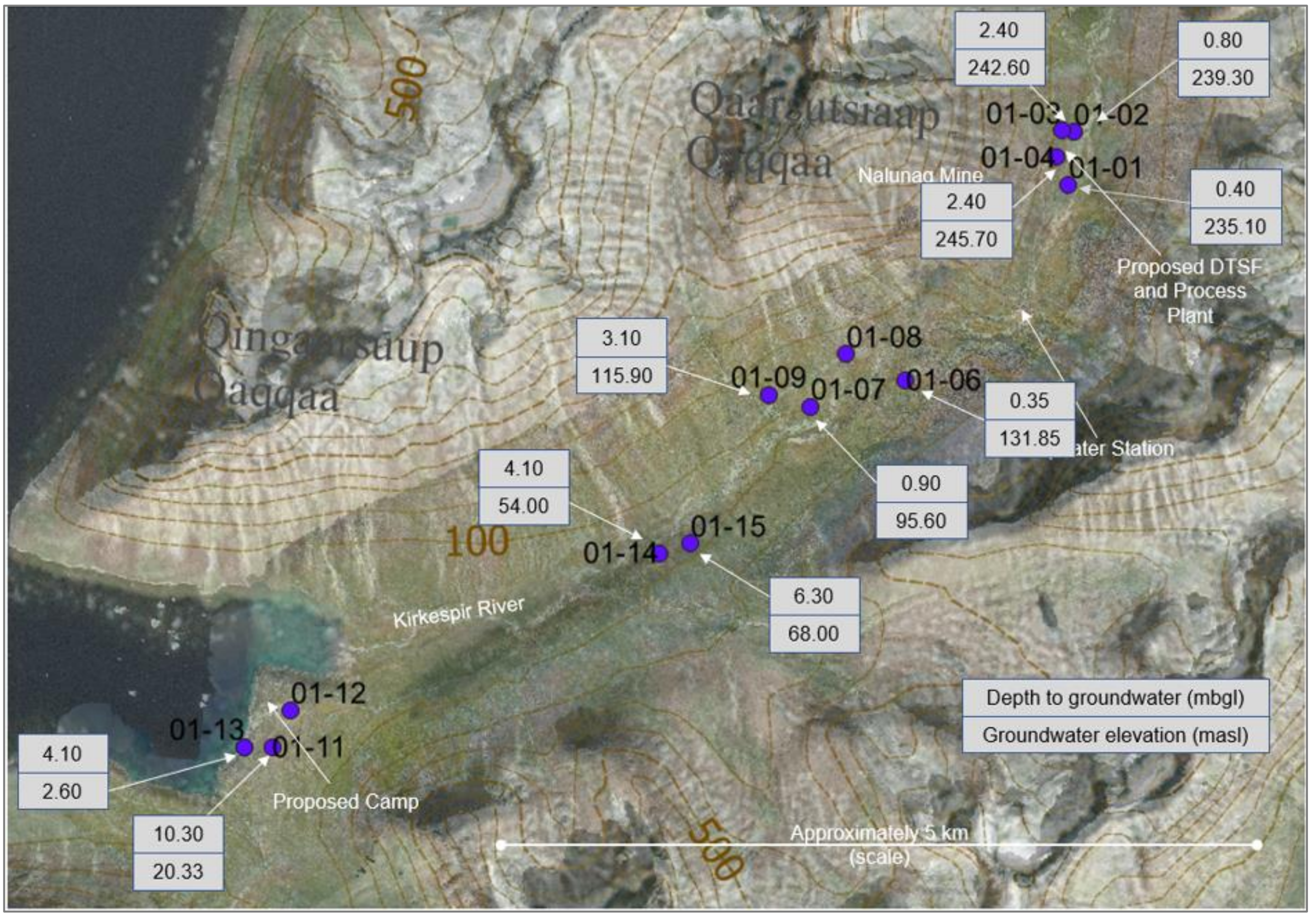

Figure 23: Groundwater depths and elevations for boreholes installed in 2002

Table 6: Trial pit locations, depth and groundwater levels for trial pits excavated in 2020

Borehole ID	Easting	Northing	Ground Elevation (masl)	Trial Pit Depth (mbgl)	Groundwater depth (mbgl)	Groundwater elevation (masl)
TP01	509359	6691080	237.0	2.7	1.94	235.06
TP02	509350.16	6691119.64	236.0	2.1	dry	dry
TP03	509408.32	6691076.46	235.0	1.8	1.00	234.00
TP04	509403	6691148	236.0	1.6	0.75	235.25
TP05	509481.54	6691374.02	240.0	1.8	1.52	238.48
TP06	509429	6691557	241.0	3.0	1.00	240.00
TP07	504134	6686811	31.0	1.8	dry	dry
TP08	504111	6686855	31.0	2.4	dry	dry

During the site visit, groundwater was encountered at shallow depth, in the fluvio-glacial deposits, in all the trial pits dug in the DTSF area at between 0.75 m below ground level (mbgl) and 1.94 mbgl, falling from approximately 240 m above sea level (masl) in TP06 to approximately 234 masl in TP03 indicating a hydraulic gradient of approximately 0.01. During previous operations a 10 m deep abstraction borehole was constructed alongside the old plant into the fluvioglacial deposits which achieved a consistent pumping rate of 45 m^3/hour (Kvaerner, 2002).

2.6.4 Hydrogeological Conceptual Model

2.6.4.1 Hydrogeological Units

Superficial Deposits

Within the superficial geology of Kirkespirdalen flow is likely to be intergranular within a number of identifiable units:

- **Talus**: colluvial deposit on the valley slopes comprising boulders, cobbles, gravel, sand and silt. In some areas these deposits may be ice cored.
- **Debris flow deposits**: comprising cobbles, gravel, sand and silt. Local in extent associated with specific gulley locations and runout onto the valley floor.
- **Fluvioglacial deposits**: comprising cobbles, gravel, sand and silt. Comprising the majority of the identified shallow valley infill in the vicinity of the proposed DTSF and process plant.
- **Moraine (till)**: comprising silt and clay with sand. Encountered interbeded with the fluvioglacial deposits in some locations.

It is considered likely that the units are interbedded as a result of various phases of talus development and glacial and fluvial activity. Resulting in an anisotropic aquifer system.

Some ranges for the above superficial valley deposits are presented in Table 7 below.

Table 7: Hydraulic properties of superficial valley deposits

Lithology	Porosity (%)	Hydraulic conductivity (m/ss)	Reference
Talus	43 – 60	6.5×10^{-3} - 9.4×10^{-3}	Clow *et al.,* 2003
Glacial Till	20%	1×10^{-12} - 2×10^{-6}	Fetter, 2001 Clow *et al*., 2003 Domenico and Schwartz, 1990
Fluvioglacial deposits	27	2.45×10^{-4}	APPENDIX C

Bedrock

The bedrock geology of the Nalunaq Mine comprises a meta-volcanic unit composed of basaltic pillow lavas and pyroclastics intruded by dolerite sills (Section 2.4 and Figure 12). Due to the crystalline nature of the bedrock, the hydrogeological regime at the site is likely to be dominated by fracture flow rather than intergranular flow with the metamorphosed and structurally deformed nature of the local geology making it difficult to distinguish one lithological unit from another in a hydrogeological context and therefore the controls over the hydrogeological system are likely to be fracture properties (frequency, aperture, length, orientation) and their spatial variation.

The shallow bedrock comprises a weathered to fresh, foliated, weak to very strong, grey and fined grained mafic rock. The highly fractured bedrock is associated with near surface weathering that is typically less than 10 m deep with poor Rock Quality Designation (RQD) values of <50% (Golder, 2020b). These conditions are also observed immediately within the portals of the development adits likely to be a result of mining activity. It is considered that groundwater flow will occur in the weathered bedrock within open fractures which are likely to exist in close proximity to the mine as an effect of blasting and stress relief both from mining but also from natural post-glacial adjustment. Due to the lack of superficial cover on the mountains, fractures are likely to be exposed to the environment and provide low retention time for precipitation to reach outflows (seepage or the mine itself) or the water table. Fracture flow is likely to be highly anisotropic and although open fractures will act as conduits to flow, fracture coatings or infills may cause fractures to act as barriers to flow potentially giving rise to perched water in places. With depth the bedrock rock quality designation (RQD) indicates good to excellent quality with values frequently over 90% (Golder, 2020b). The rock is likely to exhibit low hydraulic conductivity due the crystalline nature of the matrix although fractures are likely to facilitate fluid flow. The hydraulic conductivity is likely to be extremely variable within this unit and as stated in Section 2.6.2 is likely to be in the range of 1 x 10^{-7} m/s to 1 x 10^{-10} m/s.

Table 8: Bedrock hydraulic properties

Lithology	Porosity	Hydraulic conductivity (m/sec)	References
Fractured igneous and metamorphic rock	3 – 35%	1 x 10^{-7} to 1 x 10^{-10}	APPENDIX C Domenico and Schwartz, 1990

2.6.4.2 Groundwater Flow

Groundwater levels in the area reflect the slope of the topography. On the valley floor groundwater is present at shallow depths (0.5 to 5 mbgl) and, on the basis of increasing flows between upstream and downstream of the mine during low flows, as reported by SRK (2002), in the upper valley in the vicinity of the mine discharges to the Kirkespir River, which flows within braided river channels within the sand and gravel in the valley. During periods of heavy rainfall, the water level rises significantly flooding the valley floor.

Recent observations within the underground developments indicate the current water level to be around the 270 Level (Golder, 2020a). Based on data collected using a V-notch weir below the 300 Level portal in 2007 and 2008 the average flow over that period was approximately 50 m^3/hour (Figure 24), consistent with the average groundwater inflow reported by Angel Mining (2009). It is noted that the recorded 2007 and 2008 flows may include both natural groundwater inflows and losses from water seeping out of the underground tailings storage area. It is noted that these flows are significantly higher than the 2.6 l/s (approximately 9 m^3/hour) reported by Kvaerner (2002). The source of this discrepancy is not known but may relate to the available data and or estimation method at the time.

Following periods of heavy rainfall and or snowmelt water is likely to enter the mine via fractures and mine openings potentially flushing debris through the mine into lower levels. Historical observations from the underground workings estimate several discontinuities to produce 0.02 m^3/min to 0.04 m^3/min (1.2 m^3/hour to 2.4 m^3/hour) (Golder, 2002b).

Based on the current understanding of the hydrogeological regime of the Nalunaq Mine it is considered likely that recharge from precipitation and snowmelt from the overlying mountain catchment infiltrates through the fractured bedrock to the mine where it flows through the mine before discharging either at the 300 Level portal (during historic operations with water management systems including pumping in place (Golder, 2009a)) or (currently) infiltrating to bedrock via the flooded South Block and then through the overlying talus and waste rock deposits into the fluvioglacial deposits infilling Kirkespirdalen.

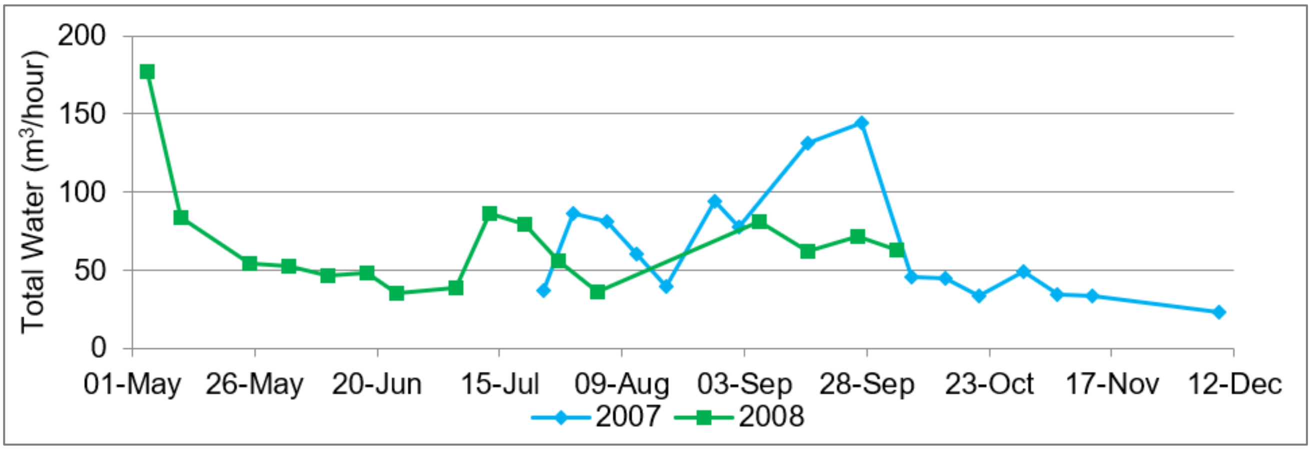

Figure 24: Available mine outflow data (Golder, 2009)

2.6.4.3 Summary of Conceptual Model

The conceptual model of the hydrogeology of the superficial deposits and bedrock in the Kirkespirdalen in the vicinity of the mine, DTSF and process plant are summarised in Figure 25 and Figure 26.

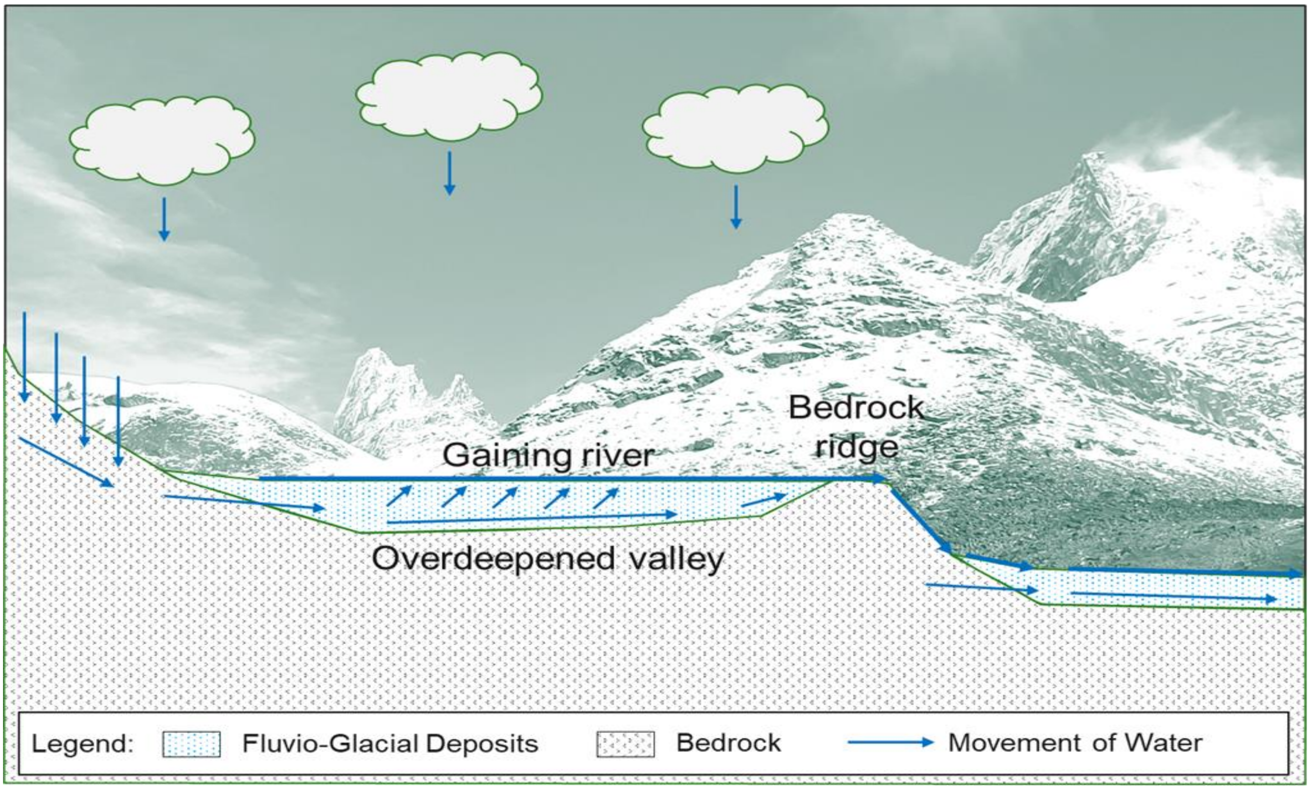

Figure 25: Conceptual model for groundwater and surface water movement in the Kirkespirdalen

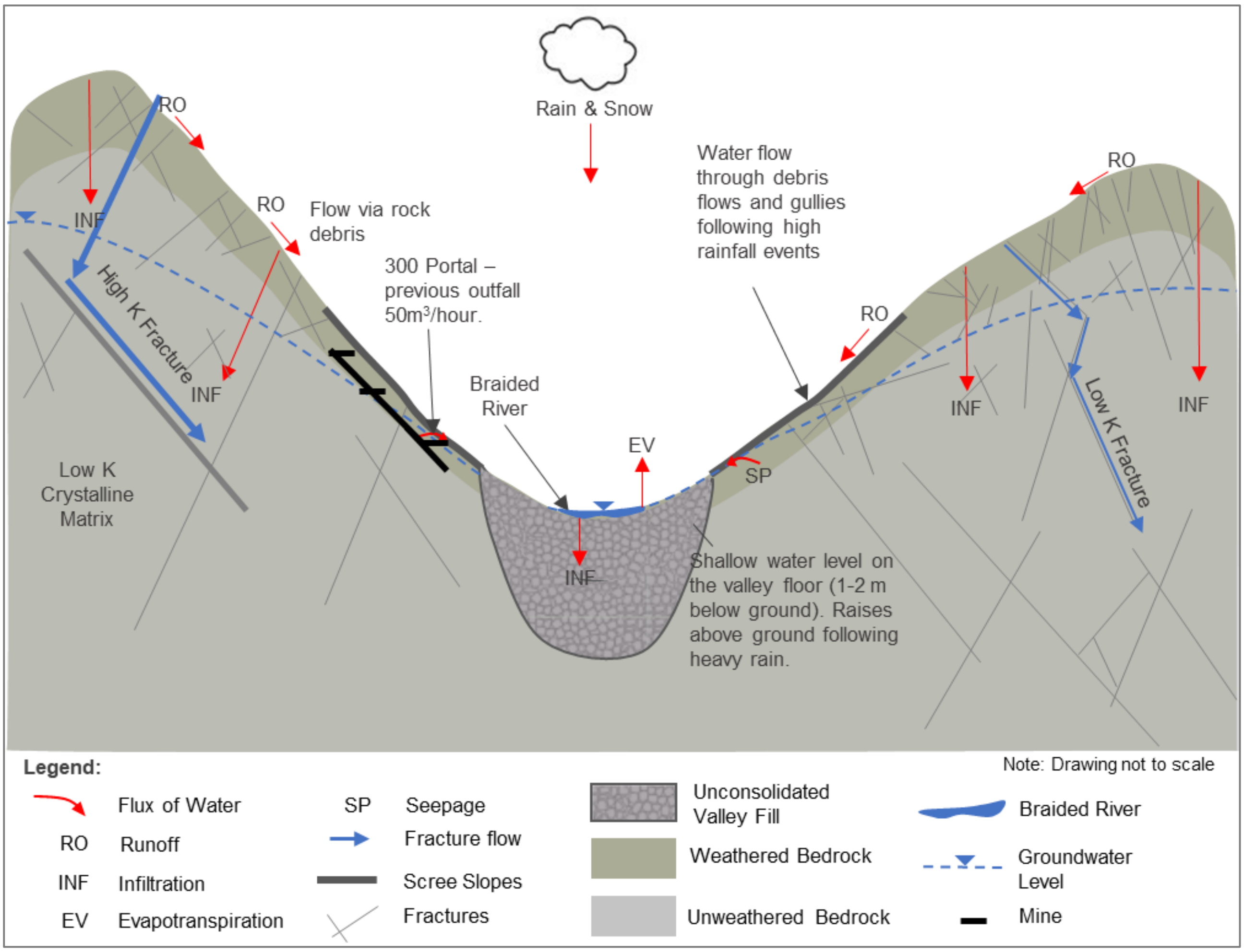

Figure 26: Conceptual model of the bedrock hydrogeology in the vicinity of the Nalunaq Mine showing the interaction with the superficial deposits

2.7 Hydrogeochemistry

2.7.1 General Deposit Mineralogy

Currently limited data exists regarding the hydrogeochemistry of the mine site. Information within this section is obtained from reports regarding environmental monitoring and from the general mineralogy and elemental composition of the deposit and the results of geochemical testing for leaching and acid-base accounting as it pertains to the potential for acid or neutral mine drainage and the leaching of metals. Nalunaq is a low-sulphide gold-quartz vein deposit where gold is hosted dominantly within quartz veining, which varies from 0.1 m to 2.0 m in thickness, within a shear zone. The host geology (Section 2.4) is a meta-volcanic unit composed of basaltic pillow lavas and pyroclastics intruded by dolerite sills, which have been metamorphosed to amphibolites, and have a pronounced foliation. Mineralisation at Nalunaq is of the mesothermal or lode-gold type.

On one or both sides of the vein an alteration zone of 0.2 m to 1 m width is present with alteration minerals including diopside, calcium rich amphibolite, biotite, calcium-rich plagioclase and calcite as ankerite. Traces of pyrite, pyrrhotite, arsenopyrite and lollingite are also present within the alteration zones and these may also contain gold where in close proximity to high grades of gold in the adjacent vein. Gold is mainly present as the native form, occasionally as a gold-bismuth alloy (maldonite, Au_2 Bi) and associated with native bismuth (Dominy *et al.*, 2006).

2.7.2 Historical Treatment of Waste Rock

During previous mining operations (c.2009 to 2014), mine waste rock was deposited directly on the mountain slope (Figure 27). The deposition of processed tailings and waste rock underground is described in the 2009 EIA (Angel Mining, 2009). In a study of the tailings of the former operation by Belmonte *et.al.* (2018) the tailings were found to be dominated by SiO_2, with Al_2O_3 as a major component, although CaO and total Fe_2O_3 were more dominant than Al_2O_3. Total cyanide detected within washed and unwashed samples was similar at 26 mg/kg and 18 mg/kg, respectively.

Figure 27: Waste rock dump exterior to the 300 Level portal, looking up the valley to the northeast (Golder, 2020a)

2.7.3 Environmental Monitoring

Environmental monitoring for the Environmental Agency for Mineral Resource Activities (EAMRA) was carried out annually from 2004 to 2019 (Glahder *et al*., 2005 to 2011 and Bach, 2020) to monitor the environmental impact of mining activities during the period 2004 to 2013 and in the period of cessation of mining 2014 to 2019. The monitoring was intended to provide data to assess the impact of wastewater discharged into the environment and monitor the potential transport of cyanide from the underground processing and tailings facilities. Water samples were taken at a number of locations as presented diagrammatically in Figure 28:

- Sample 1 was taken upstream of the mine site;
- Sample 2 was taken at the 300 Level mine entrance at the process wastewater discharge;
- Sample 3 was taken before the settlement pond after mine water mixes with the small (Figure 29); and
- Sample 4 was obtained in the waterfall pool of the Kirkespir River.

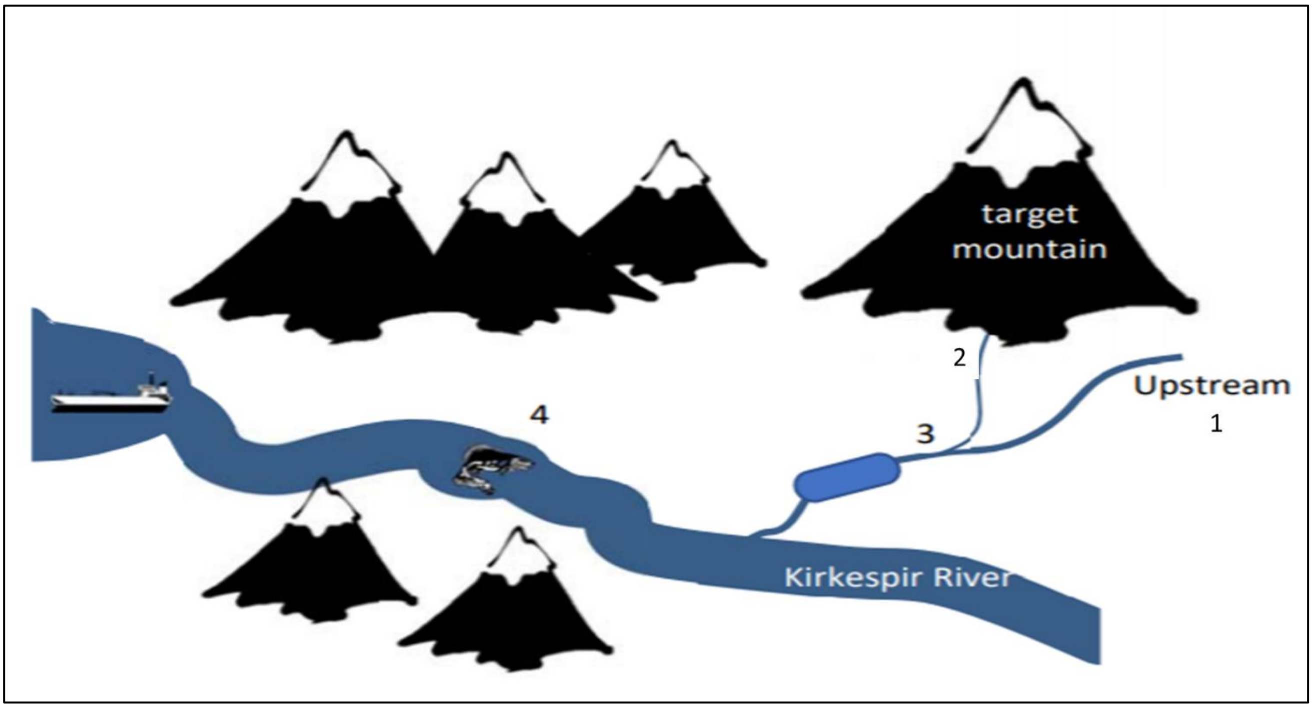

Figure 28: Freshwater sampling locations (Bach, 2020)

Figure 29: Water from the mine entering the small creek before the settling pond (Bach, 2020)

Freshwater samples were analysed for arsenic, cadmium, cobalt, chromium, copper, mercury, lead and zinc. During mining operations, elevated concentrations of arsenic, copper and nickel were detected in the wastewater sample from the mine exit (Sample location 2). Only arsenic was detected at an elevated concentration following cessation of mining. A high concentration of copper was detected in the upstream water in 2014, possibly due to contamination of the sample (Bach, 2020). Results for the years 2012 to 2019 are presented in Table 9 and **Error! Reference source not found.**. Elements were measured in filtered (<45 µm) freshwater samples at the upstream sampling point (“st 1”), the outflow from the 300 m portal (“st 2”), where process wastewater is discharged, a position where mine water mixes with the creek before the settlement pond and at the waterfall pool within the Kirkespir River (“st 4”). Corresponding values for Greenland Water Quality Criteria (GWQC) for mining activities (Mineral Resources Authority, 2015) are also shown.

Table 9: Summary of results for years 2012 to 2019 for freshwater samples (from Bach, 2020)

	As	Au	Cd	Co	Cr	Cu	Fe	Hg	Ni	Pb	Se	Zn
GWQC	*4*		*0.1*		*3*	*2*	*300*	*0.05*	*5*	*1*		*10*
2012												
Upstream (st 1)	0.862	0.001	0.005	0.012	0.102	1.86	2.62	0.013	0.289	0.030	0.232	1.06
Outflow mine (st 2)	70.1***	20.8	0.034	19.0	0.153	74.1***	22.5	0.049	19.8*	0.061	1.45	3.36
Kirkespir River (st 4)	1.65	0.016	0.008	0.155	0.121	3.65*	3.80	0.005	0.340	0.052	0.215	1.26
2013												
Upstream (st 1)	1.01	0.003	<DL	0.025	0.052	0.16	8.28	<DL	0.061	<DL	0.056	0.168
Outflow mine (st 2)	128***	26.8	0.016	50.3	0.070	65.6***	169	<DL	15.2**	0.018	0.956	0.926
Kirkespir River (st 4)	2.66	0.013	<DL	0.107	0.058	0.832	<DL	<DL	0.258	<DL	0.172	0.290
2014												
Upstream (st 1)	3.13	0.046	<DL	0.176	0.106	10.2**	12.5	<DL	0.282	0.321	<DL	4.48
Outflow mine (st 2)	273***	7.23	0.099	44.4	0.086	4.40*	387	0.030	4.90	0.008	0.576	2.56
Kirkespir River (st 4)	2.33	0.012	<DL	0.136	<DL	<DL	<DL	<DL	0.103	<DL	<DL	0.36
2015												
Upstream (st 1)	2.20	-	<DL	0.137	0.430	0.530	21.0	<DL	0.280	0.183	0.200	2.92
Outflow mine (st 2)	30.8***	-	0.018	0.205	0.415	1.40	6.05	0.004	1.29	0.089	0.160	4.34
Kirkespir River (st 4)	1.50	-	0.046	0.045	0.310	0.380	3.47	<DL	0.190	0.091	0.170	0.840
2017												
Upstream (st 1)	1.69	<DL	0.003	0.044	0.211	0.104	16.0	<DL	0.097	0.035	0.056	<DL
Outflow mine (st 2)	26.8***	<DL	0.009	0.059	0.347	0.762	3.65	<DL	0.749	0.011	0.148	1.77
Kirkespir River (st 4)	1.74	<DL	0.005	0.050	0.235	0.349	16.2	<DL	0.116	0.062	0.064	<DL
2019												
Upstream (st 1)	1.03	<DL	0.002	0.005	0.071	0.095	0.981	<DL	<DL	<DL	0.044	0.526
Outflow mine (st 2)*	157***	0.502	0.070	7.59	0.077	0.509	129	0.033	1.89	0.011	0.382	5.82
Kirkespir River (st 4)	1.77	0.011	0.002	0.057	0.087	0.208	9.96	<DL	0.085	0.004	0.092	0.606

*NOTES: * indicates slightly elevated concentrations, ** indicates concentrations 5-10 times background concentrations and *** indicates >10 x background concentration. DL = detection limit; ‘-‘ = not tested*

Low flow in the Kirkespir River is estimated to be approximately 284,256 m^3/d based on monitoring reported by SRK, 2002 (Golder, 2021c). At the waterfall pool (sample 4), element concentrations are consistently below the criteria applied.

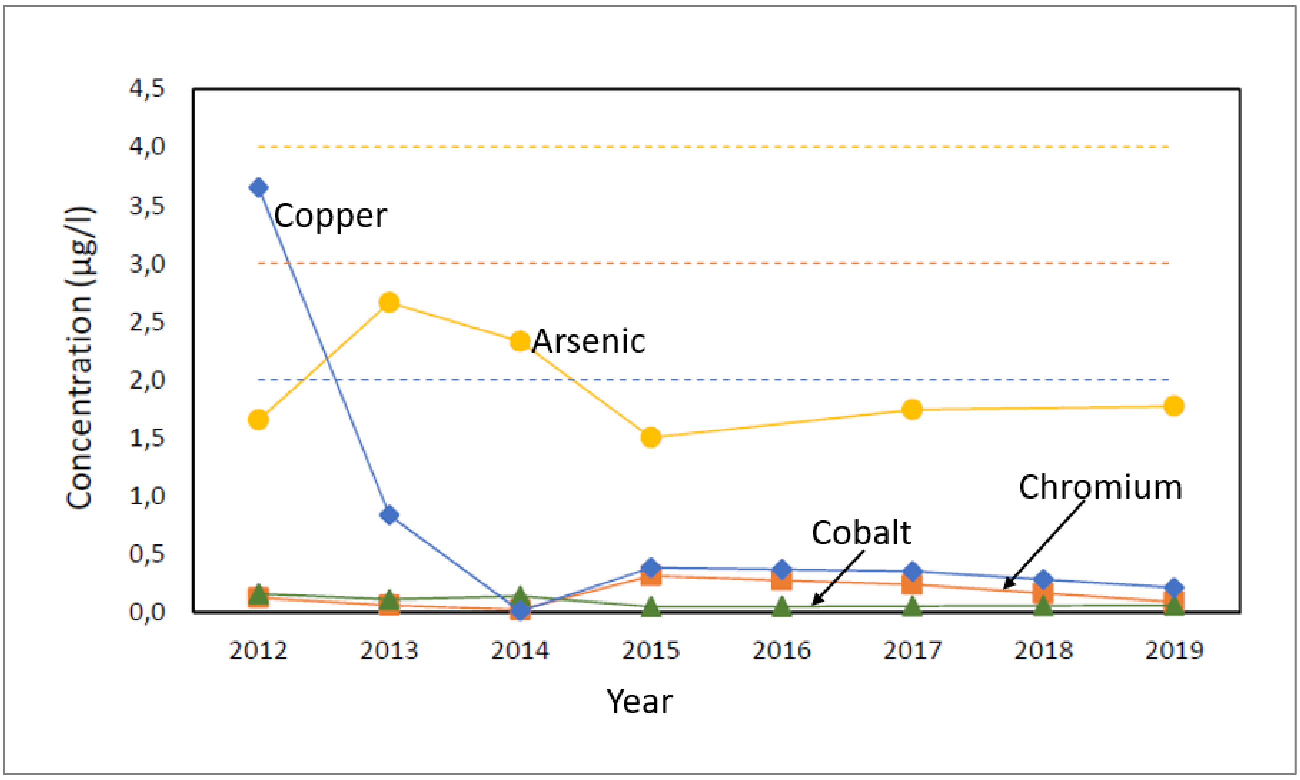

Figure 30: Concentrations of As (blue squares), Co (green triangles), Cr (orange squares) and Cu (blue diamonds) collected in water samples from the waterfall pool in the Kirkespir River. The dashed horizontal lines indicate GWQC As, Cr and Cu. (from Bach, 2020)

Monitoring of water chemistry at mine portal outflows and within monitoring wells was undertaken in 2015 (Table 10) following the cessation of mining in 2013. The monitoring demonstrates the presence of high concentrations of arsenic, cobalt, nickel and zinc compared to surface waters at the upstream camp location and there is a general increase in the concentration of these metals as the water flows down through the mountain. The higher concentrations are likely to be attributable to the groundwaters encountering different conditions of pH, oxygen and redox potential and to the higher retention time of the waters within the mine as compared to the river (Bach and Larsen, 2016).

Tailings produced by means of the Carbon-in-Pulp method were deposited within mine stopes. It is not known how sulphide minerals were treated in the milling process and these may have been concentrated in fine particles in the tailings within the mine and have the potential to release metals when exposed to oxidising conditions (Seal and Foley, 2002).

Table 10: Metal concentrations in samples of outflow water from the mine, surface water, groundwater, and underground mine (mg/l) in 2015 (from Bach and Larsen, 2016)

	As	Cd	Co	Cr	Cu	Fe	Hg	Ni	Pb	Se	Zn
Guideline values	4	0.1		3	2	300	0.05	5	1		10
Detection limit	0.100	0.009	0.003	0.25	0.060	0.740	0.017	0.110	0.010	0.17	0.100
CIP area	218	0.094	16.6	0.41	1.12	73.8	<dl	8.38	0.037	0.98	27.2
Detox area	106	0.074	9.95	0.51	4.55	86.5	<dl	3.41	0.756	0.78	546
Flow from process area	316	0.179	35.0	0.40	0.87	321	0.030	2.13	0.073	0.58	18.9
Flow to new tailing from process area	225	0.110	23.6	0.42	1.19	223	0.021	2.7	0.045	0.50	8.43
Flow to new tailing from mountain	89.8	0.029	6.23	0.47	1.36	16.5	<dl	15.2	0.066	0.50	3.03
Flow to new tailing after mix	216	0.094	22.4	0.37	0.86	229	<dl	2.8	0.062	0.54	7.53
Flow from new tailing area	285	0.037	27.6	0.045	2.45	261	0.030	0.94	0.024	0.30	1.08
Overlying water in ‘old’ tailing area	66.5	0.038	3.01	0.040	1.41	14.8	<dl	8.73	0.175	0.20	39.6
Upstream camp	2.20	0.006	0.137	0.43	0.53	21.0	<dl	0.28	0.183	0.20	2.92
Outflow 300 m mine portal 1	32.2	0.019	0.183	0.50	1.55	7.78	<dl	1.12	0.072	<dl	5.65
Outflow 300 m mine portal 2	29.3	0.017	0.227	0.33	1.24	4.31	<dl	1.46	0.106	0.19	3.03
Outflow 350 m mine portal	35.4	0.071	5.59	0.38	3.26	6.21	<dl	16.9	0.051	0.21	9.46
Monitoring well 1	12.9	0.019	0.223	0.28	1.39	6.87	<dl	1.02	0.056	0.22	1.80
Monitoring well 2	3.00	0.031	4.72	0.51	1.60	87.2	<dl	1.11	0.109	0.45	4.61
Kirkespir River	1.50	0.046	0.045	0.31	0.38	3.47	<dl	0.19	0.091	0.17	0.84

NOTES: Cells are highlighted where the Greenland Water Quality Guideline Values are exceeded. <dl= less than detection limit.

2.7.4 Cyanide Monitoring

It should be noted that the operation proposed by Nalunaq does not include the use of cyanide. The information in this section reflects the legacy of previous mining operations.

The former mining operation at Nalunaq used a cyanide leaching process where gold separation was carried out using the Carbon-In-Pulp method involving the addition of sodium cyanide. Sodium metabisulphite and air were used to decompose the cyanide into cyanate however this process is typically incomplete and residual cyanide remains within the tailings. Wastewater from the process therefore contains cyanide, which is diluted and degraded further within the settling pond and freshwater environment. Due to the dilution and retention time within the settling pond, a cyanide concentration of 0.20 mg/l was permitted by the DCE within the water leaving the mine at the 300 Level portal. Cyanide monitoring was carried out at the following locations (Figure 31):

1) From within the mine, as sample of process water / tailings water (1);
2) From wastewater leaving the mine at the 300 Level portal (2);
3) From the settlement pond (3);
4) From two monitoring wells (4 and 5); and
5) From the waterfall pond on the Kirkespir River (6).

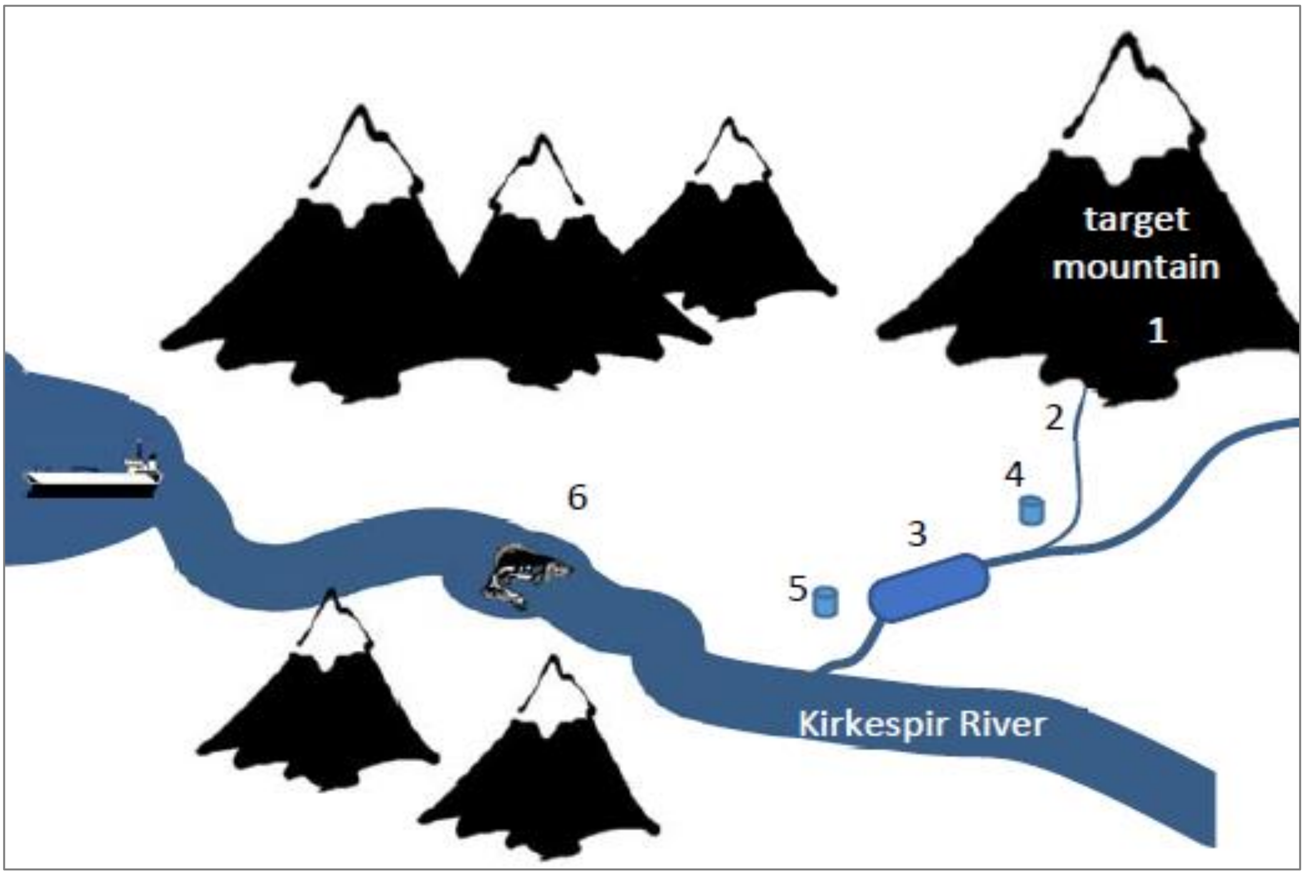

Figure 31: Cyanide sampling locations (Bach, 2020)

Results were compared to guideline values for maximum cyanide concentration set by EAMRA of 4.0 ppm in winter and 2.0 ppm in summer for the process water (location 1), 0.20 ppm for the wastewater discharge and settlement pond (locations 2 and 3) and 0.005 ppm for the monitoring wells and Kirkespir River (locations 4, 5 and 6).

During operations cyanide was monitored daily in outflowing wastewater and weekly in the Kirkespir River. Guideline values were exceeded on some occasions in 2011, however DCE did not consider that the concentrations represented a risk to the environment. Following mine closure DCE monitored cyanide levels annually at locations 2, 4, 5 and 6. Samples were also collected periodically from the underground mine. In 2015 and 2019 samples were analysed for total cyanide and free cyanide. In 2015 the total cyanide was 1.0 mg/l in the processing area and 0.003 mg/l in the tailings chamber. In 2019 total cyanide was detected at 0.36 mg/l in a sample collected inside the mine at the 300 m portal consistent with part of the cyanide remaining as complex-bound cyanide within the mine water (Bach, 2020). Based upon the results of water monitoring Bach (2020) stated that the cyanide did not represents a risk to the surrounding environment.

2.7.5 Potential for Acid Rock Drainage

In order to inform an assessment of risk from acid rock drainage (ARD) samples of tailings material that result from trials of the proposed gravity and gravity plus floatation processing circuits have been subject to elemental analysis, leaching tests and acid-base accounting. The results of the testing available to date are presented in SGS (2020a) and appended at APPENDIX D. A more detailed review will be presented under separate cover once all the analytical test work has been completed.

Major and trace element analysis has been undertaken to determine the total amount of elements in the solid phase of the tailings samples, together with identification of the mineralogy, to identify mineral assemblages and understand their influence on metal leaching and mineral reaction rates. Acid Base Accounting (ABA) is used as a screening procedure to estimate the net neutralising potential of a sample, from a calculation of the difference between the acid-neutralising potential and acid-generating potential of a sample and is interpreted alongside mineralogical data. The samples have been tested for net acid generation (NAG) and the leachate analysed to determine the potential chemistry of any seepage water from the waste.

Prior to commencement of mining in 2004 baseline water sampling indicated that water quality was ‘generally very good’ (Angus and Ross, 2009) with approximately neutral pH, odourless and with no visible suspended solids. Surface water contamination associated with this deposit type is generally not a significant due to the relatively low metal loading associated with these deposits (Seal and Foley, 2002). Sulphate is usually quickly diluted and arsenic, iron and other metals attenuated a short distance downstream as is evident in the results of the environmental monitoring undertaken to date (Section 2.7.3). Groundwater may contain high levels of arsenic and other metals. High levels of arsenic are present within samples from the gold mineralised vein and within haloes in the footwall and hanging wall (Schlatter, 2011; see below).

At Nalunaq ore minerals other than gold comprise maldonite (a gold alloy mineral, Au_2Bi), lollingite (an iron arsenide, FeAs2), arsenopyrite, pyrrhotite, pyrite, chalcopyrite and Bi-sulphosalts (Kaltoft *et al*., 2000), all of which have acid-generating potential. Due to the close association between gold and arsenic within the mineral vein, arsenic contents from samples obtained within the vein tend to be very high (Figure 32). Most samples with high arsenic contents were removed from the 400 m adit along the strike of the main vein with a few samples obtained from the footwall or hanging wall (Figure 32).

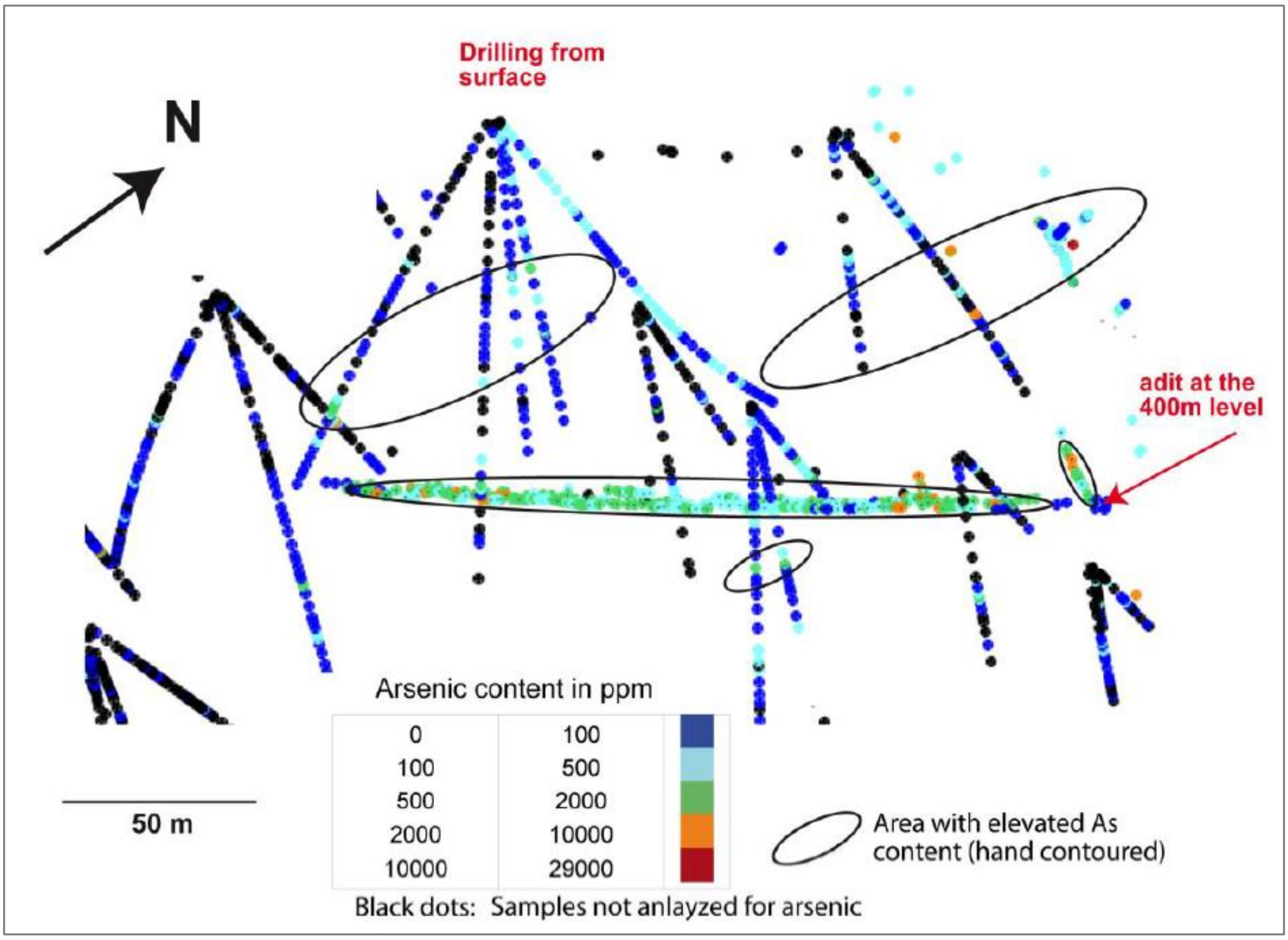

Figure 32: Arsenic contents of drill core samples, underground samples and surface samples (From Schlatter, 2011)

Tailings

Mineralogical characterisation of a tailings sample (back calculated from a concentrate (SGS, 2020b)) demonstrates that tailings mainly consist of amphibole and clinopyroxene (46.8%), followed by quartz (11.8%), arsenopyrite (8.0%), plagioclase (6.8%), epidote group minerals (4.9%) and chlorite / clay minerals (3.3%). The sulphidic mineral content was high (13.8%), with the sample containing arsenopyrite (8.0%), pyrrhotite (3.12%), pyrite (1.1%) and minor other sulphides including glaucodot, galena and chalcopyrite (total <1.5%). Total sulphur from the Leco method was 3.61%, of which 3.37% consisted of pyritic sulphur, calculated from the sulphide mineral amounts. The carbonate mineral content is relatively low, consisting mainly of calcite (2.1%) and other carbonates including dolomite (0.2%).

Based on the results of the ABA/NAG testing by SGS (2020a) it is concluded that the samples analysed are non-acid generating as illustrated in Figure 33 and are net buffering. The only identified potential contaminant of concern (PCOC) based on the elemental analysis was arsenic. The sequential extraction analyses indicate that a larger proportion of the arsenic was present in silicate and likely not susceptible to liberation. The average sulphide concentrations were low (average 0.08 wt%) consistent with the results of historic test work and environmental monitoring.

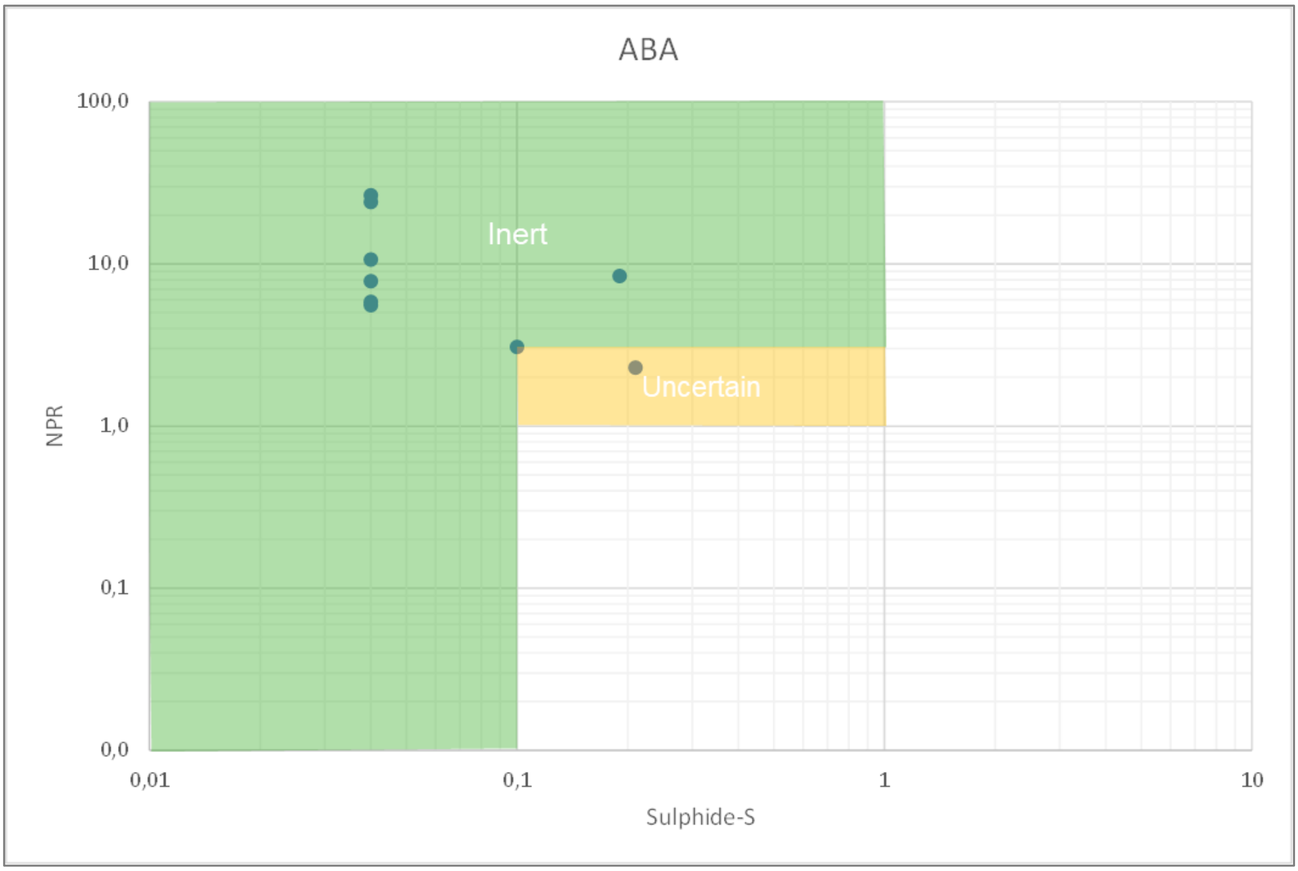

Figure 33: Results of acid base accounting (ABA) ratio and neutralisation potential ratio (NPR) (based on data from SGS, 2020a)

Waste Rock

With regard to the potential for ARD from the waste rock a review of the mineralogical data by SRK (2021) concluded that there were no significant mineralogical differences between the various mine areas (i.e. Mountain, Target, South and Valley Blocks) hence the likely geochemical behaviour of the waste rock with regard to the potential for ARD and or metal leaching will be similar to that experienced from historic waste rock. Based on the environmental monitoring undertaken to date no significant impact on the surface water environment has been noted that can be attributed to the waste rock. On this basis it is concluded that it is unlikely that the waste rock from the proposed new mining operation will present a significant risk to the surface water environment.

2.7.6 Conclusions

The results of environmental monitoring have demonstrated the presence of elevated concentrations of arsenic, copper and nickel in wastewater from the 300 Level portal. Based on the results of surface water monitoring undertaken between 2015 and 2019 there were no exceedances of the GWQS (Greenland Water Quality Standards) in the Kirkespir River at the waterfall monitoring station resulting from mining operations during between 2009 to 2013.

Monitoring of water chemistry at mine portal outflows and within monitoring wells was carried out in 2015, following cessation of mining. The monitoring demonstrated that arsenic, cobalt, nickel and zinc were elevated compared to surface waters at the upstream historical camp location and would suggest that the concentration of these metals increases as the groundwaters flow through the mountain encountering different conditions of pH, oxygen and redox potential. Water collected from the process area inside the mine in 2015 contain particularly high levels of arsenic and zinc. The source of arsenic within the process area is likely to be from the oxidation of arsenopyrite and tends to correspond with high cobalt concentrations, likely due to substitution of cobalt into the mineral arsenopyrite, forming glaucodot and, to a lesser extent, cobaltite.

Within rock the highest arsenic concentrations were detected within the area of the 400 m adit along the strike of the main vein.

Tailings have historically been deposited within chambers in the mine and these would have been deposited in a wet state, whilst waste rock was to be deposited directly on the mountain slope. An assessment of acid base accounting and acid generation has been undertaken to assess the potential for acid rock drainage from the tailings, based on which the tailings are considered to not be acid generating. It is noted that the materials historically deposited under water within mine chambers are unlikely to be under oxidising conditions and thus not be acid generating. With regard to the waste rock, based on the historic monitoring data and the similarities between the mineralogy the rock types across the Mountain, Target, South and Valley Blocks it is concluded that there will be no significant risk to surface water quality arising from the deposit of waste rock.

3.0 WATER QUALITY ASSESSMENT

Water quality criteria for mining projects in Greenland (Mineral Resources Authority, 2015) are presented in Table 11 below and compared to the monitoring results for 2015 to 2019, which is considered to represent the baseline for the project as these years follow cessation of mining in 2013. The nearest receptor is the freshwater Kirkespir River, so freshwater criteria are considered appropriate. The baseline for groundwater is presented, although the GWQC for mining activities do not include specified criteria for groundwater, based on data reported in Bach and Larson (2016). The concentrations of PCOCs recorded in the discharge (outflow) from the 300 Level portal in 2019 are presented in Table 12. It is noted however that as seen from the monitoring in the Kirkespir River there is no significant impact on the Kirkespir River.

Table 11: Water Quality Criteria and baseline concentrations in surface water and groundwater

Parameter	Freshwater GWQC (µg/L)	Sea water GWQC (µg/L)	Kirkespir River 2015-2019	Groundwater Baseline (µg/L)
Arsenic (As)	4	5	1.50 – 1.77	1.50
Cadmium (Cd)	0.1	0.2	0.002 – 0.046	0.046
Cobalt	-	-	0.045 – 0.057	0.045
Chromium (Cr(III))	3	3	*0.087 – 0.310	0.31
Copper (Cu)	2	2	0.208 – 0.380	0.38
Iron (Fe total)	300	30	3.47 – 16.2	3.47
Lead (Pb)	1	2	0.004 – 0.091	0.091
Mercury (Hg)	0.05	0.05	0.004 – 0.033	<dl
Nickel (Ni)	5	5	0.085 – 0.190	0.19
Zinc (Zn)	10	10	0.606 – 0.840	0.84
Cyanide (CN free)	5	2	-	-

Parameter	Freshwater GWQC (µg/L)	Sea water GWQC (µg/L)	Kirkespir River 2015-2019	Groundwater Baseline (µg/L)
Nitrogen (N total)	300	-	-	-
Phosphorus (P total)	20	-	-	-
Total suspended solids	50000	50000	-	-

*NOTES: Groundwater baseline applies to the borehole monitoring undertaken in 2015 (Bach and Larsen, 2016) and is unlikely to be truly representative of the baseline groundwater at this locality; <dl= below detection limit, * = values are for total chromium.*

Table 12: Mine discharge water quality from 300 Level portal in 2019

Parameter	Freshwater GWQC (µg/L)	Concentration in discharge (outflow) from 300 Level portal (2019)*
Arsenic (As)	4	157 (273)
Cadmium (Cd)	0.1	0.070 (0.099)
Cobalt	2	7.59 (50.3)
Chromium (Cr(III))	3	0.077 (0.430)
Copper (Cu)	2	0.509 (74.1)
Iron (Fe total)	300	129 (387)
Lead (Pb)	1	0.011 (0.321)
Mercury (Hg)	0.05	0.033 (0.049)
Nickel (Ni)	5	1.89 (19.8)
Zinc (Zn)	10	5.82 (5.82)
Cyanide (CN free)	5	-
Nitrogen (N total)	300	-
Phosphorus (P total)	20	-
Total suspended solids	50000	-

*NOTES: * Highest value measured since 2012 given in brackets (µg/L). Text in red identifies an exceedance of the water quality criteria.*

Where concentrations have exceeded groundwater quality criteria, values have been highlighted in red; these determinands have included arsenic, cobalt, copper, iron and nickel. The water quality results as presented above have been used to provide input to a tailings seepage assessment (Golder, 2021d).

4.0 FLOOD RISK ASSESSMENT

An assessment of flood risk to the proposed DTSF and process plant has been undertaken and is reported in Golder 2021b (APPENDIX A).

The assessment considered both existing site conditions as well as developed site conditions, accounting for various proposed layouts for the proposed DTSF and the process plant, under various ground surface conditions. The ground surface conditions considered “current” ground conditions (i.e. in which the compacted (now disused) camp platform areas remain intact), and “regraded” ground conditions (i.e. in which any areas with compacted material be removed and the underlying ground conditions returned to mimic the natural riverbed). These layouts were assessed under various climate scenarios, with key results reported for the 1-in-100 year, 1-in-200 year, 1-in-1000 year and Probable Maximum Precipitation (PMP) conditions.

Based on this assessment, the following key conclusions were made:

- The entire valley bottom is at risk of flooding, even under high-frequency (low return period) events for both existing site conditions and developed site conditions.
- Key results for the DTSF during a Probable Maximum Flood (PMF) are as follows:
  - A maximum flood depth of 3.1 m and a maximum flow velocity of 4.5 m/s can be expected for the proposed “Original” DTSF facility layout (regraded ground conditions). Localised velocities as high as 10.3 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.
  - A maximum flood depth of 2.7 m and a maximum flow velocity of 3.8 m/s can be expected for the proposed “updated” DTSF facility layout (regraded ground conditions). Localised velocities as high as 14.3 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.
- Key results for the process plant during a Probable Maximum Flood, are as follows:
  - A maximum flood depth of 2.0 m and a maximum flow velocity of 1.4 m/s can be expected for the “Original” process plant facility layout, assuming that the current camp pad is regraded and uncompacted. Localised velocities as high as 7.8 m/s can be expected at the base of the facility if the now disused camp platform is not regraded.
  - A maximum flood depth of 1.9 m and a maximum flow velocity of 2.7 m/s can be expected at the proposed “Updated” process plant facility layout (regraded ground conditions). Localised velocities as high as 6.0 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.

The following actions are recommended:

- Continuous monitoring of the Kirkespir River, as well as the highlighted tributary reporting to the river.
- Regrading of the raised camp platform areas to reduce localised velocities at the base of the facilities.
- Selection of the updated DTSF layout in order to reduce the potential obstruction to natural river flows within the Kirkespir River during flooding events.
- Selection of the updated DTSF layout in order to reduce the potential risk to the facility as a result of fluvial (river) flooding and scour.
- Establishing a platform elevation beneath the DTSF and Processing Plant that is situated above the predicted 1 in 1,000 year flood level.

- Installation of a warning system that provides immediate warning to the Site in the event of a large flood event.

There are several uncertainties in conducting flood risk assessments, as highlighted in this report, including the determination of representative climate conditions as well as the evaluation of catchment characteristics. This assessment incorporated available data and engineering judgment to inform inputs to the modelling exercise. In the event of any future changes to the proposed facility layouts, a reassessment may be required.

5.0 WATER MANAGEMENT PLAN

A water management plan for the mine operation including the proposed DTSF and process plant has been undertaken and is reported separately in Golder 2021e.

6.0 CONCLUSIONS AND RECOMMENDATIONS

In this report is presented a description of the groundwater and surface water environment in the vicinity of the Nalunaq Mine as currently understood based on the currently available data. The following broad recommendations are made with regard to further characterisation and monitoring of the water environment (see Golder, 2020a to d; and Golder, 2021a to e for further details):

- Establish continuous monitoring of the Kirkespir River flows upstream of the mine, in the tributary adjacent to the mine, at the waterfall monitoring station and the Container Bridge;
- Install a flood warning system that provides immediate warning to the site in the event of a large flood event from upstream;
- Establish a programme of water quality monitoring to include sampling upstream of the mine, from the mine portal discharge, discharge from any settlement ponds to the environment, waterfall monitoring station and the Container Bridge for a range of potential contaminants of concern as agreed with the authorities;
- Establish groundwater monitoring wells upstream and downstream of the proposed DTSF and process plant to monitor the environmental security of these facilities; and
- Establish an on-site weather station at both the mine camp and process plant, including monitoring of wind speed/direction, precipitation (rate/type), temperature, humidity and snow depth/density.

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APPENDIX A

Nalunaq Flood Risk Assessment (Golder, 2021b)

GOLDER

REPORT

Nalunaq Gold Mine

Flood Risk Assessment

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January 2021

Distribution List

Nalunaq A/S - 1 copy (pdf)

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APPENDIX E
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1.0 INTRODUCTION

Nalunaq A/S has engaged Golder Associates (UK) Ltd ("Golder") to provide technical support at its Nalunaq Gold Mine in southern Greenland. Following discovery of the Nalunaq mine in the early 1990s and development and operation by Crew Gold Corporation ("Crew Gold"), development was continued by Angus & Ross plc and Angel Mining (Gold) A/S, between 2004 and 2013. Subsequently additional exploration work has been undertaken in the Nalunaq area. It is understood that Nalunaq A/S are aiming to restart mining operations in 2021.

Golder has been contracted by Nalunaq A/S to provide support for the water and tailings management at the Nalunaq mine ("the Project"). As part of the Hydrology and Hydrogeology Assessment, Golder has prepared a Flood Risk Assessment (FRA), in support of an Options Analysis. The FRA has been based on a review of available information (including available site-based data) and hydraulic modelling to assess flood extent, water levels and flow velocities. This report outlines the model inputs, assumptions, and results for the study.

2.0 SITE DESCRIPTION

The Project is located in southern Greenland, approximately 35 kilometres (km) northeast of the town of Nanortalik, in the Municipality of Kujalleq. The mine lies on the northern slopes of the Kirkespirdalen (Kirkespir Valley) around nine km from the eastern side of the Sarqå Fjord. The Project location is shown on Figure 1.

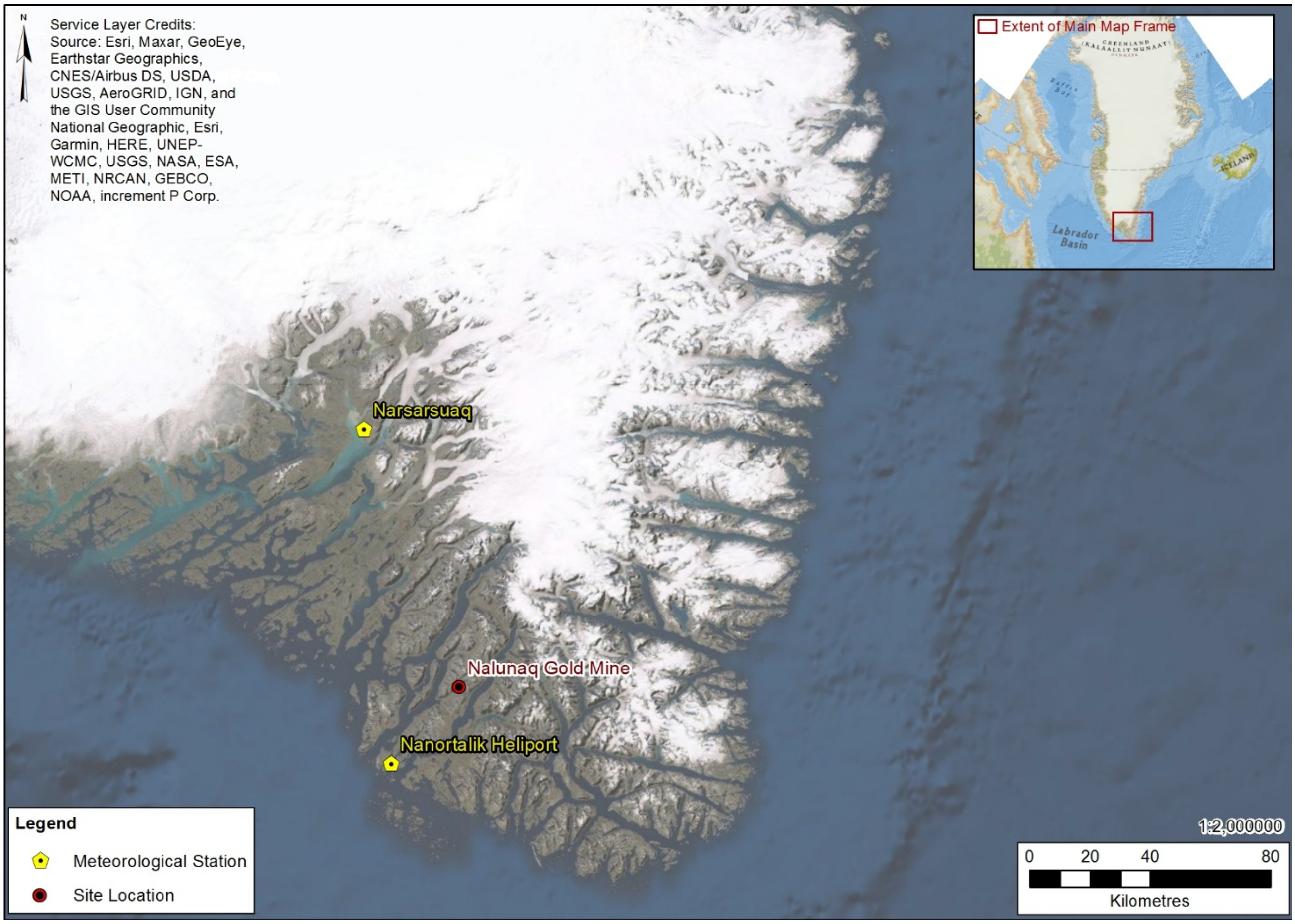

Figure 1: Project Location Plan

2.1 Proposed Site Layout

The mine facilities will consist of underground workings along the northern slopes of Kirkspirdalen, as well as several facilities along the valley bottom, namely a Dry Tailings Stack Facility (DTSF), Process Plant, Ore Pad and the underground mine. The Kirkespir River flows as a braided network of streams across the valley floor, with the centerline of the main river channel currently aligned approximately 20 to 50 metres (m) away from the proposed facility layouts.

Two facility layouts, each considering different footprints of the DTSF and Process Pad, were considered within this assessment to support the Options Analysis, as follows:

1) DTSF Layout accommodating a 5-year deposition schedule, as well as a Process Plant and Ore Pad layout, with the Ore Pad located to the east of the Process Plant. For the purposes of this report, this layout is referred to as the "Original" layout.

2) DTSF Layout accommodating a 5-year deposition schedule, approximately 30 m narrower than the "Original" layout (i.e. in order to reduce the width of the DTSF within the river valley), and the Ore Pad relocated to the west of the Process Plant. For the purposes of this report, this layout is referred to as the "Updated" layout.

The Original and Updated Layouts are shown in Figure 2.

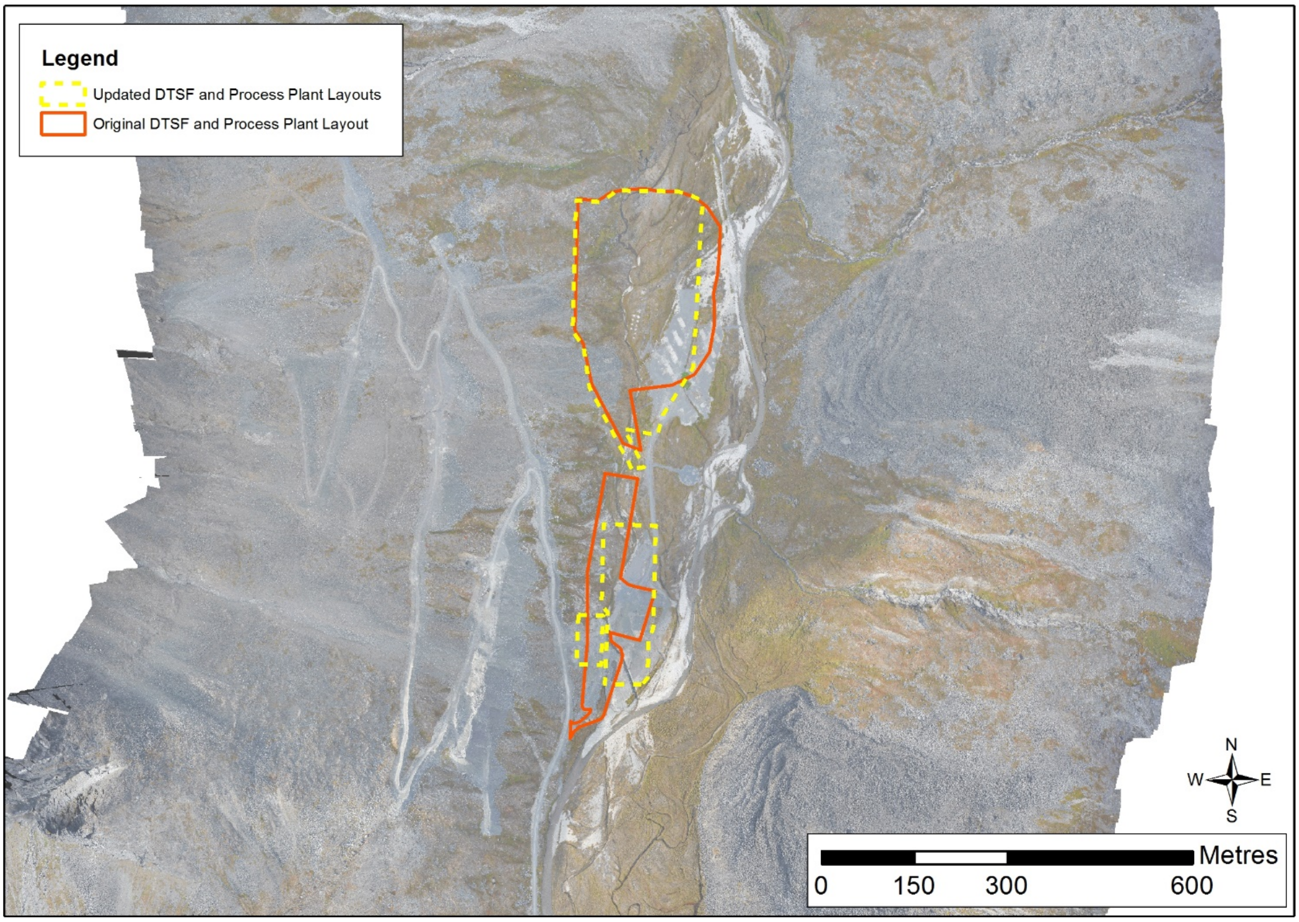

Figure 2: DTSF and Process Pad Layouts ('Original' and 'Updated')

3.0 CLIMATE ASSESSMENT

3.1 Climatic Setting

The Site location has a tundra climate with strong oceanic and polar influences (SRK Consulting, 2002). Precipitation (including both rainfall and snowfall) is moderate with an annual average cumulative depth of approximately 602 mm. Snow cover is relatively limited within southern Greenland, with an average annual snowfall depth of 194 mm. Temperatures show relatively little variation between seasons. July is the hottest month with a mean temperature of 10.7 degrees Celsius (°C) and February is the coldest month with a mean temperature of -7.9 °C.

3.2 Regional Climate Stations

There is no onsite meteorological station at the Site, with only short climate datasets available during which local data capture (e.g. rainfall) has been carried out as part of a specific site-based study. These are too short to be sufficient for hydrological analysis. As such, daily precipitation, and temperature data from two stations (Nanortalik Heliport and Narsarsuaq) were sourced from NOAA (2020) and Tutiempo (2020), respectively. The location of these stations relative to the Site are shown in Figure 1 and station details are listed in Table 1.

Table 1: Climate Station Details

Station Name	Latitude Longitude	Distance from Mine (km)	Elevation (m AD)	Record	Data Type	Portion of Record Complete
Nanortalik Heliport	60.13°N -45.23°E	35 km (SE)	5 mAD	01/01/1980 02/11/1985 (i.e. < 5 years)	Daily Precipitation	92.5%
				01/01/2014 10/07/2020 (i.e. < 6 years)	Hourly Average Air Temperature	89.7%
Narsarsuaq	61.13°N -45.41°E	91 km (NNE)	34 mAD	01/01/1973 31/12/2003 (i.e. 30 years)	Daily Precipitation	98.8%
					Daily Average Temperature	99.5%

As less than five years of daily precipitation data was available for the Nanortalik Station, this record was dismissed in favour of Narsarsuaq, which also has a longer and more complete dataset (1973 to 2003). For consistency, the same record was used for temperature.

3.3 Precipitation

Total precipitation depths (i.e. including both rainfall and snowmelt) were available for the Narsarsuaq Station as outlined in Table 1 above, and these were used as the basis for the analysis of potential flood risk to the proposed mine infrastructure.

In order to estimate rainfall and snowfall values, potential snowfall depths were derived using the degree-day method (Maidment, 1993). A base daily average air temperature of 0 °C was assumed between April and October (period of major melt), while a base daily average air temperature of 2.5 °C was assumed between November and March. Any daily recorded precipitation which occurred on days with recorded daily air temperatures that exceeded the base temperature was assumed to report to the Site as rainfall. The assignment of these base temperatures reflect lower air temperatures required to trigger snowmelt between April and October, as opposed to other times of the year. This is due in part to energy available from the sun, as well as other factors, such as warmer rainfall and higher ground temperatures. A melt factor of 0.9 mm per °C per day was also applied, which accounts for the accelerating effect of rainfall on the melting of the snowpack (and hence rate of snowmelt).

Annual total precipitation averaged 601.8 mm, of which it was determined approximately 68% was estimated to occur as rainfall and the remainder as snowfall. The wettest month was September with an average monthly total precipitation depth of 73.8 mm, and the driest month was March with an average monthly total precipitation depth of 35.6 mm. Measurable snowfall occurred from October to April, with rainfall occurring predominantly in the summer months.

Precipitation, rainfall and snowfall depths for Narsarsuaq are provided in Table 2 and Figure 3.

Table 2: Average Monthly Precipitation at Narsarsuaq Station (1973 – 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)	44.0	37.7	35.6	45.6	35.8	57.4	58.2	64.6	73.8	57.6	47.6	43.9	601.8
Rainfall (mm)	3.2	7.5	2.4	33.5	35.0	57.4	58.2	64.6	73.1	50.4	16.2	6.4	407.8
Snowfall (mm) (1)	40.7	30.3	33.3	12.2	0.8	0.0	0.0	0.0	0.6	7.2	31.4	37.5	194.0

NOTES: (1) As water equivalent.

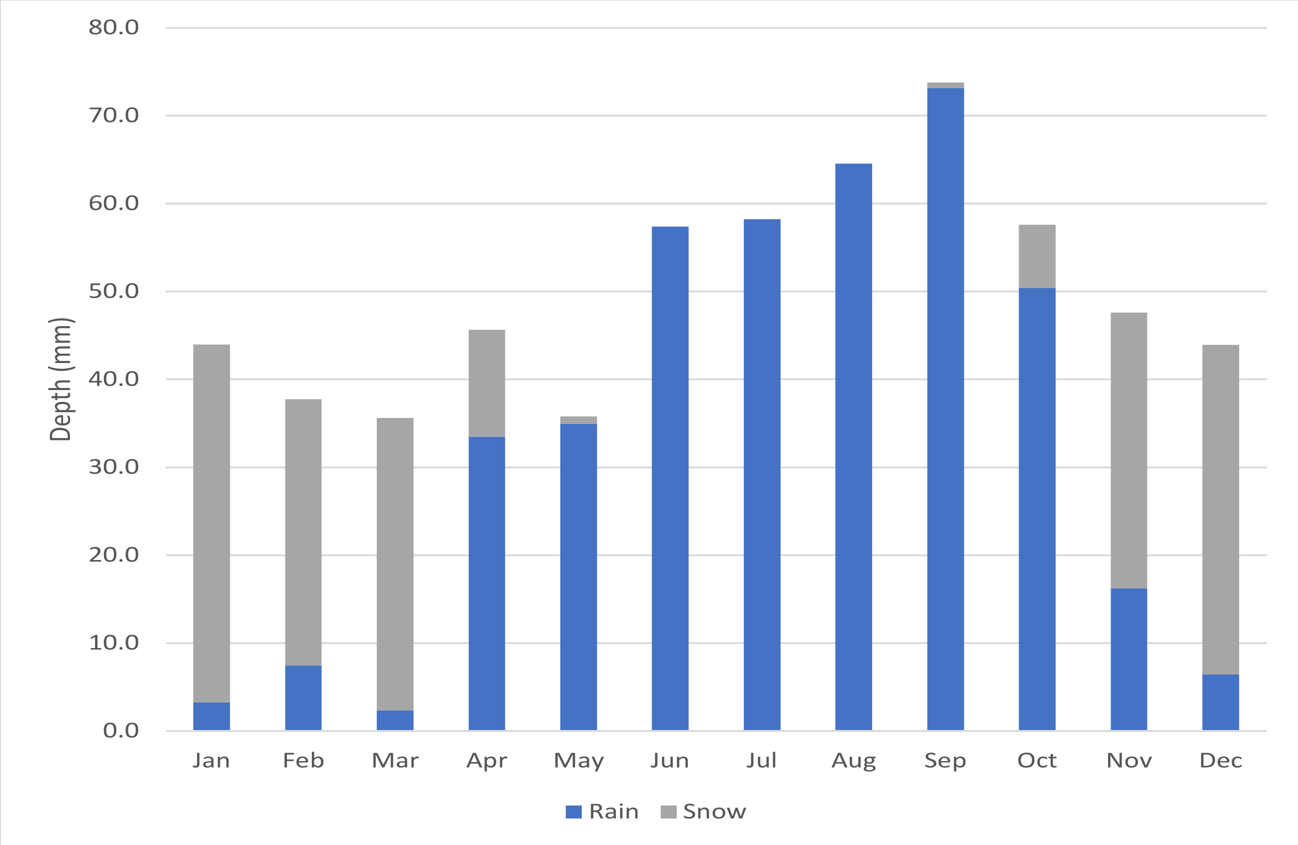

Figure 3: Average Monthly Rainfall and Snowfall at Narsarsuaq Station (1973 – 2003)

3.4 Temperature

Average temperature data recorded at the Narsarsuaq Station between 1973 and 2003 are presented in Table 3, including the mean (average) minimum, mean maximum and mean daily temperatures for the 30-year period of record. The mean annual temperature during this period was 0.9 °C. Temperatures were highest from April to October, and lowest from November to March (mean temperatures did not exceed 0 ˚C). July was the hottest month with a mean maximum temperature of 20.3 °C. February was the coldest month, with a mean minimum temperature of -24.0 °C. The highest temperature recorded in the 30-year record was 25 °C (02/04/1998) and the lowest was -39.8 °C (23/01/1984).

Table 3: Average Temperature at Narsarsuaq Station (1973 – 2003)

Parameter	Temperature (°C)												
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Mean (Average) Maximum Daily Temperature	8.4	7.0	8.6	12.3	16.2	19.2	20.3	19.2	16.6	12.8	11.4	9.1	13.4
Mean (Average) Daily Temperature	-7.5	-7.8	-6.0	0.3	5.4	8.9	10.7	9.4	5.8	0.8	-3.5	-6.0	0.9

Parameter	Temperature (°C)												
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Mean (Average) Minimum Daily Temperature	-23.3	-24.0	-21.1	-13.1	-4.3	1.3	3.4	2.3	-3.1	-9.8	-17.9	-20.7	-10.9

3.5 Evaporation

Potential evapotranspiration (PET) at the Narsarsuaq Station between 1973 and 2003 was calculated from the temperature dataset using the Thornthwaite method (Thornthwaite, 1948). Average monthly and annual PET depths are presented in Table 4. Average annual evapotranspiration over the 30-year period of record was 465.2 mm. Evapotranspiration rates were highest from June to August (over 95 mm of evaporation occurred in each month). Potential evapotranspiration rates were lowest from November to March, with little to no evaporation in these months.

Table 4: Average Potential Evapotranspiration at Narsarsuaq Station (1973 – 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Potential Evapotranspiration (PET) (mm)	0.1	0.5	0.0	14.9	64.4	100.6	118.0	96.3	56.3	12.2	1.5	0.4	465.2

4.0 HYDROLOGICAL ASSESSMENT

4.1 Hydrological Setting

The Nalunaq Mine is located in the fjords of southern Greenland. The area is mountainous and is characterised by steep topography with slopes reaching from sea level to elevations of approximately 1500 metres above sea level (masl). The mine sits on the northern slopes of Kirkespirdalen U-shaped glacial valley. The valley surface is predominantly covered in grass and scree; however, vegetation becomes more limited at higher elevations.

The Kirkespir River flows approximately 15 km along the length of the valley, originating at a small glacial lake at the head of the valley and discharging into the Sarqå Fjord at its base. The stream has no major tributaries and has an estimated catchment area of 95 km^2 (Kvaerner E&C, 2002). Flow measurements from the river are limited, though measurements taken between 25/05/1998 and 31/08/1998 give an indication of typical base flow in the river, with an average[1] flow rate of 3.95 m^3/s being recorded immediately downstream of the Site over the 3-month monitoring period (SRK Consulting, 2002).

[1] The maximum recorded stream flow rate was in late May 1998 (i.e. 4.4 m^3/s) and the minimum recorded stream flow rate was in late August 1998 (3.6 m3/s). There was no rainfall recorded during the 3 month monitoring window, with the last recorded rainfall observed on 25th April 1998

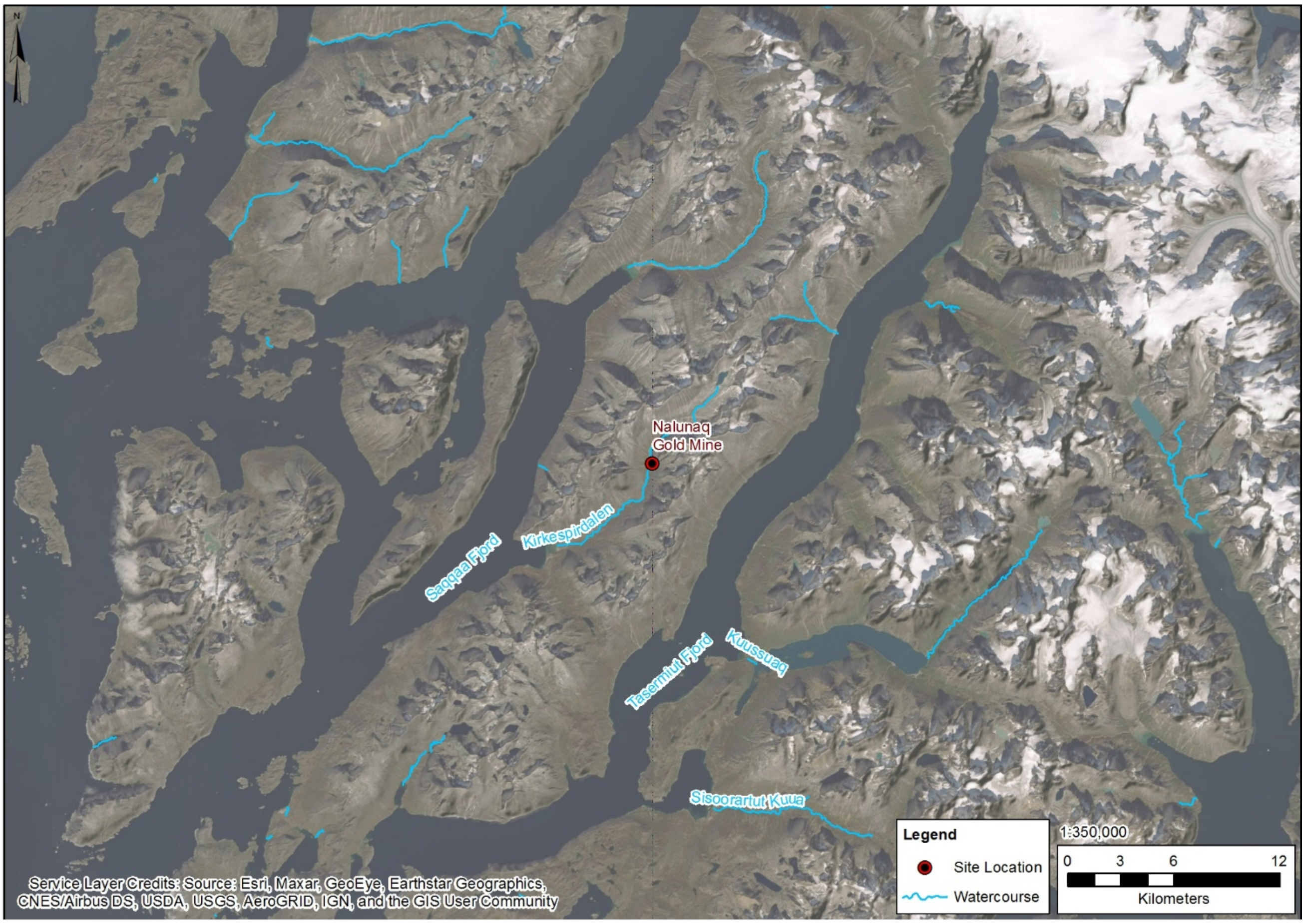

Figure 4: Location of Mine and surrounding fjords

4.2 Precipitation Analysis

4.2.1 Snowmelt

The annual spring melt plays a key part in the local hydrology and the 30-year total precipitation and temperature records for the Narsarsuaq station (1973 - 2003) were used to derive snowmelt data. As mentioned in Section 3.3, snowmelt data was derived using the degree-day method (Maidment, 1993) with a melt factor, which accounts for the accelerating effect of rainfall on the snowmelt rate. This approach allows for accumulation of a synthetic snowpack according to the daily snowfall and subsequent depletion of the snowpack, based on a potential snowmelt. A snow density of 0.1 was assumed in the calculations to convert snow depth into its water equivalent.

The calculated average monthly and annual snowmelt water equivalents from 1973 to 2003 are presented in Table 5, along with average rainfall plus snowmelt depths. Snow melt is predicted in all months barring August, however it peaks in spring (i.e. April) with a maximum average of 83.1 mm.

Table 5: Average Snowmelt and Rainfall plus Snowmelt at Narsarsuaq Station (1973 – 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Snowmelt (mm)	6.2	7.9	6.0	83.1	50.3	26.9	3.0	0.0	0.1	3.2	6.5	8.3	201.4
Rainfall plus Snowmelt (mm)	9.4	15.4	8.3	116.6	85.2	84.3	61.2	64.6	73.3	53.6	22.7	14.7	609.3

The total rainfall and snowmelt of 609.3 mm indicated in Table 5 (above) is the *calculated* value based on the degree-day method described in Section 3.3, and therefore the annual total is marginally higher than the recorded annual average precipitation of 601.8 mm (Table 2).

4.2.1 Rainfall plus Snowmelt Depths

Annual maximum daily rainfall plus snowmelt was compiled from the 30 year record for the Narsarsuaq Station. Frequency analysis of the annual maximum timeseries was undertaken to estimate rainfall and snowmelt depths for a range of return periods. The Log Normal probability distribution was deemed to provide the most representative fit to the results. The results are presented in Table 6.

Table 6: Annual Maximum Daily Rainfall plus Snowmelt Depths at Narsarsuaq Station

Return Period (Years)	Annual Exceedance Probability (%) (1)	Rainfall plus Snowmelt Depth (mm)
2	50	42.4
5	20	59.0
10	10	70.1
25	4	84.3
50	2	95.9
100	1	106.6
200	0.5	116.5
500	0.2	131.1
1,000	0.1	142.5

NOTES: (1) The Annual Exceedance Probability (AEP) refers to the probability of a flood event occurring in any given year.

4.3 Probable Maximum Precipitation

Probable Maximum Precipitation (PMP) is defined as "*theoretically the greatest precipitation for a given duration that is physically possible over a given watershed area, or size of storm area at a particular location at a certain time of year, under modern meteorological conditions*" (WMO, 2009).

To account for the possible influence of rain-on-snow events at the Site, two scenarios for assessing the PMP were considered, i.e.:

- The PMP depth derived for the first scenario considered daily rainfall data over the entire year; and
- The PMP depth for the second scenario considered rainfall limited to the snowmelt season (April to July), in which the 1-in-100 year (1% Annual Exceedance Probability (AEP)) snowmelt depth was added to the derived rainfall depth to obtain a combined PMP.

The PMP depths were calculated using the statistical procedure described by the WMO (2009), and the results are summarised in Table 7.

Table 7: Probable Maximum Precipitation Results (mm)

Assessment Basis	PMP Depth (mm)
Annual based on daily rainfall over entire year (Scenario 1)	439.3
Assessment based on spring rainfall plus 1-in-100 year snowmelt (Scenario 2)	428.6

As shown in Table 7, rainfall events which occur outside of the snowmelt season are the driving factor behind the large storm events. These rainfall events typically occur in July and August. On this basis, the PMP depth represented by the annual rainfall record (Scenario 1) of 439.3 mm was selected for use in the flood risk assessment.

4.4 Hydrology Calculations

4.4.1 Sub-Catchment Properties

To assist in the assessment of land type cover, aerial photography in the form of orthophotos was used in combination with satellite imagery. The orthophotos extend 9 km upstream of the Sarqå Fjord, covering an area of approximately 19 km^2.

While these photos cover the Site and its immediate surrounds, this detailed imagery does not extend higher than 600 masl. As such, publicly available aerial imagery was used to supplement the orthophoto imagery.

To assist in the determination of catchment boundaries, 0.6 m resolution Light Detection and Ranging (LiDAR) data (covering the same extent as the orthophotos) was used in combination with publicly available topographic data.

A study catchment with a downstream boundary located 600 m downstream of the Process Plant was selected for the assessment of peak flows (Figure 5). This boundary location was chosen to allow all contributing flows to the Site to be assessed. A minor tributary to the Kirkespir River flows through both the proposed location of both the proposed Original and Updated DTSF and Process Plant footprints. As such, peak flows for the study catchment were derived based on a combination of hydrographs derived for (i) the tributary sub-catchment and (ii) the main river sub-catchment. The boundaries of the catchments are shown in Figure 5.

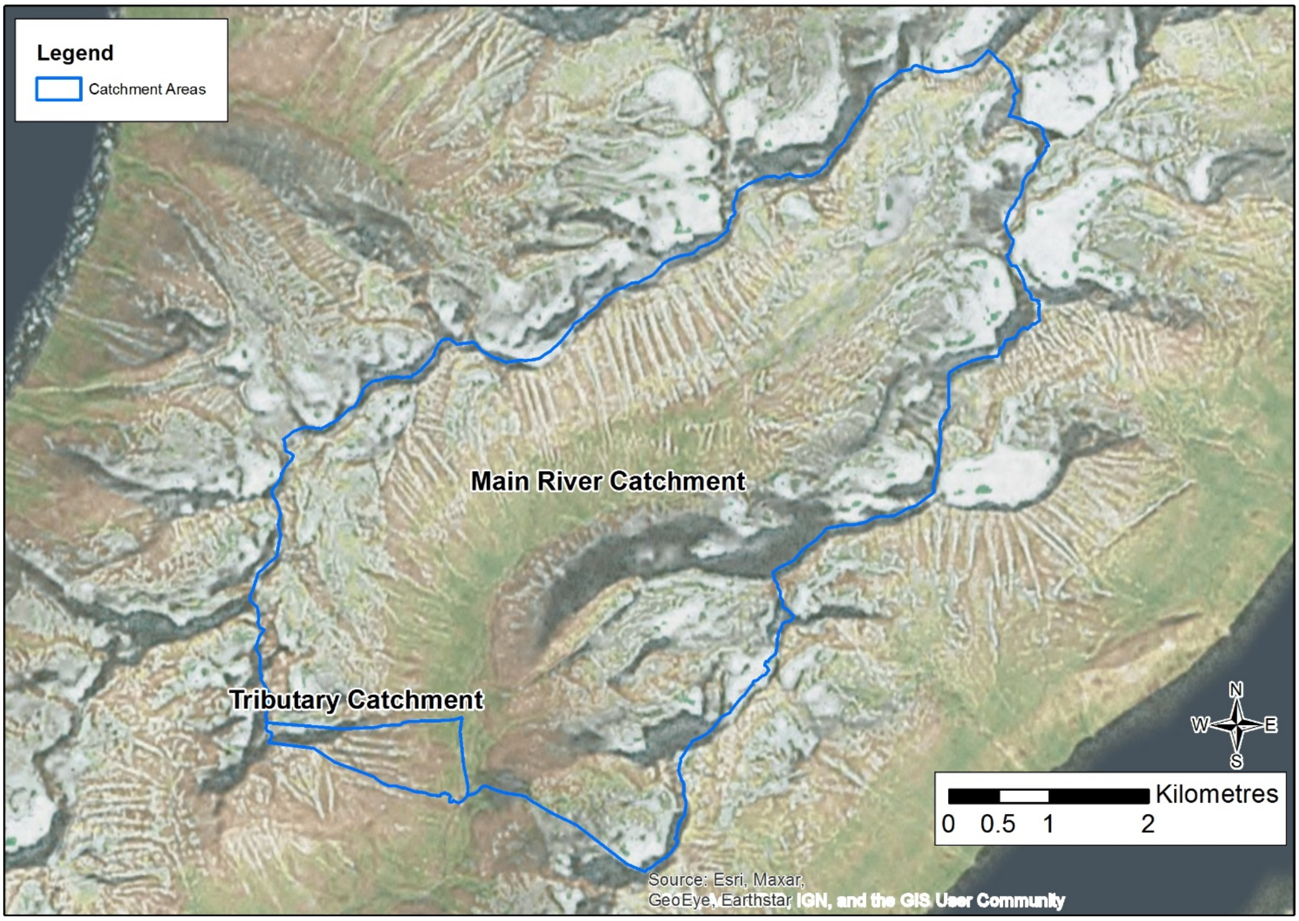

Figure 5: Sub-Catchment Extents

For each catchment, the surface area and time of concentration were assessed using the NRCS watershed Lag Method (USDA, 2010). The NRCS method uses Curve Numbers (which represent the watershed's soil and cover conditions) to define infiltration loss. In the absence of soil hydraulic conductivity data, a Hydrologic Soil Group of C was assumed for the catchments.

Based on a visual assessment of the landcover, Kirkespirdalen is predominantly composed of scree and grass. The curve numbers used to represent each land type are presented below in Table 8, and were selected based on guidance provided in Chow *et al* (1988).

Table 8: Land Type Properties

Land Type	Curve Number	Estimated Coverage (%)	Basis (1)
Grass	74	20 %	Grass, good condition, >75% cover
Scree	89	80 %	Gravel, soil group C

NOTES: (1) Based on land type description provided in Chow et al (1988).

An area-weighted average curve number of 86 was calculated using the estimated 80% to 20% ratio of scree to grass. The sub-catchment topographical properties are provided in Table 9.

Table 9: Sub-catchment Properties

Sub-Catchment	Area (km^2)	Longest Flow Path (km)	Average Land Slope (%)	Time of Concentration (mins)	Lag Time (mins)
Tributary Catchment	1.2	3.2	20.4	37.8	22.7
Main River Catchment	30.9	10.7	7.9	159.5	111.7
Combined Catchment	32.1	10.7	7.9	159.5	111.7

4.4.2 Runoff Rates

The US Army Corps of Engineers Hydrologic Engineering Centre's Hydrologic Modelling System (HEC-HMS) software (USACE 2020) was used to develop flow rate hydrographs for a range of return periods for each sub-catchment. The Soil Conservation Service (SCS) Curve Number loss method and the SCS Unit Hydrograph transform method were applied in the model. The SCS Unit Hydrograph method uses an empirical model to interrogate the relationship between excess rainfall and runoff.

In addition to the Curve Numbers described above, the hydrological model also requires the following parameters:

- Sub-catchment Properties (refer Table 9 above):
  - Catchment Area;
  - Curve Number;
  - Percent Impervious (assumed to be 0%); and
  - Lag Time (length of time between the centroid of precipitation mass and the peak flow; this was assumed to be 60% of time of concentration).
- Meteorological Data:
  - For storms ranging from the 1-in-2 year (50% AEP) to the 1-in-1000 year (0.1% AEP) event precipitation data was distributed using the Frequency Storm method in HEC-HMS. The use of this method required sub-daily precipitation depth estimates, presented in Appendix A. A 3-hour storm event was deemed representative based on the time of concentration of the contributing catchment(s).
  - For the PMP, values from Table 7 were interrogated using the Hypothetical Storm method in HEC-HMS and distributed according to the SCS Type 1 distribution. The SCS Type 1 distribution was chosen as it is representative of typical storm events in the Alaskan region (Chow, 1988). Due to the lack of available sub-daily data in Greenland, the use of this distribution for this approach was deemed appropriate.

Table 10 presents the peak runoff rates of the HEC-HMS model runs for a range of return periods.

Table 10: Peak Runoff Rates for the Design Range of Return Periods

Return Period (Years)	Peak Flow (m^3/s)		
	Tributary Catchment	Main River Catchment	Combined Catchment[(a)]
2	1.5	18.1	18.8
5	3.5	38.1	39.6
10	5.1	53.7	55.8
25	7.2	75.1	77.9
50	8.9	92.2	95.8
100	10.7	110.2	114.5
200	12.5	128.9	133.8
500	15.0	154.7	160.7
1,000	17.0	175.0	181.7
PMP	51.6	598.1	621.1

NOTES: (a) It is noted that the peak flows calculated for the Combined Catchment are not a direct summation of the peak flows for the Tributary and Main River catchments. This is a function of the hydrological characteristics of the contributing catchment areas and their respective response time in the same storm event. The smaller tributary is "flashier" and water levels will rise and fall much more quickly than the main river, and therefore the peak flows will not coincide.

5.0 HYDRAULIC MODELLING

5.1 Modelling Objectives

The assessment of the flood risk to the Site was carried out using a 2-D unsteady-state hydraulic model to simulate flows in the Kirkespir River. The model was developed using the U.S. Army Corps of Engineers' Hydraulic Engineering Center River Analysis System (HEC-RAS) (USACE 2016) Version 5.0.7.

The objectives of the hydraulic modelling were to:

- Assess the flood risk to the mine site surface facilities from the Kirkespir River; and
- Support the Options Analysis of the proposed mine site facility layouts.

5.2 Simulated Scenarios

The model was run for both existing and developed site-layouts, for the following return periods:

- 1-in-2 year (50% AEP);
- 1-in-10 year (10% AEP);

- 1-in-100 year (1% AEP);
- 1-in-200 year (0.5% AEP);
- 1-in-1000 year (0.1% AEP); and
- Probable Maximum Flood (PMF).

Climate change was not considered as part of this assessment. However, considering the short (5 to 10 year) life of mine for the facility, historical climate conditions are deemed as adequate for the basis of the climate assessment.

A hydraulic model was developed to interrogate the hydraulic response of the Kirkespir River under "existing" site conditions, i.e. ahead of the construction of the proposed mine site infrastructure.

A further hydraulic model was then constructed in order to interrogate the following "developed" scenarios:

- Original layout consisting of the "Original" DTSF layout (refer to Section 2.1 above), accommodating a 5 year deposition schedule, the Process Plant, and the Ore Pad located to the east of the plant.
- Updated layout consisting of the "Updated" DTSF layout (refer to Section 2.1 above), accommodating a narrower footprint for the 5 year deposition schedule, the Process Plant, and the Ore Pad located to the west of the plant.

For each of these scenarios, the following ground conditions were then assessed:

- Current ground conditions – In this instance, it has been assumed that the facilities will be constructed on the existing site (i.e. including the existing camp platform) without any reprofiling and/or removal of compacted material.
- Regraded ground conditions – In this instance, it has been assumed that any areas with compacted material will be removed and the underlying ground conditions returned to mimic the natural riverbed (i.e. loose gravel).

5.3 Model Inputs

5.3.1 Simulation Parameters

Simulation parameters employed for the model runs are presented in Table 11 below.

Table 11: Model Run Simulation Parameters

Model Simulation Parameter	Value
Model Maximum Timestep	10 seconds (variable, controlled by courant condition)
Model ramp-up duration	6 hours
Model simulation duration	48 hours

5.3.2 Boundary Conditions

The model extent is shown in Figure 6. The upstream boundary of the model extends approximately 300 m upstream of the proposed DTSF, while the downstream boundary extends approximately 600 m downstream of the proposed Process Plant.

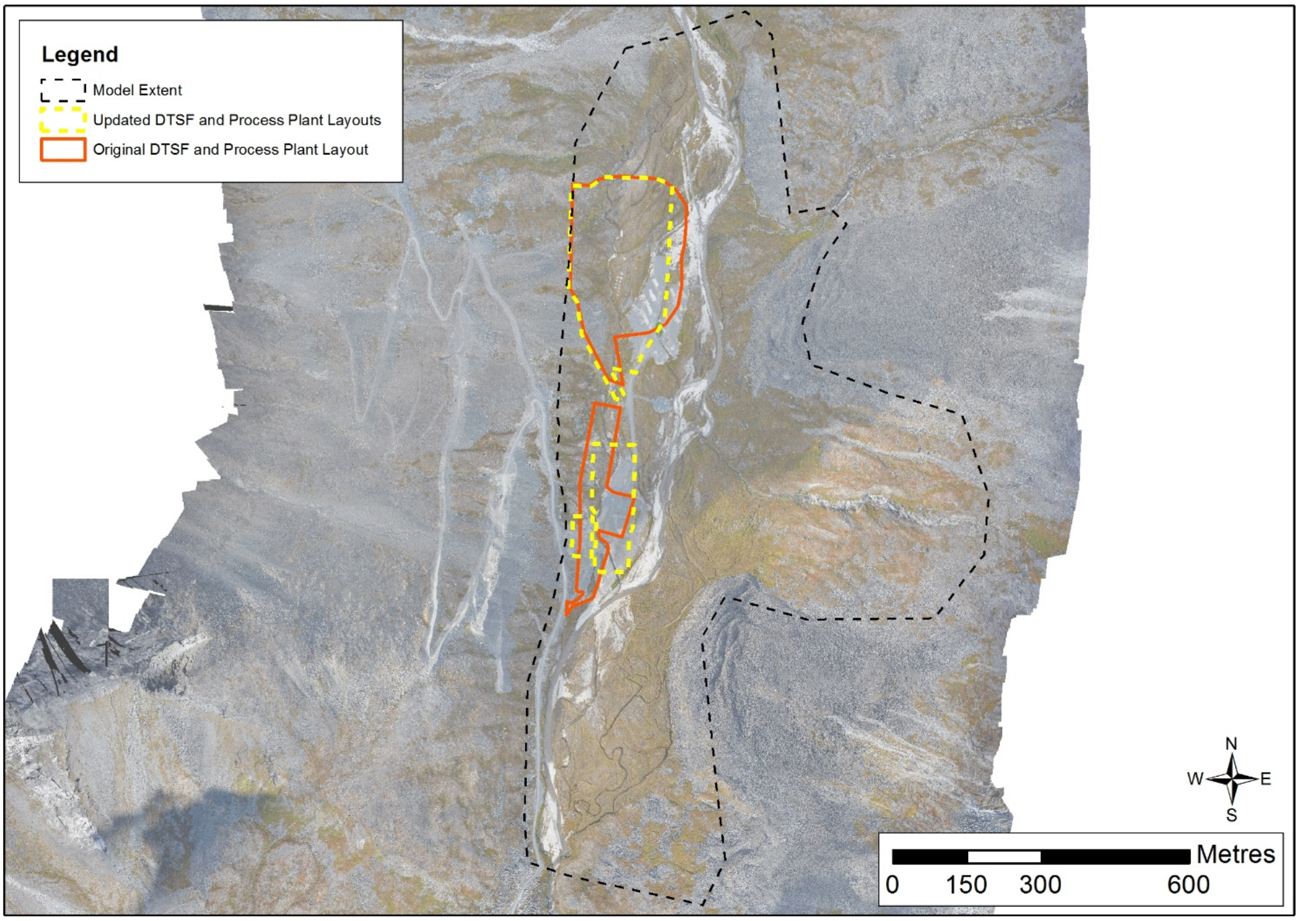

Figure 6: Model Extent

5.3.3 Inflow Hydrographs

To ensure model stability, the inflow hydrographs used in the models allowed for a gradual ramp-up of flows, starting from an initial flow of $1 m^3/s$ to the derived peak flows (Table 10) over a 24-hour simulated run-time. The model was then run at the peak flow rate for a further 24 hours of simulated run time to allow for any numerical instability within the model to be eliminated. The model results were then obtained based on the results at the end of the 48-hour model run.

5.3.4 Kirkespir River Terrain Data

The 2-D geometry of the Kirkespir River and the associated floodplain was obtained from the 0.6 m resolution LiDAR data. Manning's n values were assigned to the various ground surfaces as shown in Table 12 below.

Table 12: Assigned Manning's n Coefficients (Sturm, 2001)

Land Type	Assigned Manning's n
Riverbed and Floodplain (braided & winding stream characterised by a highly mobile gravel riverbed)	0.050
Existing Camp platform (compacted gravel)	0.011

5.4 Model Results

5.4.1 Existing (Pre-Construction) Site Conditions

Flood maps showing maximum flood depths and maximum flow velocities under existing (pre-construction) site conditions are presented in Appendix B.

It should be noted that the entire valley bottom is at risk of flooding, even under the lower order (more frequent) 1-in-2 year and 1-in-5 year events.

5.4.2 Localised Flood Risk (Tributary Flooding)

As mentioned previously in Section 4.4.1, a minor tributary flows through the proposed footprints of both the DTSF and Process Plant. As such, an analysis has been undertaken to interrogate the potential risk of flooding from the tributary under existing site conditions (i.e. prior to any development). Two (2) key points of interest have been selected within the tributary channel, i.e.:

- The first point of interest (POI) is just located upstream of where the proposed DTSF is to be constructed.
- The second POI is located upstream of where the proposed Process Plant[2] option is to be constructed.

To facilitate an understanding of depths and velocities in the tributary in its present state (i.e. assuming that it is not diverted around the facilities), peak depths and velocities are presented in Table 13 for the first POI (upstream of the proposed DTSF location) and Table 14 for the second POI (upstream of the proposed Process Plant location).

Table 13: Minor Tributary Flow Characteristics - Existing Site Conditions – First POI, i.e. Upstream of Proposed DTSF Layouts

Design Event	Design Flow Rate, m^3/s	Maximum Depth, m	Maximum Velocity, m/s
1-in-2 year	18.8	0.3	0.4
1-in-5 year	39.6	0.6	1.0
1-in-100 year	115	0.93	1.9
1-in-200 year	134	0.96	2.0
1-in-1000 year	182	1.0	2.2
PMF	621	1.5	3.2

[2] This point of interest is 10 m upstream of the Original Process Plant Layout and 80 m upstream of the Updated Process Plant Layout

Table 14: Minor Tributary Flow Characteristics - Existing Site Conditions – Second POI, i.e. Upstream of Proposed Process Plant Layouts

Design Event	Design Flow Rate, m^3/s	Maximum Depth, m	Maximum Velocity, m/s
1-in-2 year	18.8	0.6	1.0
1-in-5 year	39.6	1.0	1.7
1-in-100 year	115	1.7	2.9
1-in-200 year	134	1.8	3.0
1-in-1000 year	182	1.9	3.2
PMF	621	2.4	3.7

It is evident that the tributary flow depths and velocities upstream of the proposed Process Plant are higher than those upstream of the proposed DTSF. This is due to an existing ridge, on which a mine road has been constructed, which constrains the flow path in the proposed Process Plant area.

5.4.3 Developed (Post-Construction) Site Conditions

5.4.3.1 Original Facilities Layout

Selected model outputs, consisting of predicted flood depths and flow velocities, were extracted from the hydraulic model for the “Original” DTSF and Process Plant layouts (these layouts are described in Section 5.2). The results are presented for both current ground conditions and regraded ground conditions. As described above, the regraded ground conditions account for the camp platform being removed and the ground covered with loose gravel to mimic river conditions.

The hydraulic model outputs were extracted for the 1-in-100 year, 1-in-200 year, 1-in-1000 year and PMP events, and are presented in Table 15 and Table 16 for the proposed DTSF and Process Plant layouts, respectively.

it is noted that the predicted depths under “current” and “regraded” ground conditions are the same for the DTSF. The maximum velocities presented for the DTSF and Process Plant under “current” ground conditions are localised velocities that will be experienced at the base of these facilities in the vicinity of the now disused camp platform.

Table 15: Key Model Outputs – Original Layout - DTSF

Design Event	Design Flow Rate, m^3/s	Ground Conditions	Predicted Depth, m	Maximum Velocity, m/s
1-in-100 year	115	Current	1.6	4.5
		Regraded		2.3
1-in-200 year	134	Current	1.7	4.5
		Regraded		2.4
1-in-1000 year	182	Current	1.9	5.6

Design Event	Design Flow Rate, m^3/s	Ground Conditions	Predicted Depth, m	Maximum Velocity, m/s
		Regraded		2.5
PMF	621	Current	3.1	10.3
		Regraded		4.5

Table 16: Key Model Outputs – Original Layout - Process Plant

Design Event	Design Flow Rate, m^3/s	Ground Conditions	Predicted Depth, m	Maximum Velocity, m/s
1-in-100 year	115	Current	0.5	1.2
		Regraded	0.6	0.3
1-in-200 year	134	Current	0.6	1.6
		Regraded	0.7	0.4
1-in-1000 year	182	Current	0.7	2.1
		Regraded	0.9	0.52
PMF	621	Current	1.6	7.8
		Regraded	2.0	1.4

The historical platform that housed the previous mine camp (i.e. the “current” ground conditions) reflects a hydraulic constraint in the riverbed, causing the flow to speed up as it passes over the raised and compacted land form. For this reason, the maximum velocities presented for DTSF and Process Plant above are localised velocities that will be experienced at the base of these facilities in the vicinity of the raised and compacted camp platforms under current ground conditions. If however this platform is regraded and uncompacted (i.e. with surface conditions similar to the natural riverbed and floodplain) the velocities presented for the regraded ground conditions will no longer be constrained across the toe of the facility.

The flood maps for each of the scenarios presented in Table 15 and Table 16 are presented in Appendix C. The predicted flood depths and maximum velocities are presented on the figures for the 1-in-100 year event through to the PMP event.

5.4.3.2 Updated Facilities Layout

Selected model outputs, consisting of predicted flood depths and maximum flow velocities, were extracted from the hydraulic model for the “Updated” DTSF and Process Plant layouts (these layouts are described in Section 5.2). Similarly to the results presented in Section 5.4.3.2, the results are presented for both current ground conditions and regraded ground conditions.

These outputs were extracted for the 1-in-100 year, 1-in-200 year, 1-in-1000 year and PMP events, and are presented in Table 17 and Table 18 for DTSF and Process Plant layouts, respectively. once again it is noted that for the DTSF, the predicted depths under "current" and "regraded" ground conditions are the same. As explained previously, the maximum velocities presented for DTSF and Process Plant under "current" ground conditions are *localised* velocities that will be experienced at the base of these facilities in the vicinity of the now disused camp platforms.

Table 17: Key Model Outputs – Updated Layout - DTSF

Design Event	Design Flow Rate, m^3/s	Ground Conditions	Predicted Depth, m	Maximum Velocity, m/s
1-in-100 year	115	Current	1.2	6.3
		Regraded		2.1
1-in-200 year	134	Current	1.3	7.9
		Regraded		2.2
1-in-1000 year	182	Current	1.5	8.5
		Regraded		2.4
PMF	621	Current	2.7	14.3
		Regraded		3.8

Table 18: Key Model Outputs – Updated Layout - Process Plant

Design Event	Design Flow Rate, m^3/s	Ground Conditions	Predicted Depth, m	Maximum Velocity, m/s
1-in-100 year	115	Current	0.4	2.7
		Regraded	0.5	0.9
1-in-200 year	134	Current	0.5	3.6
		Regraded	0.6	1.0
1-in-1000 year	182	Current	0.6	4.6
		Regraded	0.8	1.2
PMF	621	Current	1.5	6.0
		Regraded	1.9	2.7

The DTSF experiences lower depths under the “Updated” facilities layout when compared to the depths experienced under the “Original” facilities layout. This is driven by the widening of the flow path in the “Updated” facilities layout, resulting in less constraint of the flow as it moves past the DTSF.

The flood maps for each of the scenarios presented in Table 17 and Table 18 are presented in Appendix D. The predicted flood depths and maximum velocities are presented for the 1-in-100 year event through to the PMP event.

5.5 Model Validation

No data was available for quantitatively calibrating or validating the model results. However, a flood event occurred in 2008 for which pictures were taken at various points throughout the mine site. These pictures are presented in Appendix E and show flooding extending to the compacted camp platform. Site-specific data was not available for the period during which the storm event occurred, however based on an analysis of rainfall from the Narsarsuaq Station, this rainfall event may have approached a 1-in-5 year event (20% AEP). Model results for baseline conditions for the associated return period also show overbank flooding extending to the compacted camp platform, and so it can be inferred that any infrastructure along the base of the old camp pad is at risk of flooding, even under relatively high-frequency events.

6.0 CONCLUSIONS AND RECOMMENDATIONS

A flood risk assessment was carried out for the Nalunaq Mine, in the Municipality of Kujalleq, Greenland. The assessment considered both existing site conditions as well as developed site conditions, accounting for various proposed layouts for the proposed Dry Tailings Stack Facility (DTSF) and the Process Plant, under various ground surface conditions. The ground surface conditions considered “current” ground conditions (i.e. in which the compacted (now disused) camp platform areas remain intact), and “regraded” ground conditions (i.e. in which any areas with compacted material be removed and the underlying ground conditions returned to mimic the natural riverbed). These layouts were assessed under various climate scenarios, with key results reported for the 1-in-100 year, 1-in-200 year, 1-in-1000 year and Probable Maximum Precipitation (PMP) conditions.

Based on this assessment, the following key conclusions were made:

- The entire valley bottom is at risk of flooding, even under high-frequency (low return period) events for both existing site conditions and developed site conditions.
- Key results for the DTSF during a Probable Maximum Flood are as follows:
  - A maximum flood depth of 3.1 m and a maximum flow velocity of 4.5 m/s can be expected for the proposed “Original” DTSF facility layout (regraded ground conditions). Localised velocities as high as 10.3 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.
  - A maximum flood depth of 2.7 m and a maximum flow velocity of 3.8 m/s can be expected for the proposed “updated” DTSF facility layout (regraded ground conditions). Localised velocities as high as 14.3 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.
- Key results for the Process Plant during a Probable Maximum Flood, are as follows:
  - A maximum flood depth of 2.0 m and a maximum flow velocity of 1.4 m/s can be expected for the “Original” Process Plant facility layout, assuming that the current camp pad is regraded and uncompacted. Localised velocities as high as 7.8 m/s can be expected at the base of the facility if the now disused camp platform is not regraded.

  - A maximum flood depth of 1.9 m and a maximum flow velocity of 2.7 m/s can be expected at the proposed "Updated" Process Plant facility layout (regraded ground conditions). Localised velocities as high as 6.0 m/s can be expected at the toe of the facility if the now disused camp platform is not regraded.

The following actions are recommended:

- Continuous monitoring of the Kirkespir River, as well as the highlighted tributary reporting to the river.
- Regrading of the raised camp platform areas to reduce localised velocities at the base of the facilities.
- Selection of the Updated DTSF Layout in order to reduce the potential obstruction to natural river flows within the Kirkespir River during flooding events.
- Selection of the Updated DTSF Layout in order to reduce the potential risk to the facility as a result of fluvial (river) flooding and scour.
- Establishing a platform elevation beneath the DTSF and Processing Plant that is situated above the predicted 1 in 1000 year (0.1% AEP) flood level.
- Installation of a warning system that provides immediate warning to the Site in the event of a large flood event.

There are several uncertainties in conducting flood risk assessments, as highlighted in this report, including the determination of representative climate conditions as well as the evaluation of catchment characteristics. This assessment incorporated available data and engineering judgment to inform inputs to the modelling exercise. In the event of any future changes to the proposed facility layouts, a reassessment may be required.

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Signature Page

Golder Associates (UK) Ltd

Karen Dingley
Principal Water Resources Engineer

Gareth Digges La Touche
Project Director

Jo Birch
Project Manager

Date: 14 January 2021

JR/KD/cr

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APPENDIX A

Derived Sub-daily Precipitation Depths

Due to the lack of sub-daily rainfall data, the calculated Annual Maximum Daily Rainfall depths required downscaling to allow assessment of sub-daily rainfall events. Several methods were considered, including the Bell (1969), Wild (1982) and Herschfield (1961) methods, each described in Adamson and Chong (1992).

The Wild method was discounted as it was developed using data collected at one rain gauge in a tropical environment. The Bell method was developed considering extreme storm rainfall patterns from various parts of the world but requires a locally specific empirical coefficient. As no coefficient was available for Greenland or any other Arctic region, this method was also found to be unsuitable.

The Herschfield method was therefore found to be the most applicable. The method was developed from storm rainfall in the US but, as it reflects a wide range of climates from tropical to arid, it has found application in many other parts of the world. Due to the region's limited rainfall, the Site may be classified as 'semi-arid'. This assumption allowed Golder to select appropriate downscaling ratios to calculate sub-daily rainfall. The selected ratios are presented in Table A1.

Table A1: Downscaling Ratios for 1 to 24 hours, Arid/Semi-arid Zones (from Adamson and Chong, 1992).

Storm Duration (Hours)	Downscaling Factor
1	0.40
2	0.50
3	0.62
6	0.80
12	0.95
24	1.00

For sub-hourly rainfall values, the derived hourly rainfall was multiplied by the downscaling ratios recommended by Hershfield (1961) which are presented in Table A2.

Table A2: Downscaling Ratios for 5 to 60 minutes (from Adamson and Chong, 1992).

Storm Duration (Minutes)	Downscaling Factor
5	0.29
10	0.45
15	0.57
30	0.79
60	1.00

Table A3 and Table A4 present the Depth-Duration-Frequency (DDF) and Intensity-Duration-Frequency (IDF) data for the rainfall plus snowmelt data generated through the use of the above-mentioned methods, respectively.

Table A3: Rainfall plus Snowmelt Depth-Duration-Frequency Table (mm)

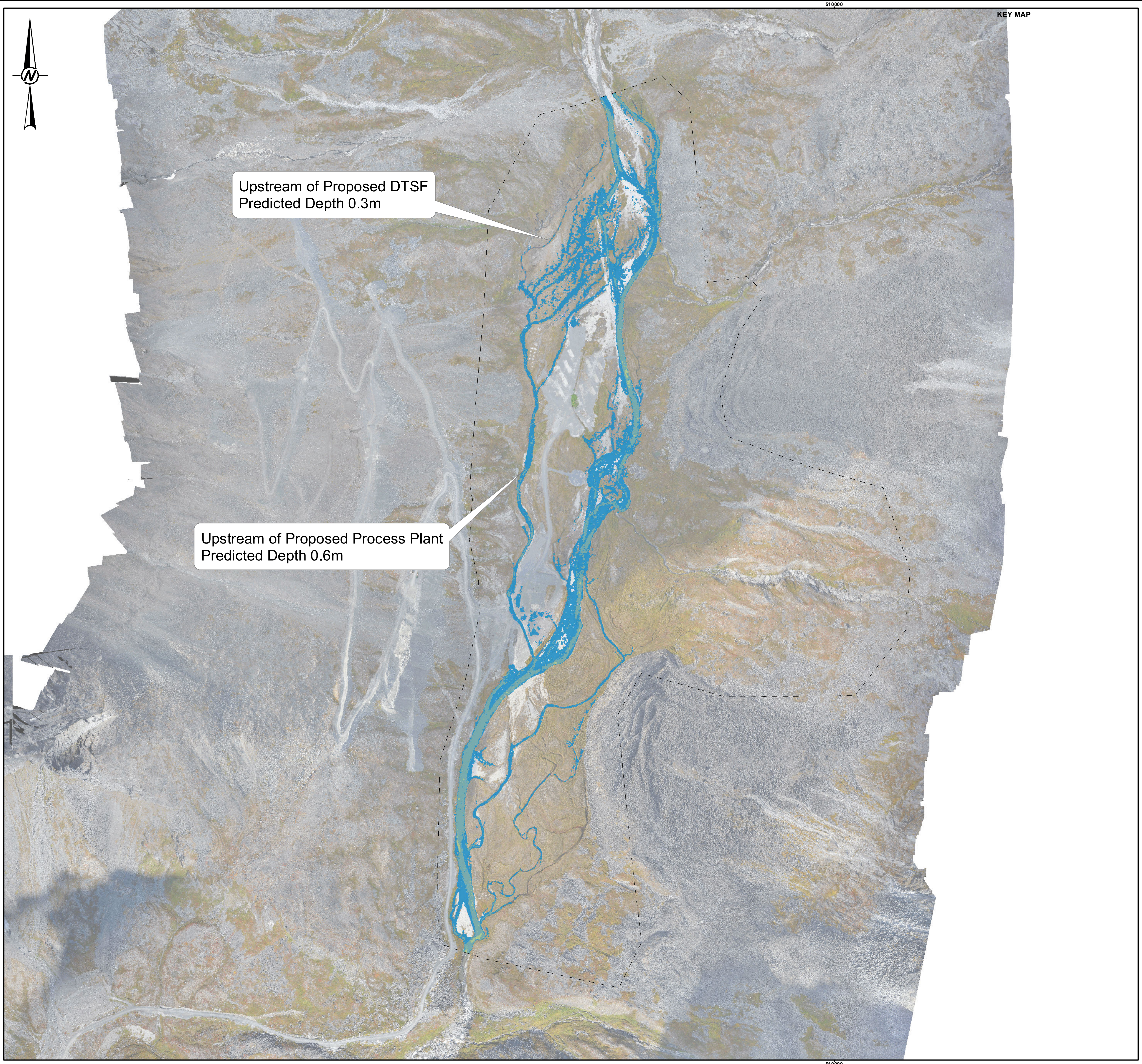
Return Period (Years)	Storm Duration (Hours)									
	0.08	0.17	0.25	0.50	1	2	3	6	12	24
2	4.9	7.6	9.7	13.4	17.0	21.2	26.3	33.9	40.3	42.4
5	6.8	10.6	13.4	18.6	23.6	29.5	36.6	47.2	56.0	59.0
10	8.1	12.6	16.0	22.2	28.0	35.1	43.5	56.1	66.6	70.1
25	9.8	15.2	19.2	26.6	33.7	42.1	52.2	67.4	80.1	84.3
50	11.0	17.1	21.6	30.0	38.0	47.5	58.8	75.9	90.2	94.9
100	12.3	19.0	24.1	33.4	42.2	52.8	65.5	84.5	100.3	105.6
200	13.5	21.0	26.6	36.8	46.6	58.2	72.2	93.2	110.6	116.5
500	15.2	23.6	29.9	41.4	52.4	65.6	81.3	104.9	124.6	131.1
1,000	16.5	25.6	32.5	45.0	57.0	71.2	88.3	114.0	135.3	142.5
10,000	21.1	32.8	41.6	57.6	72.9	91.1	113.0	145.8	173.2	182.3

Table A4: Rainfall plus Snowmelt Intensity-Duration-Frequency Table (mm/hr)

Return Period (Years)	Storm Duration (Hours)									
	0.08	0.17	0.25	0.50	1	2	3	6	12	24
2	59.0	45.8	38.7	26.8	17.0	10.6	8.8	5.7	3.4	1.8
5	82.1	63.7	53.8	37.3	23.6	14.7	12.2	7.9	4.7	2.5
10	97.6	75.7	63.9	44.3	28.0	17.5	14.5	9.3	5.5	2.9
25	117.3	91.0	76.9	53.3	33.7	21.1	17.4	11.2	6.7	3.5
50	132.1	102.5	86.5	60.0	38.0	23.7	19.6	12.7	7.5	4.0
100	147.0	114.1	96.3	66.7	42.2	26.4	21.8	14.1	8.4	4.4
200	162.1	125.8	106.2	73.6	46.6	29.1	24.1	15.5	9.2	4.9
500	182.5	141.6	119.6	82.9	52.4	32.8	27.1	17.5	10.4	5.5
1,000	198.3	153.9	129.9	90.0	57.0	35.6	29.4	19.0	11.3	5.9
10,000	253.8	196.9	166.3	115.2	72.9	45.6	37.7	24.3	14.4	7.6

APPENDIX B

Flood Maps – Existing Site Conditions



LEGEND

Model Extent

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-2 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

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APPROVED	####

PROJECT NO.	CONTROL	REV.	DRAWING
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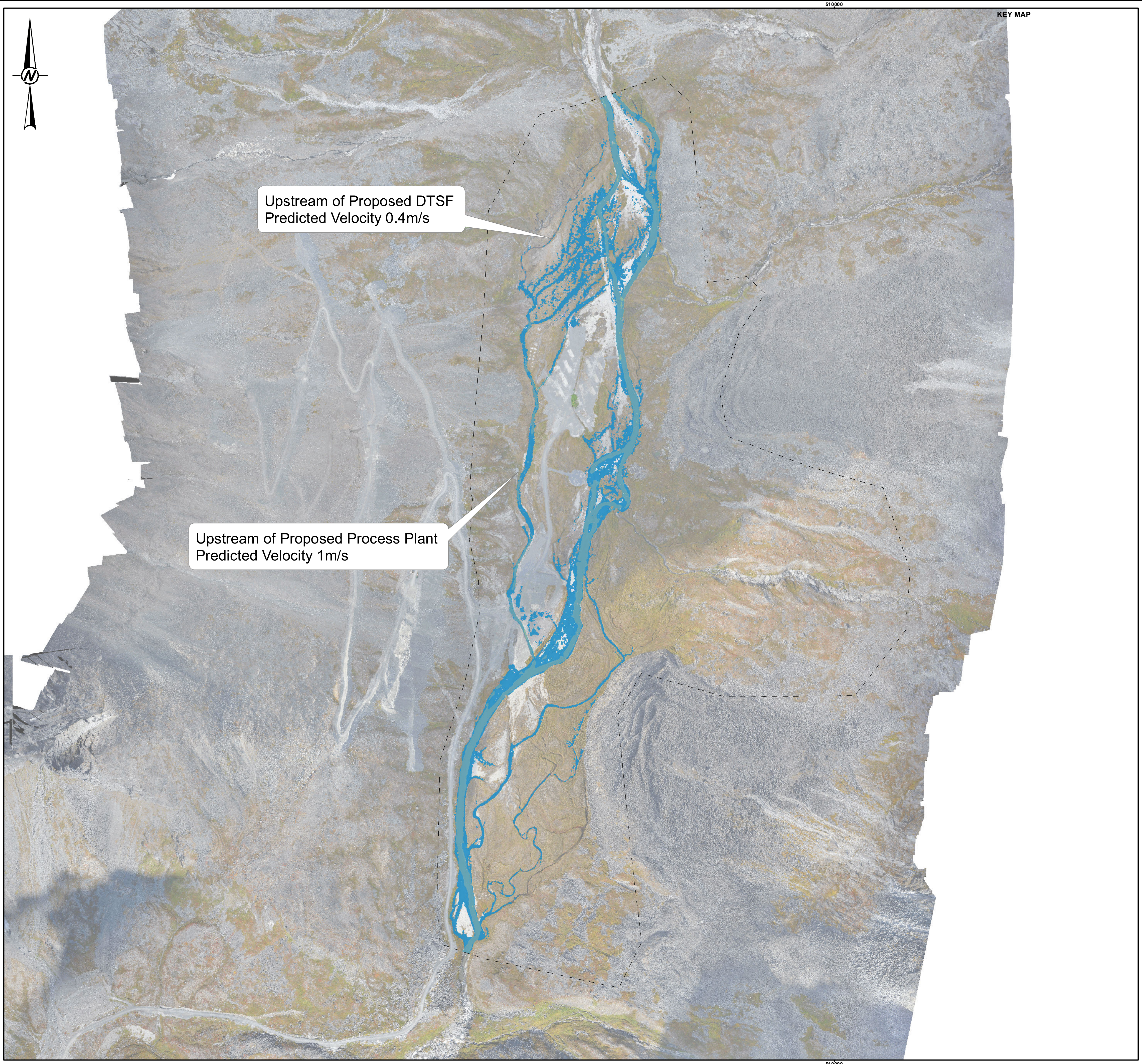

LEGEND

Model Extent

Mazimum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

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NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-2 YEAR EVENT. MAXIMUM VELOCITY.

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PROJECT NO.	CONTROL	REV.	DRAWING
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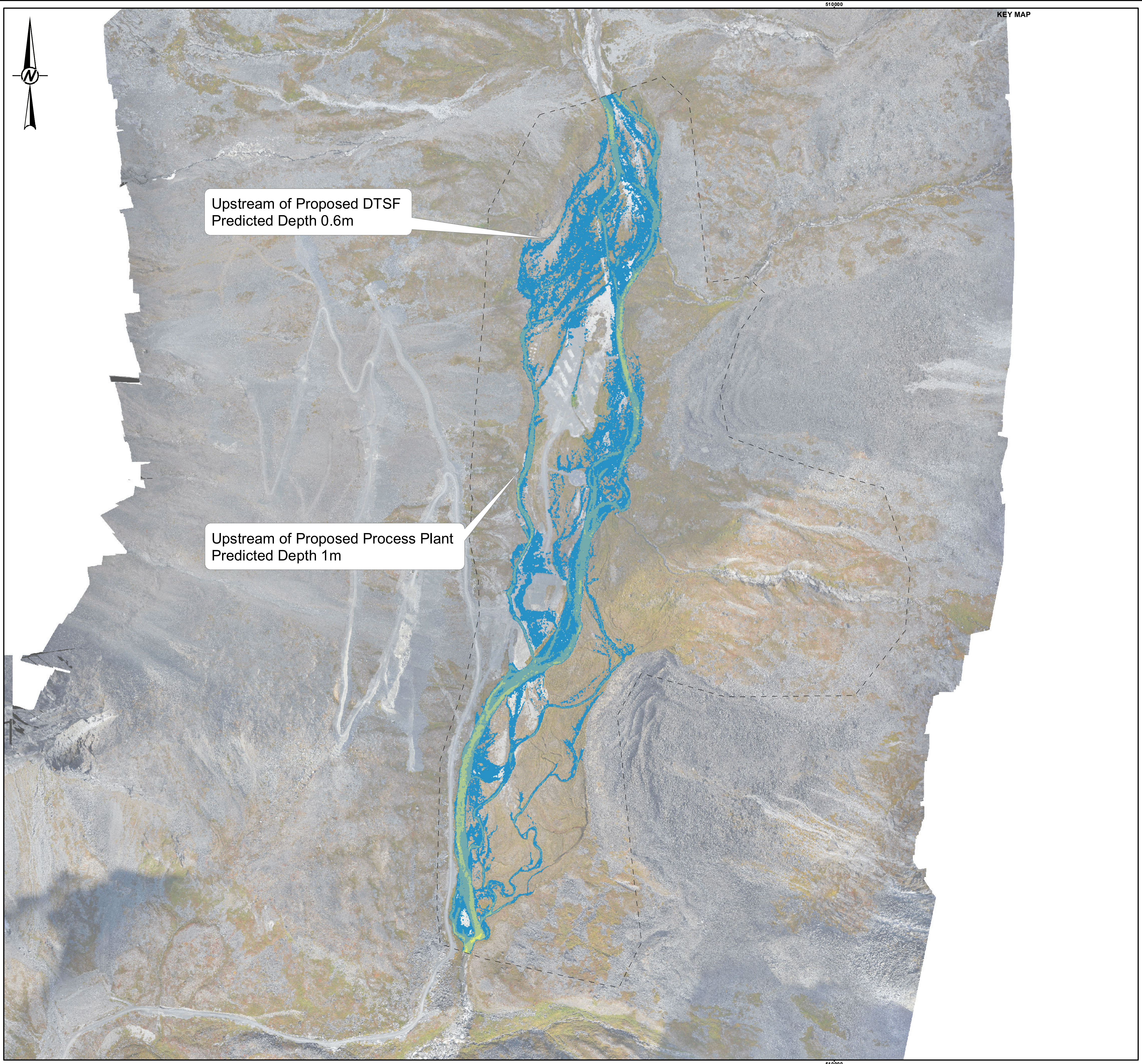

LEGEND

Model Extent

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

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TITLE
EXISTING SITE CONDITIONS, 1-IN-5 YEAR EVENT. MAXIMUM DEPTH.

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REVIEWED	JR
APPROVED	KD

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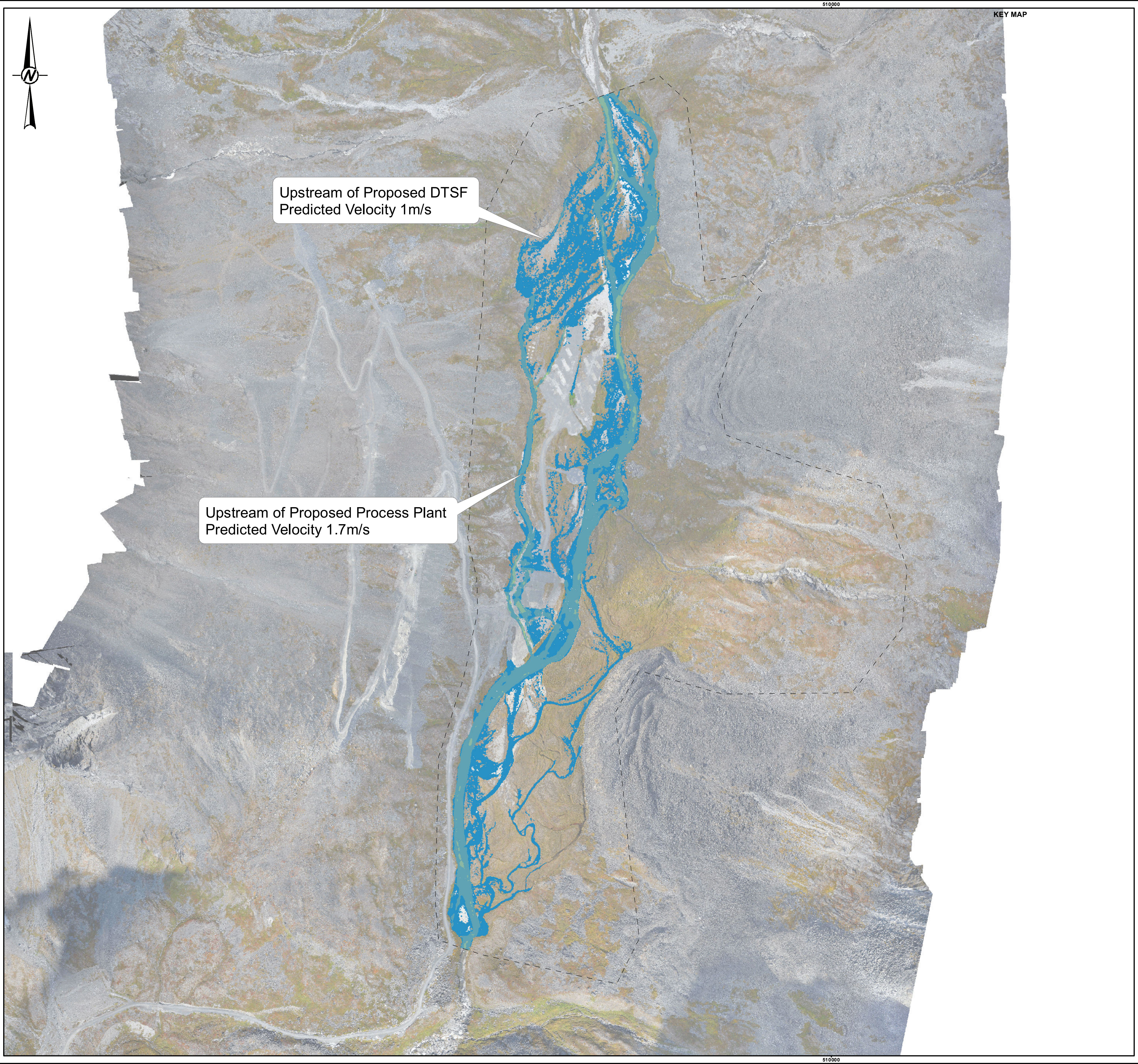

LEGEND

Model Extent

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-5 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

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YYYY-MM-DD	2021 JAN 14
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PREPARED	CB
REVIEWED	JR
APPROVED	KD

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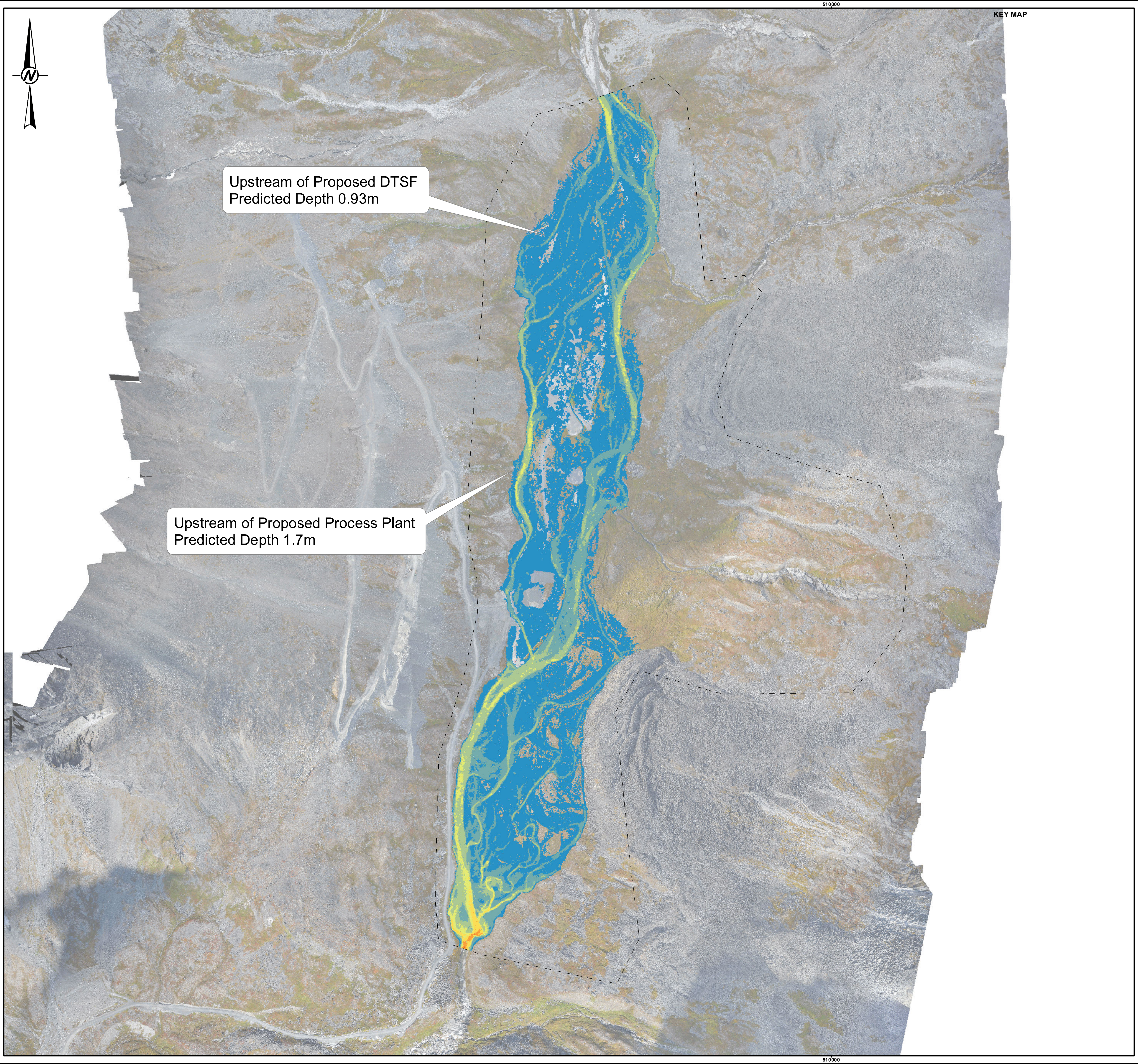

LEGEND

Model Extent

Maximum Depth (m)

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- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
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TITLE
EXISTING SITE CONDITIONS, 1-IN-100 YEAR EVENT. MAXIMUM DEPTH.

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PREPARED	CB
REVIEWED	JR
APPROVED	KD

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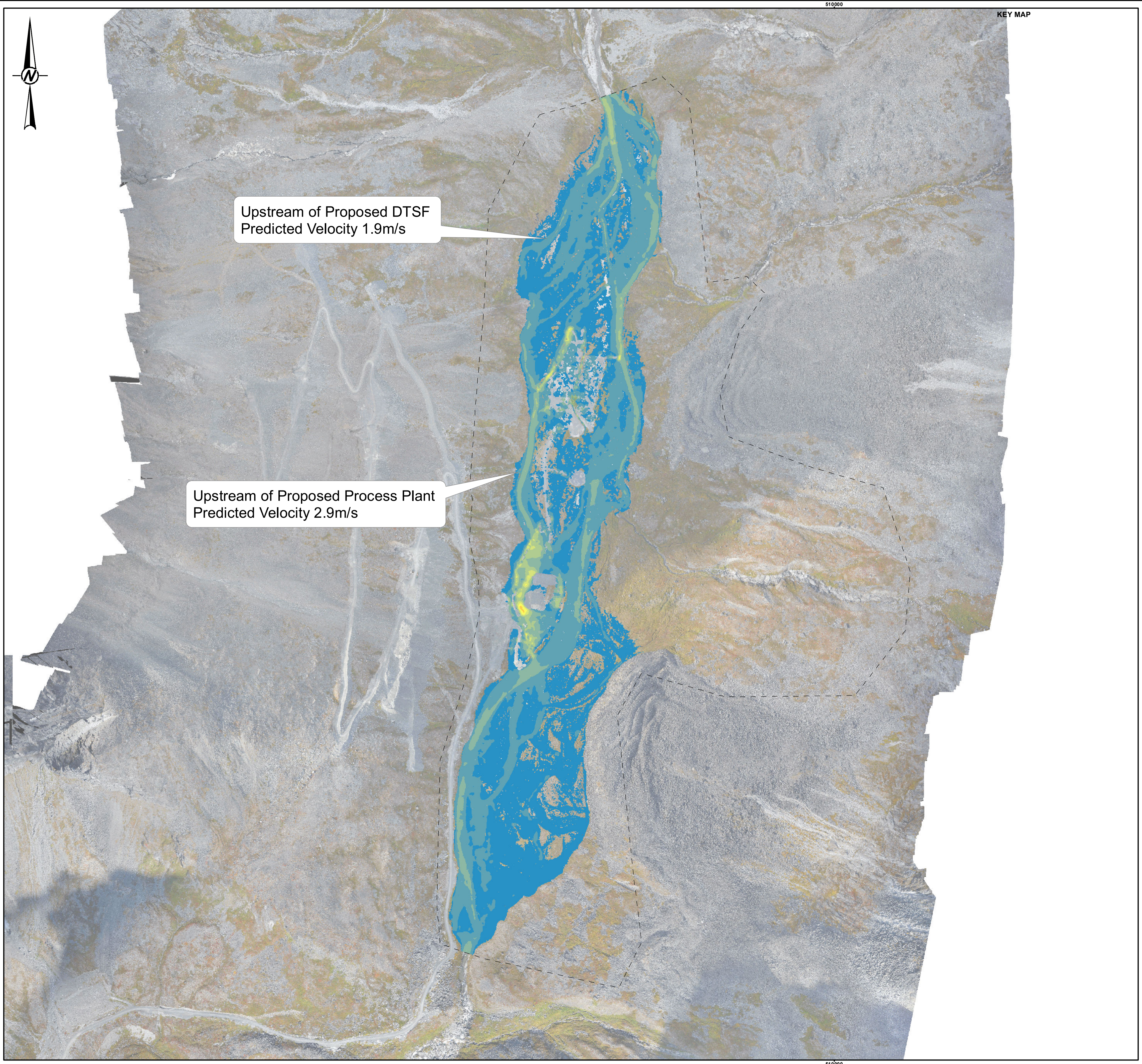

LEGEND

Model Extent

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
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TITLE
EXISTING SITE CONDITIONS, 1-IN-100 YEAR EVENT. MAXIMUM VELOCITY.

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YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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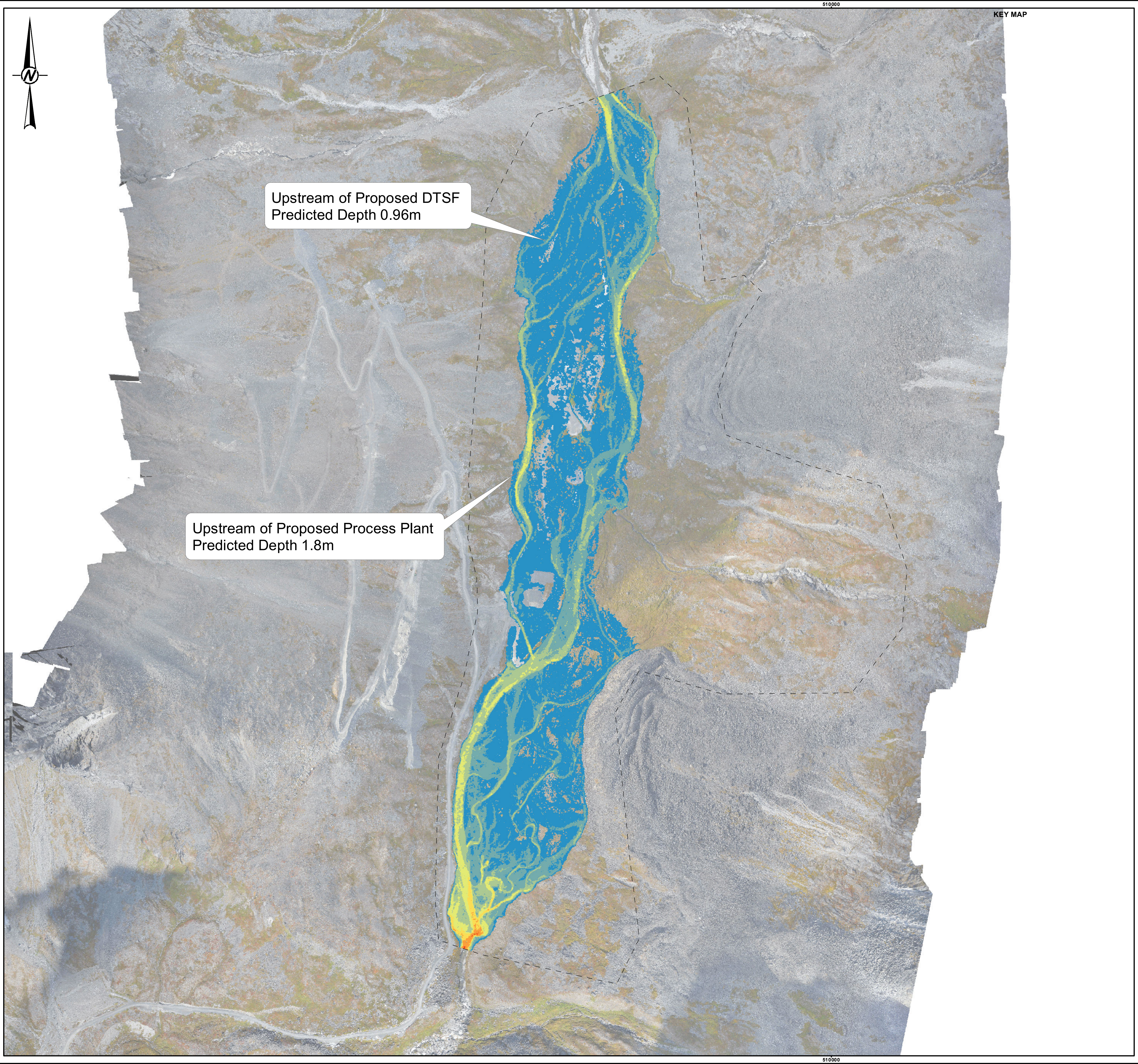

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Model Extent

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT	AEX GOLD
PROJECT	NALUNAQ
TITLE	**EXISTING SITE CONDITIONS, 1-IN-200 YEAR EVENT. MAXIMUM DEPTH.**

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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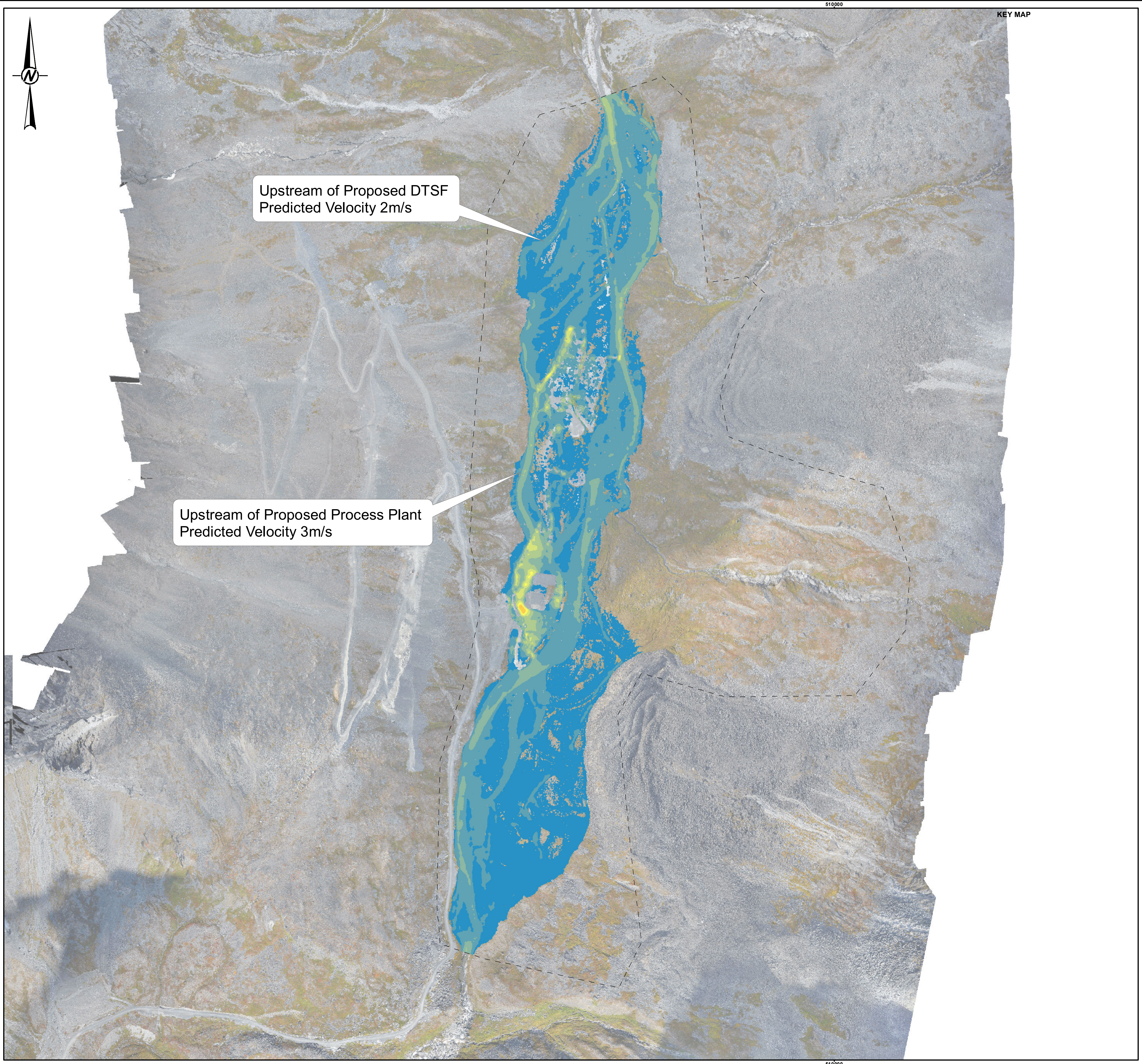

LEGEND

Model Extent

MAximum Velocity (m)

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- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-200 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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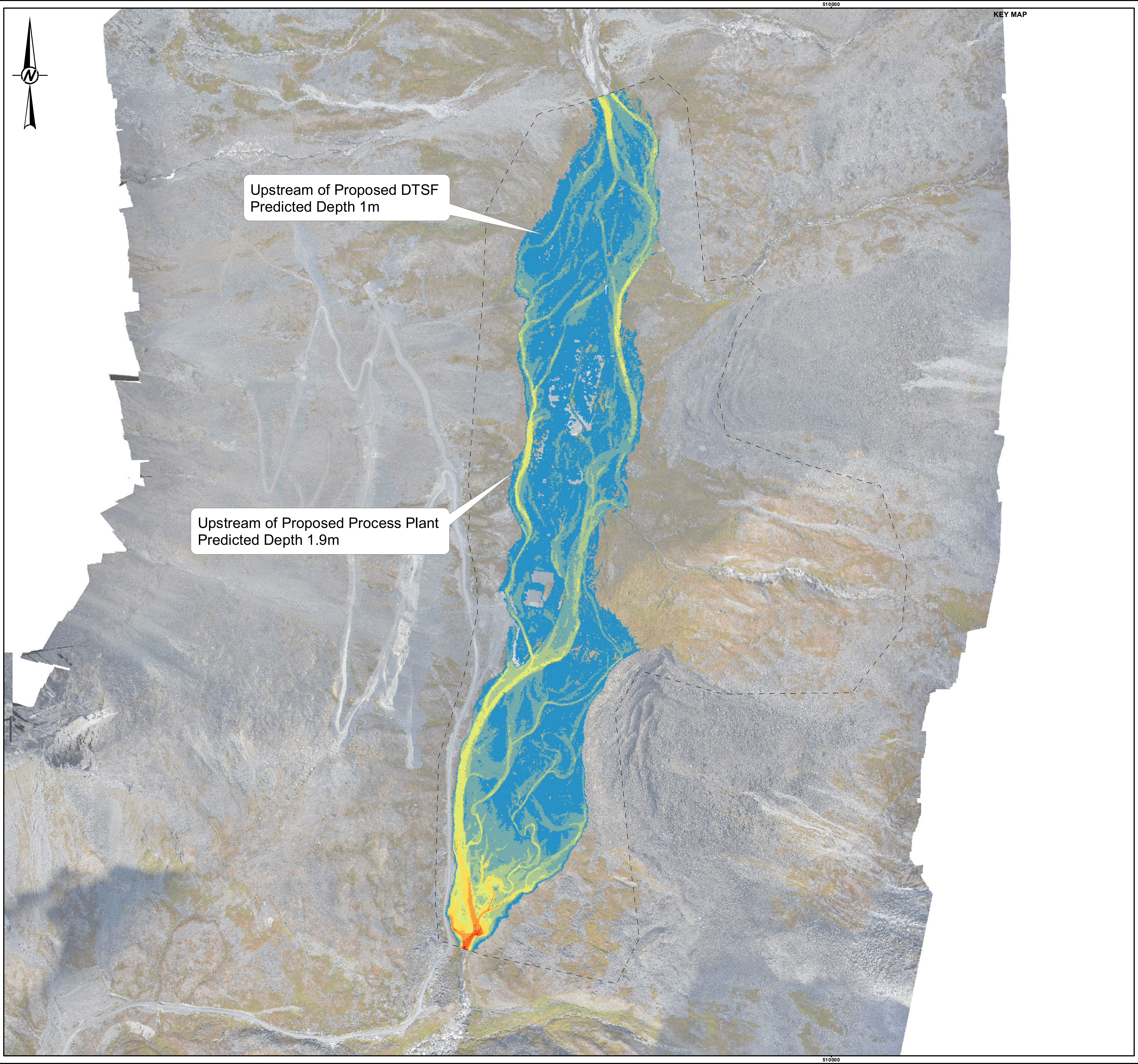

LEGEND

Model Extent

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-1000 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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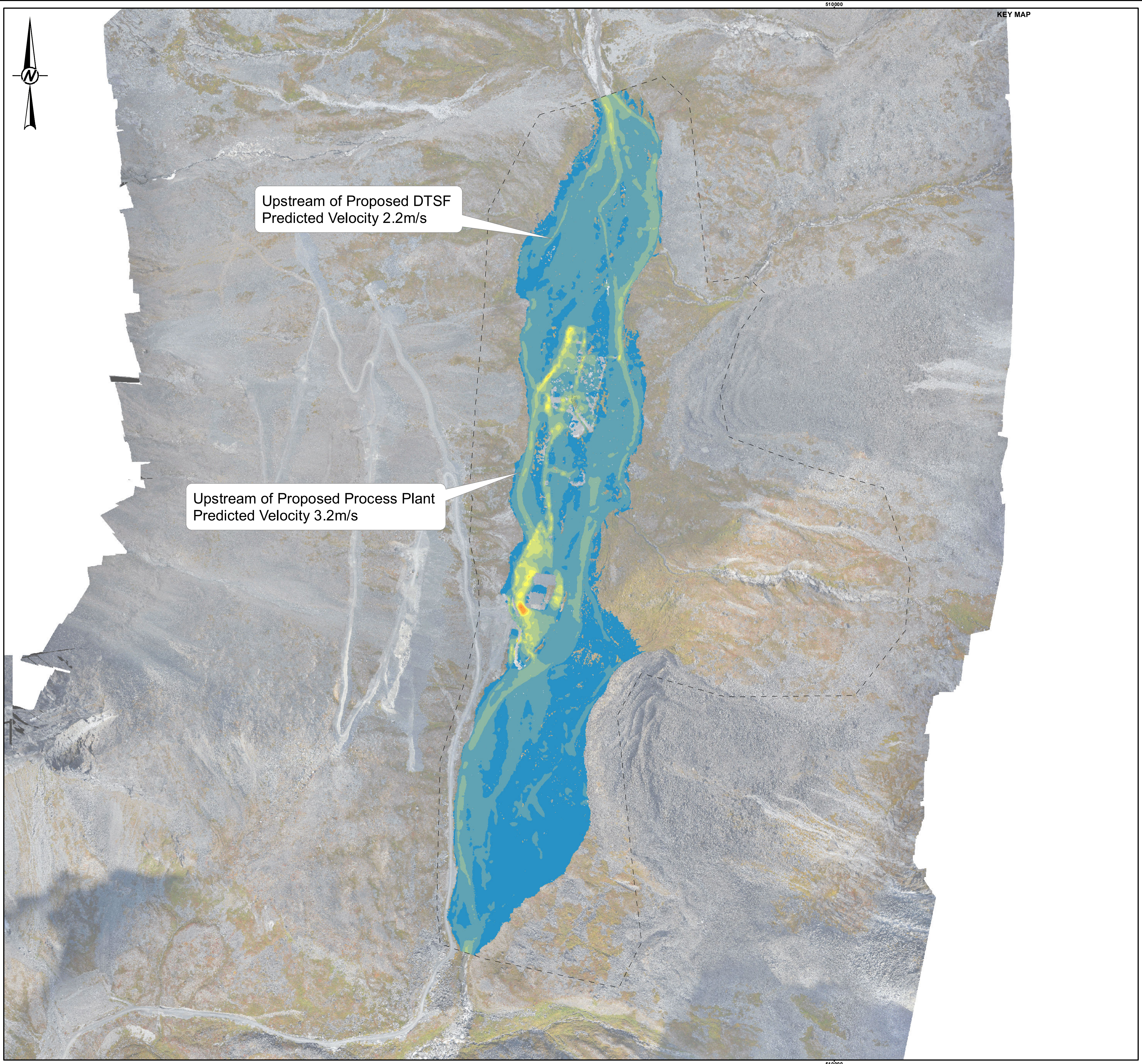

LEGEND

Model Extent

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, 1-IN-1000 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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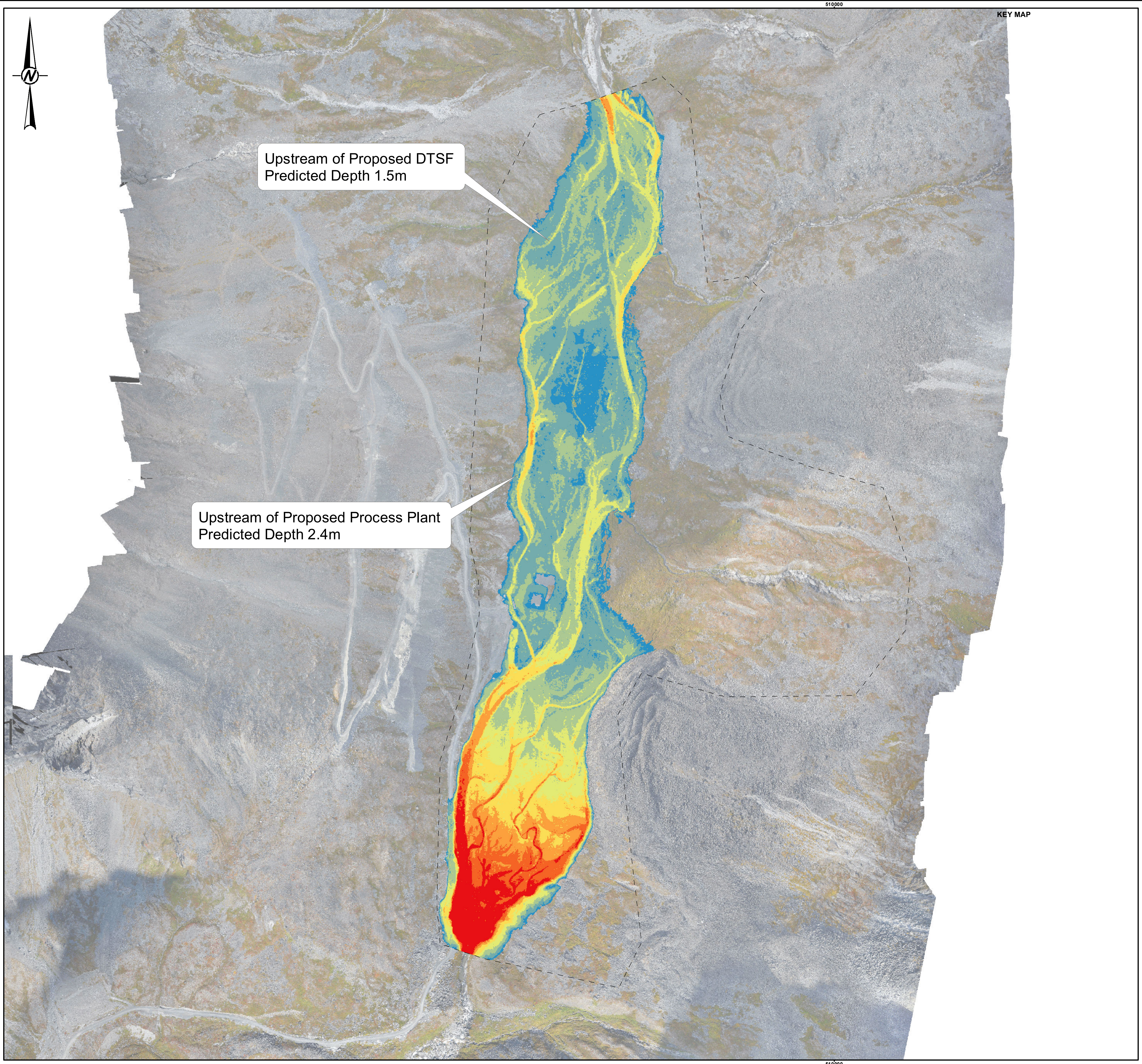

LEGEND

Model Extent

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, PMP EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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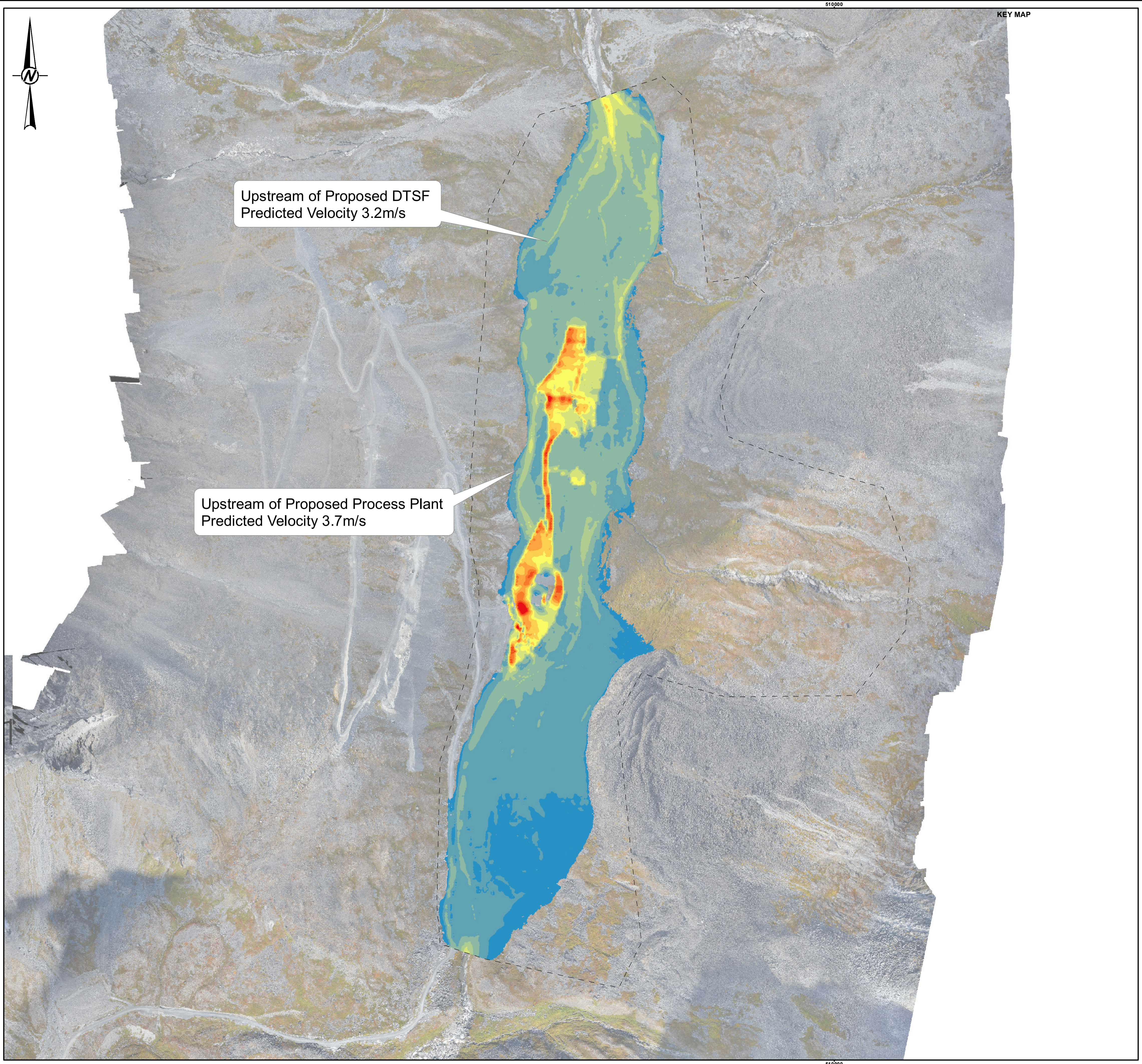


LEGEND

Model Extent

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:7,500 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
EXISTING SITE CONDITIONS, PMP EVENT. MAXIMUM VELOCITY.

CONSULTANT

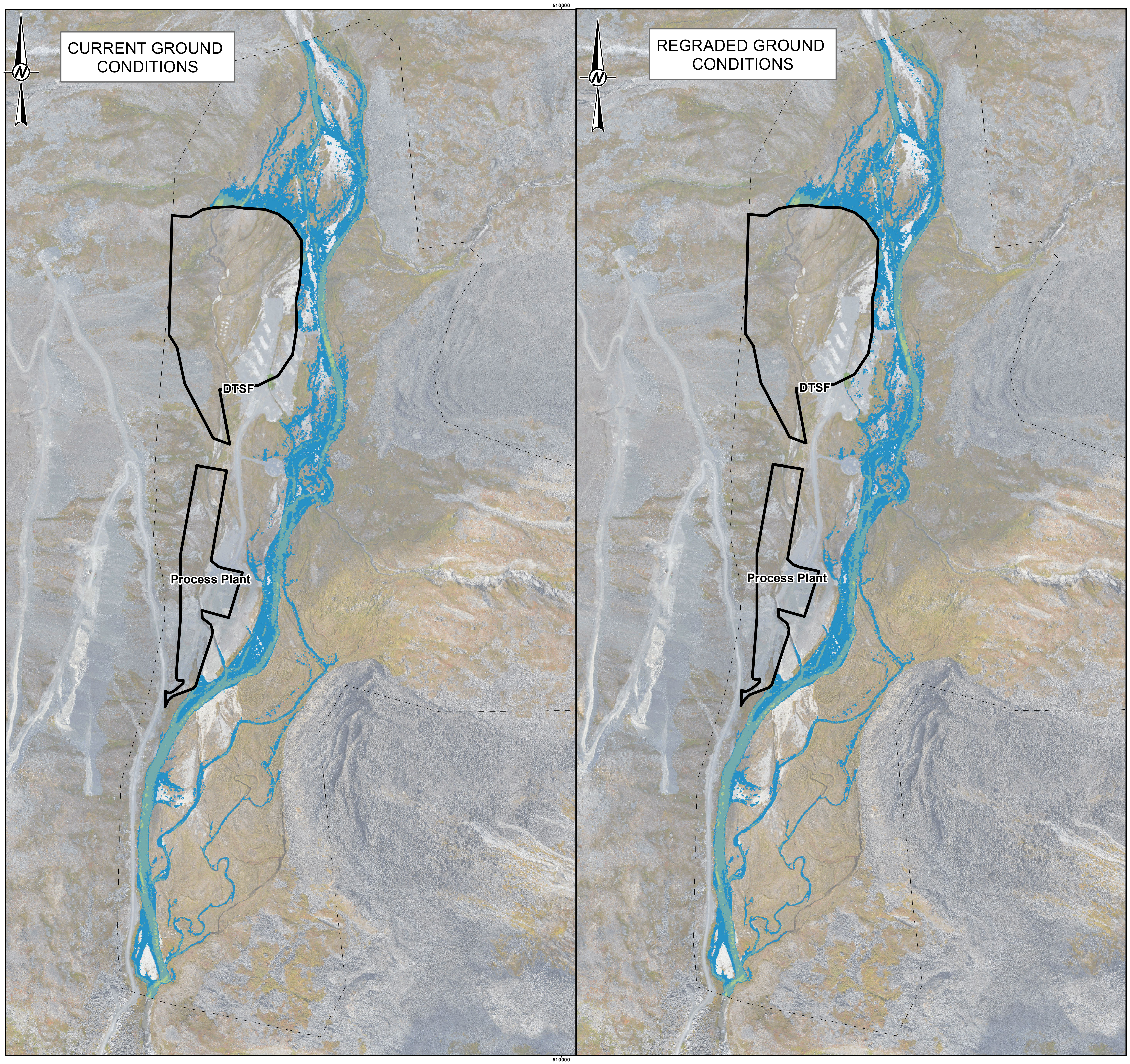
GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0019	A.0	B6-2

APPENDIX C

Flood Maps - Developed Site Conditions – Original Layout



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

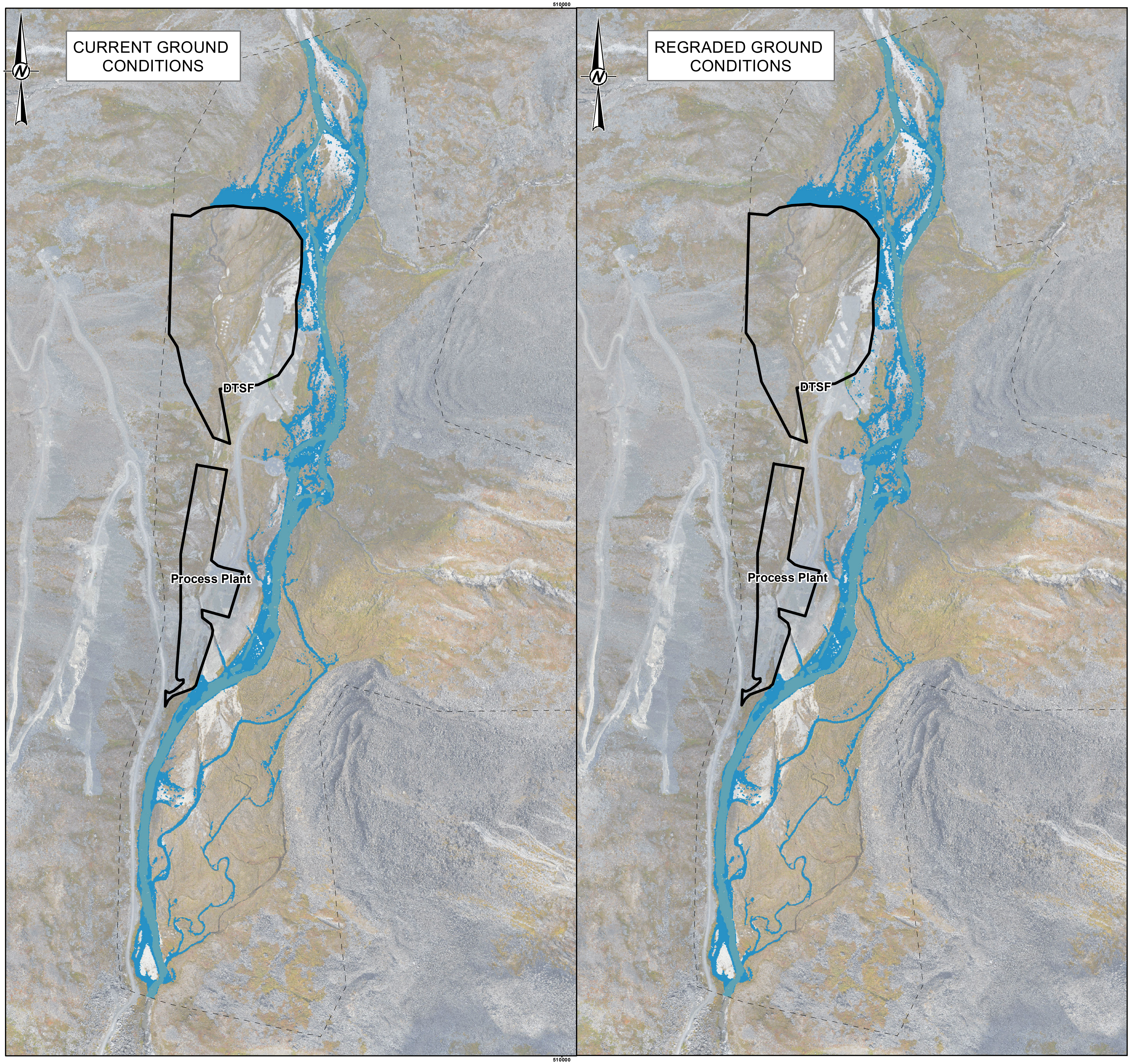
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-2 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0020	A.0	C1-1



LEGEND

Model Extent

Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

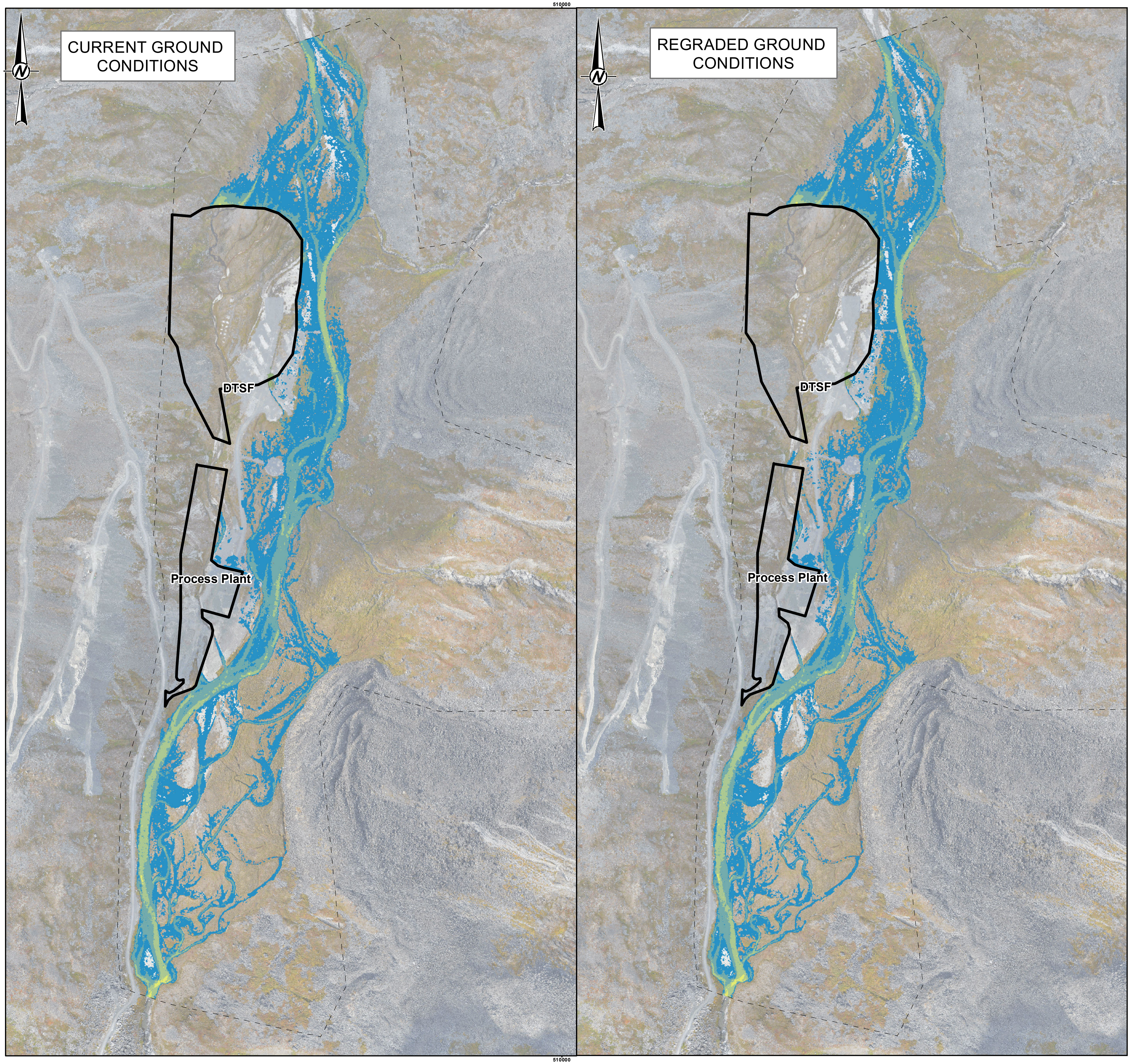
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-2 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0021	A.0	C1-2



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

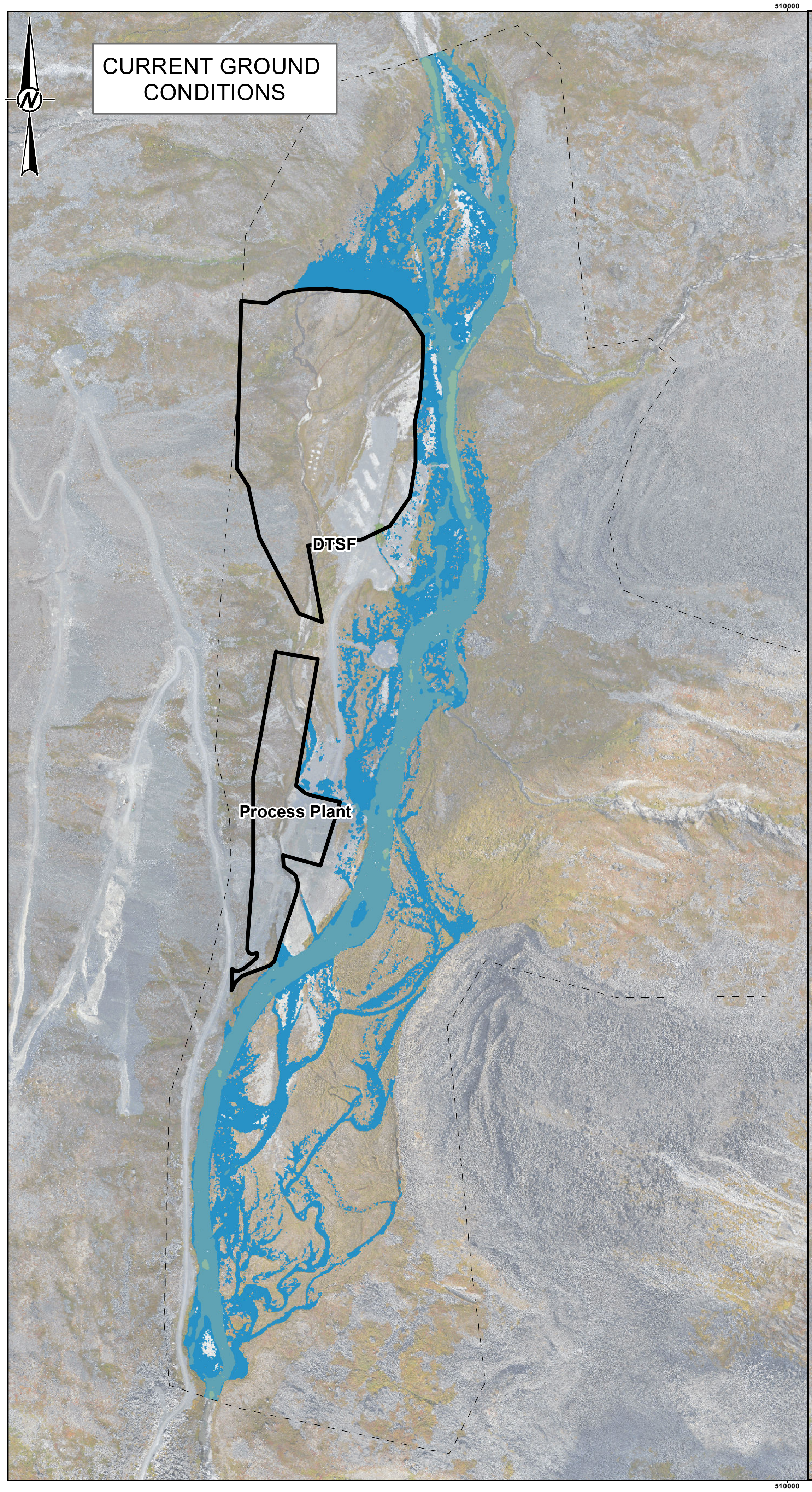
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-5 YEAR EVENT. MAXIMUM DEPTH.

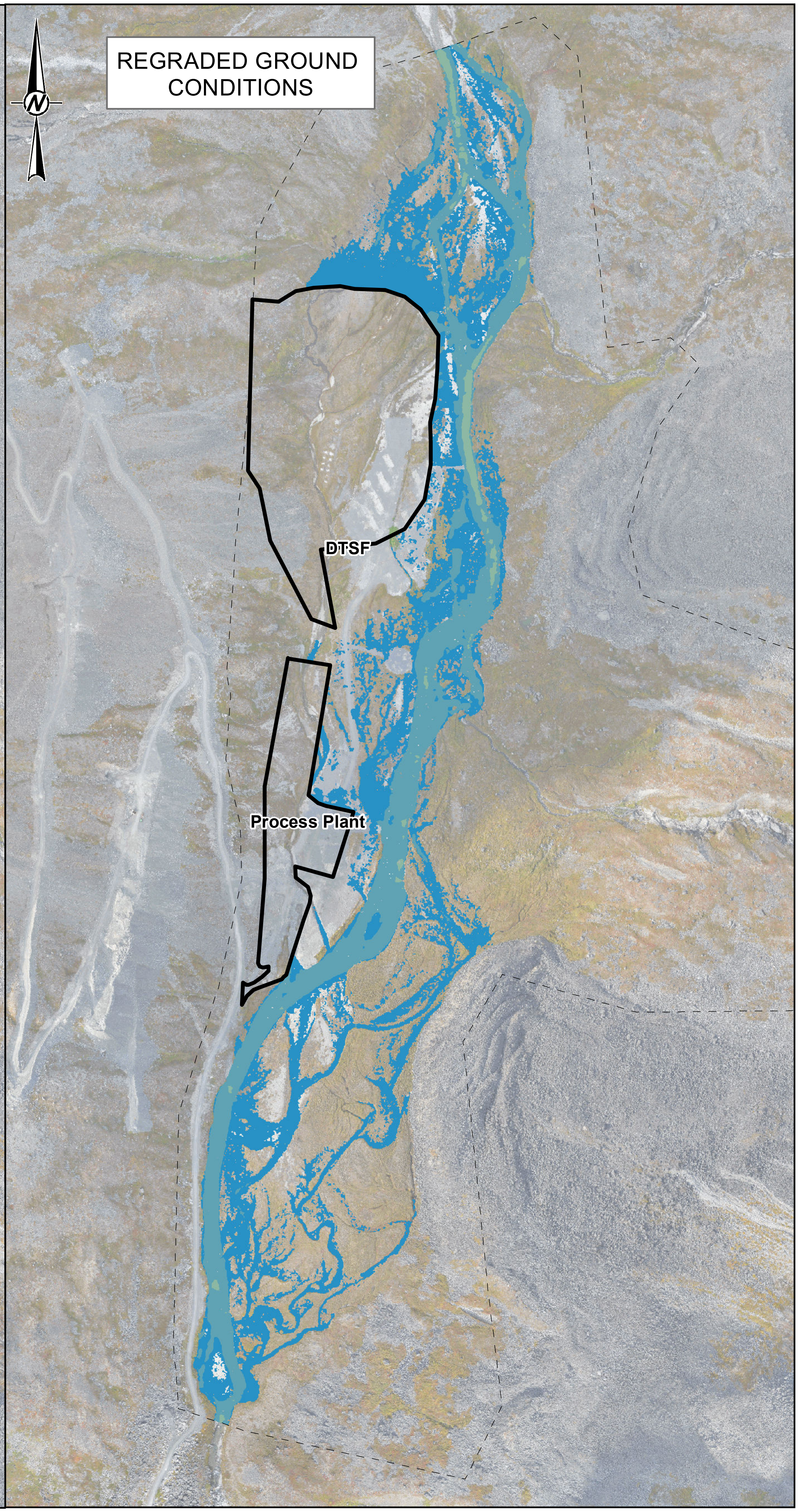
CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0022	A.0	C2-1





LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

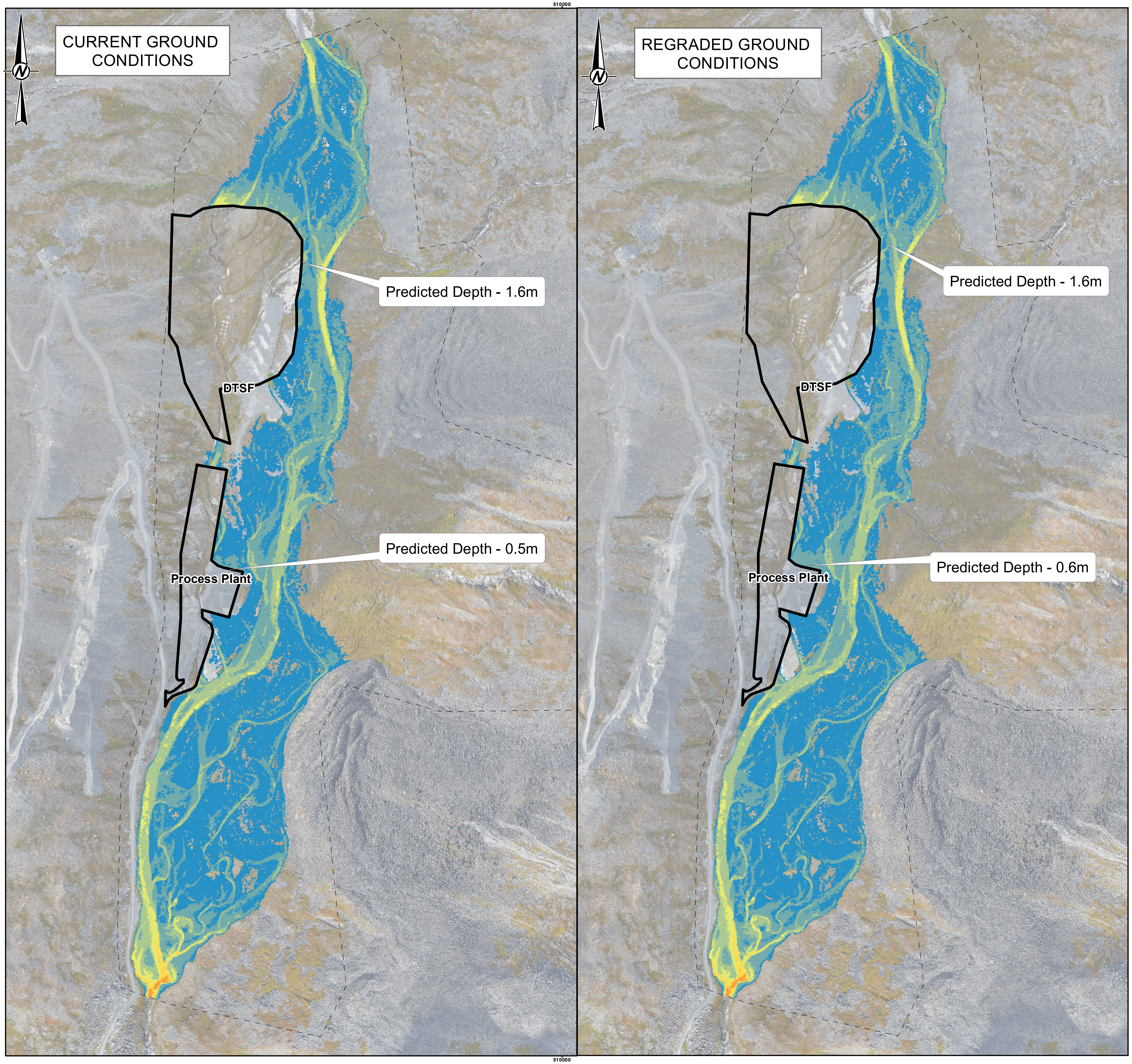
CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-5 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT	YYYY-MM-DD	2021 JAN 14
GOLDER	DESIGNED	CB
	PREPARED	CB
	REVIEWED	JR
	APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0023	A.0	C2-2



LEGEND

Model Extent

Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)

1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)

1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

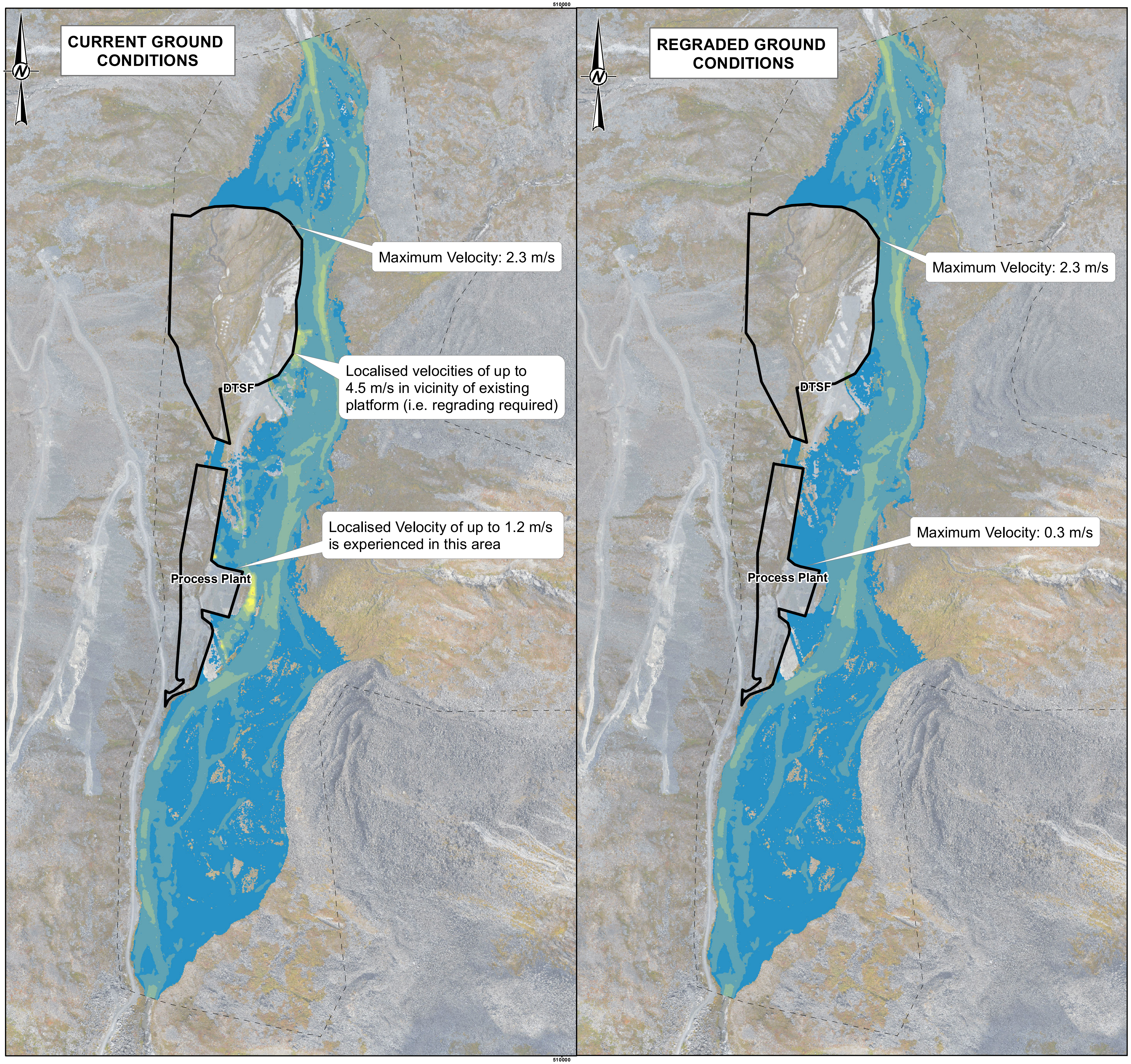
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-100 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0024	A.0	C3-1



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

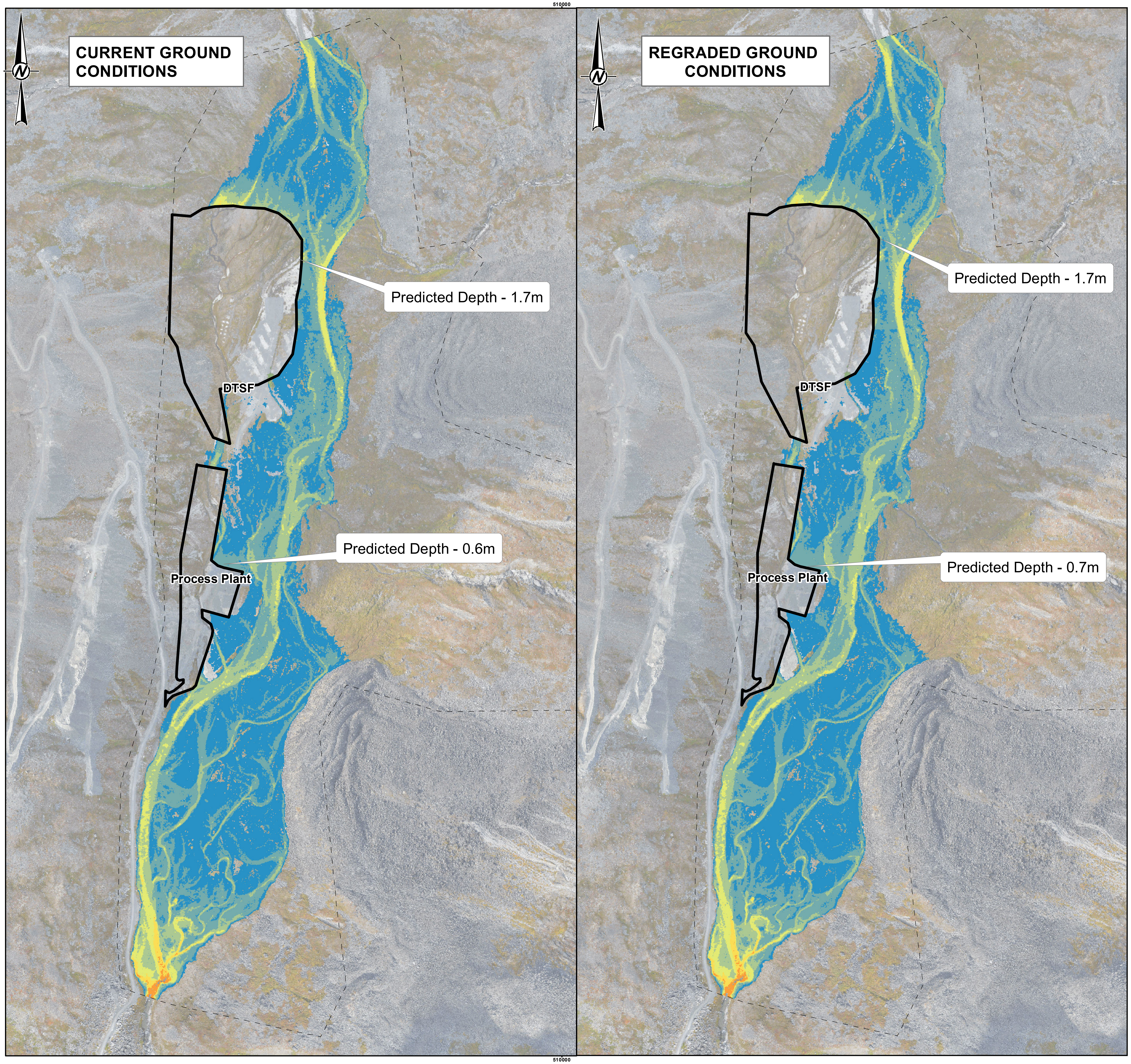
PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-100 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT
GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0025	A.0	C3-2



LEGEND

Model Extent

Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

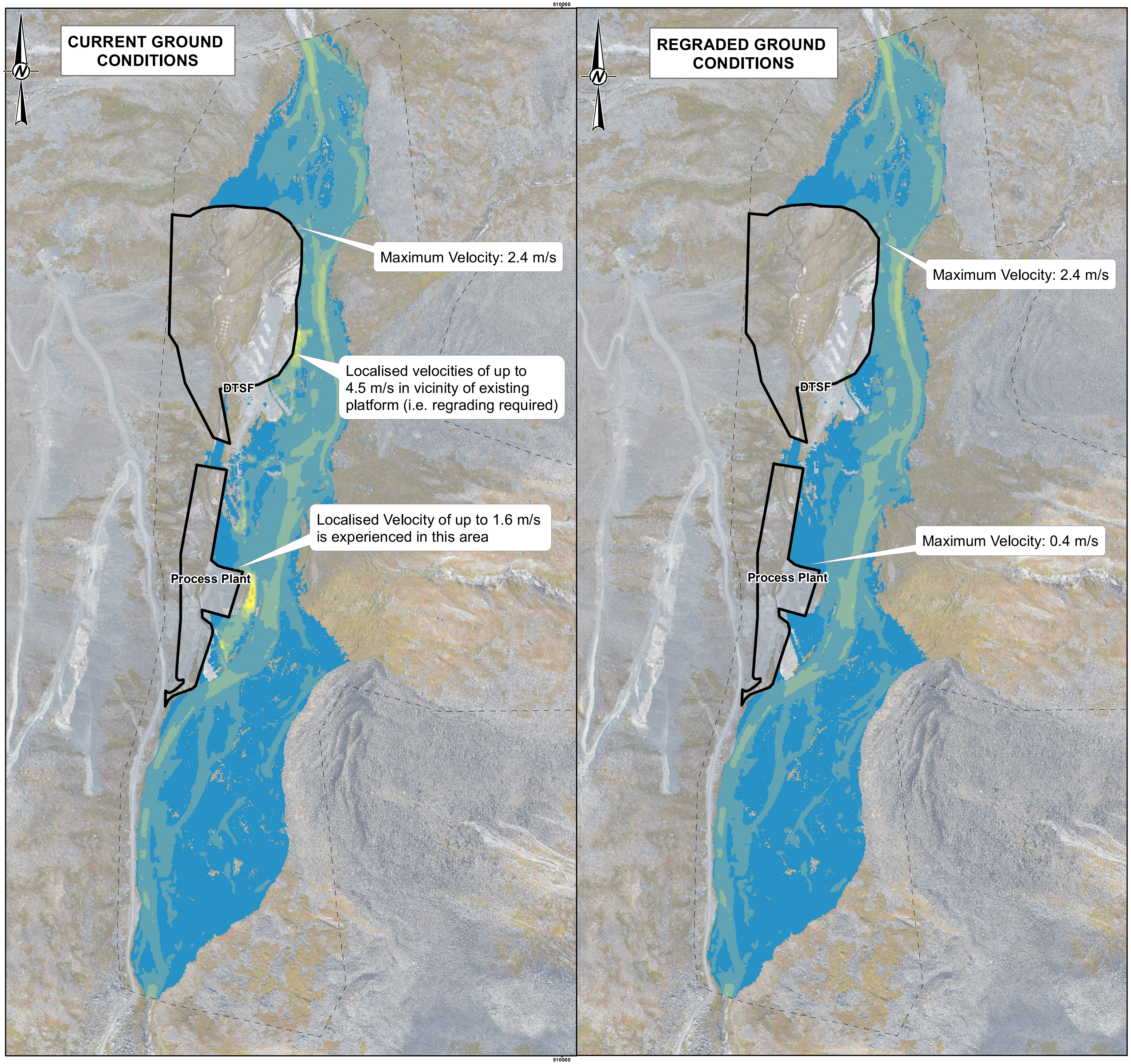
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-200 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0026	A.0	C4-1



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

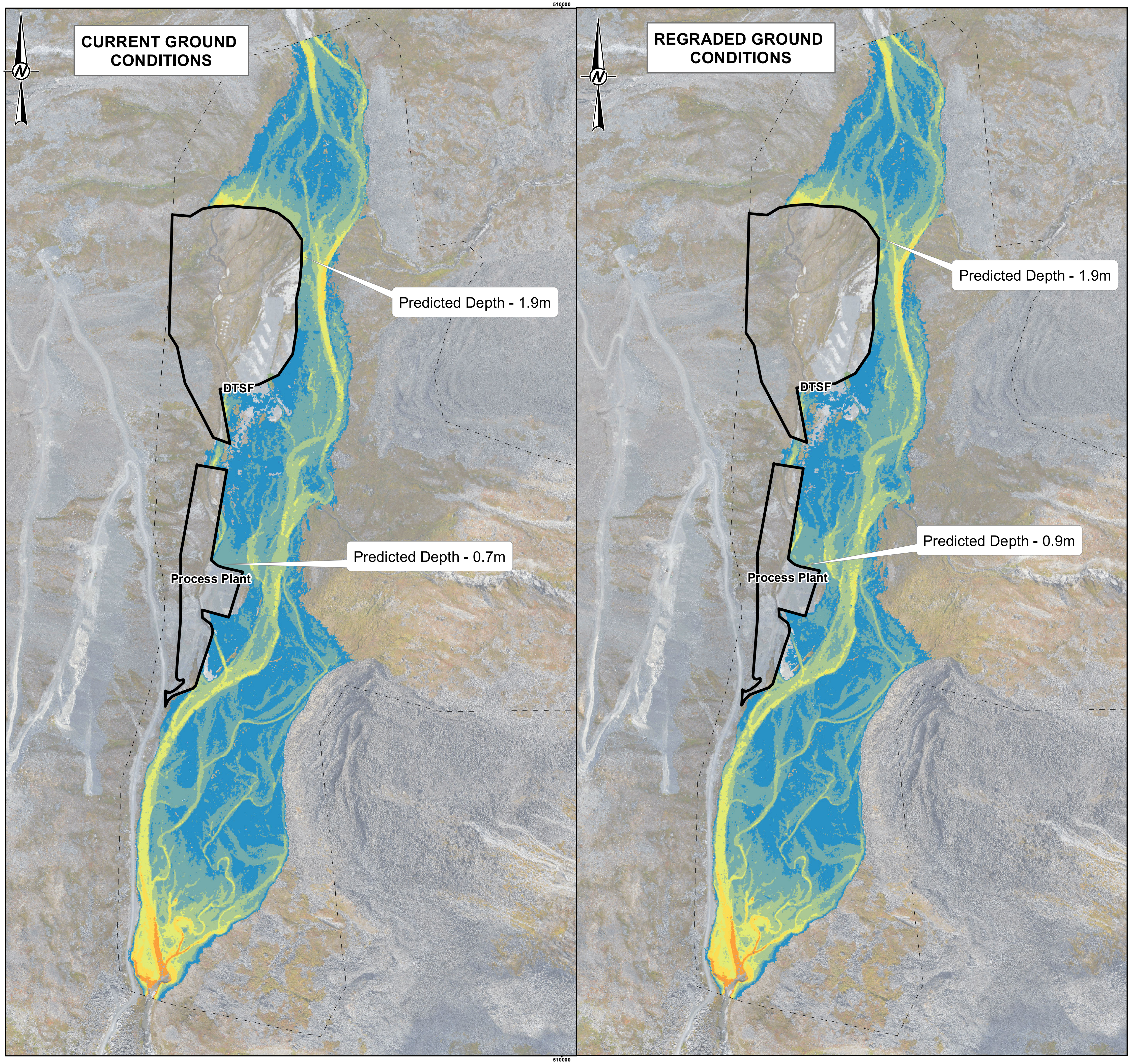
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-200 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0027	A.0	C4-2



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

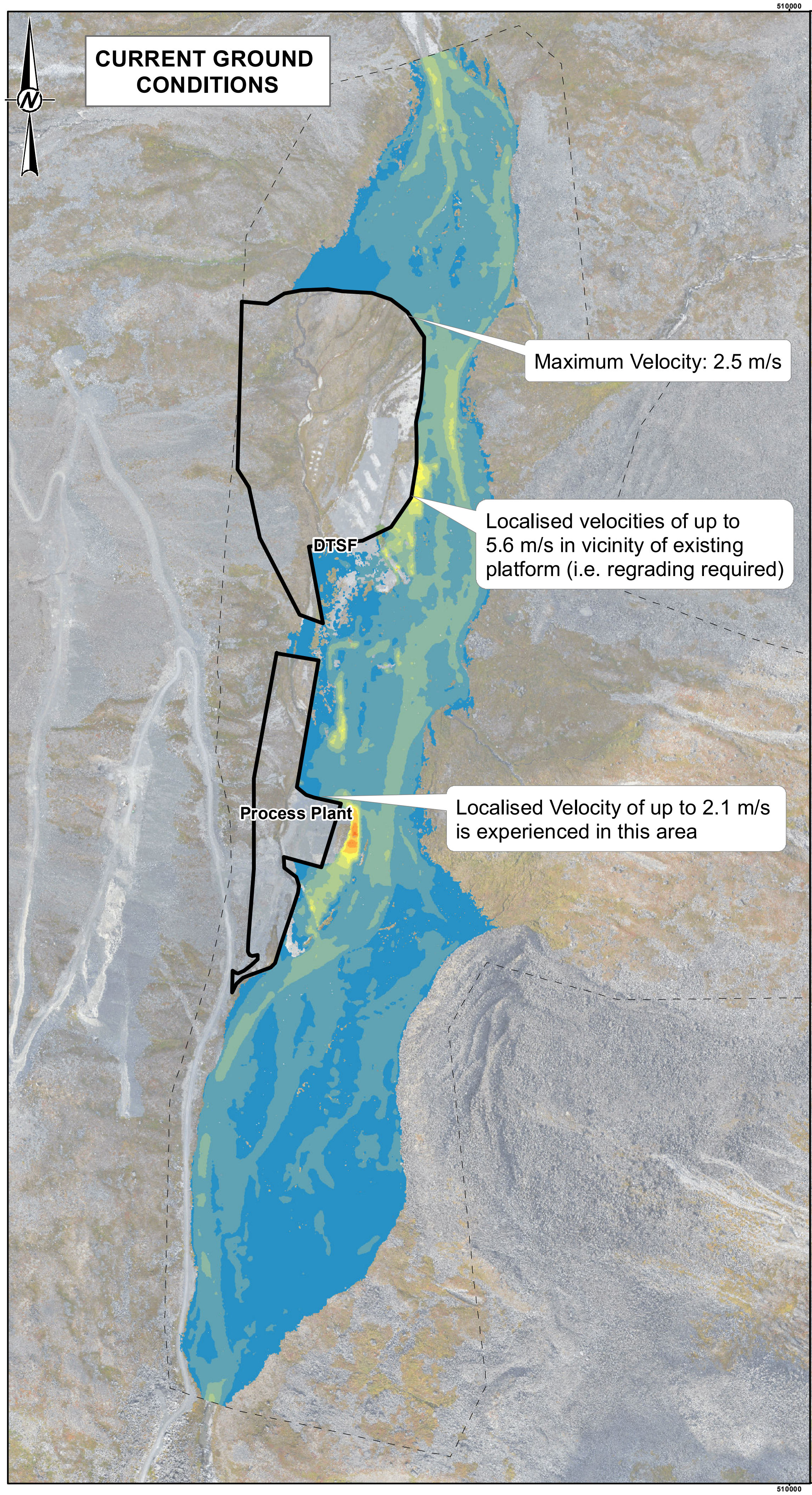
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-1000 YEAR EVENT. MAXIMUM DEPTH.

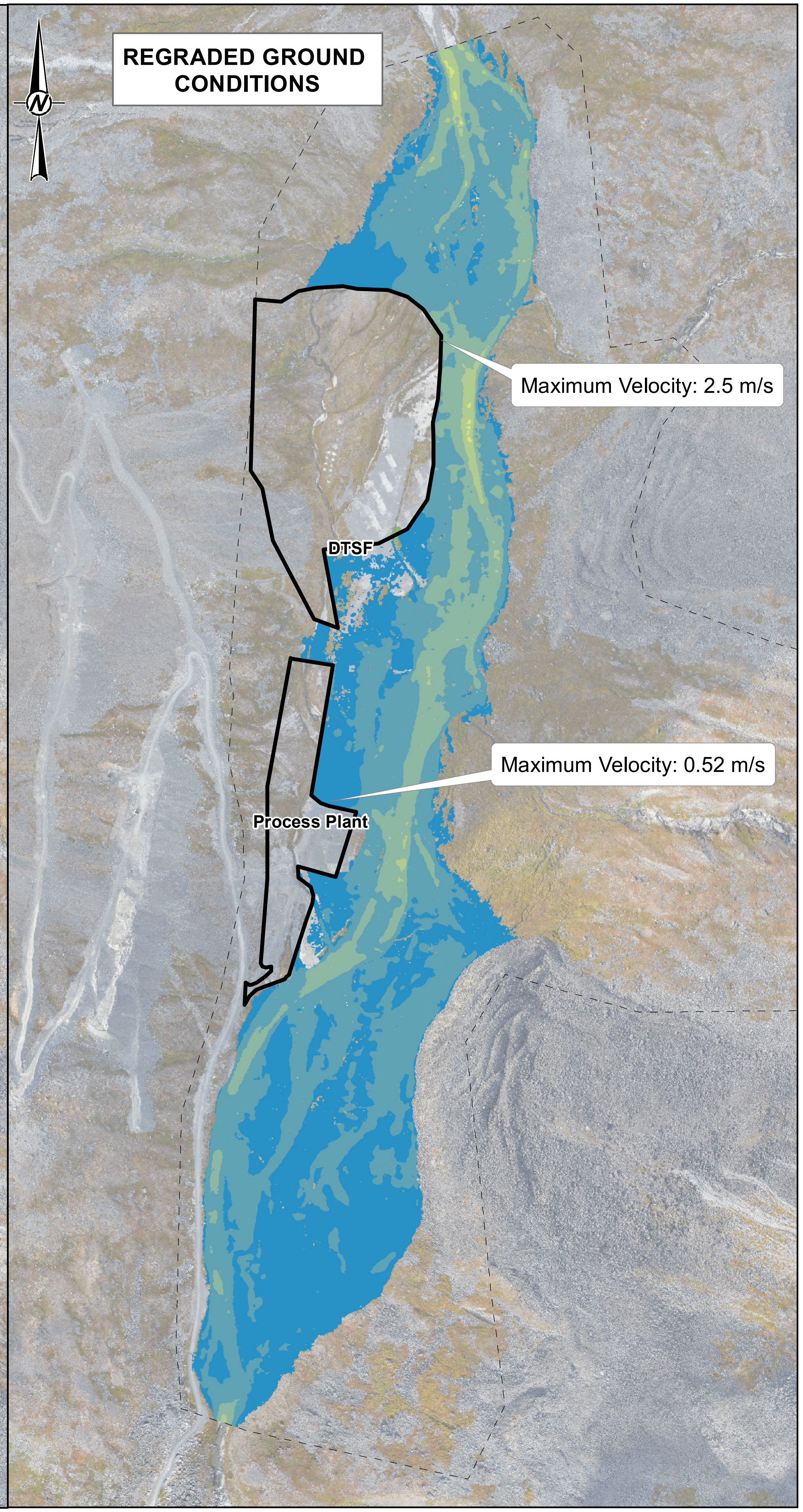
CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0028	A.0	C5-1





LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

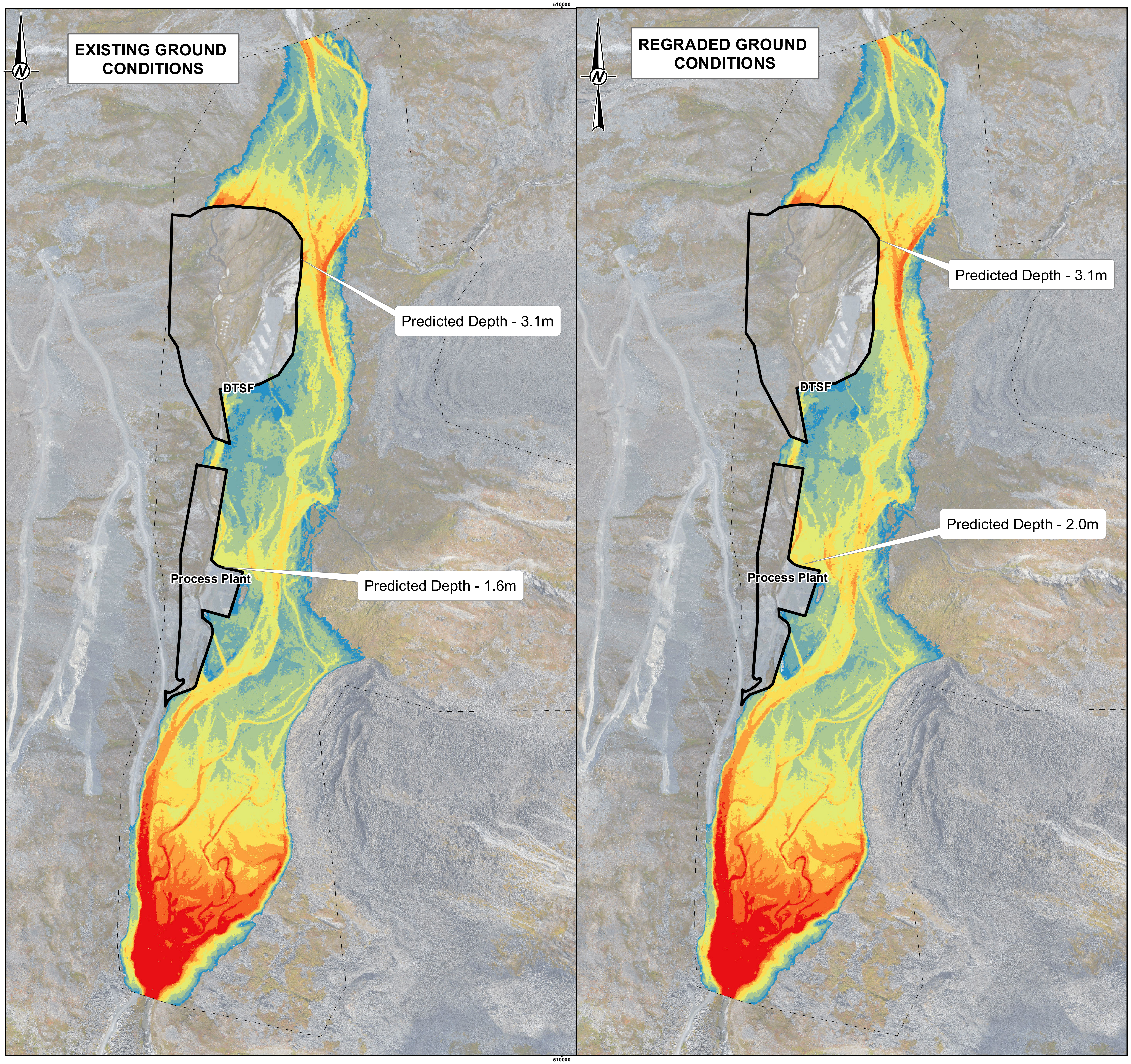
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - 1-IN-1000 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0029	A.0	C5-2



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

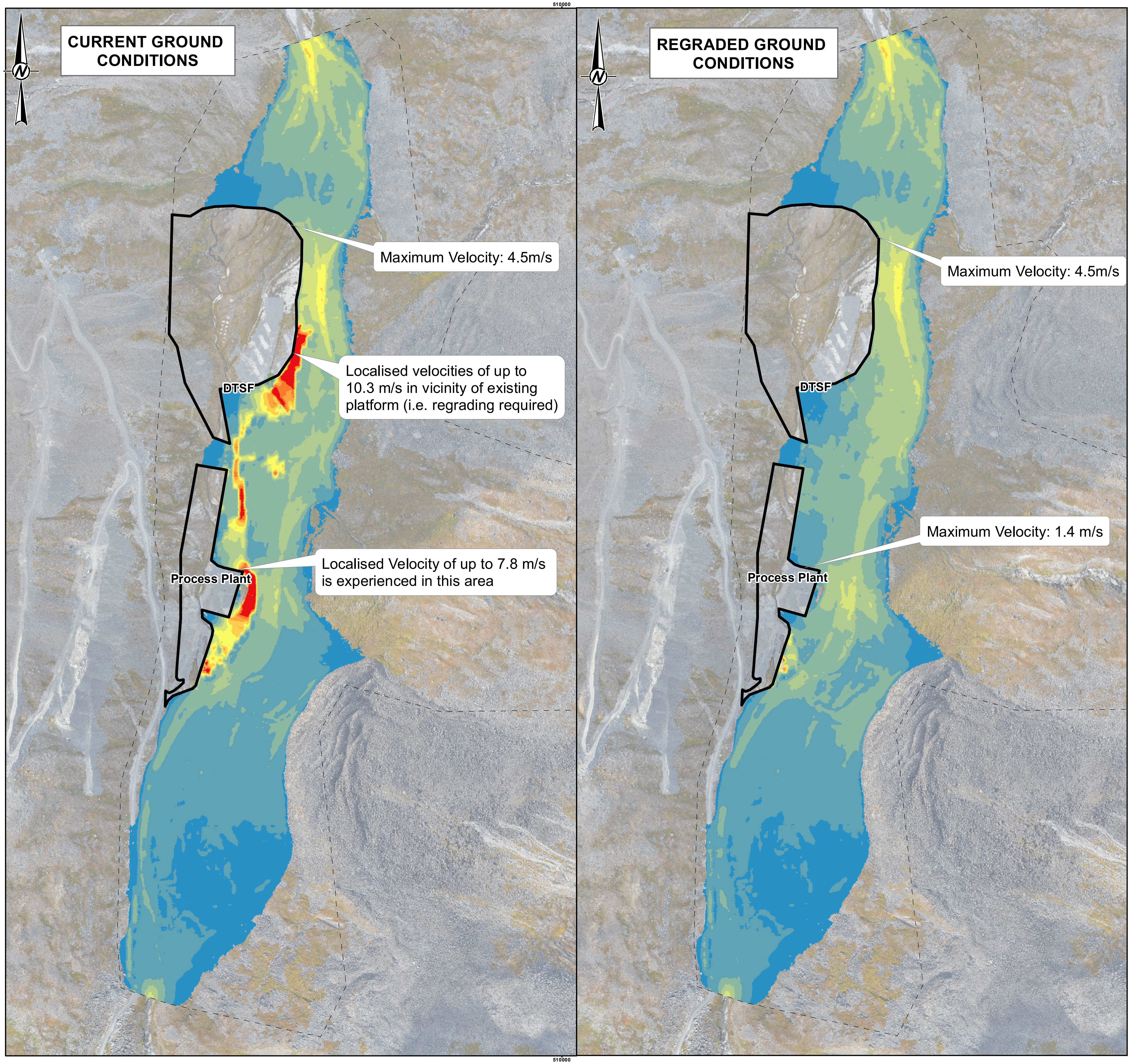
TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - PMP YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0030	A.0	C6-1



LEGEND

- Model Extent
- Proposed 'Original' Facility Footprints

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. ORIGINAL LAYOUT - PMP EVENT. MAXIMUM VELOCITY.

CONSULTANT

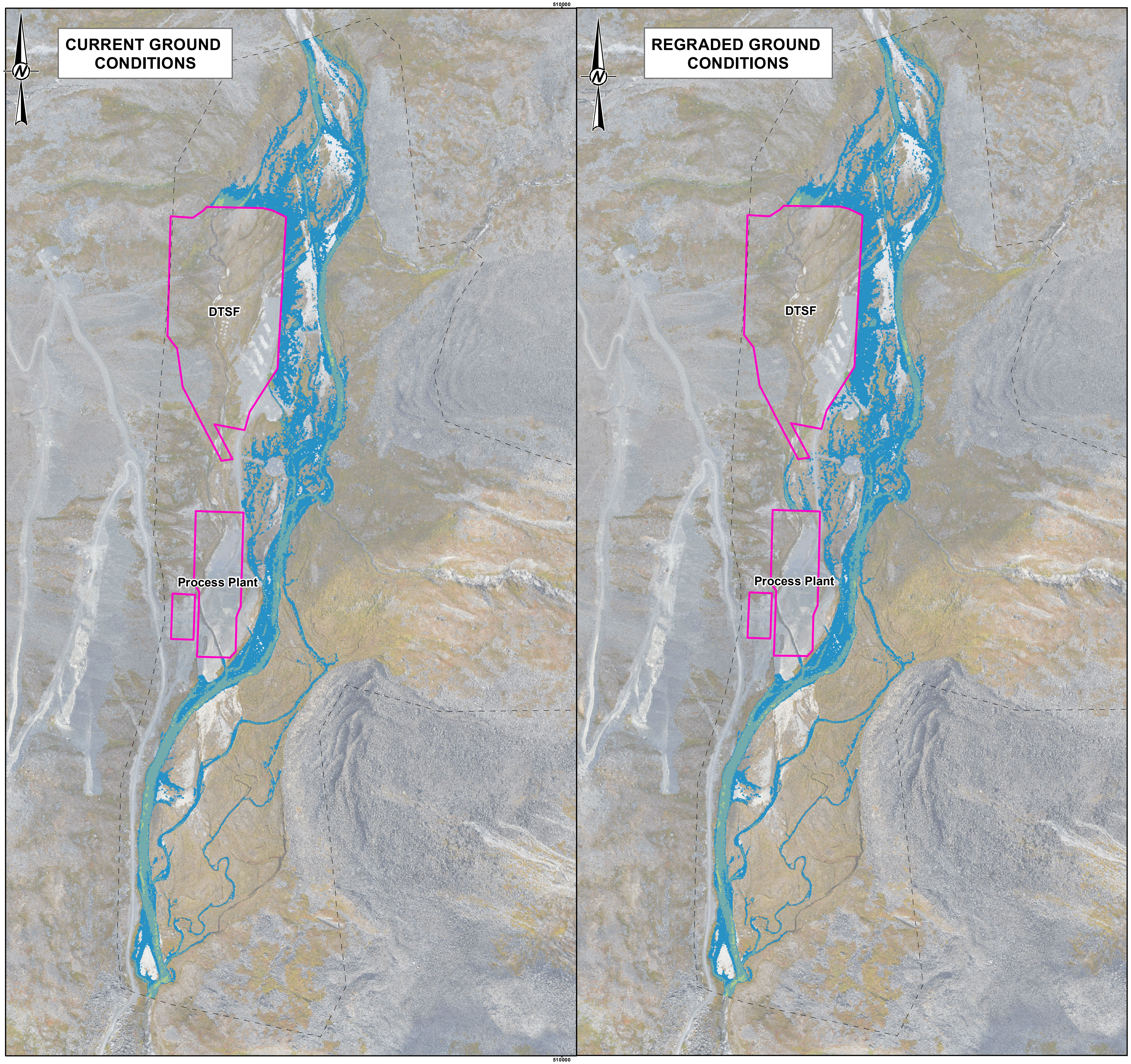
GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0031	A.0	C6-2

APPENDIX D

Flood Maps – Developed Site Conditions - Updated Layout



LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-2 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

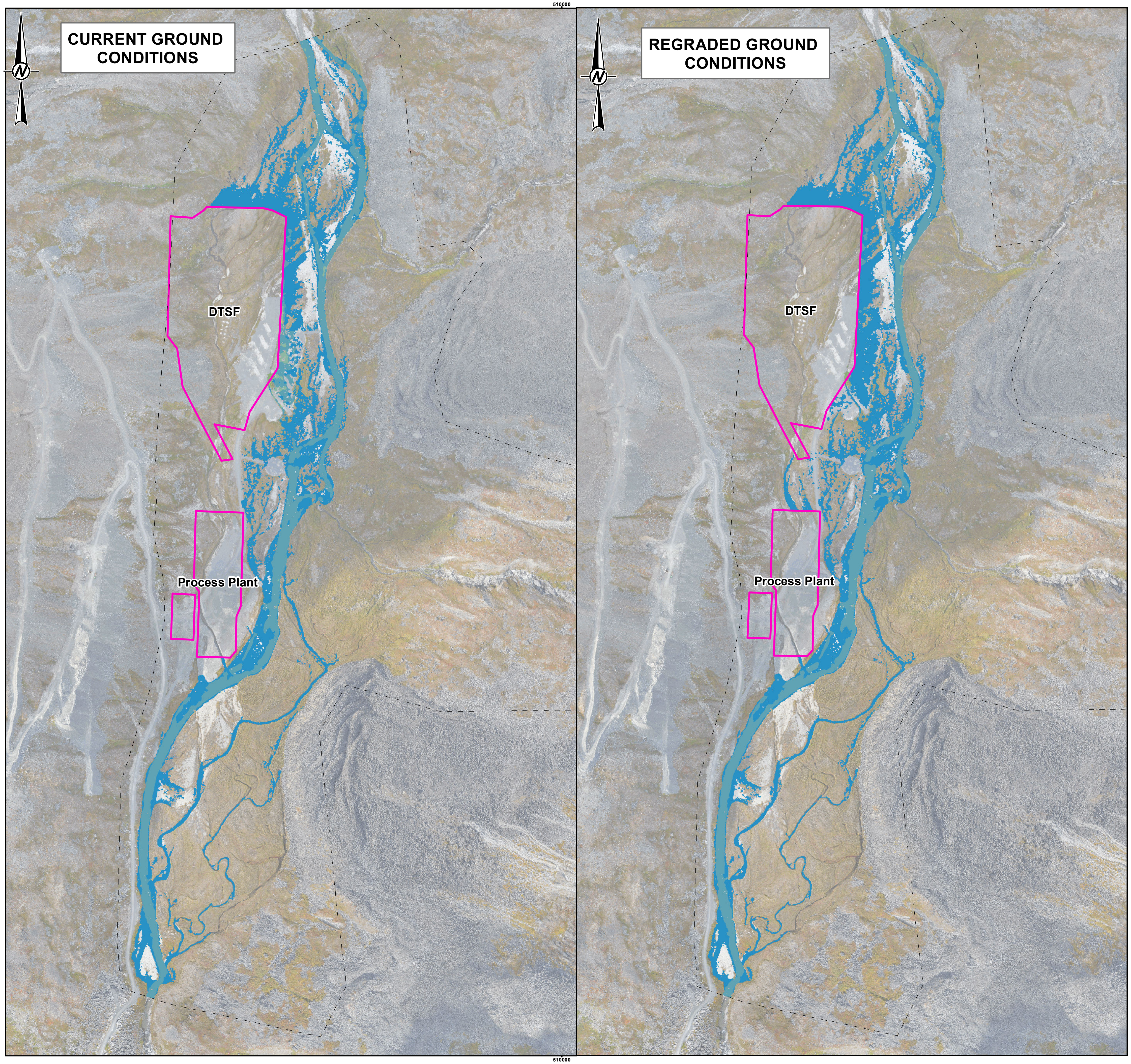
YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0032	A.0	D1-1

IF THIS MEASUREMENT DOES NOT MATCH WHAT IS SHOWN, THE SHEET SIZE HAS BEEN MODIFIED FROM: ISO A3

25mm 0

PATH: E:\AEX_Gold\Nalunaq\99_PROJECTS\20136781\Flood_Analysis\02_PRODUCTION\MXD\A.0\20136781_SW_610_0032_D1_1_Updated_1in2_MaxD.mxd PRINTED ON: 2021-01-14 AT: 9:25:42 AM



LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

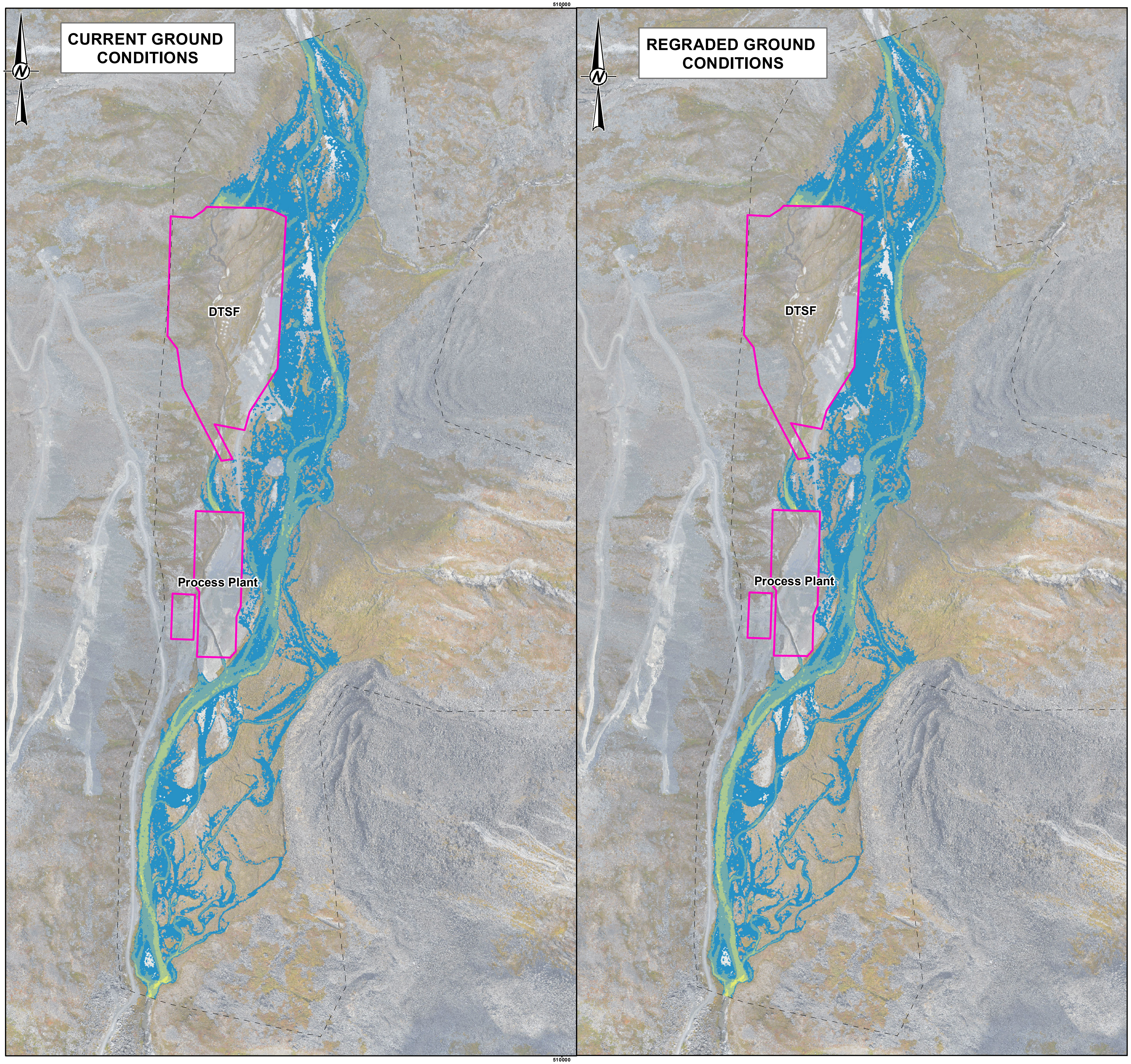
TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN2 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0033	A.0	D1-2



LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

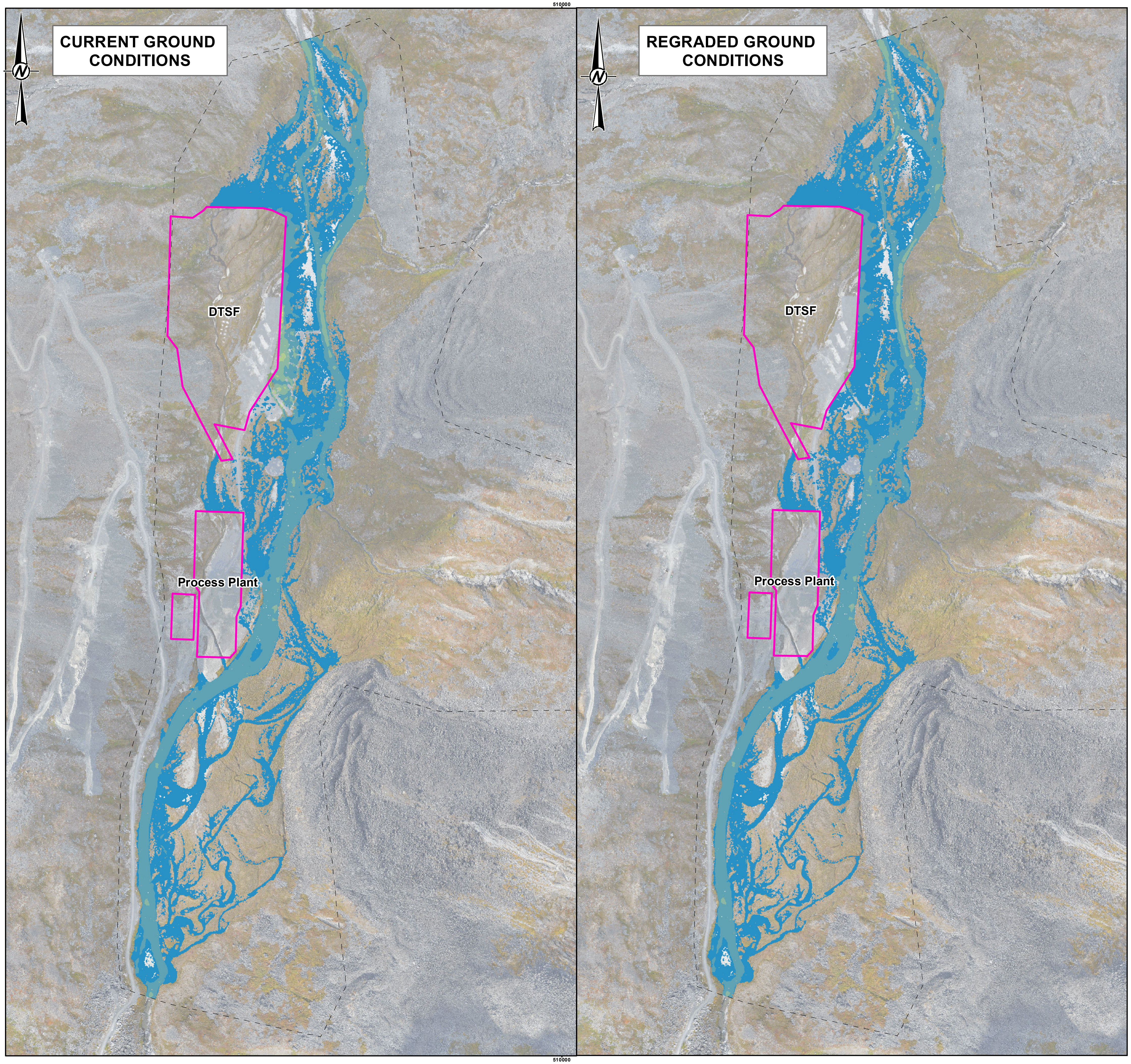
TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-5 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0034	A.0	D2-1



LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500
1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

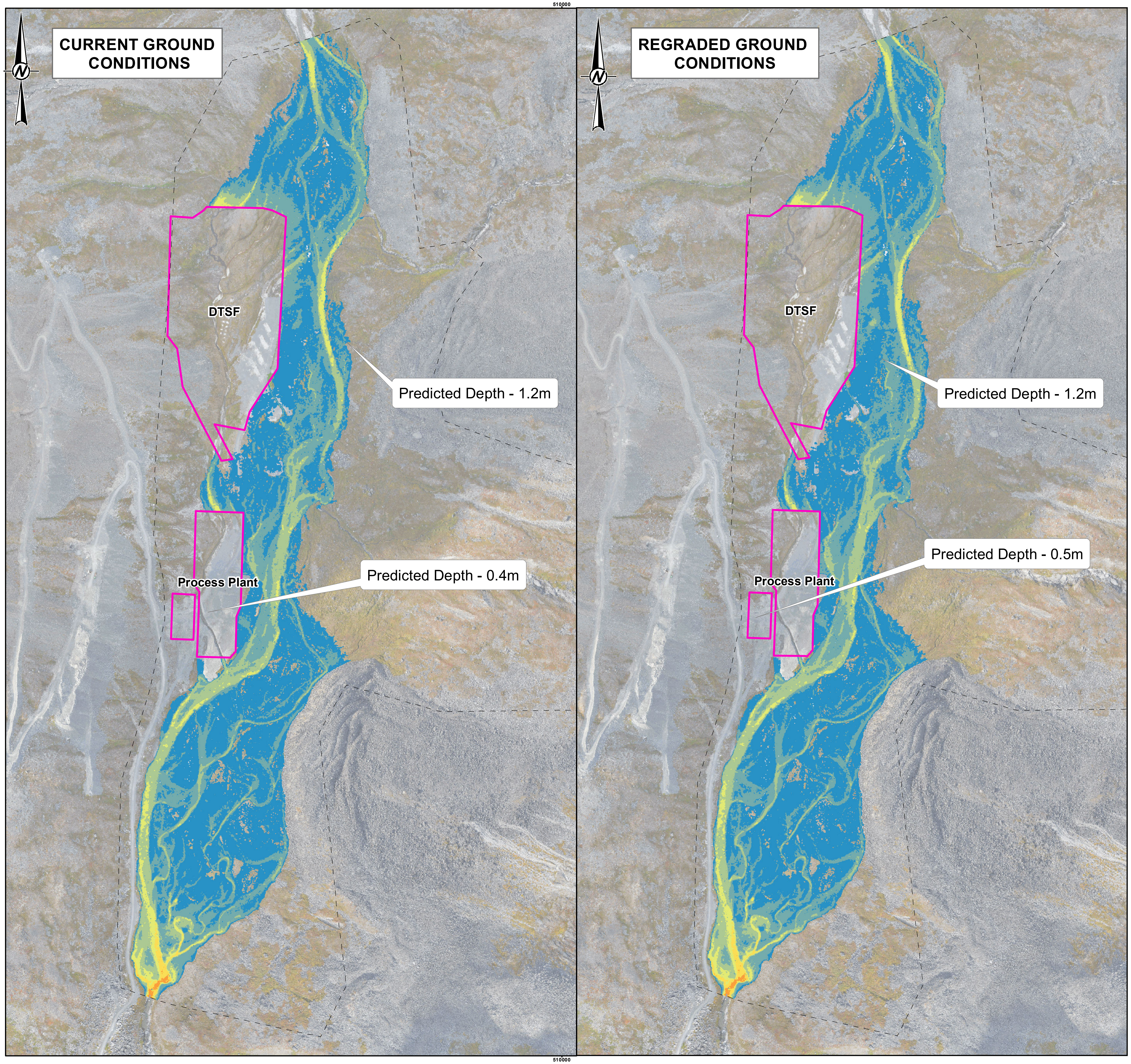
CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN5 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT	YYYY-MM-DD	2021 JAN 14
GOLDER	DESIGNED	CB
	PREPARED	CB
	REVIEWED	JR
	APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0035	A.0	D2-2



LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Depth (m)

- 0 - 0.5
- 0.5 - 1
- 1 - 1.5
- 1.5 - 2
- 2 - 2.5
- 2.5 - 3
- 3 - 3.5
- >3.5

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

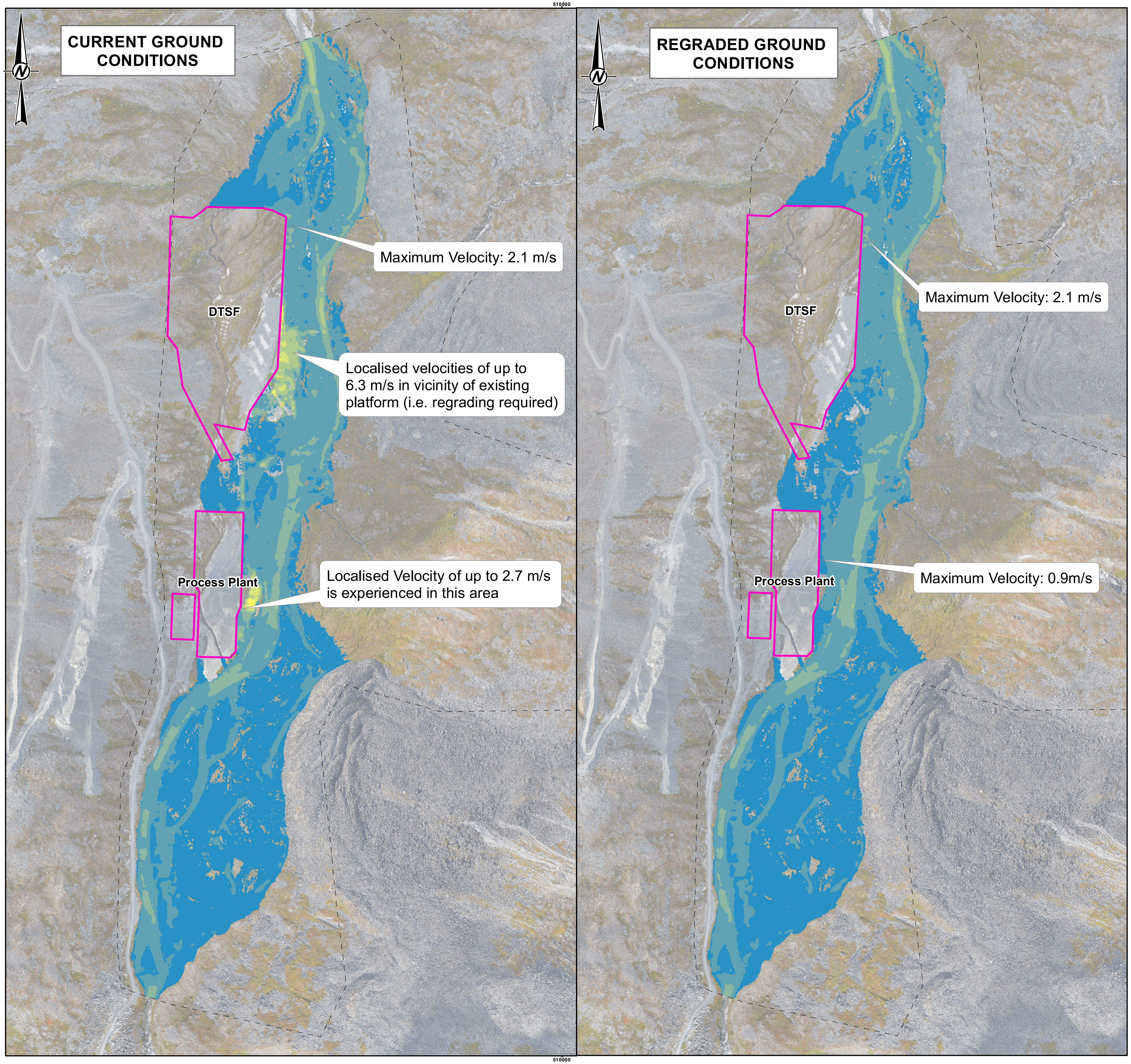
TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-100 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0036	A.0	D3-1



LEGEND

Model Extent

Updated DTSF and Process Plant

Maximum Velocity (m/s)

- 0 - 1
- 1 - 2
- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-100 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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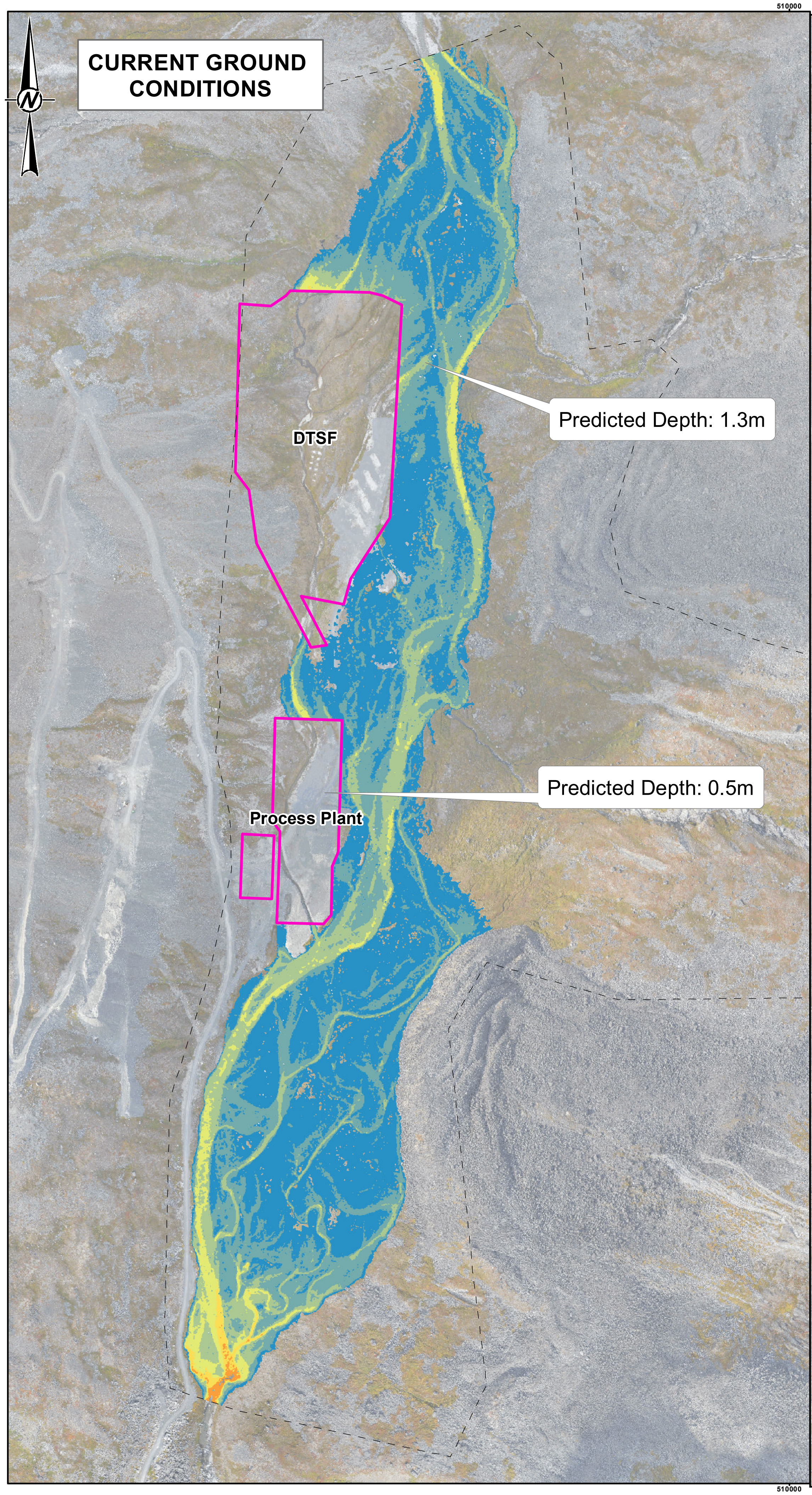

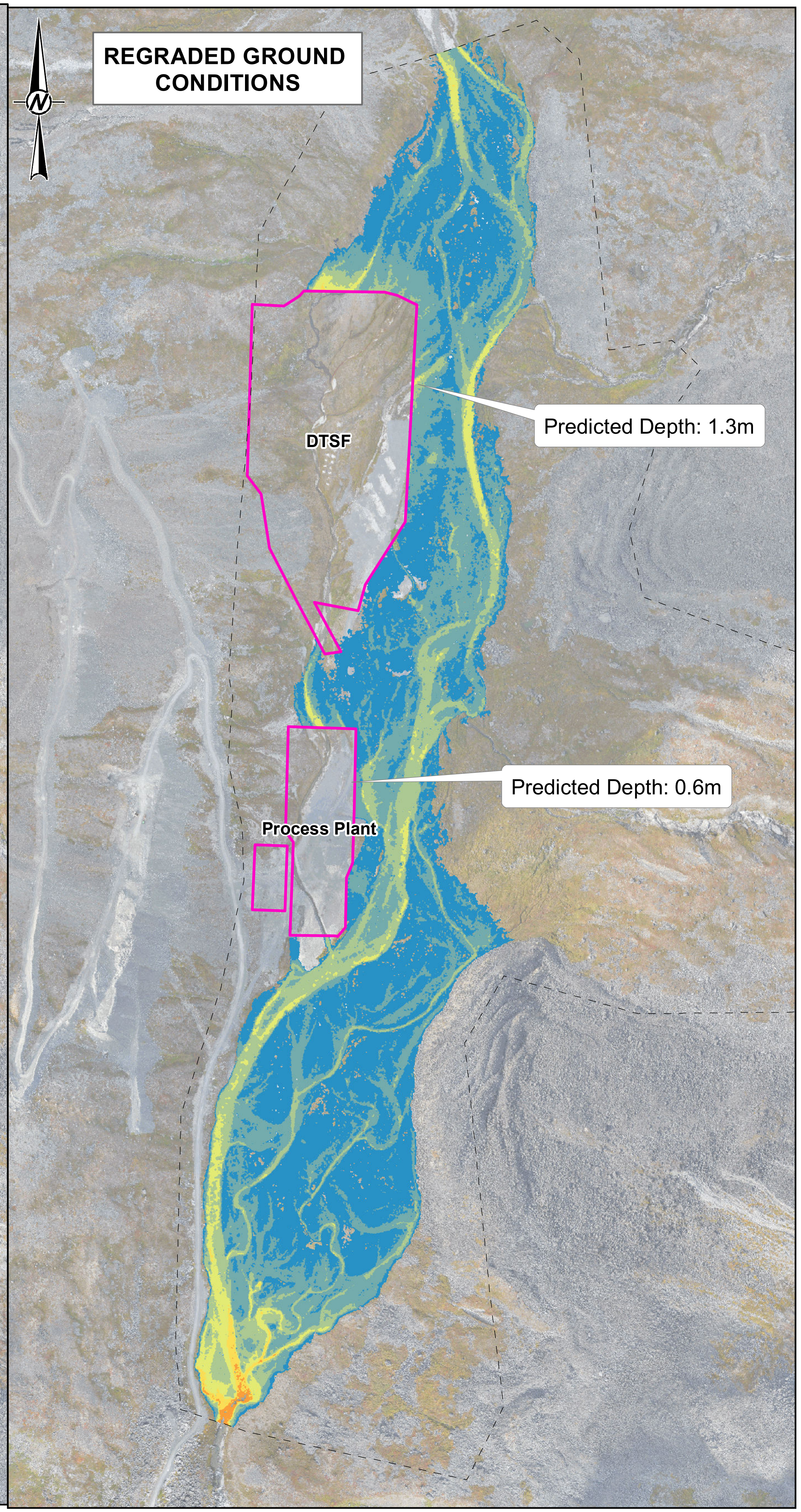

LEGEND

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Updated DTSF and Process Plant

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NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-1000 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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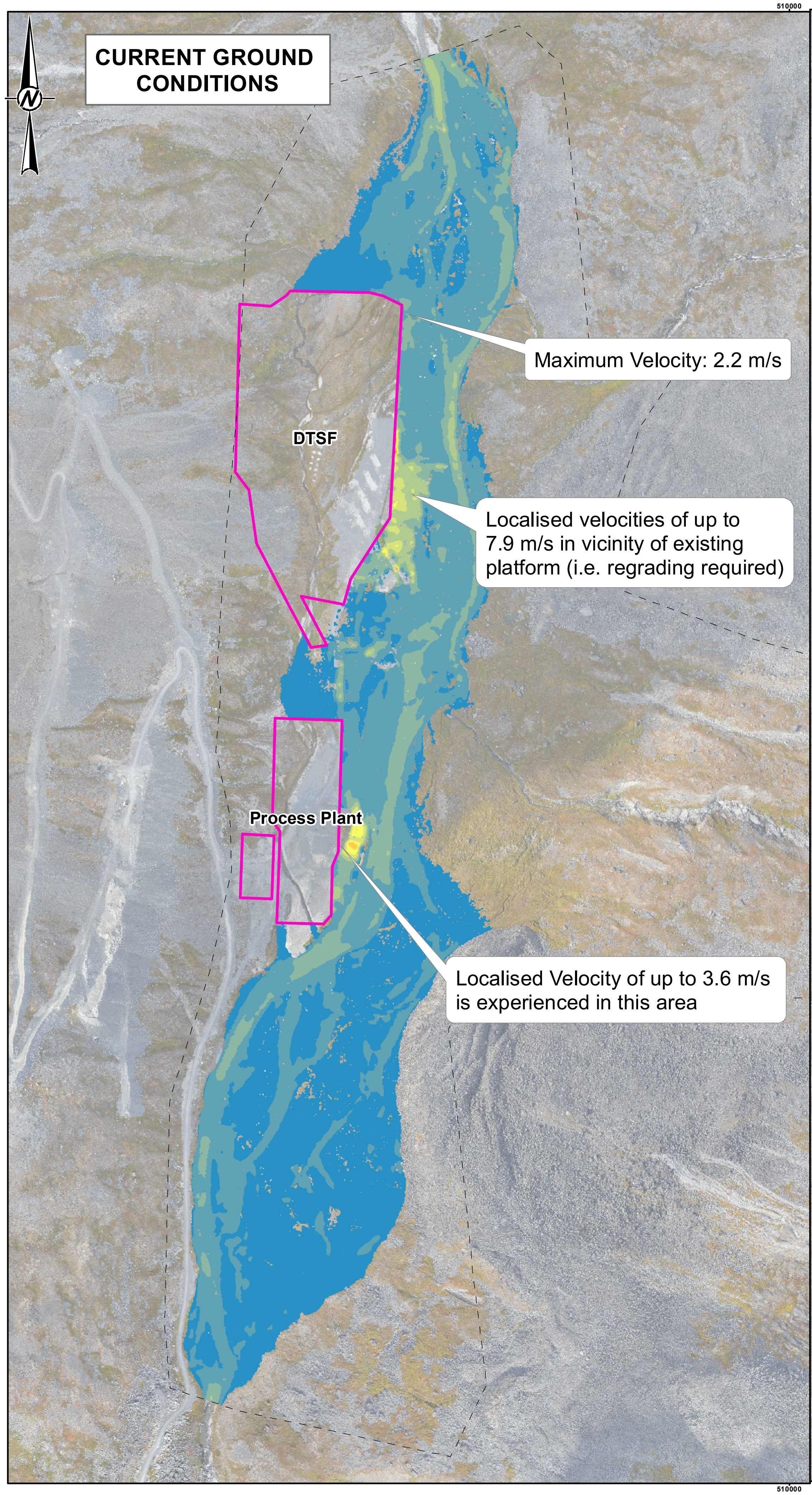

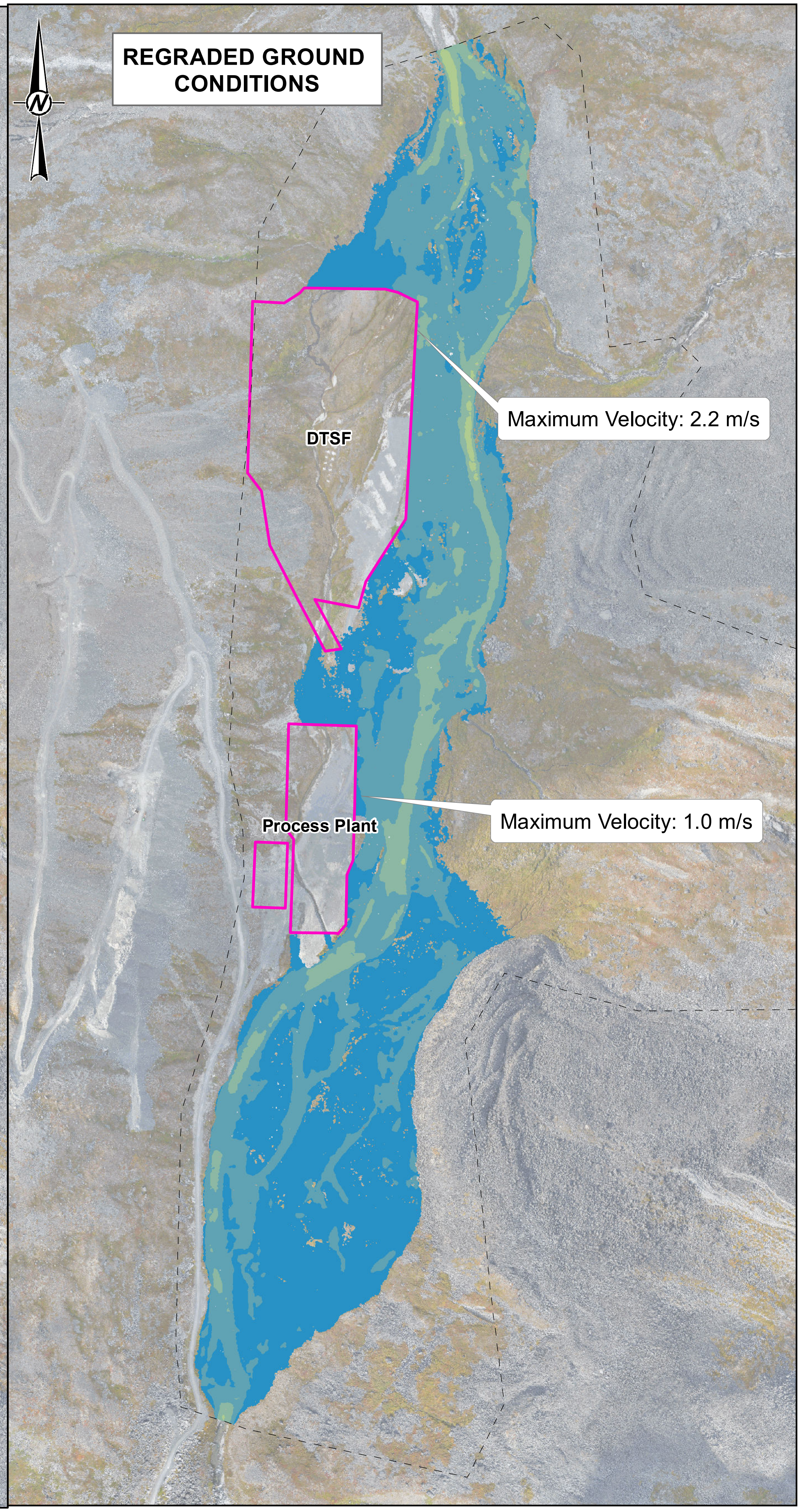

LEGEND

Model Extent

Updated DTSF and Process Plant

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- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

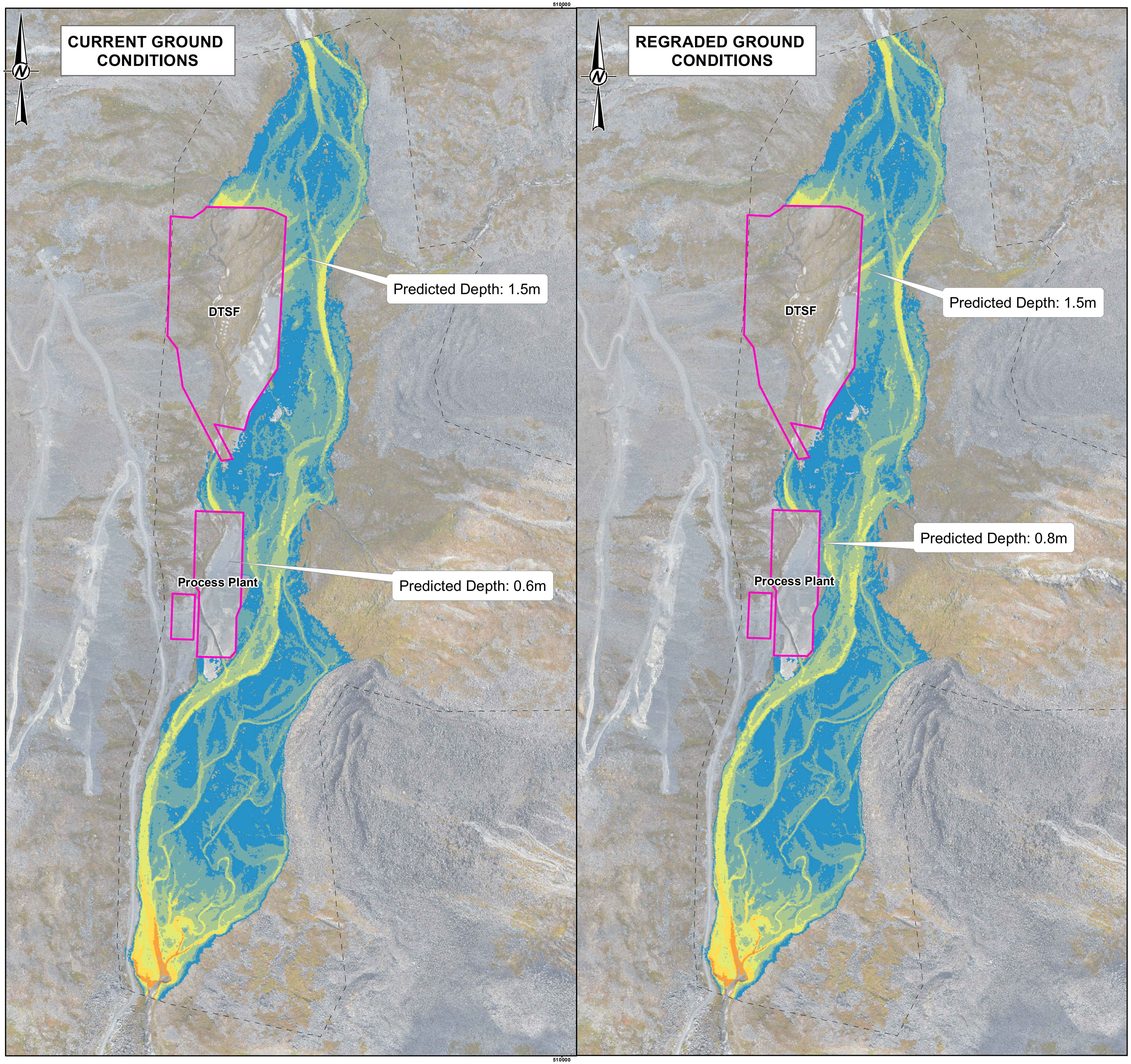
TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-200 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0039	A.0	D4-2



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- >3.5

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1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-1000 YEAR EVENT. MAXIMUM DEPTH.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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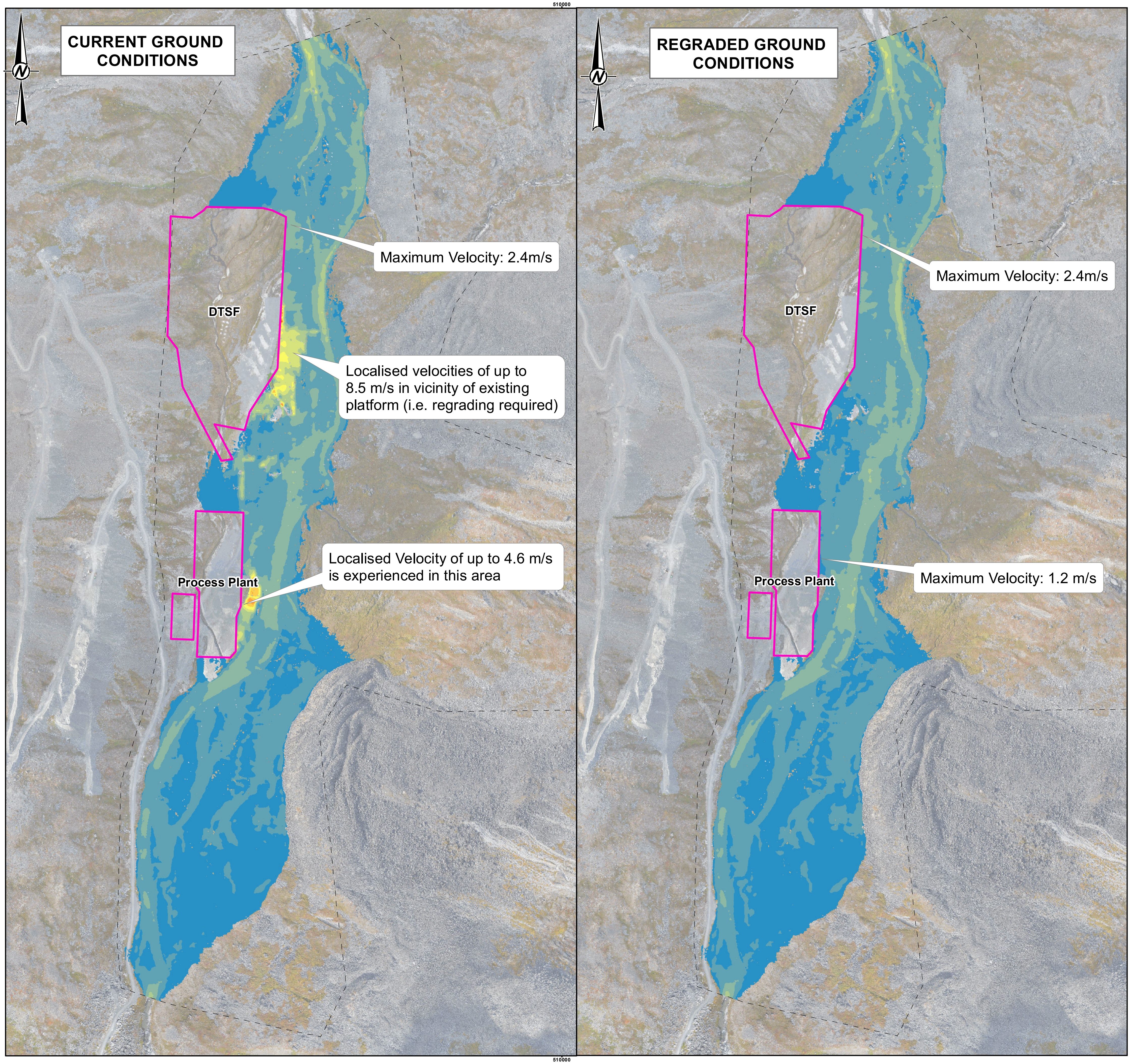

LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Velocity (m/s)

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- 2 - 3
- 3 - 4
- 4 - 5
- 5 - 6
- 6 - 7
- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500
1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

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AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - 1-IN-1000 YEAR EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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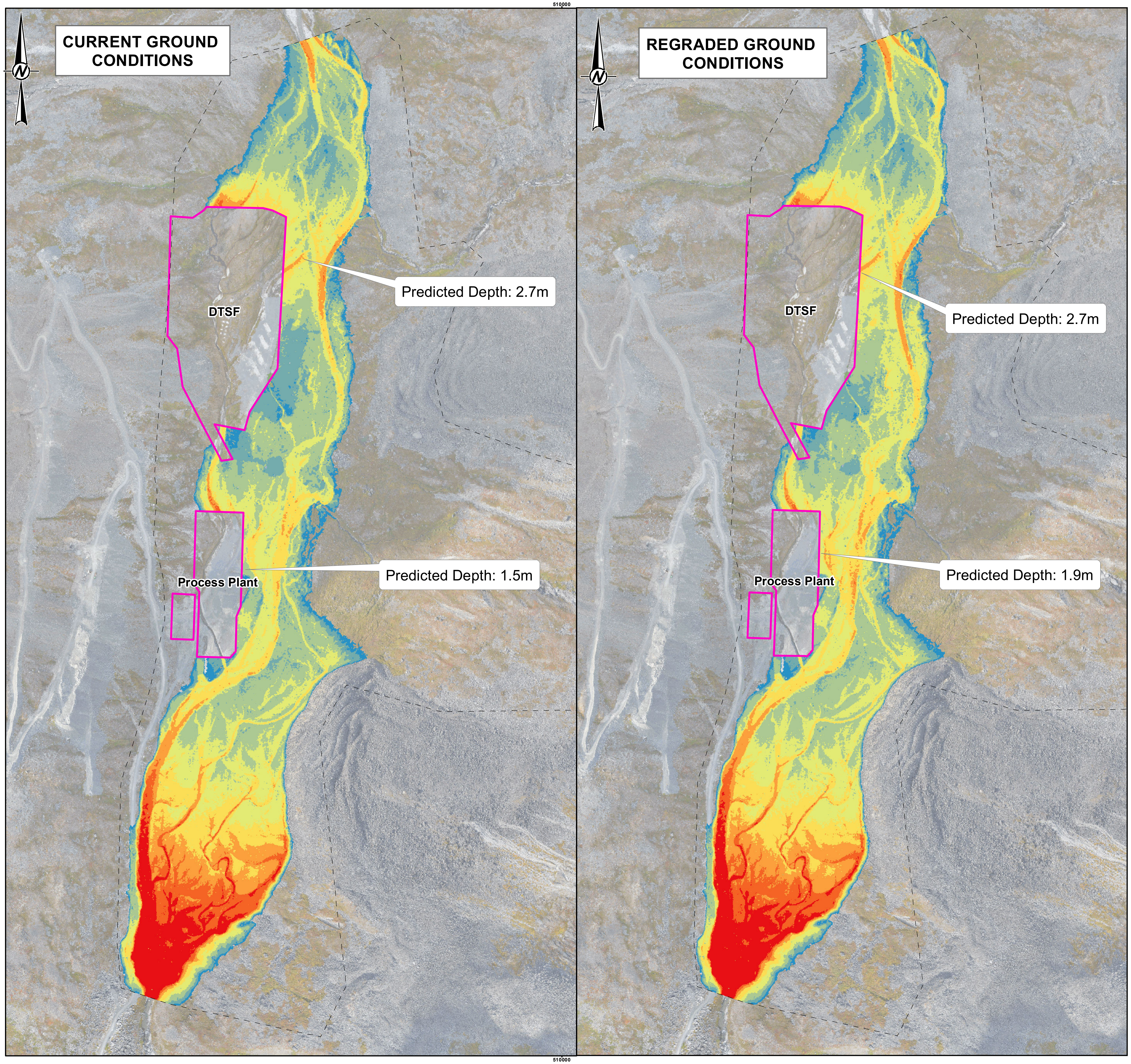

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- 2 - 2.5
- 2.5- 3
- 3 - 3.5
- >3.5

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1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - PMP EVENT. MAXIMUM DEPTH.

CONSULTANT
GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
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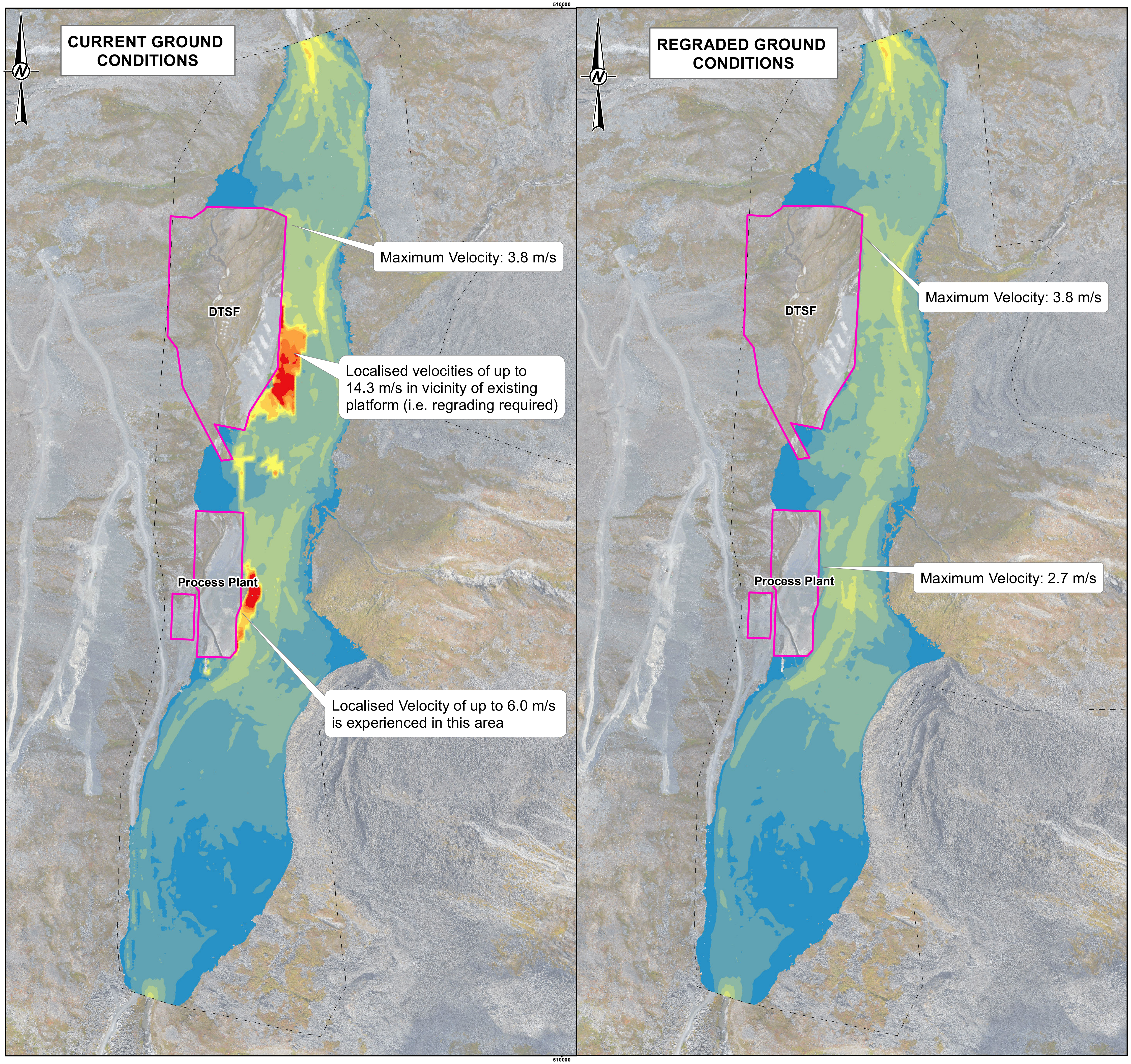

LEGEND

- Model Extent
- Updated DTSF and Process Plant

Maximum Velocity (m/s)

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- 7 - 8
- 8 - 9
- 9 - 10
- >10

0 250 500

1:6,750 METRES

NOTE(S)
1. LINE NOTES
2. LINE NOTES
3. LINE NOTES

REFERENCE(S)
1. COORDINATE SYSTEM: WGS 1984 UTM ZONE 23N
2. SERVICE LAYER CREDITS:

CLIENT
AEX GOLD

PROJECT
NALUNAQ

TITLE
DEVELOPED SITE CONDITIONS. UPDATED LAYOUT - PMP EVENT. MAXIMUM VELOCITY.

CONSULTANT

GOLDER

YYYY-MM-DD	2021 JAN 14
DESIGNED	CB
PREPARED	CB
REVIEWED	JR
APPROVED	KD

PROJECT NO.	CONTROL	REV.	DRAWING
20136781	SW-610-0043	A.0	D6-2

APPENDIX E

Historical Flooding Photos

Figure E1: Observed Flooding at the site in 2008 (photo provided by AEX Gold). Exact timing and location of the photo is unknown.

Figure E2: Observed Flooding at the site in 2008 (photo provided by AEX Gold). Exact timing and location of the photo is unknown.

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APPENDIX B

Nalunaq Mine Inflows Assessment: Groundwater and Surface Water (Golder, 2021c)

TECHNICAL MEMORANDUM

DATE 12 January 2021 **Reference No.** 20136781.618.A0

TO Samuel Martel, Martin Menard, Nalunaq A/S

CC Jo Birch

FROM Gareth Digges La Touche **EMAIL** gdltouche@golder.com

NALUNAQ GOLD PROJECT: MINE INFLOW ASSESSMENT – GROUNDWATER AND SURFACE WATER

1.0 INTRODUCTION

Following discovery of the Nalunaq Gold Mine in southern Greenland in the early 1990s and development and operation by Crew Gold Corporation ("Crew Gold"), development was continued by Angus & Ross plc and Angel Mining (Gold) A/S, between 2004 and 2013. Subsequently additional exploration work has been undertaken in the Nalunaq area. It is understood that Nalunaq A/S ("Nalunaq") are aiming to restart mining operations in 2021.

Golder Associates (UK) Ltd. ("Golder") have been contracted to Nalunaq A/S to provide support for water and tailings management at their Nalunaq Mine. More specifically, Golder has undertaken the following:

- An assessment of the potential groundwater inflow rates to the Nalunaq Mine (specifically the South, Target and Mountain Blocks (Section 2.0); and
- An assessment of the potential inflows to Valley Block (Section 3.0) comprising:
  - A qualitative assessment of the risk of groundwater inrush and the necessary standoff between the flooded South Block and the Valley Block;
  - An assessment of the potential rate of groundwater inflow to the Valley Block through the duration of the exploration drift construction (assuming no engineered connection to South Block); and
  - A qualitative assessment of risks from surface water inflows to the Valley Block 235 Level portal due to flooding of the Kirkespir River and surface water runoff from the overhanging slopes.

Groundwater inflow rates of approximately 50 m^3/hour have been reported by Angel Mining (2009) compared with an average flow of 64 m^3/hour in 2007 and 2008 and a maximum flow of 175 m^3/hour in May 2008 reported by Golder (2009; Figure 1). It is noted that the recorded 2007 and 2008 flows may include both natural groundwater inflows and losses from operational uses such a drilling water. No meteorological data is available for the period to identify the impact of precipitation events.

In this Technical Memorandum are presented the results of a number of analytical calculations to benchmark the reasonableness of these numbers based on typical hydraulic conductivity values for the fractured bedrock in the vicinity of the mine. In addition, we have assessed the potential inflow to the Valley Block development. The results of these calculations are presented in this Technical Memorandum.

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It should be noted that these calculations are order of magnitude estimates and are subject to considerable uncertainty. It should be noted that based on our current understanding of the mine environment that groundwater ingress to the current mine workings will vary both seasonally and in response to rainstorm events. We have made an estimate of the potential seasonality of these flows based on the currently available data.

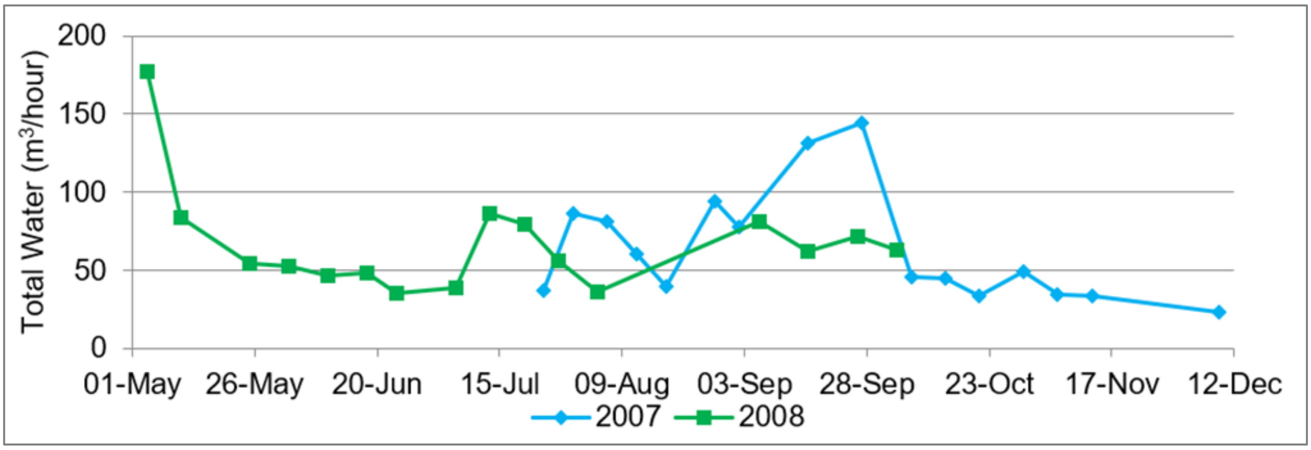


Figure 1: Available mine outflow data (Golder, 2009)

2.0 SOUTH, TARGET AND MOUNTAIN BLOCKS GROUNDWATER INFLOW

2.1 South, Target and Mountain Blocks Water Balance

The potential discharge from the mine can be estimated based on a simple water balance assuming that all the precipitation that falls on the surface catchment overlying the mine either infiltrates to the mine workings and from there is channelled to the mine portal or runs off into the Kirkespirdalen.

The average annual precipitation is estimated as approximately 602 mm (Golder, 2020a). Based on a working assumption that between 25% and 75% of the precipitation either runs off (RO) or is returned to the atmosphere via evapotranspiration (ET) or sublimation, for the purpose of this assessment we have assumed that between approximately 150 mm/year and 300 mm/year infiltrates. Given an estimated surface catchment area of approximately 661,218 m^2 (Figure 2) inflow rates of approximately 99,183 m^3/year (11 m^3/hour), 198,218 m^3/year (23 m^3/hour) and 298,540 m^3/year (34 m^3/hour) are calculated for the 75%, 50% and 25% RO and ET loss assumptions, respectively.

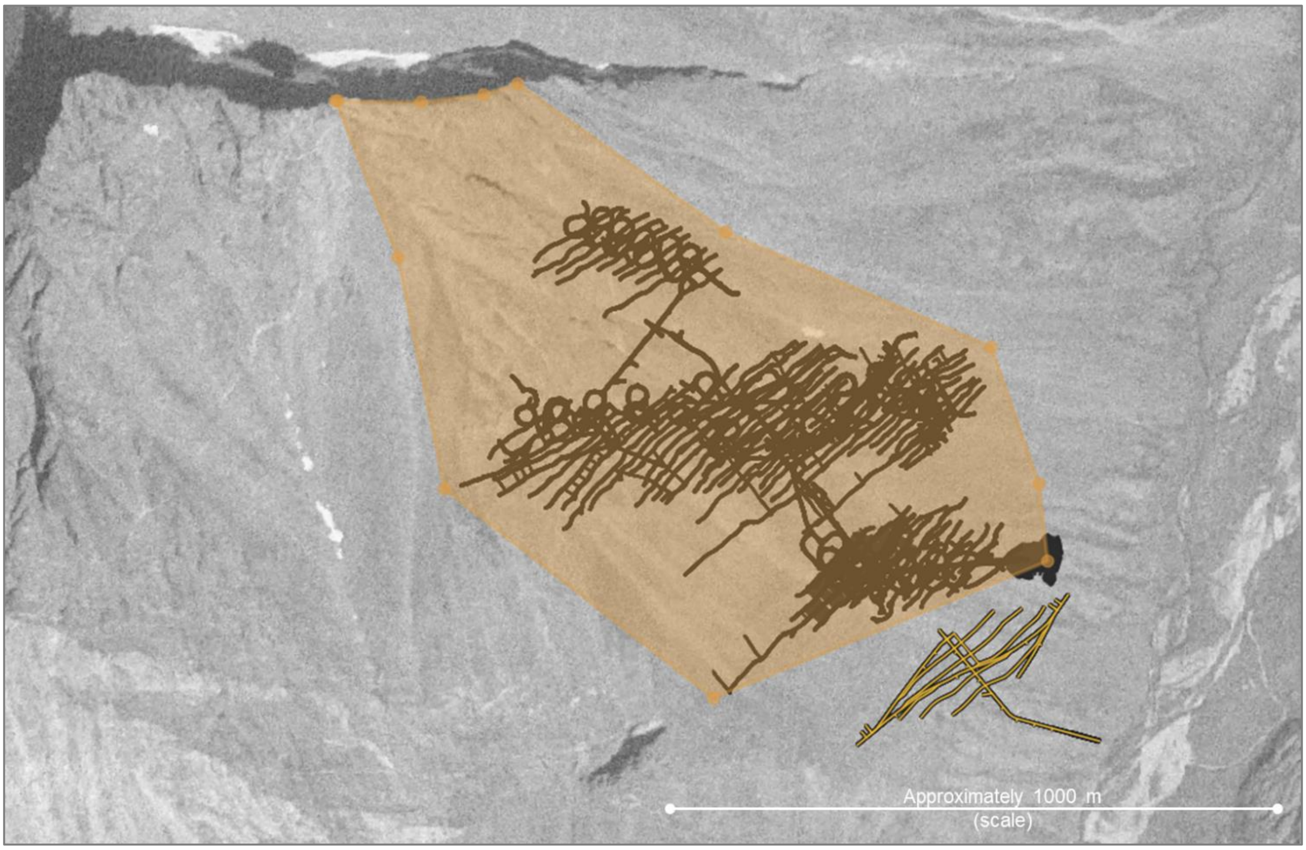


Figure 2: Estimated surface catchment area (661,218 m^2) for infiltration to the South, Target and Mountain Blocks of the Nalunaq Mine

To estimate the potential monthly variation in flows the monthly precipitation data presented in Golder 2020a and reproduced in Table 1 has been used using the same RO and ET/sublimation assumptions. For the purpose of the calculations it has been assumed that during December through March recharge is reduced to just the rainfall component of precipitation on the basis that the majority of precipitation is held in storage in the snowpack until the spring thaw, with some occurring as a result of melting at the base of the snow pack and rainfall infiltrating through the snowpack during rain on snow events. In April and May it is assumed that the snow component is not available due to sublimation and just the rainfall component is used to calculate the recharge plus in each month 50% of the precipitation that fell as snowfall during December to March to account for snow melt during the spring thaw. The results of the calculations are presented in Table 2 and Figure 3. It is noted that the assessment reflects the peak flow reported in May, as shown in Figure 1, by Golder (2009).

Table 1: Average Monthly Precipitation at Narsarsuaq Station (1973 – 2003)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)	44.0	37.7	35.6	45.6	35.8	57.4	58.2	64.6	73.8	57.6	47.6	43.9	601.8
Rainfall (mm)	3.2	7.5	2.4	33.5	35.0	57.4	58.2	64.6	73.1	50.4	16.2	6.4	407.8
Snowfall (mm)	40.7	30.3	33.3	12.2	0.8	0.0	0.0	0.0	0.6	7.2	31.4	37.5	194.0

Table 2: Water balance-based inflow assessment for South, Target and Mountain Blocks based on varying runoff, evapotranspiration, sublimation rates

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Infiltration (mm) assuming 25% RO/ET	3.2	7.5	2.4	96.0	97.2	43.1	43.7	48.5	55.4	43.2	35.7	6.4
Inflow (m^3/month)	2116	4959	1587	63493	64237	28465	28862	32036	36598	28565	23605	4232
Inflow (m^3/hour)	3	7	2	87	88	39	40	44	50	39	32	6
Infiltration assuming (mm) 50% RO/ET	3.2	7.5	2.4	87.7	88.8	28.7	29.1	32.3	36.9	28.8	23.8	6.4
Inflow (m^3/month)	2116	4959	1587	57956	58716	18977	19241	21357	24399	19043	15737	4232
Inflow (m^3/hour)	3	7	2	79	80	26	26	29	33	26	22	6
Infiltration (mm) assuming 75% RO/ET	3.2	7.5	2.4	79.3	79.9	14.4	14.6	16.2	18.5	14.4	11.9	6.4
Inflow (m^3/month)	2116	4959	1587	52418	52798	9488	9621	10679	12199	9522	7868	4232
Inflow (m^3/hour)	3	7	2	72	72	13	13	15	17	13	11	6

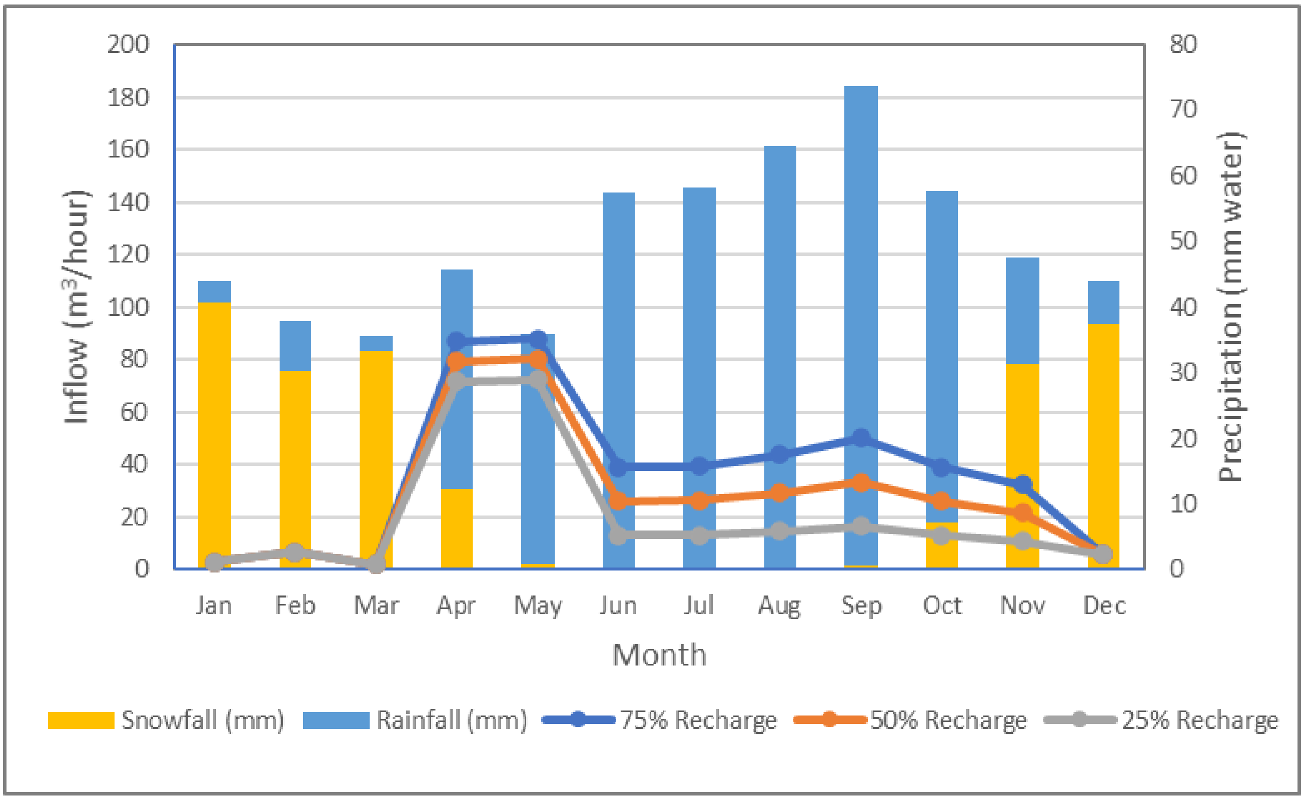


Figure 3: Calculated inflows to South, Target and Mountain Blocks plotted by month

The calculated inflows are of the same order of magnitude as the average inflow rate stated by Angel Mining (2009) (50m^3/hour), however the peak inflows are less than the maxima reported by Golder (2009). The underground mine will have a larger groundwater catchment than surface water catchment, due to the depressurisation effect of the draining workings on the surrounding rock mass which is considered likely to extend the radius of influence of the mine drainage on groundwater, so the number stated by Angel Mining (2009) is not considered unreasonable in this context, although there are no data to support the value. In addition it is possible that the recorded higher flow values include drill water which has been supplied to the mine, thus artificially increasing the outflows.

2.2 South, Target and Mountain Block Inflow Calculation Methods

The potential groundwater inflows to the mine have been calculated using the methods of Goodman (1965) and Hantush (Singh and Atkins, 1985). These methods are designed for calculating inflow to tunnels and single underground voids respectively but may be applied to give order of magnitude estimates to mine workings.

2.2.1 Goodman

The steady state inflow (Q) to a single linear tunnel may be calculated using the method of Goodman (1965) as follows:

$$Q = \frac{2\pi K L H_0}{ln\left(\frac{4H_0}{D}\right)}$$

Where:

K is the hydraulic conductivity (m/s);

L is the tunnel length (m);

H_o is the head of water above the tunnel (m); and

D is the tunnel diameter (m).

The input assumptions are used across a range of hydraulic conductivities (1 x 10^{-7} m/s to 1 x 10^{-10} m/s) and are presented on the calculation sheets presented as APPENDIX A. The calculated inflows ranged from approximately 0.074 m^3/hour to approximately 74 m^3/hour.

For the purpose of comparison only, assuming the average discharge rate of 50 m^3/hour reported by Angel Mining (2009) is valid the hydraulic conductivity value was varied such that the calculation returned a flow rate of 50 m^3/hour. The resulting calculated bulk hydraulic conductivity of the bedrock is approximately 6.73 x 10^{-8} m/s based on the assumptions used such as adit length and head of water remaining constant. This is within the range of 1 x 10^{-7} m/s to 1 x 10^{-10} m/s assumed as likely for the bedrock of the Nalunaq Mine.

2.2.2 Hantush

The steady state inflow (Q) to an underground void tunnel may be calculated using the method of Hantush (Singh and Atkins, 1985) as follows:

$$Q = 2\pi TDG\left(\lambda, \frac{r}{B}\right)$$

Where:

T is the transmissivity (m^2/s);

D is the depth of the workings below the piezometric surface (m);

λ is the Hantush well function;

r is the hydraulic gradient (m/m);

B is the leakage factor; and

G is derived from λ and r/B.

The input assumptions are used across a range of hydraulic conductivities (1 x 10^{-7} m/s to 1 x 10^{-10} m/s) and are presented on the calculation sheets presented as APPENDIX B. The calculated inflows ranged from approximately 1 m^3/hour to approximately 97 m^3/hour.

2.3 South, Target and Mountain Block Groundwater Inflows

As set out above the range of inflows presented in Figure 3 range between approximately 2 m^3/hour to 88 m^3/hour. These inflows, based on a water balance, are of a similar order of magnitude to those calculated using the methods of Goodman and Hantush 0.074 m^3/hour to 97 m^3/hour as set out in Section 2.2. On the basis of the calculations presented above, the average annual flow rates reported by Angel Mining (2009) and the maximum flow rates reported (Golder, 2009) it is recommended that the upper bound value is scaled by a factor of safety of 2 and that for the purpose of water balance modelling the values presented in Figure 4 and Table 3 are used. It is noted that the assessment reflects the peak flow reported in May, as shown in Figure 1, by Golder (2009).

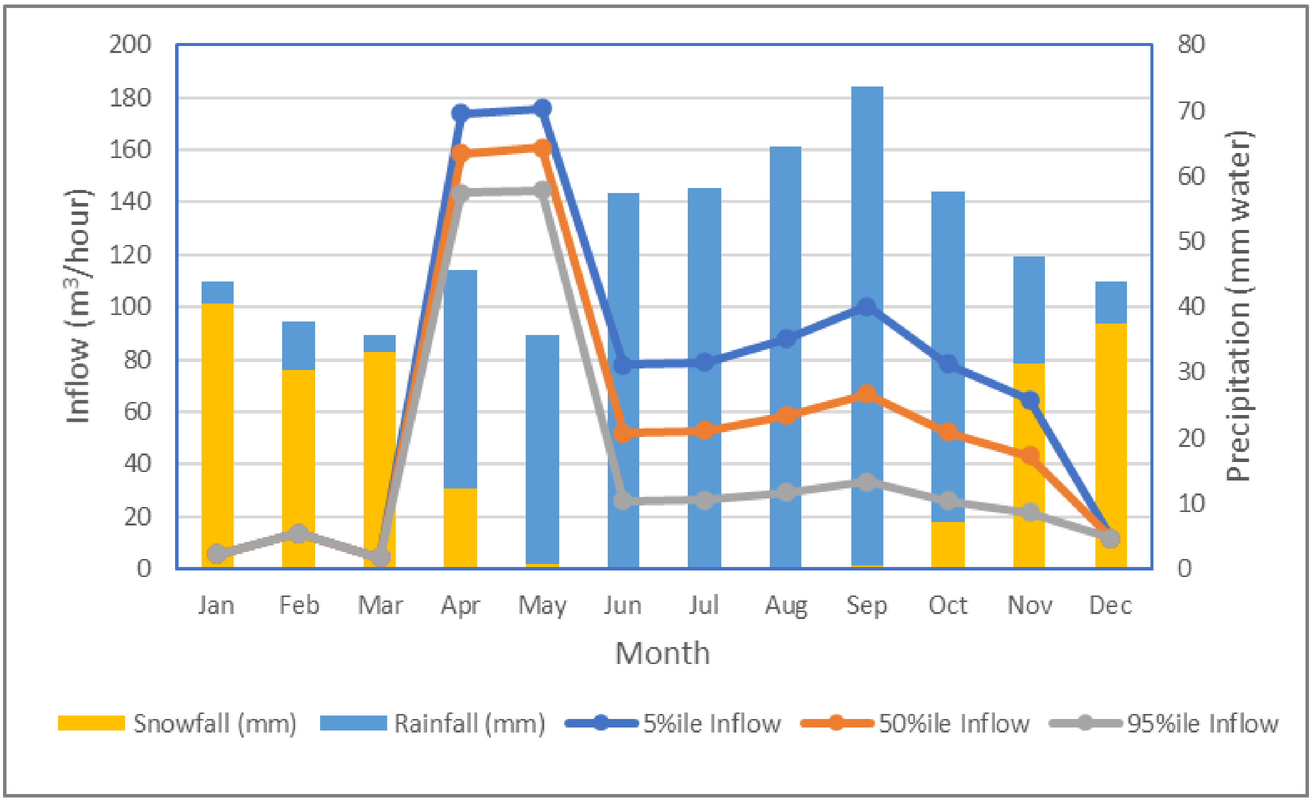


Figure 4: Assessed inflow rates to South, Target and Mountain Blocks for the purpose of water management modelling

Table 3: Assumed inflow rates to South, Target and Mountain Blocks for the purpose of water management modelling

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Assumed 5%ile Inflow (m^3/hour)	6	14	4	174	176	78	79	88	100	78	65	12
Assumed 50%ile Inflow (m^3/hour)	6	14	4	159	161	52	53	59	67	52	43	12
Assumed Minimum (95%ile) Inflow (m^3/hour)	6	14	4	144	145	26	26	29	33	26	22	12

3.0 VALLEY BLOCK INFLOWS

3.1 Introduction

The purpose of the inflow assessment for the Valley Block has the following elements:

- A qualitative assessment of the risk of groundwater inrush and the necessary standoff between the flooded South Block and the Valley Block;
- An assessment of the potential rate of groundwater inflow to the Valley Block through the duration of the exploration drift construction (assuming no engineered connection to South Block); and
- A qualitative assessment of risks from surface water inflows to the Valley Block 235 Level portal due to flooding of the Kirkespir River and surface water runoff from the overhanging slopes.

3.2 Conceptual Model

As set out in Golder 2020b the mine is situated in the basement rocks of south Greenland. Dominey *et al*. (2006) report that the site lies in the Psammite Zone which is a supracrustal succession of psammites with pelites and interstratified mafic volcanic rocks with gold mineralisation at Nalunaq hosted by a meta-volcanic unit composed of basaltic pillow lavas and pyroclastics intruded by dolerite sills. The volcanic rocks are reported (Dominey *et al*., 2006) to be metamorphosed to amphibolites and the area is intruded by late- and post-tectonic granitoid plutons. A geological map of the area in the vicinity of the mine is presented at Figure 5. The bedrock in the area is variably weathered at surface but becomes fresh at shallow depth, typically 20 m to 30 m from surface.

The Nalunaq deposit is divided into four main structural blocks. From southeast to northwest these are Valley Block, South Block, Target Block and Mountain Block. South Block and Target Block are separated by the Pegmatite Fault causing approximately 80 m of vertical offset of South Block relative to Target Block, and dextral displacement of approximately 85 m (SRK, 2016).

Two further faults crosscut the orebody, the shallow dipping Your Fault and the more steeply dipping Clay Fault. Both faults typically show less than 5 m of displacement (Golder, 2020c). The immediate zone around the Clay Fault is described (Golder, 2020c) as being highly disturbed whilst the ground leading up to it and beyond does not appear to be any more heavily fractured than surrounding areas.

The bedrock porosity is provided by fractures. Fracture flow is likely to be highly anisotropic and although open fractures will act as conduits to flow, fracture coatings or infills may cause fractures to act as barriers to flow potentially giving rise to perched water in places. With depth the bedrock rock quality designation (RQD) indicates good to excellent quality with values frequently over 90% (Golder, 2020d). The rock is likely to exhibit low hydraulic conductivity due the crystalline nature of the matrix although fractures are likely to facilitate fluid flow. The hydrogeological conceptual model is presented in Golder 2020d and is summarised in Figure 6.

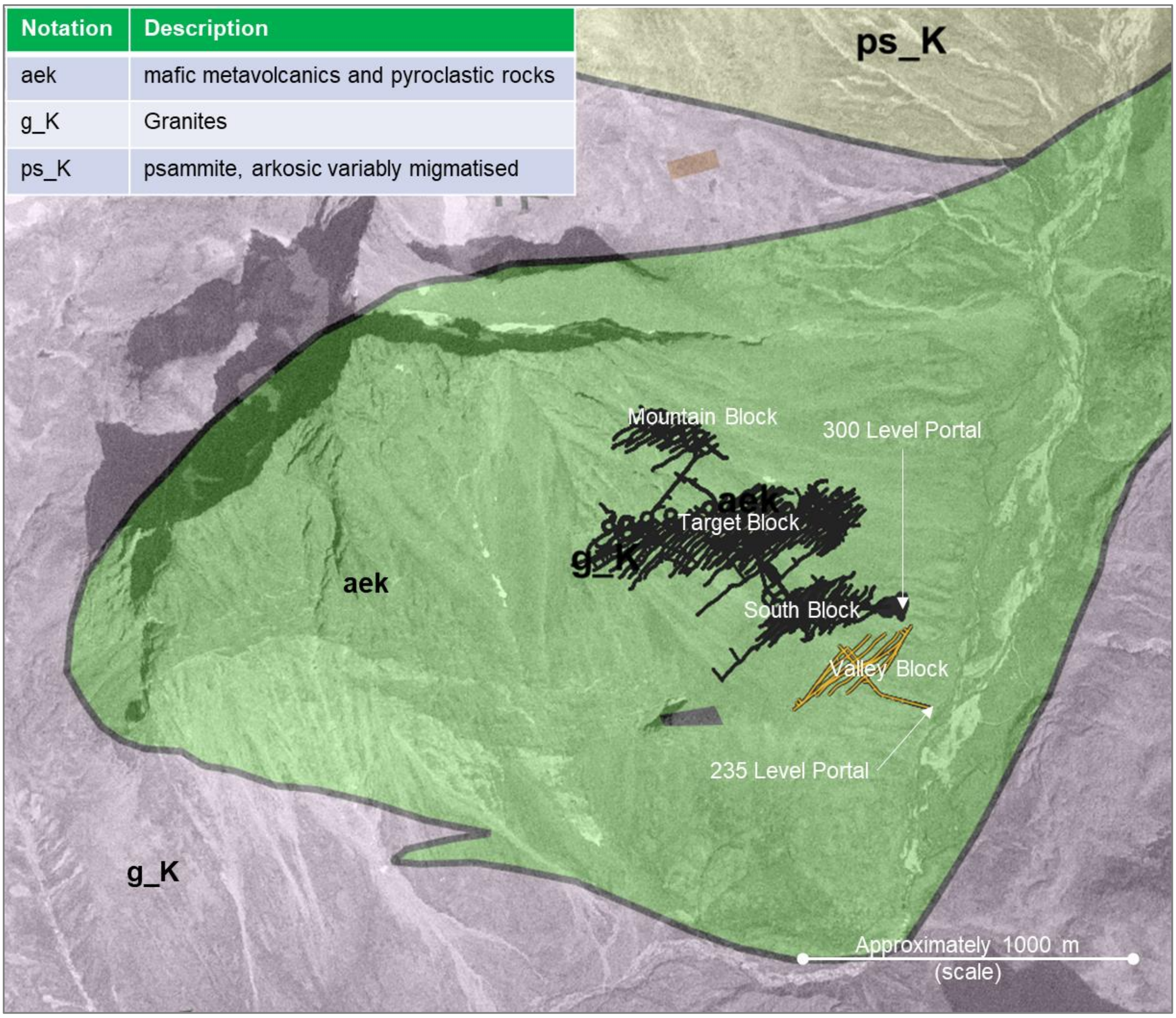


Figure 5: Geological map of the area in the vicinity of the Nalunaq Mine (GEUS, 2019)

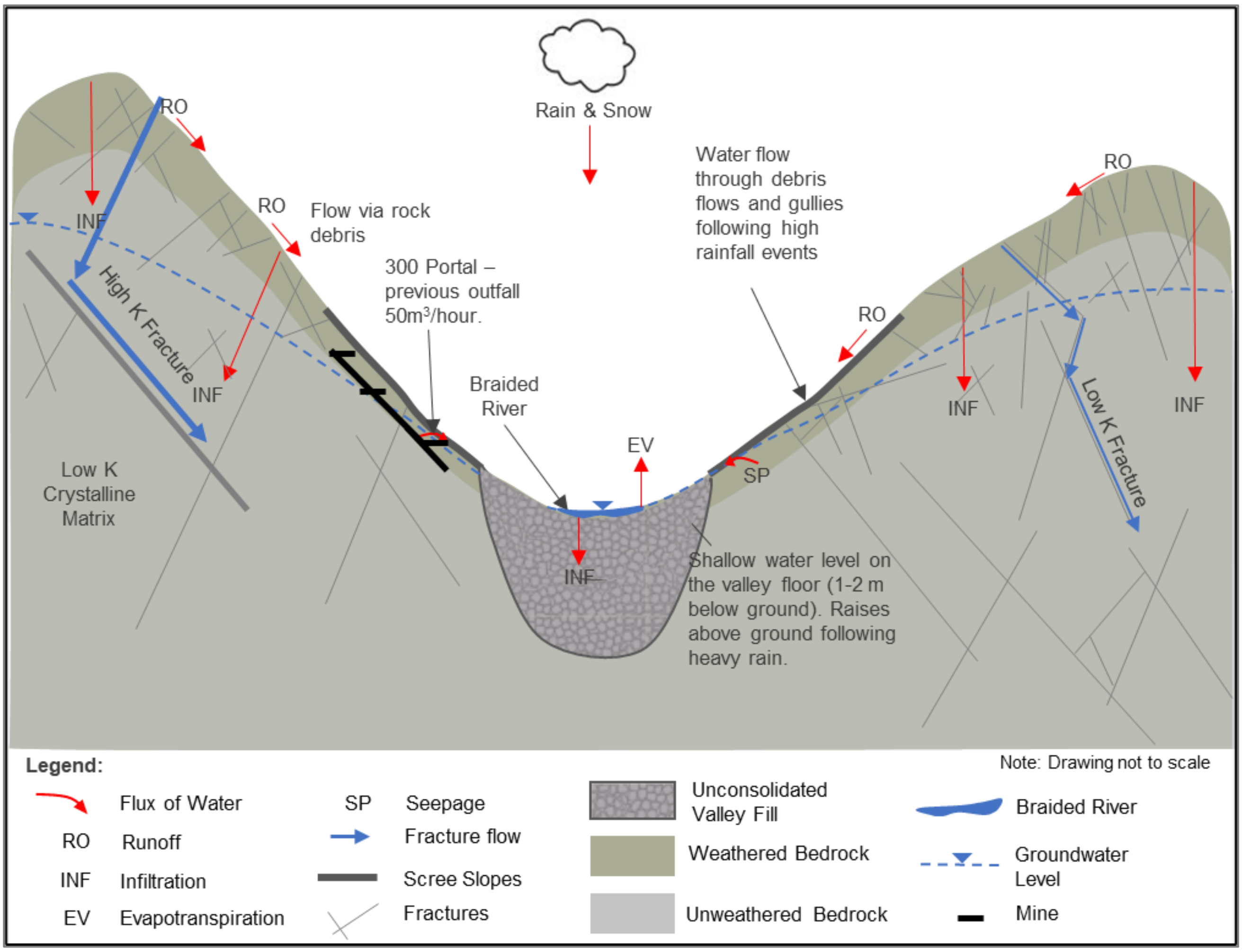


Figure 6: Conceptual model of the bedrock hydrogeology in the vicinity of the Nalunaq Mine showing the interaction with the superficial deposits

The inflow of groundwater to the Valley Block will be derived from a number of sources:

- Infiltration of recharged precipitation through the mountain;
- Inflow from the fluvioglacial deposits infilling Kirkespirdalen; and
- Inflows from the flooded South Block.

The potential for rapid inflows from the flooded South Block to the Valley Block has also been assessed and the results and recommendations of that assessment are presented below.

3.3 Groundwater Inrush Hazard

Due to the proximity of the Valley Block to the flooded South Block an assessment of the potential inrush hazard has been undertaken. For an inrush hazard to be realised the ground between the two areas of working needs to either be weak from a rock mechanics perspective and thus fail resulting in a connection via highly permeable ground or there needs to be a high permeability connection via fractures/faults, or other permeable ground, or other means such as exploration boreholes.

As shown on Figure 5 and Figure 8 the Valley Block is proposed to be developed to within approximately 47 m of the South Block, but that at no point does the South Block directly overlie the Valley Block. As shown on

Figure 8 the Valley Block is bounded by the Justinas Fault. Fracture mapping has been undertaken in South Block with fracture trace lengths of 0.2 m to 10 m being reported, with an average trace length of 2.3 m with a standard deviation of 2.6 m (Golder, 2020c). Based on this data it is considered unlikely that there will be a direct fracture-controlled pathway linking the two working areas. No fault structures are currently known to directly connect the Valley Block and South Block.

The United Kingdom Health and Safety Executive (HSE) has developed an Approved Code of Practice (ACoP) with respect to the prevention of inrushes (HSE, 1993) which provides statutory guidance on the *Mines (Precautions Against Inrushes) Regulations 1979* (PAIR) and the *Management and Administration of Safety and Health at Mines Regulations 1993* (MASHAM). As set out in the ACoP, Regulation 6 of PAIR prohibits a mine working which would be within 37 m of any disused mine workings or 45 m of any disused workings (which includes disused shafts and boreholes) or 45 m of any other potentially hazardous areas specified in the Regulations unless the manager follows laid down procedures. “Other potentially hazardous” areas are defined in the ACoP as the ground “surface, water bearing strata, unconsolidated deposits and disused workings not being mine workings”. As stated above the Valley Block is separated from the South Block by approximately 47 m, hence meets the requirements of the ACoP assuming that there are no adverse geotechnical conditions (i.e. weak ground).

An assessment of potential inflows assuming high permeability ground does exist between the Valley Block and South Block has been undertaken. For the purpose of the assessment, it is assumed that the ground has a hydraulic conductivity of 1 x 10^{-5} m/s, which is two orders of magnitude greater than the hydraulic conductivity reported in Golder (2020d). A high hydraulic conductivity is used to provide a conservative assessment of inflows. The potential inflows were calculated using a range of methods (Darcy’s Law, Goodman (1965) and Heuer (1995, 2005) as set out in APPENDIX C). A worst case inflow of 0.38 m^3/s (approximately 1,365 m^3/hour) is calculated using the method of Goodman (1965). The inflows calculated using the other methods were of a similar magnitude.

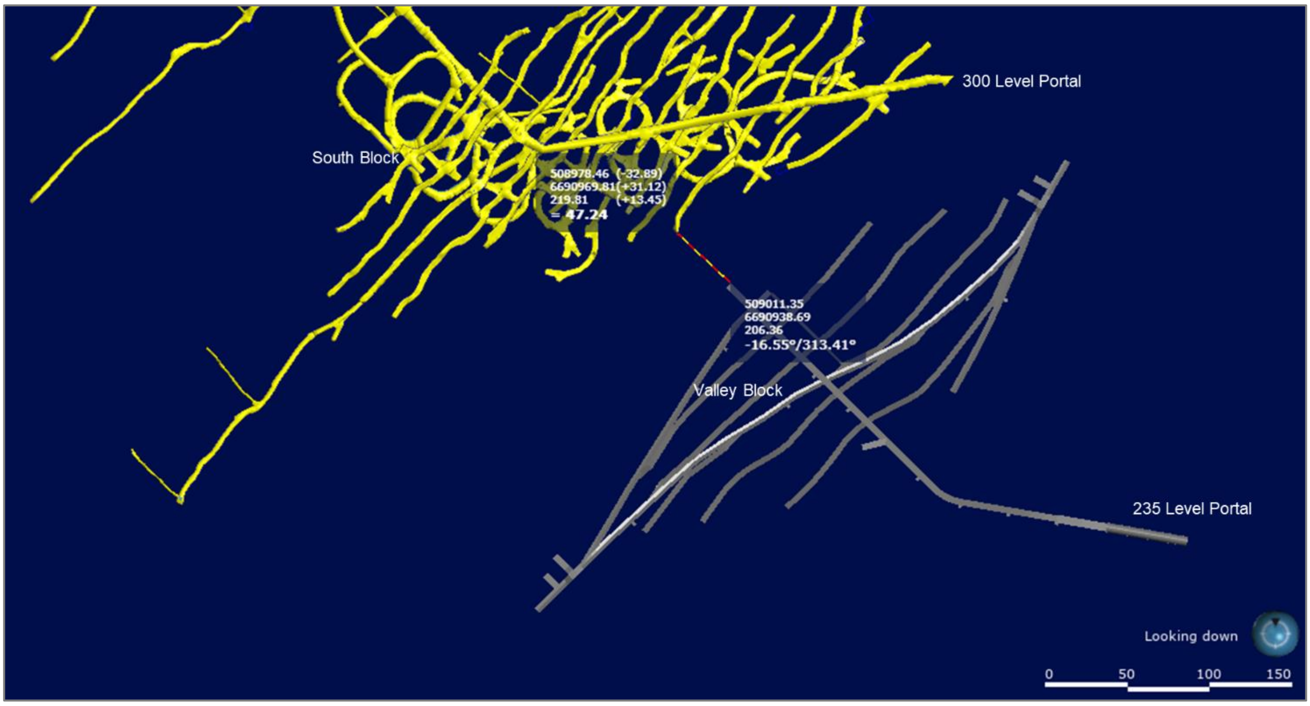


Figure 7: Vertical view of Valley Block and South Block

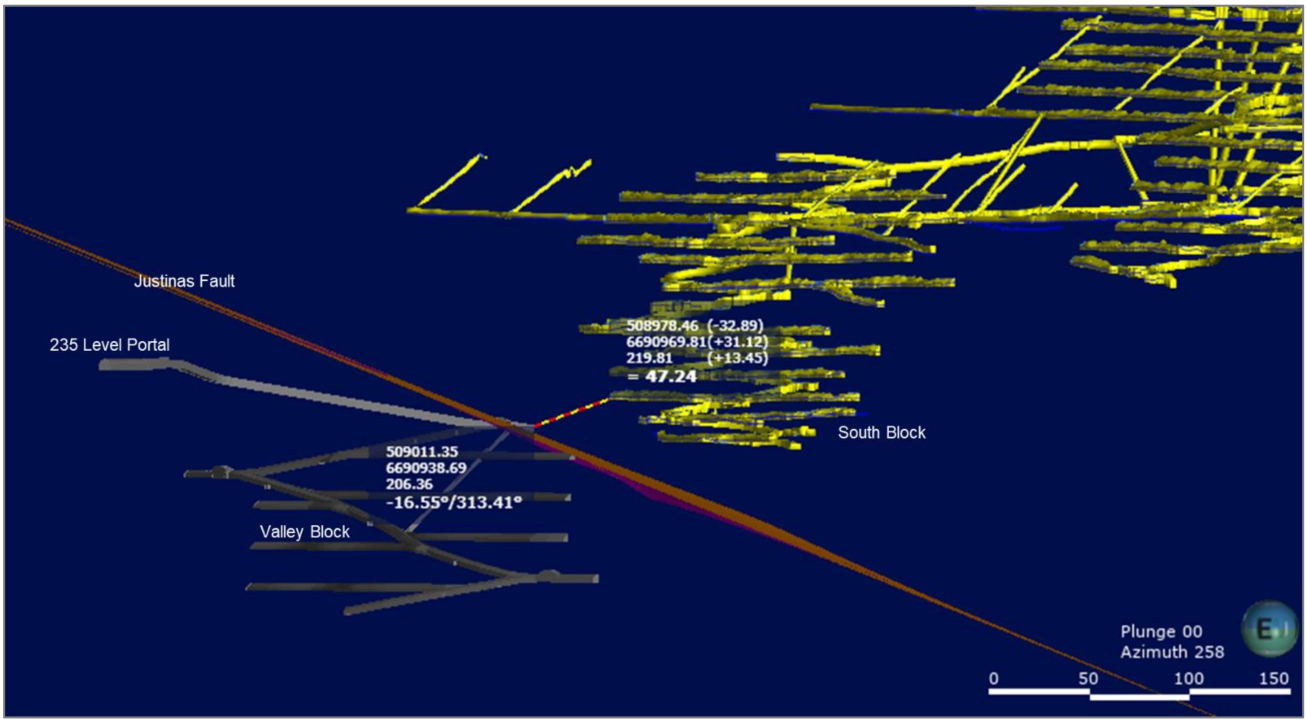


Figure 8: View of Valley Block and South Block (showing the Justinas Fault (250 Level Fault not shown)) in the direction of 258º

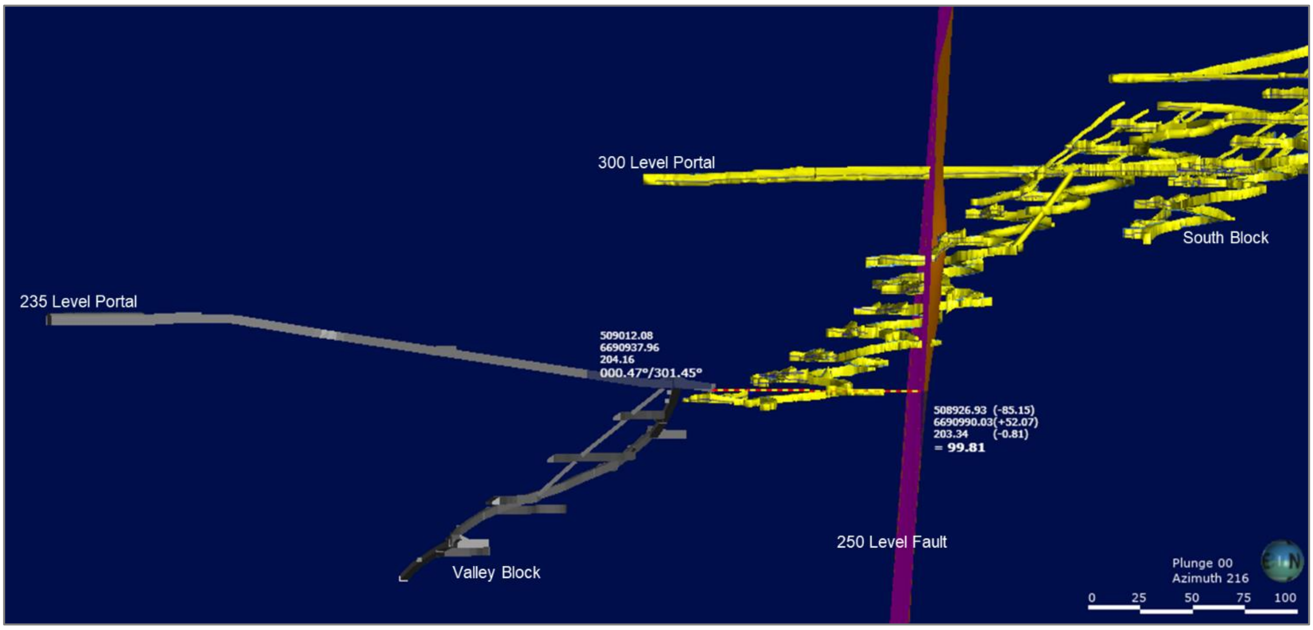


Figure 9: View of Valley Block and South Block (showing the 250 Level Fault (Justinas Fault not shown)) in the direction of 216º

3.4 Groundwater Inflows to the Valley Block

As stated in Section 3.2 there are three potential sources of inflow to the Valley Block:

- Infiltration of recharged precipitation through the mountain;
- Inflow from the fluvioglacial deposits infilling Kirkespirdalen; and
- Inflows from the flooded South Block.

These are assessed separately. Flows from the flooded South Block will be relatively constant as there will be a constant pressure gradient between the two Blocks. Likewise, the inflows from the fluvioglacial deposits are not anticipated to vary greatly with time, although some increase will occur as the development gets deeper. The main variation, as with South, Target and Mountain Block will result from seasonal variations in recharge through the rock mass above the open workings. The calculation of inflows from the three components is set out below.

3.4.1 Recharge Infiltration

As set out in Section 2.1, with regard to South, Target and Mountain Blocks, the direct recharge component of the potential discharge from the mine can be estimated based on a simple water balance assuming that a proportion of the precipitation that falls on the surface catchment overlying the mine infiltrates to the Valley Block and from there is channelled to the mine portal, while the remainder runs off into the Kirkespirdalen.

The average annual precipitation is estimated as approximately 602 mm (Golder, 2020a) and the surface catchment area is estimated as 146,933 m^2 (Figure 10). To estimate the potential monthly variation in flows the monthly precipitation data presented in Golder 2020a and reproduced in Table 1 has been used using the same RO and ET/sublimation assumptions as set out in Section 2.1. The results of the calculations are presented in Figure 11 and Table 4.

Figure 10: Estimated surface catchment area (146,933 m²) for infiltration to the Valley Block of the Nalunaq Mine

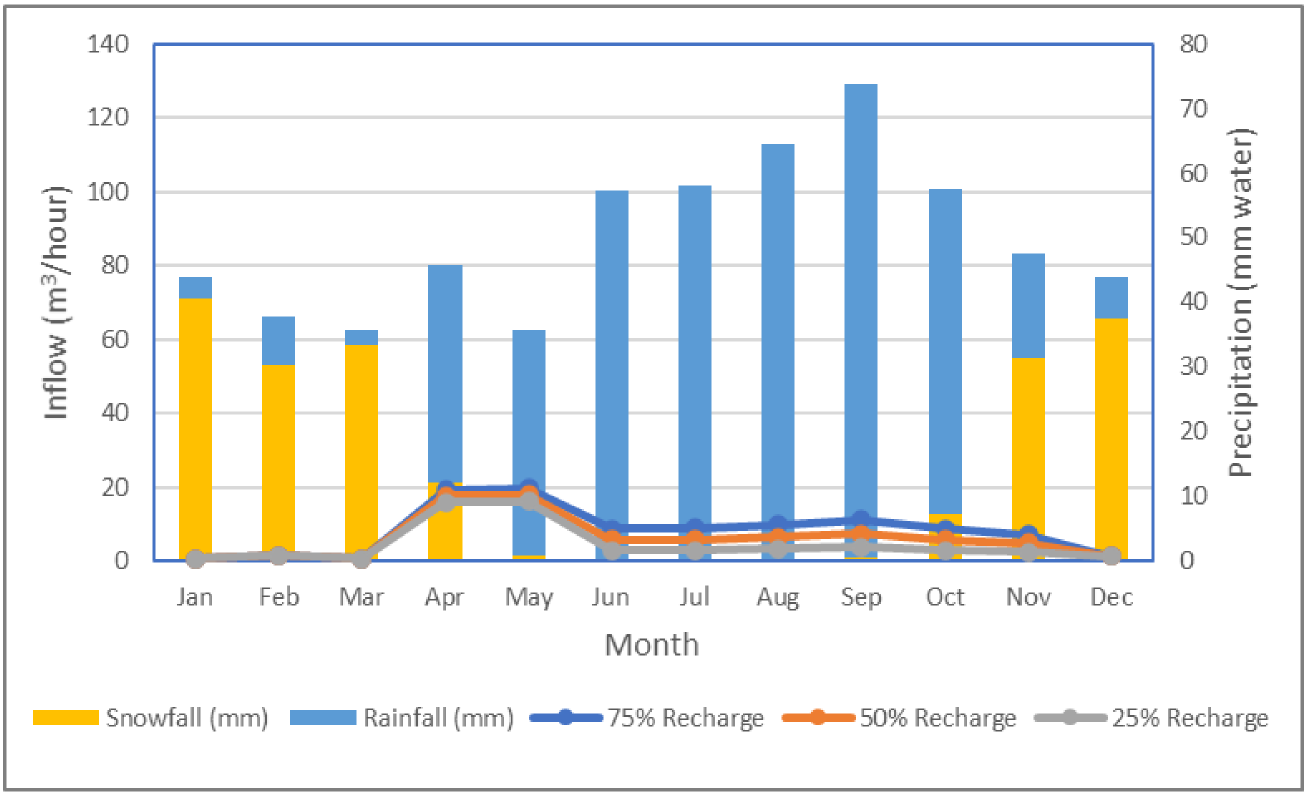


Figure 11: Calculated recharge groundwater inflows to Valley Block plotted by month

Table 4: Water balance-based groundwater inflow assessment for Valley Block based on varying runoff, evapotranspiration, sublimation rates

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Infiltration assuming 25% RO/ET	3.2	7.5	2.4	96.0	97.8	43.1	43.7	48.5	55.4	43.2	35.7	6.4
Inflow (m^3/month)	470	1102	353	14109	14363	6325	6414	7119	8133	6348	5246	940
Inflow (m^3/hour)	1	2	0.5	19	20	9	9	10	11	9	7	1
Infiltration assuming 50% RO/ET	3.2	7.5	2.4	87.7	88.8	28.7	29.1	32.3	36.9	28.8	23.8	6.4
Inflow (m^3/month)	470	1102	353	12879	13048	4217	4276	4746	5422	4232	3497	940
Inflow (m^3/hour)	1	2	0.5	18	18	6	6	7	7	6	5	1
Infiltration assuming 75% RO/ET	3.2	7.5	2.4	79.3	79.9	14.4	14.6	16.2	18.5	14.4	11.9	6.4
Inflow (m^3/month)	470	1102	353	11648	11733	2108	2138	2373	2711	2116	1749	940
Inflow (m^3/hour)	1	2	0.5	16	16	3	3	3	4	3	2	1

3.4.2 Inflow from the Fluvioglacial Deposits

The inflows to the Valley Block from the fluvioglacial deposits are controlled by the hydraulic conductivity of the intact bedrock (assumed to be 1 x 10^{-7} m/s) and the head difference (120 m) between groundwater levels in the fluvioglacial deposits (approximately 234 masl) and the base of the Valley Block (approximately 114 masl). The inflows are evaluated using the methods of Heuer (1995, 2005) and Goodman (1965). The calculated rate of inflows ranged from 0.012 m^3/s (approximately 43 m^3/hour) to 0.026 m^3/s (approximately 93 m^3/hour). The results of the calculation are presented in APPENDIX C.

3.4.3 Inflows from South Block

The inflows from the South Block have been evaluated using the methods of Heuer (1995, 2005) and Goodman (1965) assuming a hydraulic conductivity of 1 x 10^{-7} m/s. The calculated rate of inflows ranged from 0.002 m^3/s (approximately 6 m^3/hour) to 0.004 m^3/s (approximately 14 m^3/hour). The results of the calculation are presented in APPENDIX C.

3.4.4 Total Groundwater Inflows

The total inflows are derived by combining the three identified components to derive the flow rates for the purpose of water balance modelling and are presented in Table 5 and Figure 12.

Table 5: Assumed groundwater inflow rates to Valley Block for the purpose of water management modelling

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Assumed 5%ile Inflow (m^3/hour)	108	109	107	127	116	116	116	117	118	116	114	108
Assumed 50%ile Inflow (m^3/hour)	108	109	107	125	113	113	113	114	114	113	112	108
Assumed Minimum (95%ile) Inflow (m^3/hour)	108	109	107	123	110	110	110	110	111	110	109	108

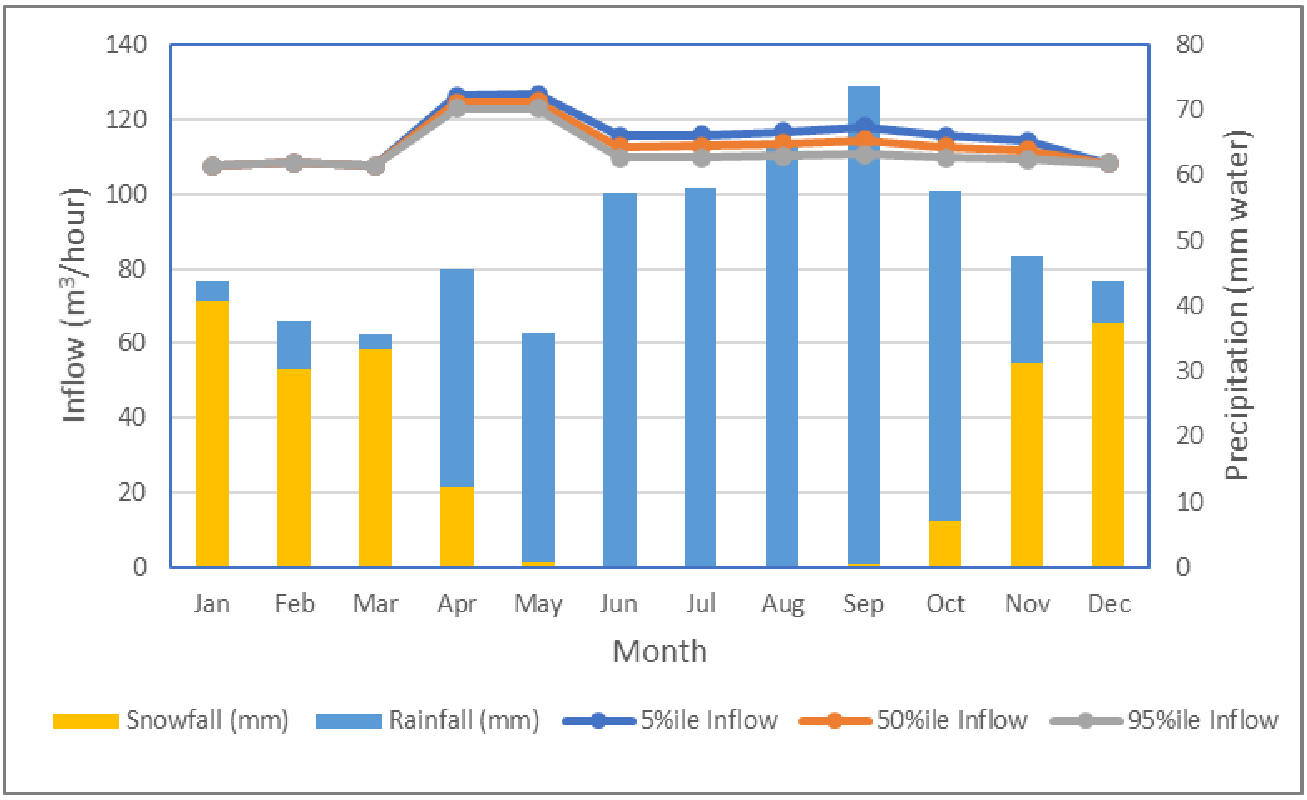
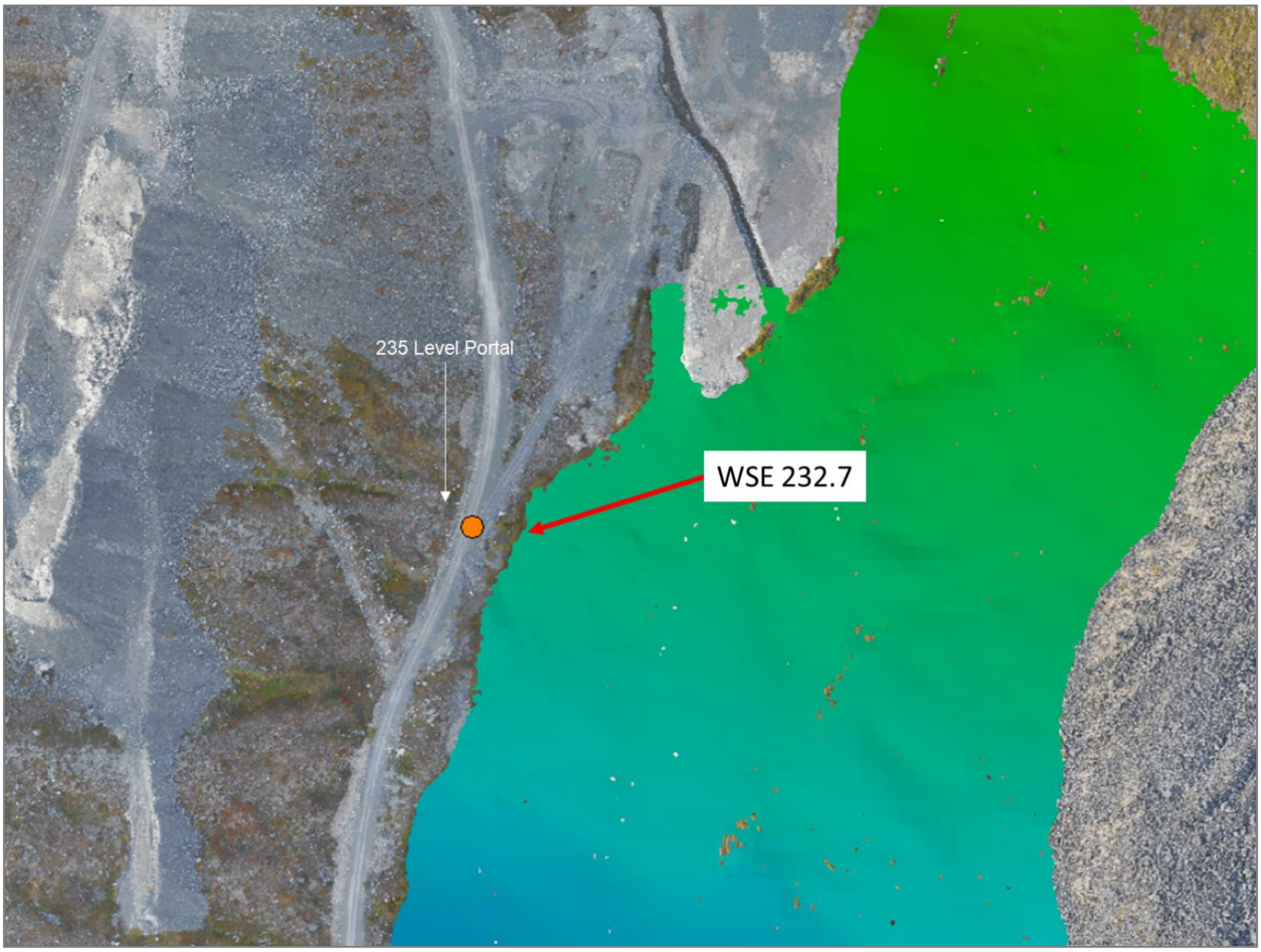


Figure 12: Assessed groundwater inflow rates to Valley Block for the purpose of water management modelling

3.5 Surface Water Ingress to Valley Block

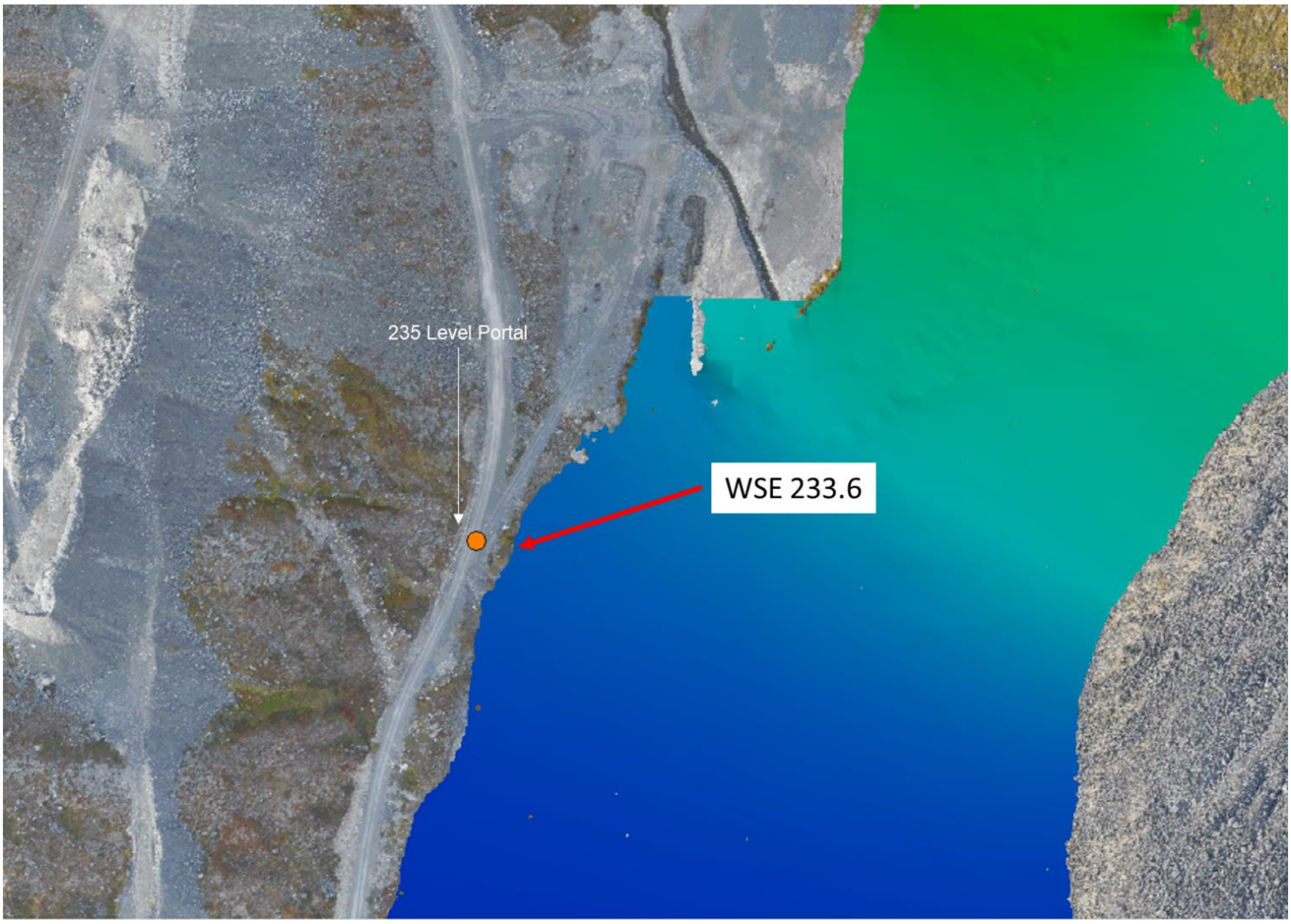
The proposed 235 Level portal is situated approximately 2 m above the level (232.7 masl) of the modelled 1:1000 year return period flood (Golder, 2020a) (Figure 13) (i.e. the flood event with a 0.1% probability of occurrence in any one year) and 1.4 m above the modelled level of the probable maximum flood (PMF 233.6 masl) (Golder, 2020a) (Figure 13). For the purpose of design, it is recommended that the initial entry is inclined upwards for the first 45 m to 75 m horizontal length of the adit at a gradient of 0.088 (5º) to allow free drainage of water from the drive and to provide a margin of safety with regard to flood levels.

Surface water diversion measures should be put in place to ensure that water from the road to the 300 Level portal is not inadvertently channelled into the 235 Level portal.



Note: WSE = water surface elevation.

Figure 13: Location of the 235 Level Portal relative to the 1:1000 year return period flood extent



Note: WSE = water surface elevation.

Figure 14: Location of the 235 Level Portal relative to the Probable Maximum Flood extent

4.0 CONCLUSIONS AND RECOMMENDATIONS

Groundwater inflows to the Nalunaq Mine have been calculated for the purpose of informing water management requirements. These have been calculated by month as follows for South, Target and Mountain Blocks; and for Valley Block, respectively (as originally presented in Table 3 and Table 5 above):

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
South, Target and Mountain Blocks												
Assumed 5%ile Inflow (m^3/hour)	6	14	4	174	176	78	79	88	100	78	65	12
Assumed 50%ile Inflow (m^3/hour)	6	14	4	159	161	52	53	59	67	52	43	12
Assumed Minimum (95%ile) Inflow (m^3/hour)	6	14	4	144	145	26	26	29	33	26	22	12
Valley Block												
Assumed 5%ile Inflow (m^3/hour)	108	109	107	112	112	116	116	117	118	116	114	108
Assumed 50%ile Inflow (m^3/hour)	108	109	107	110	111	113	113	114	114	113	112	108
Assumed Minimum (95%ile) Inflow (m^3/hour)	108	109	107	109	109	110	110	110	111	110	109	108

It is recommended that on the restart of operations a number of monitoring points are established in the mine and that v-notch weirs (see APPENDIX D for typical arrangements) are used to monitor the inflows to allow a refinement of this estimate and to establish the magnitude of seasonal variation and the response of the mine to rainstorm events.

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APPENDIX A

Groundwater Inflow Calculation Worksheet (Goodman) For Areas Above The 300 Level

Calculation of Groundwater Inflow to Underground Mine Workings

Inflows to 300 level

Parameter	Notation	Units	Most Likely	Justification
Hydraulic conductivity	K	m/s	6.73E-08	Value optimised to calculate the desired discharge rate of 50 m3/hour
Adit length	L_1	m	600	Approximate width of workings
Head of water	H_{o1}	m	300	Assuming inflows at 300 level and a water level at 600 masl
Adit diameter	D_1	m	5	Approximation of drive diameter
Inflow	Q_1	m^3/s	0.0139	Goodman et al (1965)
Total Inflow (m^3/day)	Q_T	$m^3/hour$	50.00	from Total inflow
Total Inflow (m^3/day)	Q_T	m^3/day	1200	from Total inflow
Total Inflow (Ml/day)	Q_T	Ml/day	1.20	from Total inflow

Inflows to 300 level

Parameter	Notation	Units	Minimum	Maximum	Justification
Hydraulic conductivity	K	m/s	1.00E-10	1.00E-07	Typical range for fractured bedrock
Adit length	L_1	m	600	600	Approximate width of workings
Head of water	H_{o1}	m	300	300	Assuming inflows at 300 level and a water level at 600 masl
Adit diameter	D_1	m	5	5	Approximation of drive diameter
Inflow	Q_1	m^3/s	0.000021	0.021	Goodman et al (1965)
Total Inflow (m^3/day)	Q_T	$m^3/hour$	0.074	74	from Total inflow
Total Inflow (m^3/day)	Q_T	m^3/day	2	1783	from Total inflow
Total Inflow (Ml/day)	Q_T	Ml/day	0.00	1.78	from Total inflow

Inflow, Q, calculated from Goodman et al (1965):

$$Q = \frac{2\pi K L H_0}{\ln\left(\frac{4H_0}{D}\right)}$$

Method from: Goodman, R.E., Moye, D.G., van Schalkwyk, A. and Javandel, I., 1965. Ground water inflows during tunnel driving. Engineering Geology, 2(1), pp. 39-56.

APPENDIX B

Groundwater Inflow Calculation Worksheet (Hantush) For Areas Above The 300 Level

Notation	Parameter	Units	Value		Comments
L	Thickness of mined hydrogeological unit	m	100.000	100.000	
L'	Thickness of saturated zone in the overlying formation(	m	300	300	
K	Hydraulic conducitivity of the mined hydrogeological un	m/s	1.00E-08	1.00E-08	
K'	Hydraulic Conductivity of the overling formation(s)	m/s	1.00E-07	1.00E-07	
S	Storativity of the mined hydrogeological unit	-	1.00E-03	1.00E-03	
D	Depth from base of workings to piezometric surface	m	300	300	
r	Radius of the excavation	m	600	300	
B		m	5.48E+01	5.48E+01	
r/B			1.10E+01	5.48E+00	
B	Check calculation only. Should = B above		5.48E+01	5.48E+01	
T	Transmisivity of the mined hydrogeological unit	m^2/s	1.00E-06	1.00E-06	
λ			4.E-01	2.E+00	
t	Elapsed time	years	5	5	
t	Elapsed time	s	1.58E+08	1.58E+08	
G	Hantush well function		1.81	1.44	From Table 4 using values of λ and r/B above
Q	Inflow (i.e. Pumping Rate)	m^3/s	3.41E-03	2.71E-03	
Q	Inflow (i.e. Pumping Rate)	$m^3/hour$	12	10	

Note: Parameter is entered as a single value
Value is calculated

Notation	Parameter	Units	Value		Comments
L	Thickness of mined hydrogeological unit	m	100.000	100.000	
L'	Thickness of saturated zone in the overlying formation(	m	300	300	
K	Hydraulic conducitivity of the mined hydrogeological ur	m/s	1.00E-07	1.00E-07	
K'	Hydraulic Conductivity of the overling formation(s)	m/s	1.00E-07	1.00E-07	
S	Storativity of the mined hydrogeological unit	-	1.00E-03	1.00E-03	
D	Depth from base of workings to piezometric surface	m	300	300	
r	Radius of the excavation	m	600	300	
B		m	1.73E+02	1.73E+02	
r/B			3.46E+00	1.73E+00	
B	Check calculation only. Should = B above		1.73E+02	1.73E+02	
T	Transmisivity of the mined hydrogeological unit	m^2/s	1.00E-05	1.00E-05	
λ			4.E+00	2.E+01	
t	Elapsed time	years	5	5	
t	Elapsed time	s	1.58E+08	1.58E+08	
G	Hantush well function		1.44	1.43	From Table 4 using values of λ and r/B above
Q	Inflow (i.e. Pumping Rate)	m^3/s	2.70E-02	2.70E-02	
Q	Inflow (i.e. Pumping Rate)	$m^3/hour$	97	97	

Note: Parameter is entered as a single value
Value is calculated

Notation	Parameter	Units	Value		Comments
L	Thickness of mined hydrogeological unit	m	100.000	100.000	
L'	Thickness of saturated zone in the overlying formation(	m	300	300	
K	Hydraulic conducitivity of the mined hydrogeological un	m/s	1.00E-09	1.00E-09	
K'	Hydraulic Conductivity of the overling formation(s)	m/s	1.00E-08	1.00E-08	
S	Storativity of the mined hydrogeological unit	-	1.00E-03	1.00E-03	
D	Depth from base of workings to piezometric surface	m	300	300	
r	Radius of the excavation	m	600	300	
B		m	5.48E+01	5.48E+01	
r/B			1.10E+01	5.48E+00	
B	Check calculation only. Should = B above		5.48E+01	5.48E+01	
T	Transmisivity of the mined hydrogeological unit	m^2/s	1.00E-07	1.00E-07	
λ			4.E-02	2.E-01	
t	Elapsed time	years	5	5	
t	Elapsed time	s	1.58E+08	1.58E+08	
G	Hantush well function		2.43	1.96	From Table 4 using values of λ and r/B above
Q	Inflow (i.e. Pumping Rate)	m^3/s	4.58E-04	3.69E-04	
Q	Inflow (i.e. Pumping Rate)	$m^3/hour$	2	1	

Note: Parameter is entered as a single value
Value is calculated

Notation	Parameter	Units	Value		Comments
L	Thickness of mined hydrogeological unit	m	100.000	100.000	
L'	Thickness of saturated zone in the overlying formation(	m	300	300	
K	Hydraulic conducitivity of the mined hydrogeological ur	m/s	6.73E-08	6.73E-08	
K'	Hydraulic Conductivity of the overling formation(s)	m/s	6.73E-07	6.73E-07	
S	Storativity of the mined hydrogeological unit	-	1.00E-03	1.00E-03	
D	Depth from base of workings to piezometric surface	m	300	300	
r	Radius of the excavation	m	600	300	
B		m	5.48E+01	5.48E+01	
r/B			1.10E+01	5.48E+00	
B	Check calculation only. Should = B above		5.48E+01	5.48E+01	
T	Transmisivity of the mined hydrogeological unit	m^2/s	6.73E-06	6.73E-06	
λ			3.E+00	1.E+01	
t	Elapsed time	years	5	5	
t	Elapsed time	s	1.58E+08	1.58E+08	
G	Hantush well function		1.44	1.43	From Table 4 using values of λ and r/B above
Q	Inflow (i.e. Pumping Rate)	m^3/s	1.82E-02	1.81E-02	
Q	Inflow (i.e. Pumping Rate)	$m^3/hour$	66	65	

Note: Parameter is entered as a single value
Value is calculated

Notation	Parameter	Units	Value		Comments
L	Thickness of mined hydrogeological unit	m	100.000	100.000	
L'	Thickness of saturated zone in the overlying formation(	m	300	300	
K	Hydraulic conducitivity of the mined hydrogeological un	m/s	5.13E-08	5.13E-08	
K'	Hydraulic Conductivity of the overling formation(s)	m/s	5.13E-08	5.13E-08	
S	Storativity of the mined hydrogeological unit	-	1.00E-03	1.00E-03	
D	Depth from base of workings to piezometric surface	m	300	300	
r	Radius of the excavation	m	600	300	
B		m	1.73E+02	1.73E+02	
r/B			3.46E+00	1.73E+00	
B	Check calculation only. Should = B above		1.73E+02	1.73E+02	
T	Transmisivity of the mined hydrogeological unit	m^2/s	5.13E-06	5.13E-06	
λ			2.E+00	9.E+00	
t	Elapsed time	years	5	5	
t	Elapsed time	s	1.58E+08	1.58E+08	
G	Hantush well function		1.44	1.43	From Table 4 using values of λ and r/B above
Q	Inflow (i.e. Pumping Rate)	m^3/s	1.39E-02	1.38E-02	
Q	Inflow (i.e. Pumping Rate)	$m^3/hour$	50	50	

Note: Parameter is entered as a single value
Value is calculated

Hantush Equation

$$Q = 2\pi TDG\left(\lambda, \frac{r}{B}\right) \quad [4a]$$

$$\lambda = \frac{Tt}{r^2 S} \quad [4b]$$

$$\frac{r}{B} = r\sqrt{\frac{K'}{KLL'}} \quad [4c]$$

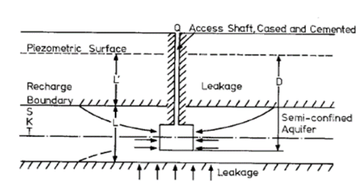

Fig. 7. Dewatering of a large underground chamber—non-steady.

The notations in the formulae given in Figs. 3–7 are defined as follows:

B	Leakage factor (m) = $\sqrt{KLL'/K'}$
D	Draw down (m) (Fig. 1)
$G(\lambda, r/B)$	Hantush well function (Table 4)
i	Hydraulic gradient (dimensionless)
K	Aquifer permeability or hydraulic conductivity (m/d)
K'	Hydraulic conductivity of aquitard (m/d)
$K_0(r/B)$	Hantush–Jacob well function for steady state leaking aquifer (Table 3)

TABLE 4

Values of the function $G(\lambda, r/B)$ [8]

$\lambda, r/B$	0	1×10^{-2}	2×10^{-2}	4×10^{-2}	6×10^{-2}	8×10^{-2}	1×10^{-1}	2×10^{-1}	4×10^{-1}	6×10^{-1}	8×10^{-1}	1
1×10^{-1}	2.24	2.24	2.25	2.25	2.25	2.25	2.25	2.26	2.28	2.31	2.36	2.43
2	1.71	1.71	1.71	1.72	1.72	1.72	1.72	1.73	1.76	1.81	1.87	1.96
5	1.23	1.23	1.23	1.23	1.23	1.24	1.24	1.25	1.30	1.38	1.48	1.81
1×10^{0}	0.983	0.983	0.984	0.985	0.986	0.987	0.900	1.01	1.07	1.18	1.32	1.49
2	0.800	0.800	0.801	0.802	0.804	0.806	0.809	0.834	0.929	1.07	1.25	1.44
5	0.632	0.628	0.629	0.630	0.633	0.637	0.642	0.682	0.824	1.01	1.22	1.43
1×10^{1}	0.534	0.534	0.535	0.537	0.541	0.547	0.554	0.661	0.793	1.01	1.22	1.43
2	0.461	0.461	0.462	0.466	0.472	0.481	0.401	0.569	0.785			
5	0.389	0.389	0.390	0.397	0.407	0.431	0.436	0.546	0.784			
1×10^{2}	0.346	0.346	0.349	0.359	0.374	0.394	0.417	0.545	0.784			
2	0.311	0.312	0.316	0.331	0.353	0.380	0.408	0.545				
5	0.274	0.276	0.284	0.309	0.341	0.374	0.406					
1×10^{3}	0.251	0.255	0.266	0.301	0.339	0.374	0.406					
2	0.232	0.238	0.255	0.299	0.330							
5	0.210	0.222	0.249	0.290								
1×10^{4}	0.196	0.216	0.248									
2	0.185	0.213	0.248									
5	0.170	0.212										
1×10^{5}	0.161	0.212										
2	0.152											
5	0.143											

Method from Singh, R.N. and Atkins, A.S., 1985. Application of idealised analytical techniques for prediction of mine water inflow. Mining Science and Technology, 2, pp.131-138.

APPENDIX C

235 Level Portal Inflow Calculations

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Value	Units	Justification
Hydraulic conductivity	K	1.00E-05	m/s	Assume K is 2 order of magnitude greater than maximum K reported in Golder, 2020
Area	A	3750	m^2	Nominal 150m width x 25m height
Area separation	x	47	m	Minimum distance between South and Valley Block
Head difference	dh	80	m	Elevation between top of water at 270m in South Block and the 190m level in Valley Block
Flow	Q	0.06	m^3/s	Calculated using Darcy's Law

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Units	Minimum	Maximum	Worst Case	Justification
Hydraulic conductivity	K	m/s	1.00E-07	1.00E-10	1.00E-05	From Golder, 2020
Adit length	L_1	m	300	300	300	Nominal overlap length
Head of water	H_{o1}	m	80	80	80	Elevation between top of water at 270m in South Block and the 190m level in Valley Block
Adit diameter	D_1	m	6	6	6	Approximate width
Inflow	Q_1	m^3/s	0.004	0.000004	0.38	Goodman et al (1965)
Total Inflow (m^3/day)	Q_T	$m^3/hour$	14	0.014	1365	from Inflow
Total Inflow (m^3/day)	Q_T	m^3/day	328	0.328	32764	from Inflow

Inflow, Q, calculated from Goodman et al (1965):

$$Q = \frac{2\pi K L H_0}{\ln\left(\frac{4H_0}{D}\right)}$$

Method from: Goodman, R.E., Moye, D.G., van Schalkwyk, A. and Javandel, I., 1965. Ground water inflows during tunnel driving. Engineering Geology, 2(1), pp. 39-56.

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Units	Minimum	Maximum	Justification
Hydraulic conductivity	K	m/s	1.00E-07	1.00E-10	From Golder, 2020
Adit length	L_1	m	1500	1500	Nominal development length
Head of water	H_{o1}	m	120	120	Elevation between Groundwater in the fluvioglacial deposits of Kirkespirdalen and the base of the Valley Block
Adit diameter	D_1	m	6	6	Approximate width
Inflow	Q_1	m^3/s	0.026	0.000026	Goodman et al (1965)
Total Inflow (m^3/day)	Q_T	$m^3/hour$	93	0.093	from Inflow
Total Inflow (m^3/day)	Q_T	m^3/day	2230	2.230	from Inflow

Inflow, Q, calculated from Goodman et al (1965):

$$Q = \frac{2\pi K L H_0}{\ln\left(\frac{4H_0}{D}\right)}$$

Method from: Goodman, R.E., Moye, D.G., van Schalkwyk, A. and Javandel, I., 1965. Ground water inflows during tunnel driving. Engineering Geology, 2(1), pp. 39-56.

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Units	Values								Justification
Adit/tunnel length	L	m	300								*Nominal Length*
Head of water	H_o	m	80								*Elevation between top of water at 270m in South Block and the 190m level in Valley Block*
Hydraulic conductivity	K	m/s	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	*Assume 2 orders of magnitude greater than intact rock reported in Golder, 2020*
		cm/s	1.00E-03	1.00E-03	1.00E-03	1.00E-03	1.00E-03	1.00E-03	1.00E-03	1.00E-03	
Inflow factor	F_h	-	4	4	4	4	4	4	4	4	*From Figure 4, Heuer (2005)*
Length of adit/tunnel in interval	L_i	m	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	*Assumption*
Percent adit/tunnel in interval	L_{ip}	%	13%	13%	13%	13%	13%	13%	13%	13%	*Calculated*
	q_s/H	l/min/100 m/m	40	40	40	40	40	40	40	40	*From Figure 4 based on* F_h
Inflow per unit length of adit/tunnel	q_s	l/min/m	32	32	32	32	32	32	32	32	*Calculated*
Flow for each length of tunnel	ΔQ_s	l/min	1200	1200	1200	1200	1200	1200	1200	1200	*Calculated*
Total inflow	$\Sigma\Delta Q_s$	l/min	**9600**								
		m^3/s	**0.16**								
Initial inflow											
Length of initial heading	L_{ih}	m	25	*Assumption*							
Initial heading inflow (worst case)	Q_h	l/min	**3200**	*Calculated*							
		m^3/s	**0.053**	*Calculated*							
Assessment of grouting											
Trigger for grouting	G_t	l/min/100 m/m	240	*From Heuer, 2005*							
Inflow through ungrouted section	$\Sigma\Delta Q_{ug}$	l/min	[redacted]	*Calculated*							
Grouted inflow	q_{sg}/H	l/min/100 m/m	[redacted]	*Assume grouted to average of K of 1st two division above trigger*							
	q_{sg}	l/min/m	[redacted]	*Calculated*							
Pre-grout inflow to grouted section	ΔQ_{pg}	l/min	[redacted]	*Calculated*							
Inflow through grouted section	$\Sigma\Delta Q_g$	l/min	[redacted]	*Calculated*							
Inflow to tunnel after grouting	$\Sigma\Delta Q_{hg}$	l/min	[redacted]	*Calculated*							
Method from:	Heuer, R.E., 1995. Estimating rock tunnel water inflow. in Proceedings of the rapid excavation and tunneling conference (12th Rapid excavation and tunneling conference), Society for Mining, Metallurgy, https://www.tib.eu/en/search/id/BLCP%3ACN012010035/Estimating-Rock-Tunnel-Water-Inflow/ Heuer, 2005. Estimating rock tunnel water inflow - II. Society for Mining, Metallurgy & Exploration 36886, pp.394-407. https://www.onemine.org/document/abstract.cfm?docid=36886&title=Estimating-Rock-Tunnel-Water-InflowII										

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Units	Values								Justification
Adit/tunnel length	L	m	300								*Nominal Length*
Head of water	H_o	m	80								*Elevation between top of water at 270m in South Block and the 190m level in Valley Block*
Hydraulic conductivity	K	m/s	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	*Golder, 2020*
		cm/s	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	
Inflow factor	F_h	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	*From Figure 4, Heuer (2005)*
Length of adit/tunnel in interval	L_i	m	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	*Assumption*
Percent adit/tunnel in interval	L_{ip}	%	13%	13%	13%	13%	13%	13%	13%	13%	*Calculated*
	q_s/H	l/min/100 m/m	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	*From Figure 4 based on F_h*
Inflow per unit length of adit/tunnel	q_s	l/min/m	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	*Calculated*
Flow for each length of tunnel	ΔQ_s	l/min	12	12	12	12	12	12	12	12	*Calculated*
Total inflow	$\Sigma\Delta Q_s$	l/min	**96**								
		m^3/s	**0.002**								
Initial inflow											
Length of initial heading	L_{ih}	m	37.5	*Assumption*							
Initial heading inflow (worst case)	Q_h	l/min	**14.4**	*Calculated*							
		m^3/s	**0.0002**	*Calculated*							
Assessment of grouting											
Trigger for grouting	G_t	l/min/100 m/m	240	*From Heuer, 2005*							
Inflow through ungrouted section	$\Sigma\Delta Q_{ug}$	l/min	[redacted]	*Calculated*							
Grouted inflow	q_{sg}/H	l/min/100 m/m	[redacted]	*Assume grouted to average of K of 1st two division above trigger*							
	q_{sg}	l/min/m	[redacted]	*Calculated*							
Pre-grout inflow to grouted section	ΔQ_{pg}	l/min	[redacted]	*Calculated*							
Inflow through grouted section	$\Sigma\Delta Q_g$	l/min	[redacted]	*Calculated*							
Inflow to tunnel after grouting	$\Sigma\Delta Q_{hg}$	l/min	[redacted]	*Calculated*							
Method from:	Heuer, R.E., 1995. Estimating rock tunnel water inflow. in Proceedings of the rapid excavation and tunneling conference (12th Rapid excavation and tunneling conference), Society for Mining, Metallurgy, https://www.tib.eu/en/search/id/BLCP%3ACN012010035/Estimating-Rock-Tunnel-Water-Inflow/ Heuer, 2005. Estimating rock tunnel water inflow - II. Society for Mining, Metallurgy & Exploration 36886, pp.394-407. https://www.onemine.org/document/abstract.cfm?docid=36886&title=Estimating-Rock-Tunnel-Water-InflowII										

Calculation of Groundwater Inflow to Underground Mine Workings

Parameter	Notation	Units	Values								Justification
Adit/tunnel length	L	m	1500								*Nominal development length*
Head of water	H_o	m	120								*Elevation between Groundwater in the fluvioglacial deposits of Kirkespirdalen and the base of the Valley Block*
Hydraulic conductivity	K	m/s	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	1.00E-07	*Golder, 2020*
		cm/s	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	1.00E-05	
Inflow factor	F_h	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	*From Figure 4, Heuer (2005)*
Length of adit/tunnel in interval	L_i	m	187.5	187.5	187.5	187.5	187.5	187.5	187.5	187.5	*Assumption*
Percent adit/tunnel in interval	L_{ip}	%	13%	13%	13%	13%	13%	13%	13%	13%	*Calculated*
	q_s/H	l/min/100 m/m	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	*From Figure 4 based on F_h*
Inflow per unit length of adit/tunnel	q_s	l/min/m	0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	*Calculated*
Flow for each length of tunnel	ΔQ_s	l/min	90	90	90	90	90	90	90	90	*Calculated*
Total inflow	$\Sigma\Delta Q_s$	l/min	**720**								
		m^3/s	**0.012**								

Method from:	Heuer, R.E., 1995. Estimating rock tunnel water inflow. in Proceedings of the rapid excavation and tunneling conference (12th Rapid excavation and tunneling conference), Society for Mining, Metallurgy, https://www.tib.eu/en/search/id/BLCP%3ACN012010035/Estimating-Rock-Tunnel-Water-Inflow/ Heuer, 2005. Estimating rock tunnel water inflow - II. Society for Mining, Metallurgy & Exploration 36886, pp.394-407. https://www.onemine.org/document/abstract.cfm?docid=36886&title=Estimating-Rock-Tunnel-Water-InflowII

APPENDIX D

Typical Weir Arrangements

Typical Weir Arrangements

The following illustrations are provided to illustrate typical weir arrangements. These would need to be appropriately scaled for use at Nalunaq. The dimensions and operation of thin plate weirs are set out in British Standard 3680 Part 4A.

Figure 1: Concrete weir tank with steel 90° plate weir.

Figure 2: V-notch weir for measuring flows from a piped flow.

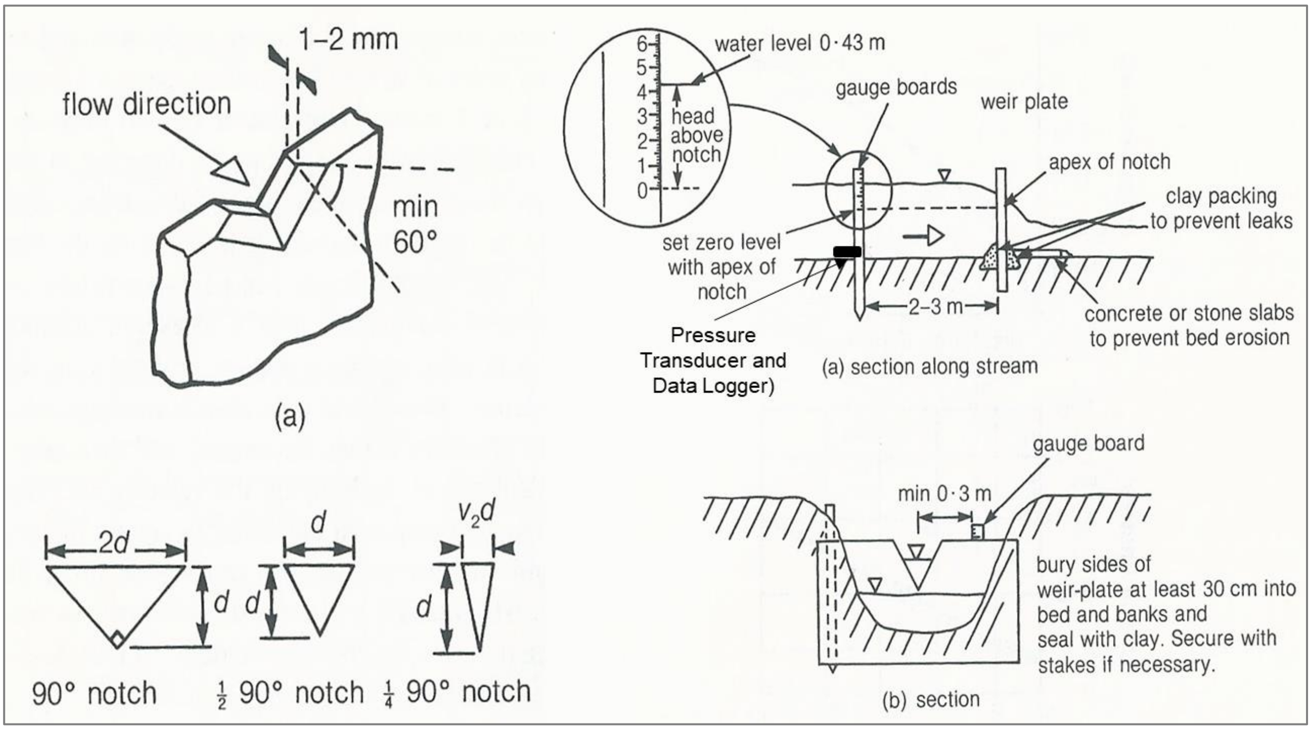

Figure 3: Notch dimensions and installation arrangements (from Brassington, 2007)

APPENDIX C

Hydraulic Conductivity and Porosity from Grain Size Analyses

Summary of Particle Size Density (PSD) Results (after Golder, 2002)

		Borehole Location										
		BH01-02	BH01-07	BH01-09	BH01-10	BH01-11 (2)	BH01-11 (5)	BH01-12	BH01-13	BH01-14 (1)	BH01-14 (3)	BH01-15
		Sample Depth (mbgl)										
US Sieve Size	Sieve Size	7.3	17.2	3.3	1.9	2	6.6	3.4	6.3	4.7	10.1	1.6
-	mm	Percent Sediment Passing Sieve (%)										
2"	50	-	-	100	-	-	-	-	-	-	-	-
1.5"	38.9	100	-	84	100	-	-	-	-	-	-	-
1"	25.4	86	100	66	81	100	100	-	-	100	-	100
3/4"	19	81	95	58	76	89	96	100	-	91	-	79
3/8"	9.51	50	85	46	72	78	88	94	100	83	100	66
4	4.76	34	80	24	66	67	81	87	99	79	99	49
10	2	23	77	17	58	44	53	80	97	76	98	32
20	0.841	18	76	11	48	26	27	56	96	71	98	22
40	0.42	14	74	7	35	14	13	37	88	65	98	15
60	0.25	10	69	5	21	9	8	25	62	53	98	8
140	0.105	5	43	2	7	5	3	12	18	27	79	4
200	0.074	4	34	1	4	4	2	9	9	17	55	3
400	0.037	-	19	-	-	-	-	-	-	3	16	-
-	0.0225	-	17	-	-	-	-	-	-	2	11	-
-	0.015	-	13	-	-	-	-	-	-	2	8	-
-	0.0095	-	10	-	-	-	-	-	-	2	7	-
-	0.00675	-	7	-	-	-	-	-	-	1	5	-
-	0.00325	-	3	-	-	-	-	-	-	1	2	-
-	0.0015	-	2	-	-	-	-	-	-	1	2	-

Notes: "-" = sieve size not used; "(1)" = sample number where multiple PSD analyses were carried out at different depths at the same borehole location.

Summary of Calculated Hydraulic Conductivity and Porosity Calculations

Borehole ID	Sample Depth (mbgl)	Description	Location	Calculated Porosity (%)	Geomean K (without porosity)		Geomean K (with porosity)		Average K (with and without porosity)	
					m/sec	m/day	m/s	m/day	m/s	m/day
BH01-02	7.3	Sandy gravel (Till)	Kirkespir River	26	5.64E-04	4.88E+01	3.98E-04	3.44E+01	4.81E-04	4.16E+01
BH01-07	17.2	Silty sand	Kirkespir River	30	2.08E-06	1.80E-01	1.50E-06	1.29E-01	1.79E-06	1.54E-01
BH01-09	3.3	Sandy gravel	Kirkespir River	26	3.61E-03	3.12E+02	2.13E-03	1.84E+02	2.87E-03	2.48E+02
BH01-10	1.9	Sandy gravel	Fjord beach	26	8.37E-05	7.23E+00	4.01E-05	3.47E+00	6.19E-05	5.35E+00
BH01-11 (2)	2	Sand and gravel	Fjord beach	28	4.94E-04	4.27E+01	2.74E-04	2.36E+01	3.84E-04	3.32E+01
BH01-11 (5)	6.6	Sand and gravel	Fjord beach	31	6.03E-04	5.21E+01	4.10E-04	3.54E+01	5.07E-04	4.38E+01
BH01-12	3.4	Sandy gravel	Fjord beach	28	3.65E-05	3.15E+00	2.23E-05	1.92E+00	2.94E-05	2.54E+00
BH01-13	6.3	Sand	Fjord beach	39	2.56E-05	2.22E+00	3.58E-05	3.09E+00	3.07E-05	2.65E+00
BH01-14 (1)	4.7	Gravelly sand	Kirkespir River	31	8.83E-06	7.63E-01	7.21E-06	6.23E-01	8.02E-06	6.93E-01
BH01-14 (3)	10.1	Silty sand	Kirkespir River	33	7.86E-07	6.79E-02	7.77E-07	6.71E-02	7.81E-07	6.75E-02
BH01-15	1.6	Sandy gravel	Kirkespir River	26	4.23E-04	3.66E+01	2.31E-04	2.00E+01	3.27E-04	2.83E+01

Notes: "(1)" = sample number where multiple PSD analyses were carried out at different depths at the same borehole location.

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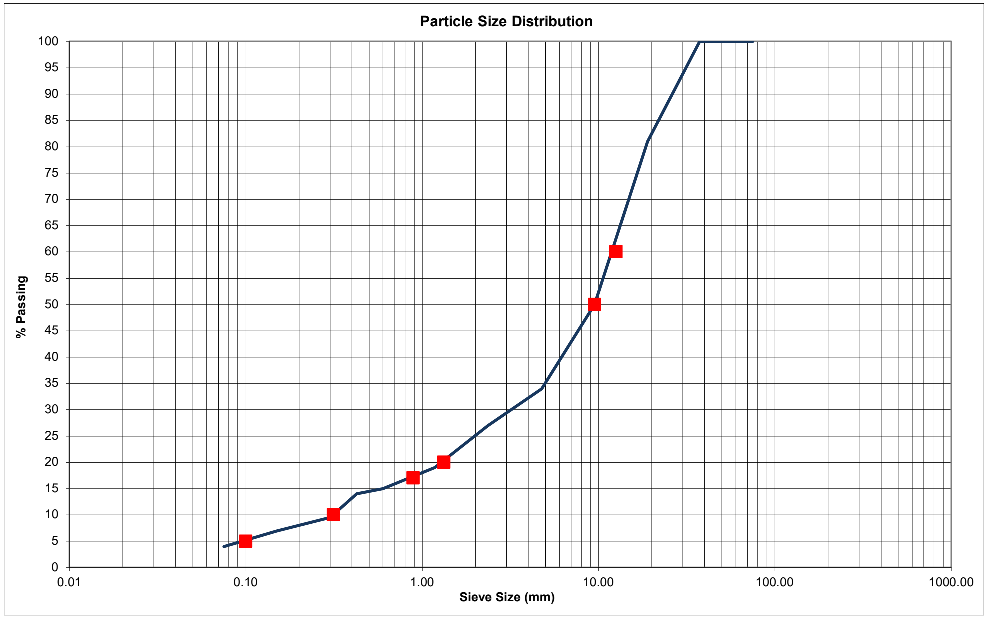
Sample Location: BH01-02
Sample Depth (m): 7.3
SOIL TYPE: Sandy gravel (Till)

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	81
9.5	50
4.75	34
2.36	27
1.18	19
0.60	15
0.425	14
0.30	9.5
0.15	7
0.075	4

Particle Size (mm)	
D_{60}	12.56
D_{50}	9.50
D_{20}	1.33
D_{17}	0.89
D_{10}	0.31
D_5	0.10

Uniformity Coefficient

D_{60} / D_{10} **40.0**



Note: grading curve based on that for BH01-02 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **26%**

Calculated Porosity (n) **26%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	3.1E-04	4.5.E-04	39.2
Amer & Awad (1974)	2.3E-01	3.0E-02	3.1E-04	6.6.E-04	57.0
Shahabi et. al. (1984)	2.1E-01	3.0E-02	3.1E-04	6.1.E-04	52.3
Kenney et. al. (1984)	5.1E-02	1.0E+00	1.0E-04	5.0.E-04	43.2
Slichter (1898)	1.0E-01	1.1E-02	3.1E-04	1.1.E-04	9.7
Terzaghi (1925)	2.1E-02	1.9E-02	3.1E-04	3.8.E-05	3.3
Beyer (1964)	5.1E-03	1.0E+00	3.1E-04	4.9.E-04	42.0
Sauerbrei (1992)	3.6E-02	3.0E-02	8.9E-04	8.3.E-04	71.5
Pavchich (1966)	1.2E-01	3.0E-02	8.9E-04	2.8.E-03	244.3
USBR (1992)	5.4E-04	1.0E+00	1.3E-03	9.2.E-04	79.6
Average				**7.4.E-04**	**64.2**
Geometric Mean				**4.6.E-04**	**39.6**

Porosity not used in calculation

Geo Mean (without porosity)	**5.6E-04**	**48.8**
Geo Mean (with porosity)	**4.0E-04**	**34.4**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

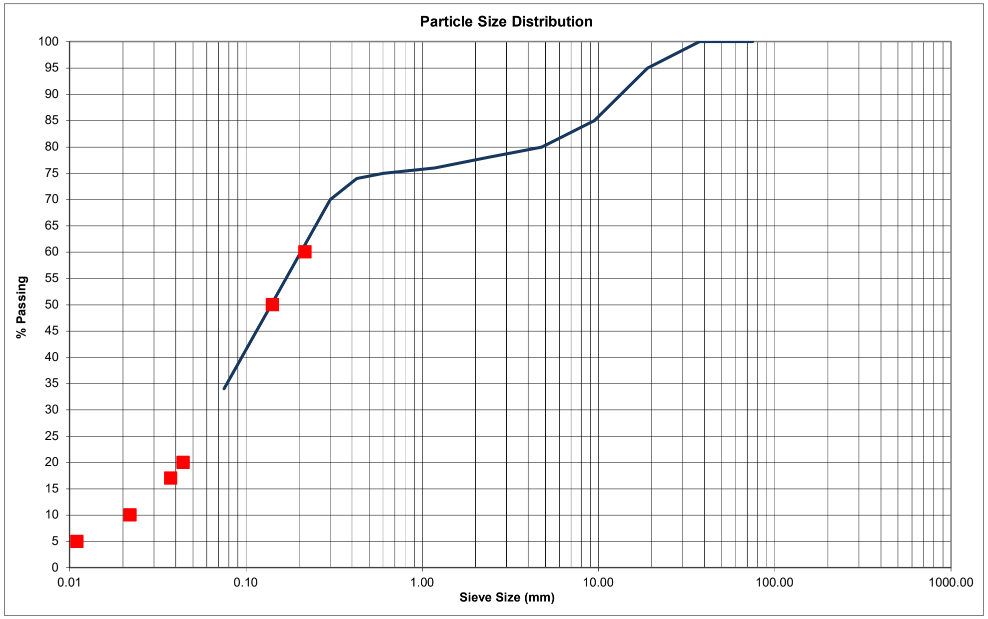
Sample Location:	BH01-07
Sample Depth (m):	17.2
SOIL TYPE:	Silty sand

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	95
9.5	85
4.75	80
2.36	78
1.18	76
0.60	75
0.425	74
0.30	70
0.15	52
0.075	34

Particle Size (mm)	
D_{60}	0.22
D_{50}	0.14
D_{20}	0.04
D_{17}	0.04
D_{10}	0.02
D_{5}	0.01

Uniformity Coefficient

D_{60} / D_{10} **9.8**



Note: grading curve based on that for BH01-07 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **30%**

Calculated Porosity (n) **30%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	2.2E-05	2.2.E-06	0.2
Amer & Awad (1974)	4.2E-02	5.2E-02	2.2E-05	1.0.E-06	0.1
Shahabi et. al. (1984)	1.0E-01	5.2E-02	2.2E-05	2.5.E-06	0.2
Kenney et. al. (1984)	5.1E-02	1.0E+00	1.1E-05	6.1.E-06	0.5
Slichter (1898)	1.0E-01	1.8E-02	2.2E-05	9.0.E-07	0.1
Terzaghi (1925)	2.1E-02	3.5E-02	2.2E-05	3.4.E-07	0.0
Beyer (1964)	7.9E-03	1.0E+00	2.2E-05	3.7.E-06	0.3
Sauerbrei (1992)	3.6E-02	5.2E-02	3.8E-05	2.6.E-06	0.2
Pavchich (1966)	7.7E-02	5.2E-02	3.8E-05	5.5.E-06	0.5
USBR (1992)	1.9E-04	1.0E+00	4.4E-05	3.7.E-07	0.0
Average				**2.5.E-06**	**0.2**
Geometric Mean				**1.7.E-06**	**0.1**

Porosity not used in calculation

Geo Mean (without porosity)	**2.1E-06**	**0.2**
Geo Mean (with porosity)	**1.5E-06**	**0.1**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

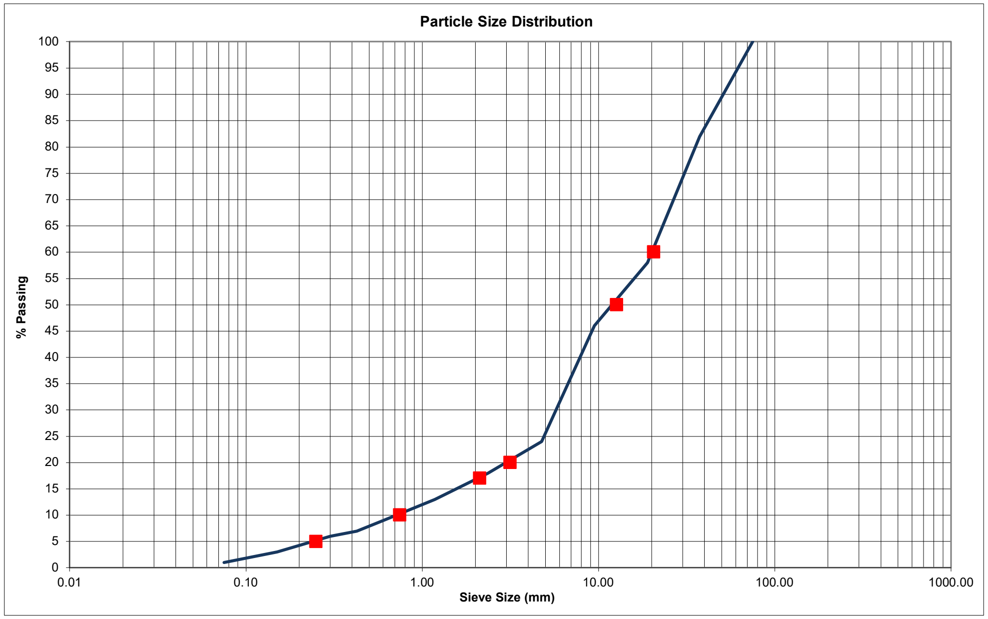
Sample Location:	**BH01-09**
Sample Depth (m):	**3.3**
SOIL TYPE:	**Sandy gravel**

Sieve Size (mm)	% Passing
75	100
37.5	82
19.0	58
9.5	46
4.75	24
2.36	18
1.18	13
0.60	9
0.425	7
0.30	6
0.15	3
0.075	1

Particle Size (mm)	
D_{60}	20.54
D_{50}	12.67
D_{20}	3.16
D_{17}	2.12
D_{10}	0.75
D_5	0.25

Uniformity Coefficient

D_{60} / D_{10} **27.6**



Note: grading curve based on that for BH01-09 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **26%**

Calculated Porosity (n) **26%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	7.5E-04	2.6.E-03	220.6
Amer & Awad (1974)	2.4E-01	3.1E-02	7.5E-04	4.0.E-03	345.0
Shahabi et. al. (1984)	1.5E-01	3.1E-02	7.5E-04	2.4.E-03	207.6
Kenney et. al. (1984)	5.1E-02	1.0E+00	2.5E-04	3.1.E-03	270.0
Slichter (1898)	1.0E-01	1.1E-02	7.5E-04	6.4.E-04	55.4
Terzaghi (1925)	2.1E-02	1.9E-02	7.5E-04	2.2.E-04	19.0
Beyer (1964)	5.8E-03	1.0E+00	7.5E-04	3.1.E-03	271.6
Sauerbrei (1992)	3.6E-02	3.1E-02	2.1E-03	4.8.E-03	415.3
Pavchich (1966)	1.1E-01	3.1E-02	2.1E-03	1.5.E-02	1253.2
USBR (1992)	7.0E-04	1.0E+00	3.2E-03	6.8.E-03	583.5
Average				**4.2.E-03**	**364.1**
Geometric Mean				**2.6.E-03**	**227.4**

Porosity not used in calculation

Geo Mean (without porosity)	**3.6E-03**	**311.7**
Geo Mean (with porosity)	**2.1E-03**	**184.3**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

Sample Location: BH01-10
Sample Depth (m): 1.9
SOIL TYPE: Sandy gravel

Sieve Size (mm)	% Passing
75	100
37.5	98
19.0	76
9.5	72
4.75	66
2.36	60
1.18	51
0.60	41
0.425	35
0.30	28
0.15	12
0.075	4

Note: grading curve based on that for BH01-10 (Golder, 2002) with missing sieve sizes extrapolated

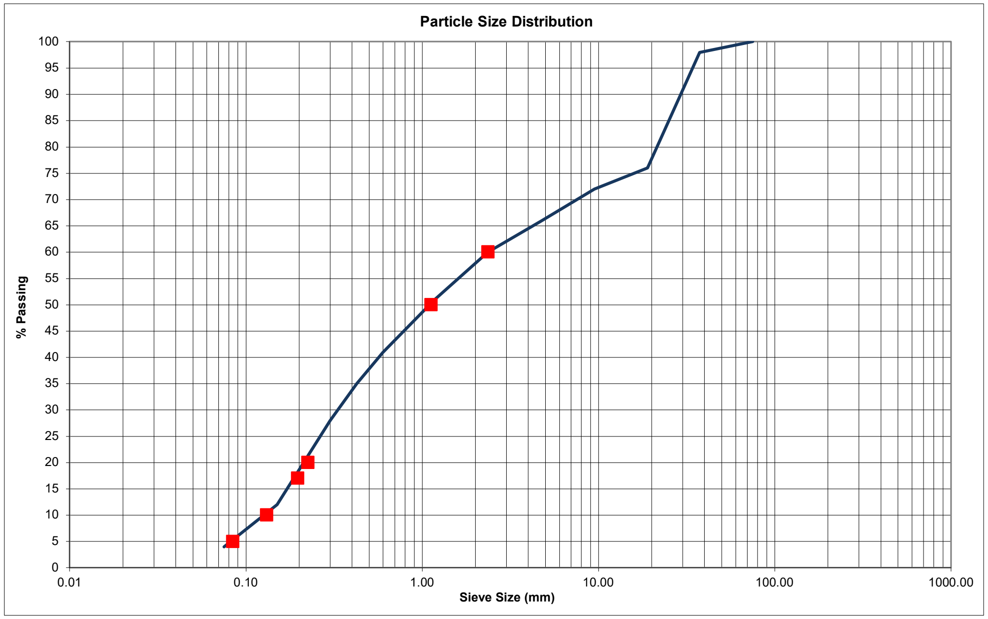
Estimated Porosity (ε) **26%**

Calculated Porosity (n) **26%** fraction

Particle Size (mm)	
D_{60}	2.36
D_{50}	1.12
D_{20}	0.23
D_{17}	0.20
D_{10}	0.13
D_5	0.08

Uniformity Coefficient

D_{60} / D_{10} **18.0**



Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	1.3E-04	7.9.E-05	6.8
Amer & Awad (1974)	1.1E-01	3.4E-02	1.3E-04	6.1.E-05	5.3
Shahabi et. al. (1984)	1.3E-01	3.4E-02	1.3E-04	7.3.E-05	6.3
Kenney et. al. (1984)	5.1E-02	1.0E+00	8.4E-05	3.6.E-04	30.8
Slichter (1898)	1.0E-01	1.3E-02	1.3E-04	2.2.E-05	1.9
Terzaghi (1925)	2.1E-02	2.2E-02	1.3E-04	7.7.E-06	0.7
Beyer (1964)	6.7E-03	1.0E+00	1.3E-04	1.1.E-04	9.7
Sauerbrei (1992)	3.6E-02	3.4E-02	2.0E-04	4.6.E-05	4.0
Pavchich (1966)	9.4E-02	3.4E-02	2.0E-04	1.2.E-04	10.4
USBR (1992)	3.1E-04	1.0E+00	2.3E-04	1.6.E-05	1.3
Average				**8.9.E-05**	**7.7**
Geometric Mean				**5.4.E-05**	**4.7**

Porosity not used in calculation

Geo Mean (without porosity)	**8.4E-05**	**7.2**
Geo Mean (with porosity)	**4.0E-05**	**3.5**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

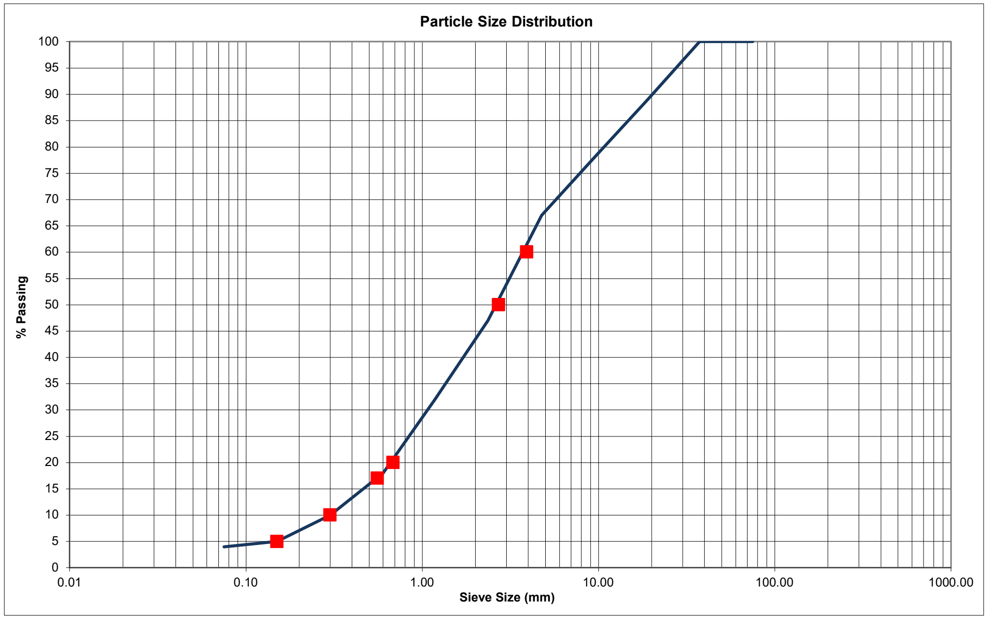
Sample Location: BH01-11
Sample Depth (m): 2
SOIL TYPE: Sand and gravel

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	89
9.5	78
4.75	67
2.36	47
1.18	32
0.60	18
0.425	14
0.30	10
0.15	5
0.075	4

Particle Size (mm)	
D_{60}	3.91
D_{50}	2.72
D_{20}	0.68
D_{17}	0.56
D_{10}	0.30
D_5	0.15

Uniformity Coefficient

D_{60} / D_{10} **13.0**



Note: grading curve based on that for BH01-11, Sample 2 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **28%**

Calculated Porosity (n) **28%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	3.0E-04	4.1.E-04	35.8
Amer & Awad (1974)	1.2E-01	4.1E-02	3.0E-04	4.1.E-04	35.8
Shahabi et. al. (1984)	9.3E-02	4.1E-02	3.0E-04	3.3.E-04	28.8
Kenney et. al. (1984)	5.1E-02	1.0E+00	1.5E-04	1.1.E-03	97.2
Slichter (1898)	1.0E-01	1.5E-02	3.0E-04	1.3.E-04	11.6
Terzaghi (1925)	2.1E-02	2.7E-02	3.0E-04	4.9.E-05	4.3
Beyer (1964)	7.3E-03	1.0E+00	3.0E-04	6.4.E-04	55.4
Sauerbrei (1992)	3.6E-02	4.1E-02	5.6E-04	4.4.E-04	38.2
Pavchich (1966)	8.4E-02	4.1E-02	5.6E-04	1.0.E-03	89.8
USBR (1992)	4.4E-04	1.0E+00	6.8E-04	2.0.E-04	17.2
Average				**4.8.E-04**	**41.4**
Geometric Mean				**3.5.E-04**	**29.9**

Porosity not used in calculation

Geo Mean (without porosity)	**4.9E-04**	**42.7**
Geo Mean (with porosity)	**2.7E-04**	**23.6**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

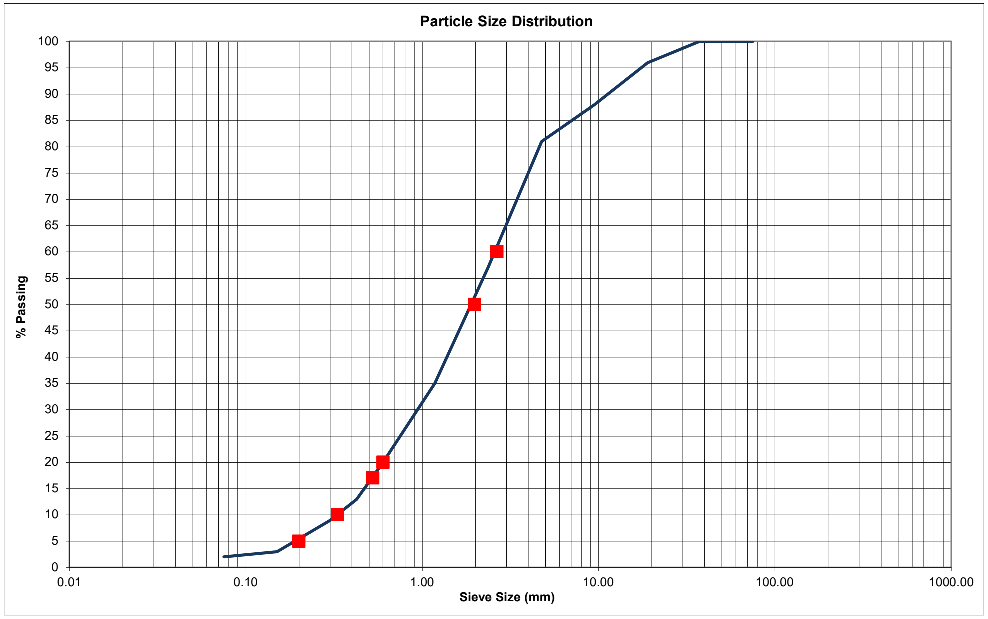
Sample Location: BH01-11
Sample Depth (m): 6.6
SOIL TYPE: Sand and gravel

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	96
9.5	88
4.75	81
2.36	57
1.18	35
0.60	20
0.425	13
0.30	9
0.15	3
0.075	2

Particle Size (mm)	
D_{60}	2.66
D_{50}	1.98
D_{20}	0.60
D_{17}	0.53
D_{10}	0.33
D_{5}	0.20

Uniformity Coefficient

D_{60} / D_{10} **8.0**



Note: grading curve based on that for BH01-11, Sample 5 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **31%**

Calculated Porosity (n) **31%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	3.3E-04	5.0.E-04	43.6
Amer & Awad (1974)	8.9E-02	6.4E-02	3.3E-04	6.1.E-04	52.9
Shahabi et. al. (1984)	6.4E-02	6.4E-02	3.3E-04	4.4.E-04	38.2
Kenney et. al. (1984)	5.1E-02	1.0E+00	2.0E-04	2.0.E-03	172.8
Slichter (1898)	1.0E-01	2.2E-02	3.3E-04	2.4.E-04	20.9
Terzaghi (1925)	2.1E-02	4.3E-02	3.3E-04	9.5.E-05	8.2
Beyer (1964)	8.3E-03	1.0E+00	3.3E-04	8.9.E-04	76.6
Sauerbrei (1992)	3.6E-02	6.4E-02	5.3E-04	6.2.E-04	53.4
Pavchich (1966)	7.2E-02	6.4E-02	5.3E-04	1.2.E-03	106.9
USBR (1992)	4.2E-04	1.0E+00	6.0E-04	1.5.E-04	12.8
Average				**6.8.E-04**	**58.6**
Geometric Mean				**4.8.E-04**	**41.3**

Porosity not used in calculation

Geo Mean (without porosity)	**6.0E-04**	**52.1**
Geo Mean (with porosity)	**4.1E-04**	**35.4**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

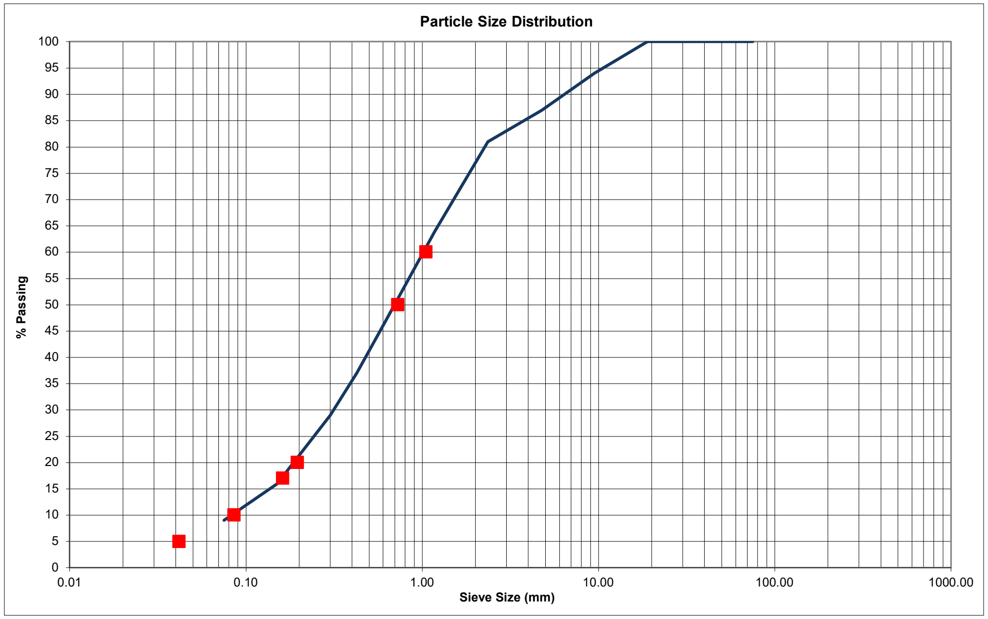
Sample Location:	**BH01-12**
Sample Depth (m):	**3.4**
SOIL TYPE:	**Sandy gravel**

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	100
9.5	94
4.75	87
2.36	81
1.18	64
0.60	46
0.425	37
0.30	29
0.15	16
0.075	9

Particle Size (mm)	
D_{60}	1.05
D_{50}	0.73
D_{20}	0.20
D_{17}	0.16
D_{10}	0.09
D_5	0.04

Uniformity Coefficient

D_{60} / D_{10} **12.3**



Note: grading curve based on that for BH01-12 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **28%**

Calculated Porosity (n) **28%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	8.6E-05	3.4.E-05	2.9
Amer & Awad (1974)	7.4E-02	4.3E-02	8.6E-05	2.3.E-05	2.0
Shahabi et. al. (1984)	1.0E-01	4.3E-02	8.6E-05	3.1.E-05	2.7
Kenney et. al. (1984)	5.1E-02	1.0E+00	4.2E-05	8.7.E-05	7.5
Slichter (1898)	1.0E-01	1.5E-02	8.6E-05	1.1.E-05	1.0
Terzaghi (1925)	2.1E-02	2.8E-02	8.6E-05	4.2.E-06	0.4
Beyer (1964)	7.4E-03	1.0E+00	8.6E-05	5.3.E-05	4.6
Sauerbrei (1992)	3.6E-02	4.3E-02	1.6E-04	3.9.E-05	3.4
Pavchich (1966)	8.2E-02	4.3E-02	1.6E-04	9.0.E-05	7.8
USBR (1992)	3.0E-04	1.0E+00	2.0E-04	1.1.E-05	1.0
Average				**3.8.E-05**	**3.3**
Geometric Mean				**2.7.E-05**	**2.3**

Porosity not used in calculation

Geo Mean (without porosity)	**3.6E-05**	**3.2**
Geo Mean (with porosity)	**2.2E-05**	**1.9**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

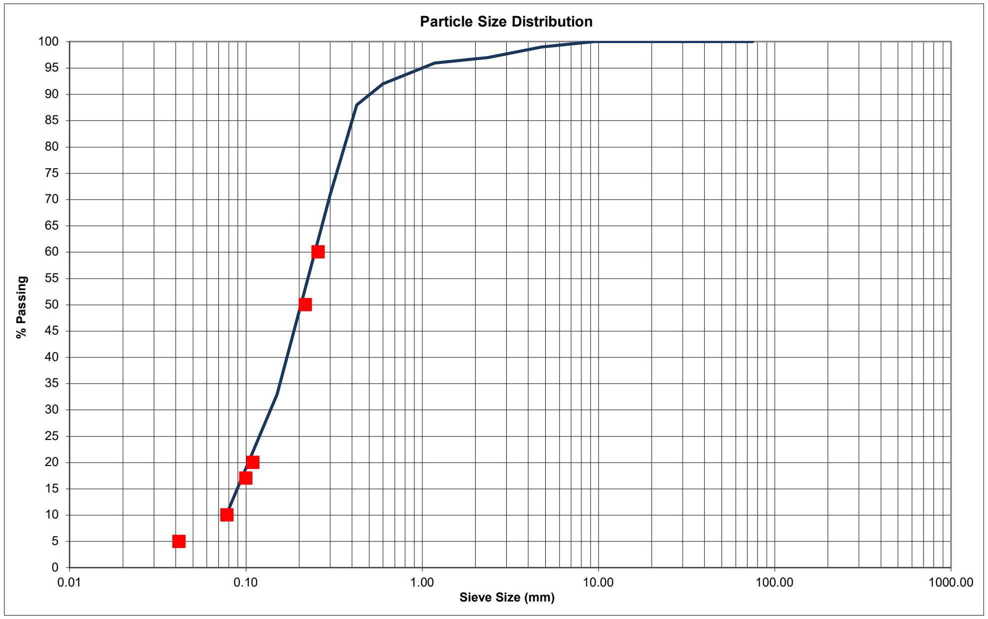
Sample Location:	BH01-13
Sample Depth (m):	6.3
SOIL TYPE:	Sand

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	100
9.5	100
4.75	99
2.36	97
1.18	96
0.60	92
0.425	88
0.30	71
0.15	33
0.075	9

Particle Size (mm)	
D_{60}	0.26
D_{50}	0.22
D_{20}	0.11
D_{17}	0.10
D_{10}	0.08
D_{5}	0.04

Uniformity Coefficient

D_{60} / D_{10} 3.3



Note: grading curve based on that for BH01-13 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **39%**

Calculated Porosity (n) **39%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	7.8E-05	2.8.E-05	2.4
Amer & Awad (1974)	3.3E-02	1.7E-01	7.8E-05	3.2.E-05	2.8
Shahabi et. al. (1984)	3.9E-02	1.7E-01	7.8E-05	3.8.E-05	3.3
Kenney et. al. (1984)	5.1E-02	1.0E+00	4.2E-05	8.7.E-05	7.5
Slichter (1898)	1.0E-01	4.7E-02	7.8E-05	2.9.E-05	2.5
Terzaghi (1925)	2.1E-02	9.7E-02	7.8E-05	1.2.E-05	1.0
Beyer (1964)	1.0E-02	1.0E+00	7.8E-05	6.0.E-05	5.2
Sauerbrei (1992)	3.6E-02	1.7E-01	1.0E-04	5.8.E-05	5.0
Pavchich (1966)	5.3E-02	1.7E-01	1.0E-04	8.6.E-05	7.4
USBR (1992)	2.5E-04	1.0E+00	1.1E-04	3.0.E-06	0.3
Average				**4.3.E-05**	**3.7**
Geometric Mean				**3.1.E-05**	**2.7**

Porosity not used in calculation

Geo Mean (without porosity)	**2.6E-05**	**2.2**
Geo Mean (with porosity)	**3.6E-05**	**3.1**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

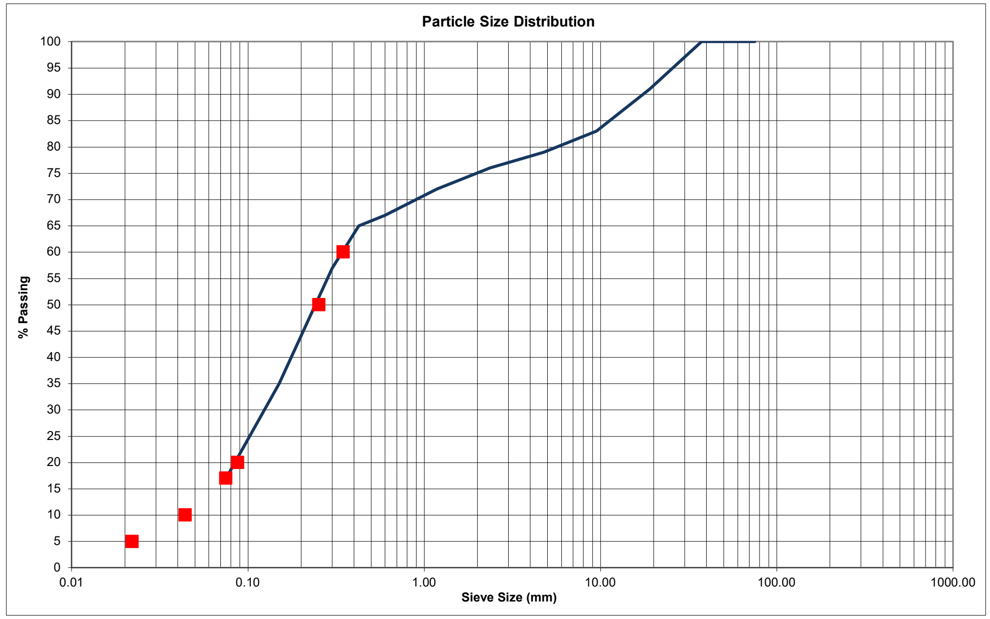
Sample Location:	BH01-14
Sample Depth (m):	4.7
SOIL TYPE:	Gravelly sand

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	91
9.5	83
4.75	79
2.36	76
1.18	72
0.60	67
0.425	65
0.30	57
0.15	35
0.075	17

Particle Size (mm)	
D_{60}	0.35
D_{50}	0.25
D_{20}	0.09
D_{17}	0.08
D_{10}	0.04
D_5	0.02

Uniformity Coefficient

D_{60} / D_{10} **7.9**



Note: grading curve based on that for BH01-14, Sample 1 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **31%**

Calculated Porosity (n) **31%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	4.4E-05	9.0.E-06	0.8
Amer & Awad (1974)	4.6E-02	6.6E-02	4.4E-05	5.7.E-06	0.5
Shahabi et. al. (1984)	7.9E-02	6.6E-02	4.4E-05	9.8.E-06	0.9
Kenney et. al. (1984)	5.1E-02	1.0E+00	2.2E-05	2.4.E-05	2.1
Slichter (1898)	1.0E-01	2.2E-02	4.4E-05	4.4.E-06	0.4
Terzaghi (1925)	2.1E-02	4.3E-02	4.4E-05	1.7.E-06	0.1
Beyer (1964)	8.3E-03	1.0E+00	4.4E-05	1.6.E-05	1.4
Sauerbrei (1992)	3.6E-02	6.6E-02	7.5E-05	1.3.E-05	1.1
Pavchich (1966)	7.1E-02	6.6E-02	7.5E-05	2.6.E-05	2.2
USBR (1992)	2.4E-04	1.0E+00	8.8E-05	1.8.E-06	0.2
Average				**1.1.E-05**	**1.0**
Geometric Mean				**7.8.E-06**	**0.7**

Porosity not used in calculation

Geo Mean (without porosity)	**8.8E-06**	**0.8**
Geo Mean (with porosity)	**7.2E-06**	**0.6**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

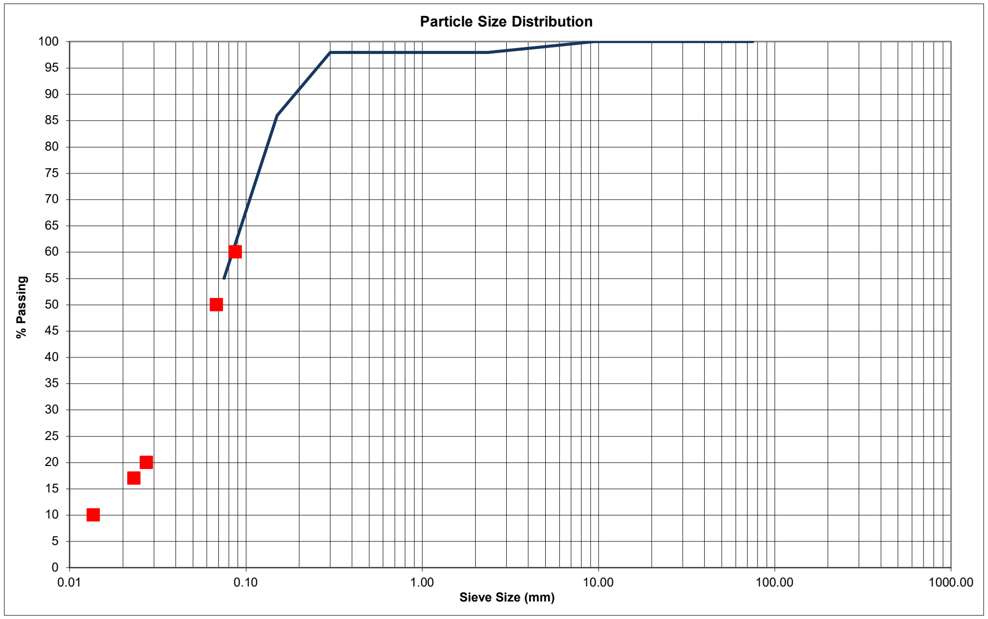
Sample Location:	**BH01-14**
Sample Depth (m):	**10.1**
SOIL TYPE:	**Silty sand**

Sieve Size (mm)	% Passing
75	100
37.5	100
19.0	100
9.5	100
4.75	99
2.36	98
1.18	98
0.60	98
0.425	98
0.30	98
0.15	86
0.075	55

Particle Size (mm)	
D_{60}	0.09
D_{50}	0.07
D_{20}	0.03
D_{17}	0.02
D_{10}	0.01
D_5	0.01

Uniformity Coefficient

D_{60} / D_{10} **6.4**



Note: grading curve based on that for BH01-14, Sample 3 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **33%**

Calculated Porosity (n) **33%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	1.4E-05	8.6.E-07	0.1
Amer & Awad (1974)	2.8E-02	8.3E-02	1.4E-05	4.2.E-07	0.0
Shahabi et. al. (1984)	7.7E-02	8.3E-02	1.4E-05	1.2.E-06	0.1
Kenney et. al. (1984)	5.1E-02	1.0E+00	6.8E-06	2.3.E-06	0.2
Slichter (1898)	1.0E-01	2.7E-02	1.4E-05	5.0.E-07	0.0
Terzaghi (1925)	2.1E-02	5.4E-02	1.4E-05	2.0.E-07	0.0
Beyer (1964)	8.7E-03	1.0E+00	1.4E-05	1.6.E-06	0.1
Sauerbrei (1992)	3.6E-02	8.3E-02	2.3E-05	1.5.E-06	0.1
Pavchich (1966)	6.6E-02	8.3E-02	2.3E-05	2.9.E-06	0.2
USBR (1992)	1.7E-04	1.0E+00	2.7E-05	1.2.E-07	0.0
Average				**1.2.E-06**	**0.1**
Geometric Mean				**7.8.E-07**	**0.1**

Porosity not used in calculation

Geo Mean (without porosity)	**7.9E-07**	**0.1**
Geo Mean (with porosity)	**7.8E-07**	**0.1**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

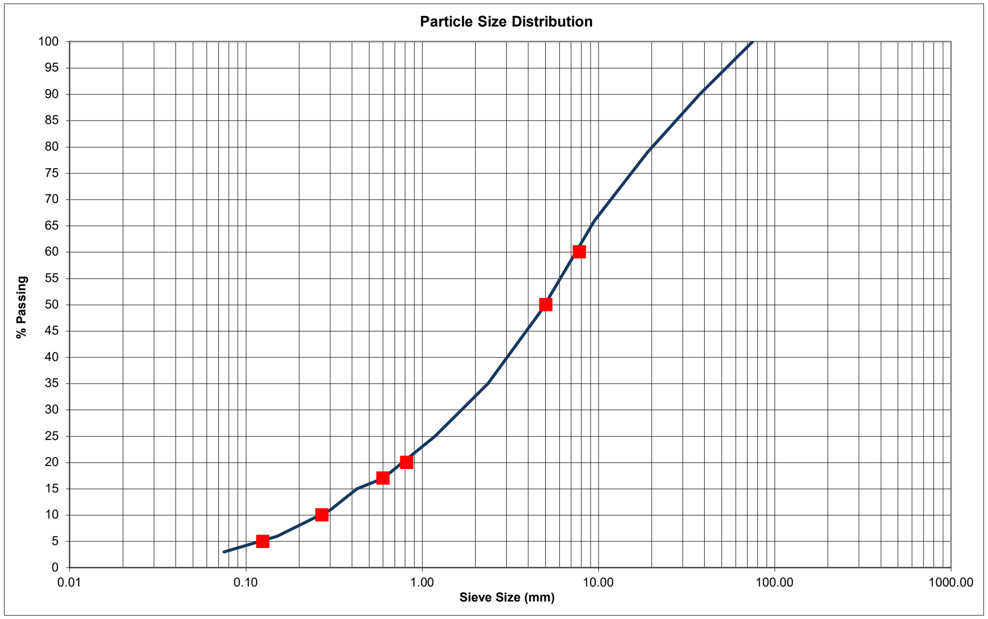
Sample Location: BH01-15
Sample Depth (m): 1.6
SOIL TYPE: Sandy gravel

Sieve Size (mm)	% Passing
75	100
37.5	90
19.0	79
9.5	66
4.75	49
2.36	35
1.18	25
0.60	17
0.425	15
0.30	11
0.15	6
0.075	3

Particle Size (mm)	
D_{60}	7.82
D_{50}	5.03
D_{20}	0.82
D_{17}	0.60
D_{10}	0.27
D_{5}	0.13

Uniformity Coefficient

D_{60} / D_{10} **29.0**



Note: grading curve based on that for BH01-15 (Golder, 2002) with missing sieve sizes extrapolated

Estimated Porosity (ε) **26%**

Calculated Porosity (n) **26%** fraction

Permeability Calculations

Author	C_1	f(n)	D_e (m)	K (m/s)	K (m/d)
Hazen (1930)	4.7E-03	1.0E+00	2.7E-04	3.4.E-04	29.0
Amer & Awad (1974)	1.8E-01	3.0E-02	2.7E-04	3.9.E-04	33.6
Shahabi et. al. (1984)	1.7E-01	3.0E-02	2.7E-04	3.6.E-04	31.5
Kenney et. al. (1984)	5.1E-02	1.0E+00	1.3E-04	7.8.E-04	67.5
Slichter (1898)	1.0E-01	1.1E-02	2.7E-04	8.4.E-05	7.2
Terzaghi (1925)	2.1E-02	1.9E-02	2.7E-04	2.9.E-05	2.5
Beyer (1964)	5.7E-03	1.0E+00	2.7E-04	4.1.E-04	35.1
Sauerbrei (1992)	3.6E-02	3.0E-02	6.0E-04	3.8.E-04	33.0
Pavchich (1966)	1.1E-01	3.0E-02	6.0E-04	1.2.E-03	101.2
USBR (1992)	4.6E-04	1.0E+00	8.2E-04	3.0.E-04	26.1
Average				**4.2.E-04**	**36.7**
Geometric Mean				**2.9.E-04**	**25.4**

Porosity not used in calculation

Geo Mean (without porosity)	**4.2E-04**	**36.6**
Geo Mean (with porosity)	**2.3E-04**	**20.0**

Porosities for Common Materials

Soil	Porosity
Clay	45-55
Silt	35-50
Sand	25-40
Gravel	25-40
Sand & gravel mixes	10-35
Glacial till	10-25
Sandstone	5-30
Limestone/Dolomite	1-20
Fractured Crystalline Rock	0-10
Vesicular Basalt	0-10
Dense, solid rock	<1

APPENDIX D

SGS Geochemical Analyses

Synthetic Precipitation Leaching Procedure - EPA Method 1312

Parame	Unit	CCME FAL	CCME Marine	MDMER	ENV #1-300-18	GRG-2 Knelson Tl	GDG-3 Knelson Tl	GDG-4 Knelson Tl	GDG-5 Knelson Tl	F2 Ro Tl	F3 Ro Tl	F4 Ro Tl	F5 Ro Tl
LIMS				Effective	14682-NOV20	14748-NOV20	14682-NOV20	14682-NOV20	14682-NOV20	14748-NOV20	14682-NOV20	14682-NOV20	14682-NOV20
Sample	g	-	-	01-Jun-2021	25	100	25	25	25	100	25	25	25
Ext Fluic	#1 or #2	-	-	-	1	1	1	1	1	1	1	1	1
Ext Volu	mL	-	-	-	500	2000	500	500	500	2000	500	500	500
Final pH	no unit	6.0-9.5	7.0-8.7	6.0-9.5	9.27	9.17	9.31	9.01	9.25	9.47	9.32	9.28	9.43
Hg	mg/L	0.000026	-	-	< 0.00001	0.00001	< 0.00001	< 0.00001	< 0.00001	< 0.00001	< 0.00001	< 0.00001	< 0.00001
Al	mg/L	0.1@pH>6.5	-	-	**0.609**	**0.27**	**0.904**	**0.352**	**0.612**	**0.39**	**0.748**	**0.711**	**0.746**
As	mg/L	0.005	0.013	0.10	**0.0646**	**0.154**	**0.154**	**0.0549**	**0.0785**	**0.054**	**0.0512**	**0.0291**	**0.0413**
Ag	mg/L	0.00025	0.0075	-	< 0.00005	< 0.0005	< 0.00005	< 0.00005	< 0.00005	< 0.0005	< 0.00005	< 0.00005	< 0.00005
Ba	mg/L	-	-	-	0.00625	0.0148	0.00511	0.00494	0.00568	0.0173	0.00386	0.00508	0.00647
Be	mg/L	-	-	-	0.000035	< 0.00007	< 0.000007	< 0.000007	0.000016	< 0.00007	< 0.000007	0.000013	0.000022
B	mg/L	1.5	-	-	0.012	< 0.02	0.008	0.008	0.010	< 0.02	0.007	0.012	0.014
Bi	mg/L	-	-	-	0.000505	< 0.00007	0.000104	0.000586	0.000331	< 0.00007	0.000056	0.000885	0.000356
Ca	mg/L	-	-	-	9.05	10.0	7.87	13.3	9.58	7.94	7.11	8.69	7.67
Cd	mg/L	0.00009	0.00012	-	0.000008	< 0.00003	0.000007	0.000007	0.000008	< 0.00003	0.000011	0.000008	0.000015
Co	mg/L	-	-	-	0.00140	0.00012	0.00115	0.000334	0.00102	0.00014	0.000657	0.000611	0.000946
Cr	mg/L	-	-	-	0.00467	0.0011	0.00553	0.00337	0.00908	0.0015	0.00404	0.00726	0.0122
Cu	mg/L	0.002	-	0.10	**0.0053**	< 0.002	**0.0064**	0.0014	**0.0024**	< 0.002	**0.0031**	**0.0020**	**0.0021**
Fe	mg/L	0.3	-	-	**1.10**	< 0.07	**0.819**	**0.357**	**0.909**	0.12	**0.593**	**0.937**	**1.13**
K	mg/L	-	-	-	0.151	0.12	0.119	0.091	0.229	0.06	0.116	0.096	0.216
Li	mg/L	-	-	-	0.0031	0.003	0.0038	0.0030	0.0047	0.002	0.0025	0.0027	0.0036
Mg	mg/L	-	-	-	0.778	0.69	0.929	0.788	0.671	0.50	0.664	0.587	0.626
Mn	mg/L	0.43	-	-	0.0132	0.0033	0.0108	0.00404	0.0118	0.0033	0.00761	0.00979	0.0133
Mo	mg/L	0.073	-	-	0.00133	0.0016	0.00090	0.00191	0.00178	0.0011	0.00055	0.00107	0.00142
Na	mg/L	-	-	-	6.24	5.06	6.24	4.95	4.96	5.88	5.10	5.51	5.68
Ni	mg/L	0.03	-	0.25	0.0035	< 0.001	0.0029	0.0023	0.0037	< 0.001	0.0019	0.0029	0.0027
P	mg/L	-	-	-	0.006	< 0.003	0.004	< 0.003	< 0.003	< 0.003	0.006	< 0.003	0.007
Pb	mg/L	0.001	-	0.08	**0.00110**	< 0.0001	0.00035	0.00004	0.00015	< 0.0001	0.00061	0.00012	0.00036
Sb	mg/L	-	-	-	< 0.0009	< 0.009	0.0108	0.0035	0.0011	< 0.009	0.0026	0.0015	0.0009
Se	mg/L	0.001	-	-	0.00023	0.0004	0.00033	0.00072	0.00029	< 0.0004	0.00009	0.00023	0.00011
Si	mg/L	-	-	-	3.10	2.49	3.90	2.40	3.12	2.57	3.00	3.23	3.63
Sn	mg/L	-	-	-	0.00017	< 0.0006	0.00010	< 0.00006	0.00012	< 0.0006	0.00008	0.00013	0.00014
Sr	mg/L	-	-	-	0.00917	0.0107	0.0106	0.0117	0.0120	0.0083	0.00918	0.00833	0.00925
Ti	mg/L	-	-	-	0.0396	0.0036	0.0415	0.0103	0.0314	0.0062	0.0270	0.0259	0.0406
Th	mg/L	-	-	-	0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	< 0.0001	0.0002
Tl	mg/L	0.0008	-	-	0.000005	< 0.00005	0.000005	< 0.000005	0.000007	< 0.00005	< 0.000005	< 0.000005	0.000014
U	mg/L	0.015	-	-	0.000142	0.00010	0.000049	0.000029	0.000281	0.00003	0.000052	0.000025	0.000322
V	mg/L	-	-	-	0.00352	0.0026	0.00551	0.00187	0.00364	0.0023	0.00382	0.00324	0.00405
W	mg/L	-	-	-	0.00078	0.0018	0.00100	0.00024	0.00065	0.0020	0.00060	0.00021	0.00095
Y	mg/L	-	-	-	0.000177	< 0.00002	0.000144	0.000033	0.000117	0.00002	0.000113	0.000118	0.000158
Zn	mg/L	0.007	-	0.40	0.003	< 0.02	< 0.002	< 0.002	0.003	< 0.02	0.002	< 0.002	0.004

Canadian Council of Ministers of the Environment (CCME), *Water Quality Guidelines for the Protection of Freshwater Aquatic Life (FAL).*
CCME. Water Quality Guidelines for the Protection of Marine Aquatic Life (Marine).
Department of Justice Canada. 2002. *Metal and Diamond Mining Effluent Regulations*, Fisheries Act SOR-2002-222. Effective June 2021.

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