UPDATED PRELIMINARY ECONOMIC ASSESSMENT
NI 43-101 TECHNICAL REPORT
CHIDLIAK PROJECT
NUNAVUT, CANADA

JDS Energy & Mining Inc.

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This report entitled Updated Preliminary Economic Assessment NI 43-101 Technical Report Chidliak Project, Nunavut, Canada, effective as of May 23, 2018 was prepared and signed by the following authors:

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Peregrine Diamonds Ltd. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Peregrine Diamonds Ltd. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

Peregrine Diamonds Ltd. (Peregrine or Peregrine Diamonds) commissioned JDS Energy and Mining Ltd. (JDS) to complete an updated Preliminary Economic Assessment (PEA) for the Chidliak Phase 1 Diamond Project ("Chidliak Project" or "the Project"), located 120 km northeast of Iqaluit, the capital of Nunavut, on Canada's Baffin Island. The requirement for an updated PEA was driven primarily by the drilling of an additional 15 holes totalling 5,300 m in the CH-6 pipe, which expanded the CH-6 resource estimate, as reported by Peregrine in "2018 Technical Report: Mineral Resource Update for the Chidliak Project, Baffin Island, Nunavut, Canada", dated effective February 15, 2018.

The structure and content of this PEA report uses National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) guidelines.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized. In this PEA, 100% of mine plan tonnes are Inferred Mineral Resources.

1.2 Project Description

The Chidliak Phase 1 Diamond Development Project comprises 74 kimberlite pipes (71 on CHI claims and three on AN claims) (refer to Section 4 and Figure 4-2). Of these, 45 are known to be diamondiferous, and two kimberlites, CH-6 and CH-7, have been adequately sampled and explored in order to define an Inferred Mineral Resource estimate.

The CH-6 and CH-7 deposits are conducive to open pit mining and are envisioned to produce a total of 9.5 million tonnes (Mt) of processing plant feed, and 75.1 Mt of waste (7.9:1 overall strip ratio), over a 13-year mine life.

The kimberlite processing plant and tailings management facility (TMF) will be located within 2.5 km of the CH-6 pit, as it contains the higher value resource, and would therefore be mined first. The TMF will be designed as a conventional facility, located adjacent to the processing plant.

The processing plant will operate year-round at 2,000 tonnes per day (t/d) using three stages of crushing, scrubbing and de-gritting, dense media separation, high intensity magnetic separation and X-ray recovery.

It is projected that approximately 16.7 million (M) carats (ct) of diamonds will be recovered over the 13-year mine life, yielding an average life of mine grade of 1.8 carats/tonne (ct/t).

Access to the site will be by all-weather road from Iqaluit. Annual supplies, except for specialty and perishable items, will be transported to Iqaluit during the summer ocean transport season.

The Project is located in the Arctic and as such will be designed to withstand severe cold conditions, as experienced by other Canadian diamond mines.

1.3 Location, Access and Ownership

1.3.1 Location

The Chidliak project is centered at 64° 28’ 26” N latitude and 66° 21’ 43” W longitude (see Figure 4-1), located on the Hall Peninsula of southern Baffin Island in Nunavut, Canada, approximately 120 km northeast of the city of Iqaluit, the capital city of Nunavut (population 6,699 in 2011 census).

1.3.2 Mineral Tenure

The Chidliak project consists of 266 CHI claims covering 277,997 hectares (ha) and 53 AN claims covering 37,126 ha. The CHI and AN claims are collectively referred to as the Chidliak project, which comprises a combined total of 319 claims covering 315,123 ha. All mineral claims are in good standing and are registered to Peregrine Diamonds Ltd.

Approximately 83% of the contiguous CHI and AN mineral claims are located on Crown Lands with the remaining 17% of the claims located on Inuit Owned Lands (IOLs). The area of the Project considered for Phase 1 development at Chidliak lies entirely on Crown Lands.

1.3.3 Access

There is no permanent road access to the Project and it is accessed primarily by air from Iqaluit via helicopter or fixed-wing aircraft. Work at Chidliak is staged from any of four camps established on the Project site (Discovery, Sunrise, Aurora and CH-6 Temporary Camp), the primary camp being Discovery camp, which has a 570 m long natural gravel landing strip for fixed-wing aircraft and a helicopter pad to support drilling and logistical operations. During winter months, when sufficient ice and snow is present, the Project can also be accessed via the Iqaluit-Chidliak winter trail, which supports the transport of equipment, supplies and samples, if required.

Access to Iqaluit is by commercial shipping and by commercial aircraft, with flights scheduled daily from Ottawa, and thrice weekly from both Montreal and Edmonton (via Yellowknife).

1.3.4 Ownership

Peregrine currently has 100% ownership of all claims at Chidliak and is only subject to a Crown Royalty as prescribed in the Nunavut Mining Regulations.

1.4 History, Exploration and Drilling

In 1996 and 1997, an area of Hall Peninsula, including some of the ground now covered by Chidliak, was explored by International Capri Resources Ltd. for nickel-copper-platinum group elements, lead-zinc-copper, and lode gold deposits. No diamond exploration is known to have occurred in the area of the Chidliak project prior to 2005, when a regional till sampling survey of the southern Baffin Island was undertaken by BHP Billiton (BHPB) and Peregrine, with BHPB as operator. This sampling work discovered kimberlite indicator minerals and led to the establishment of the Chidliak project and all subsequent exploration work.

Exploration work conducted on the Chidliak project to date has resulted in the discovery of 74 kimberlites. Work has included:

- Heavy mineral sampling and compositional analysis of kimberlitic indicator minerals;
- Airborne and ground geophysical surveys, including magnetic, electromagnetic and gravimetric methods;
- Core and reverse circulation (RC) drilling for discovery, delineation and sampling of kimberlites;
- Processing of samples of drill core for microdiamonds, bulk density, petrography, whole rock geochemistry and macrodiamonds[1] (mini-bulk sample);
- Bulk sampling through large-diameter RC drill sampling and processing of recovered material for macrodiamonds; and
- Bulk sampling through surface trenching and processing of excavated material for macrodiamonds.

Key datasets used as a basis for the Mineral Resource estimates in the CH-6 and CH-7 pipes include:

- Drill logs, petrography and representative samples for 75 core holes (16,662 m) and 85 small-diameter RC holes (1,155 m);
- Microdiamond sample assay results for 12,411 kg of drill core and surface bulk samples;
- Bulk density assay results for 3,386 samples;
- Drill logs and related data for six large-diameter RC holes (1,212 m) in CH-7 from which a total of 329 m3 (809.5 t [dry]) of kimberlite was collected and processed for macrodiamonds; and
- Macrodiamond recoveries from 14 t of CH-6 drill core (2010), 47.2 t of CH-7 surface trench material (2010), 404.3 t of CH-6 surface trench material (2013), and 809.5 t of CH-7 RC-drilled material (2015).

1.5 Geology and Mineralization

Much of the Chidliak area comprises upland surfaces and stepped plain or dissected upland surfaces. Glacial tills are found throughout the area, generally as thin veneers on bedrock. Ice flow directions in the area are dominated by the Hall Ice Divide, parallel to the length of the peninsula, with the primary ice flow direction parallel to the ice divide and then emanating to the north and south away from it.

The majority of the Chidliak area is underlain by Archean and Proterozoic orthogneisses, paragneisses and metavolcanics. Paleoproterozoic metasediments occur in north-south trending, discontinuously mapped belts on the western part of the Project area. Rocks of the Paleoproterozoic Cumberland Batholith occur along the far western margin of the Project.

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[1] The term macrodiamond is used throughout this Report to refer to diamonds recovered by commercial diamond production plants, which typically recover diamonds larger than the Diamond Trading Company (DTC) sieve category 1 (~ 0.01 ct). The DTC+1 sieve is roughly equivalent to 0.85 mm square-mesh sieve.

The Jurassic-aged Chidliak kimberlites occur as pipes and rare sheet-like bodies. Two main types of pipes are present: those with volcaniclastic kimberlite infill only and those infilled by a combination of volcaniclastic, coherent and apparent coherent kimberlite deposits (VK, CK and ACK, respectively). The VK-only pipes tend to be larger (125 m to 150 m radius) than the combined-infill pipes (50 m to 75 m radius).

CH-6, and CH-7 are two of several kimberlites at Chidliak with potential economic interest, and are both combined-infill pipes. In addition to basement xenoliths, most of the pipes contain xenoliths of now-eroded Late Ordovician to Early Silurian carbonate and clastic rocks.

The CH-6 kimberlite is a steep-sided, slightly southwest-plunging, kidney shaped to elliptical body with a surface area of approximately 1.0 ha. It is infilled by two main geological units, KIM-L and KIM C, which are most readily distinguished megascopically by the respective presence or paucity of Paleozoic carbonate xenoliths. KIM-L, the dominant pipe infill, is dark, competent and texturally heterogeneous ranging from PK (pyroclastic kimberlite, a variety of VK) to ACK with depth in the body. Crustal dilution is generally low (<5%) and heterogeneously distributed. Mantle xenoliths and xenocrysts are common throughout. Volumetrically minor discontinuous CK units of uncertain origin occur intercalated with KIM-L. KIM-C is a comparatively homogeneous CK that occurs along the north and northeast margin below 80 m depth. The two geological domains modelled at CH-6 each consist primarily of the kimberlite unit of the same name: KIM-L and KIM-C, and comprise 89% and 11% of the modelled pipe volume respectively.

The CH-7 kimberlite is a steep-sided, southwest-plunging body comprised of at least two coalescing lobes with a combined surface area of approximately 1.0 ha. Five main geological units have been recognized at CH-7: KIM-1 to KIM-5, each characterized by distinct physical, and in some cases geochemical, characteristics. KIM-1 is coarse-grained CK whereas KIM-2, KIM-3 and KIM-4 are PK. KIM-5 is texturally variable (PK, ACK) and variably lateritized. A variety of other minor units are present, such as a gneiss xenolith-bearing CK (KIM-6) and blocks of gneiss and carbonate. Five of the seven geological domains modelled at CH-7 consist primarily of the kimberlite unit of the same name: KIM 1 to KIM-5. Of these, KIM-3 and KIM-4 display the greatest variability in terms of presence of other different units, and in the case of KIM-4 is poorly constrained by drilling to date. The remaining two domains, R and S, each comprise several units in roughly equal proportions. Subdivision of these domains is not possible based on current drilling and geological information. KIM-2 comprises 61% of the modelled pipe volume with the smaller domains in the central and northern portion of the pipe together making up the remaining 39%.

1.6 Mineral Resource Estimates

The Mineral Resource estimate presented in this report was originally published in 2018 (Mineral Resource Update for the Chidliak Project, Baffin Island, Nunavut, Canada dated February 15, 2018, Fitzgerald et al., 2018).

The Mineral Resource estimate for CH-6 comprises a portion of the KIM-L and KIM-C geological domains between surface and 525 metres below surface (mbs), equivalent to 155 metres above sea level (masl). Mineral Resources in CH-7 extend from surface to a maximum depth of 240 mbs (450 masl). These represent the portions of the pipes that are well constrained by drilling and for which sufficient evaluation data are available.

Grade estimates in CH-6 are based on an assessed effectively constant diamond size frequency distribution (SFD) throughout CH-6 and a calibration of the ratio of microdiamond[2] abundance (stones per kilogram, st/kg) to recoverable macrodiamond grade. The calibration is anchored to microdiamond and macrodiamond data from a corresponding volume of KIM-L material (the 2013 CH-6 trench bulk sample). Drill core microdiamond results were used in conjunction with the established ratio of microdiamond abundance to recoverable grade to derive average grade estimates for the CH-6 resource domains. Grade estimates for the resource domains of CH-7 are either based on a calibrated microdiamond approach (as described above for CH-6), or based directly on recovered large-diameter drill (LDD) sample grades. Average grades have been used for all Mineral Resource estimates. Diamond values are based on valuation of parcels of 1,013.54 ct from CH-6 and 735.75 ct from CH-7. Average modelled values as stated by WWW International Diamond Consultants Ltd. (2018a and 2018b) have been adopted for the Mineral Resource estimates.

A Mineral Resource statement for the Chidliak project that includes all currently defined Mineral Resources is presented in Table 1-1. All grades are reported as those recoverable above a 1.18 mm bottom cut off, assuming the recovery efficiency achieved in the sample process plants used to treat Chidliak kimberlite and recover diamonds from surface excavation and LDD samples. The recoverable grade estimates may be adjusted for the expected recovery efficiency of the planned production processing plant. Average diamond values were derived by applying best estimate value distribution models to models of recoverable diamond SFD, and therefore, also represent “recoverable” values that correlate with the +1.18 mm grades reported. Changing process plant efficiency (relative liberation and recovery of diamonds) may also require an adjustment to diamond values.

Table 1-1: Mineral Resource Statement for the Chidliak Project

Body	Resource Classification	Depth Range	Volume (Mm^3)	Density (g/cm^3)	Tonnage (Mt)	Grade (ct/t)	Carats (Mct)	Value (US $/ct)
CH-6	Inferred	0 to 525 mbs	2.85	2.62	7.46	2.41	17.96	151
CH-7	Inferred	0 to 240 mbs	1.94	2.57	4.99	0.85	4.23	114
All	Inferred		4.79	2.60	12.45	1.78	22.19	132

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Target for Further Exploration (TFFE) volume and tonnage range estimates have been made for CH-6, CH-7 (Fitzgerald et al., 2018). However, the TFFE material was not included in the PEA.

1.7 Mining

It is planned to open pit (OP) mine the Chidliak CH-6 and CH-7 deposits. Mining of the deposits will produce a total of 9.5 Mt of processing plant feed and 75.1 Mt of waste (7.9:1 overall strip ratio) over a 13-year mine life. The mine design process for the deposits commenced with the development of OP optimization design

[2] The term microdiamond is used throughout this report to refer to diamonds recovered through caustic fusion of kimberlite at a bottom screen size cut-off of 105 µm (~0.00002 ct). Rare larger diamonds that may be recovered by a commercial production plant may be recovered through this process but are still referred to as microdiamonds.

parameters. These parameters included estimates of diamond price, mining dilution, process recovery, off-site costs, geotechnical constraints (slope angles) and royalties (see The current life of mine (LOM) plan focuses on achieving consistent plant feed production rates, and early mining of higher value material, as well as balancing grade and strip ratios.

Table 1-2).

The current life of mine (LOM) plan focuses on achieving consistent plant feed production rates, and early mining of higher value material, as well as balancing grade and strip ratios.

Table 1-2: Mine Planning Optimization Design Parameters*

Parameter	Unit	Values	
		CH-6	CH-7
Average LOM Diamond Price	US$/carat	173	147
Exchange Rate	C$:US$	1.28	
Selling Cost	% of price	4	
Selling Cost	US$/carat	6.92	5.88
Net Average LOM Diamond Price	C$/carat	213	181
Waste Mining Cost	C$/t waste mined	3.80	5.20
Mineralized Material Mining Cost	C$/t processed	4.40	7.90
Strip Ratio (tonnes waste : tonnes processed) (Estimated)	t:t	10	4
Mining Cost	C$/t processed	39.90	27.04
Processing	C$/t processed	18.00	
Freight	C$/t processed	16.75	
Coarse PK haulage	C$/t processed	5.00	
General and Administrative (G&A)	C$/t processed	18.10	
Site Services	C$/t processed	9.92	
Government Royalty	C$/t processed	10.00	7.00
Subtotal: Processing/G&A (Excluding Waste Mining)	C$/t processed	78.37	77.47
Total Operating Cost w/ Mining	C$/t processed	118.27	104.51
Process Recovery	%	98	
Mining Dilution	%	5	6
Mining Recovery	%	95	
Process Throughput Rate	t/d	2,000	
Overall Pit Slope Angle	°	35 - 52	35 - 48

Note: *The values in this table vary slightly from those used in the economic model as parameters were further refined in the economic model as the Project progressed. The differences are not considered material to pit shape definition.

NPV Scheduler pit optimization software was used to determine the optimal mining shells with the assumed overall slope angles shown in the previous table. Preliminary mining stages were selected and mine planning and scheduling were then conducted on the selected optimal shells. The mineable resources for the Chidliak deposit are presented in Table 1-3.

Table 1-3: PEA Mine Plan Summary

Description	Unit	Value
Mine Production Life	yr	13
Process Feed Material	Mt	9.5
Diluted Diamond Grade	ct/t	1.79
Contained Carats	Mct	17.0
Waste	Mt	75.1
Total Material	Mt	84.6
Strip Ratio (tonnes waste : tonnes processed)	t:t	7.9

Note: Mineral Resources are reported at a bottom cut off of 1.18 mm
Mineable Resources are not Mineral Reserves and do not have demonstrated economic viability.

The mining sequence was divided into a number of stages at each of CH-6 and CH-7 and designed to maximize grade and value, reduce pre-stripping requirements in the early years, and maintain the plant at the full production capacity of 2,000 t/d. The mine life of approximately 13 years produces from an open pit at the CH-6 kimberlite pipe for the first nine years, followed by production via open pit at the CH-7 kimberlite. The mine and plant production schedule is summarized in Table 1-4.

Table 1-4: Mine and Plant Production Schedule

Description	Unit	Total	Year													
			-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Plant Feed	Mt	9.5	0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Grade	ct/t	1.79	2.57	2.44	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.80	0.74	0.79
Contained Carats	Mct	17.0	0.3	1.5	1.8	1.9	1.6	1.8	1.5	1.6	1.6	1.2	0.6	0.6	0.5	0.6
Waste	Mt	75.1	3.7	9.7	9.5	9.6	9.6	8.9	10.1	1.9	0.7	3.4	3.0	3.1	1.7	0.2
Strip Ratio	t:t	7.9	33.3	15.6	13.1	13.1	13.1	12.2	13.9	2.6	1.0	4.6	4.2	4.3	2.4	0.2
Total Material	Mt	84.6	3.8	10.3	10.3	10.3	10.3	9.6	10.8	2.6	1.4	4.1	3.8	3.9	2.5	0.9
Total Processed	Mt	9.5	-	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Processed Grade	ct/t	1.79	-	2.46	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.80	0.74	0.79
Recovered Carats	Mct	16.7	-	1.8	1.7	1.9	1.5	1.7	1.5	1.6	1.6	1.1	0.6	0.6	0.5	0.6

1.8 Metallurgical Testing and Mineral Processing

Specific process and equipment manufacturer metallurgical testing has not been undertaken on material from the CH-6 and CH-7 kimberlites to date. An Ore Dressing Study (ODS) will be delineated at the next engineering stage to refine and optimize the process flowsheet and equipment selection.

Processing of Chidliak kimberlite from the mini-bulk and bulk samples of CH-6 and CH-7 provided preliminary treatability information for the different kimberlite types (e.g. geological units) through the various metallurgical processes i.e. crushing, scrubbing, screening, de-sliming, Dense Media Separation (DMS), final diamond recovery (X-ray and grease), re-crush, de-grit, fines thickening and materials handling. The preliminary treatability information indicates that a conventional DMS-based mineral processing flowsheet for the kimberlite is appropriate to effectively capture diamonds greater than 1.0 mm.

Initial sample treatment indicated that the amount of clay in the weathered surface kimberlite would not impact the material handling characteristics of the mineralized feed in a production processing plant. The weathered surface kimberlite was observed to be fine grained with low clay content.

Based on JDS experience from other Northern Canadian diamond operations, a 98% diamond recovery has been assumed for the proposed processing facility.

1.9 Recovery Methods

The Chidliak process plant will have a capacity of 2,000 t/d (0.7 Mt/a).

The preliminary flowsheet is based on existing Canadian diamond mine operations and will be optimized at a later engineering stage when process and equipment specific test work results become available.

Kimberlite processing and diamond recovery at Chidliak will involve:

- Run of Mine (ROM) stockpiles - to allow kimberlite blending and oversize waste rock removal;
- Primary and secondary crushing - jaw and cone crusher plants;
- Crushed feed stockpile - to provide buffer capacity between primary crushing and the main plant;
- Scrubbing and de-gritting – fines removal, friable rock and clay deagglomeration;
- Tertiary crushing – high-pressure grinding rolls (HPGR), diamond liberation, fines generation;
- DMS - heavy mineral, diamond concentration;
- Recovery:
  - Wet high intensity magnetic separation;
  - Wet X-ray sorting;
- Drying;
- Dry single particle X-ray sorting;
- Grease table;

- Diamond cleaning; and
- Diamond preparation - for valuation prior to shipment off-site.

A batch plant planned to be located near the process plant building will be used for main plant audits, mine grade control and exploration sample treatment.

1.10 Infrastructure

The Project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 157 km of all-seasonal access road from Iqaluit to the plant site location;
- Process facilities;
- Diesel power plant;
- TMF and waste rock storage facilities (WRSF);
- Permanent camp (established for the construction stage);
- Administration building;
- Truck shop and warehouse;
- Mine dry and office complex;
- 20 ML of fuel storage and containment in Iqaluit;
- 300,000 L of on-site fuel storage and distribution;
- Industrial waste management facilities such as the incinerator; and
- Site water management facilities.

For the purposes of this study, it has been assumed that a deep-water port facility will be constructed in Iqaluit by a third party.

1.11 Environment and Permitting

One of Peregrine's highest priorities is to manage and mitigate the potential effects of the Project to the surrounding environment. Peregrine is committed to exploration and mining practices that are environmentally responsible and socially acceptable, and dedicated to creating and maintaining a safe environment for the land, employees, and nearby communities.

The permitting process for exploration and development of any mineral project is continuous and changes as the Project scope changes. As a project advances from exploration to mining, the activities take on a smaller geographic extent as work becomes more intensive. The permitting framework is set out in both the Nunavut Lands Claim Agreement (NLCA) and the Nunavut Planning and Project Assessment Act (NUPPA).

Peregrine will design, construct, operate, and decommission the Project to meet or, where possible, exceed all applicable environmental and safety standards practices and commitments. Peregrine will develop and implement an environmental management system that will define the processes by which compliance will be met and demonstrated, and will include ongoing monitoring and reporting to relevant parties at the various stages of the Project. The Project is expected to be non-acid generating. However, water management will be a critical component, as the most likely avenue for transport of any contaminants that may occur into the natural environment will be through surface or groundwater. As such, Peregrine will develop a water management plan that applies to all mining activities undertaken during all phases of the Project.

1.12 Operating and Capital Cost Estimates

1.12.1 Capital Cost Estimate

The capital cost estimate (CAPEX) includes all costs required to develop, sustain, and close the operation for a planned Phase 1, 13-year operating life. This study is limited to the Phase 1 development, and any potential additional resource development would require future studies. The construction schedule is based on an approximate 18-month all-weather road build period followed by approximately 14 months of on-site building. The accuracy of this estimate is -20% / +30% in accordance with the level of detail for a Class 4 estimate.

The high-level CAPEX estimate is shown in Table 1-5. The sustaining capital is carried over operating Years 1 through 13, and closure costs are projected from Year 14 to Year 20.

Table 1-5: Summary of Capital Cost Estimate

Capital Costs	Pre-Production ($M)	Sustaining / Closure ($M)	Total ($M)
Pre-Stripping	16.8	-	16.8
Mining Equipment	27.0	6.9	33.9
Mining Infrastructure / Ancillary	25.8	2.5	28.3
Site Development and Roadworks	112.8	-	112.8
Process Facilities	67.6	22.3	89.9
Utilities	27.0	-	27.0
Ancillary Facilities	28.3	-	28.3
Indirect Costs	53.8	-	53.8
Engineering and Project Management	28.4	-	28.4
Owners Costs	22.0	-	22.0
Salvage Value	-	-	-
Closure Costs	-	16.1	16.1
Subtotal	**409.5**	**47.8**	**457.3**
Contingency	54.9	8.5	63.4
Total Capital Cost	**464.4**	**56.3**	**520.7**

1.12.2 Operating Cost Estimate

The operating cost estimate (OPEX) for the Chidliak project has been prepared incorporating both off-site and on-site infrastructure as related to the mine plan and processing schedule. Table 1-6 summarizes the LOM OPEX estimate. The total unit operating cost of approximately $105/t is made up of about 42% support costs (general and administration, site services and freight), which is typical for remote Northern projects. Table 1-7 shows the main OPEX assumptions.

Table 1-6: Summary of Operating Estimate

Operating Costs	$/t processed	LOM ($M)
Mining	42.8	406
Processing	17.9	170
Freight / Logistics	14.9	141
Site Services	10.2	97
G&A	18.9	179
Total	**104.8**	**994**

Table 1-7: Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.232
Overall power consumption (all facilities)	kWh/t processed	20
Diesel cost (delivered)	$/litre	0.913
LOM average man-power (including contractors, excluding corporate)	employees	299

1.13 Economic Analysis

1.13.1 Main Assumptions

An economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations that represent an indicative value of the after-tax cash flows of the Project.

The economic model starts at the point of the Project when the decision has been made to proceed with construction - i.e. after the feasibility study, permitting and financing. The other primary assumptions used in the cash flow model are outlined in Table 1-8.

Table 1-8: Economic Assumptions

Item	Unit	Value
Base Model CH-6 Diamond Valuation	US$/carat	151
Base Model CH-7 Diamond Valuation	US$/carat	114
Annual Diamond Price Escalation (from 2018)	%	1.75
Exchange (FX) Rate	US$:C$	0.78
NPV Discount Rate	%	7.5
Operating Days	days/a	365
Non-Governmental Royalties	%	0
Diamond Recovery	%	98
Selling Cost	% of price	4
	US$/carat	6.9

1.13.2 Results

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized. All of the mine plan tonnes in this PEA are Inferred Mineral Resources.

The economic results for the Project based on the assumptions made are shown in Table 1-9. The Project meets the criteria (after tax) usually associated with financeable projects: a quick payback period of two years, a high internal rate of return (IRR) of 31% and a net present value (C$669 M) greater than the pre-production CAPEX of C$464 M.

Table 1-9: Economic Results

Parameter	Unit	Pre-tax Results	After-tax Results
$NPV_{0\%}$	C$M	2,002	1,332
$NPV_{7.5\%}$	C$M	1,052	669
IRR	%	39	31
Payback period	Production years	1.9	2.0

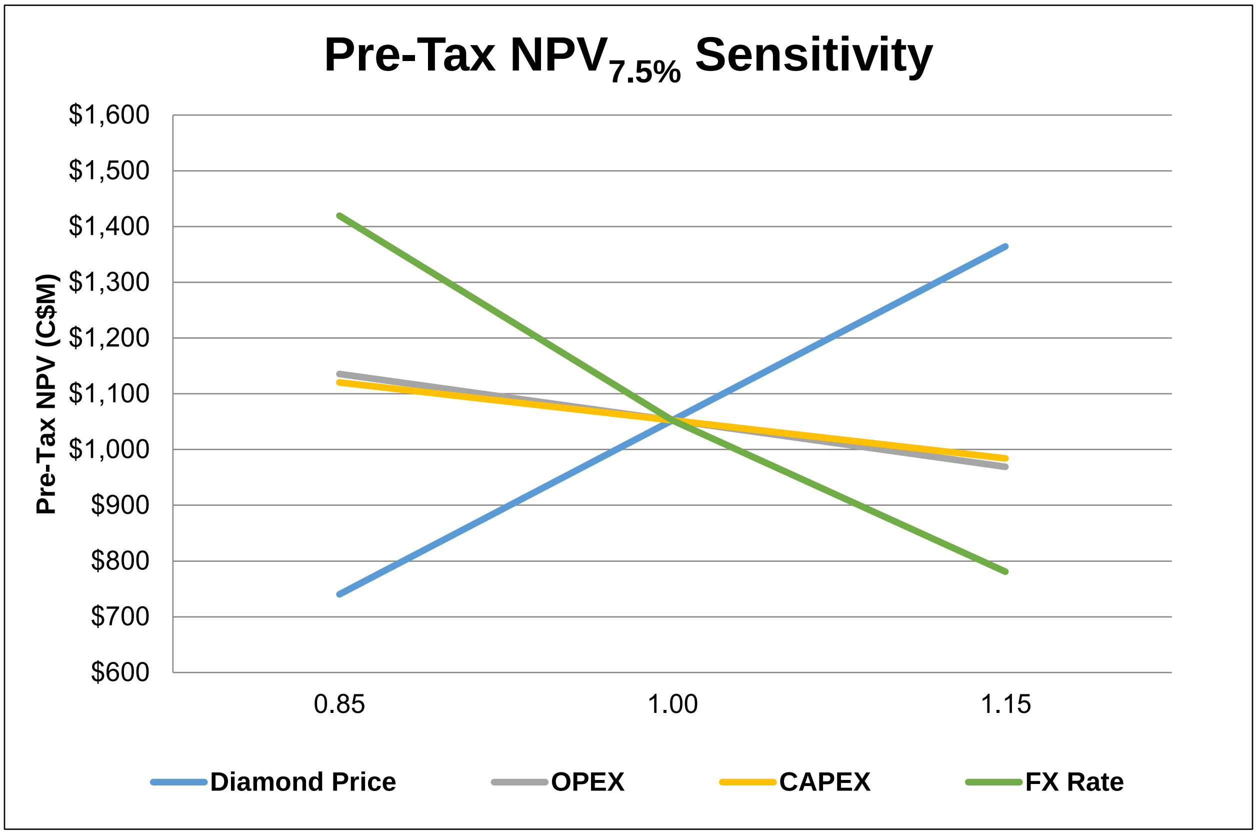
1.13.3 Sensitivities

Sensitivities to diamond price/grade, foreign exchange (FX) rate, OPEX, and CAPEX were conducted by increasing or decreasing each variable by 15%, while all other variables were held constant. As with most mining projects, the Chidliak Project is most sensitive to diamond price, grade and exchange rate. The Project is slightly more sensitive to CAPEX than OPEX. The results of the sensitivity analyses are shown in Table 1-10 and Figure 1-1.

Table 1-10: Sensitivities Analyses

Variable	Pre-tax $NPV_{7.5\%}$ (C$M)			Post-tax $NPV_{7.5\%}$ (C$M)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Diamond Price / Grade	740	1,052	1,364	467	669	868
CAPEX	1,120	1,052	984	737	669	601
OPEX	1,136	1,052	969	723	669	614
FX Rate	1,419	1,052	781	903	669	494

Figure 1-1: Pre-Tax NPV @ 7.5% Sensitivity Graph



1.14 Conclusions

It is the conclusion of the Qualified Persons (QPs) that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result shown by this study. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

1.14.1 Risks

The most significant potential risks associated with the Project are development of appropriate off-site infrastructure (primarily port construction in Iqaluit), operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, retention of mining personnel due to the remote location, ability to raise financing and diamond price variability.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active management.

The study is based on Inferred Mineral Resources, which are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is a risk that further work will not enable these resources to be categorized as Mineral Reserves.

1.14.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the Project. The major opportunities that have been identified at this time, excluding those typical to all mining projects, such as changes in diamond prices, exchange rates, etc. are:

- Expansion of the mine through delineation or development of additional economic kimberlites;
- Development of an underground bulk mining operation in CH-6, thereby increasing resource recovery and improving overall economics;
- Pit slope steepening resulting in an improved waste to plant feed strip ratio;
- Project strategy and optimization of mine plans and development schedule; and
- Tailings characterization to eliminate the need for a liner in the tailings management facility.

Further information and assessments are required before these opportunities can be included in the Project outcomes. This further work should be part of a Pre-Feasibility study.

1.15 Recommendations

In the opinion of JDS, the Chidliak Phase 1 Diamond Development Project is of sufficient merit to proceed to the Preliminary Feasibility Study (PFS) stage. This more advanced study will further detail:

- Mineral resources and diamond value;
- Engineering design;
- Project scheduling;
- Process flowsheet parameters; and
- Capital and operating costs.

The study will improve the confidence in the Project design and execution, and will result in an improved accuracy of project economics.

It is also recommended that environmental monitoring and program planning, and permitting continue as needed to support Peregrine's project development plans.

It is estimated that a PFS and its supporting work programs will cost approximately C$26.7M. Major components of future work recommended are:

- Deep core drilling at CH-6 to better quantify the TFFE and possibly upgrade it to a Mineral Resource;
- Bulk sample CH-6 via surface trench and LDD in order to potentially upgrade the Inferred Mineral Resource to Indicated level;
- Maintaining the collection of geotechnical data to refine mine design; and
- Additional sampling and ore dressing study test to be conducted for processing flow-sheet optimization.

2 Introduction

2.1 Basis of Technical Report

Peregrine Diamonds Ltd. (Peregrine) commissioned JDS Energy and Mining Ltd. (JDS) to update the report entitled “NI 43-101 Preliminary Economic Assessment Technical Report on the Chidliak Project, Nunavut, Canada” dated July 5, 2016. The requirement for an update was driven primarily by the drilling of an additional 15 holes totaling 5,300 m in the CH-6 pipe, which expanded the CH-6 resource estimate, as reported by Peregrine in “2018 Technical Report: Mineral Resource Update for the Chidliak Project, Baffin Island, Nunavut, Canada”, with an effective date of February 15, 2018.

This report was prepared in order to fulfill the reporting requirements stipulated by Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

It must be noted that this PEA is preliminary in nature and includes the use of Inferred Mineral Resources (100% of mine plan tonnes) for which the geology is considered too speculative to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

2.2 Scope of Work

This report summarizes the work carried out by several consultants and the scope of work for each company is listed below, and combined, makes up the total project scope.

JDS scope of work includes:

- Compile the technical report including historical data and information provided by other consulting companies;
- Establish an economic framework for potentially mineable resources;
- Provide geotechnical recommendations to support mining operations;
- Open pit mine planning and scheduling;
- Select mining equipment;
- Develop a conceptual flowsheet, specifications and selection of process plant equipment;
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities;
- Estimate mining, process plant and infrastructure OPEX and CAPEX;
- Prepare a financial model and conduct an economic evaluation including sensitivity and project risk analysis; and
- Interpret the results and make conclusions and recommendations to improve value and reduce risks.

Peregrine's scope of work includes:

- Data verification of geology, sample collection and sample processing; and
- Estimation of the Mineral Resource.

Knight Piésold's (KP) scope of work includes:

- Scoping and selection of appropriate waste rock storage facilities; and
- Scoping and selection of appropriate mine water management facilities.

2.3 Qualifications and Responsibilities

The results of this PEA are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Peregrine and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practices.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions / associations. The QPs are responsible for the specific report sections are listed in Table 2-1 as follows:

Table 2-1: QP Responsibilities

Qualified Person	Company	Report Section(s)
Richard Goodwin, P.Eng.	JDS	All Sections except those noted below
Dino Pilotto, P.Eng.	JDS	Sections 14.7 and 16, excepting 16.1
Mike Levy, P.E.	JDS	Section 16.1
Ken Embree, P. Eng.	KP	Section 18.9 and 18.10
Catherine Fitzgerald, P.Geo.	Peregrine	Sections 10 and 14, excepting 14.7
Herman Grütter, P.Geo.	Peregrine	Sections 4 through 9, 11 and 12

2.4 Site Visit

In accordance with NI 43-101 guidelines, QPs Doerksen and Pilotto visited the Chidliak Project on April 12, 2016 accompanied by Alan O'Connor of Peregrine, Trevor Herd, JDS Infrastructure Manager and Jeff Stibbard, P.Eng., JDS CEO. The other QPs relied upon the observations of the JDS personnel for their work in the 2016 PEA. Table 2-2 lists site visits for the QPs from the 2016 PEA work.

Table 2-2: QP Site Visits

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Gord Doerksen, P.Eng.	JDS	April 12, 2016	Alan O'Connor	Chidliak Project site, Iqaluit infrastructure
Dino Pilotto, P.Eng.	JDS	April 12, 2016	Alan O'Connor	Chidliak Project site

JDS did not consider that a new site visit was required for this report for the following reasons:

- The April 2016 site visit is considered to be recent and current with regard to project development;
- All four JDS employees that participated in the 2016 site visit are still employees of the company; and
- One of the primary independent QPs for this Report (Dino Pilotto) participated in the 2016 site visit.

2.5 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in the mining industry.

All dollar figures quoted in this report refer to Canadian dollars (C$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.6 Sources of Information

This report is based on information collected by JDS during a site visit performed on April 12, 2016 and on additional information provided by Peregrine throughout the course of JDS's investigations. Other information was obtained from the public domain. JDS has no reason to doubt the reliability of the information provided by Peregrine. This technical report is based on the following sources of information:

- Discussions with Peregrine personnel:
    - Tom Peregoodoff, President and CEO;
    - Dr. Herman Grütter, Vice President, Technical Services;
    - Dr. Jennifer Pell, Chief Geoscientist;
    - Alan O'Connor, Project Manager, Chidliak;
    - Catherine Fitzgerald, Project Geologist - Resource Definition;
    - Greg Shenton, Chief Financial Officer; and
    - David Willis, Manager, Land and Community.

- Review of geological exploration data collected by Peregrine;
- The updated geological resource estimate contained in the report: Mineral Resource Update for the Chidliak Project, Baffin Island, Nunavut, Canada, with an effective date of February 15, 2018);
- Review of bulk sample processing and diamond recovery data compiled by Peregrine;
- Diamond pricing data from Re-Pricing of the Chidliak CH-6 Diamonds & Modelling of the Average Price, April 2018 for and on behalf of Peregrine Diamonds Ltd., by WWW International Diamond Consultants Ltd. and Re-Pricing of the Chidliak CH-7 Diamonds & Modelling of the Average Price, April 2018 for and on behalf of Peregrine Diamonds Ltd., by WWW International Diamond Consultants Ltd.; and
- Additional information from public domain sources.

The documentation received and the sources of information are listed in Section 28.

3 Reliance on Other Experts

The QPs opinions contained herein are based on information provided by Peregrine and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are:

- Wentworth Taylor, CPA for taxation guidance (used for Section 23);
- WWW International Diamond Consultants (WWW) for diamond valuations (used for Section 23); and
- David Willis of Peregrine and Warren Nimchuk of JDS for environment and permitting comments (used for Section 20).

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

4 Property Description and Location

This section is an updated version of information contained in Fitzgerald et al., 2018.

4.1 Location

The Chidliak Project is located on the Hall Peninsula of southern Baffin Island in Nunavut, Canada, approximately 120 km northeast of Iqaluit, Nunavut, centered at 64° 28' 26" N latitude and 66° 21' 43" W longitude (Figure 4-1). The Project comprises 74 kimberlite pipes (71 on CHI claims and three on AN claims) (Figure 4-2). Of these, 45 are known to be diamondiferous and two kimberlites, the CH-6 and CH-7 bodies, are sufficiently understood and sampled to support Mineral Resource estimates.

Figure 4-1: Location of the Chidliak Project

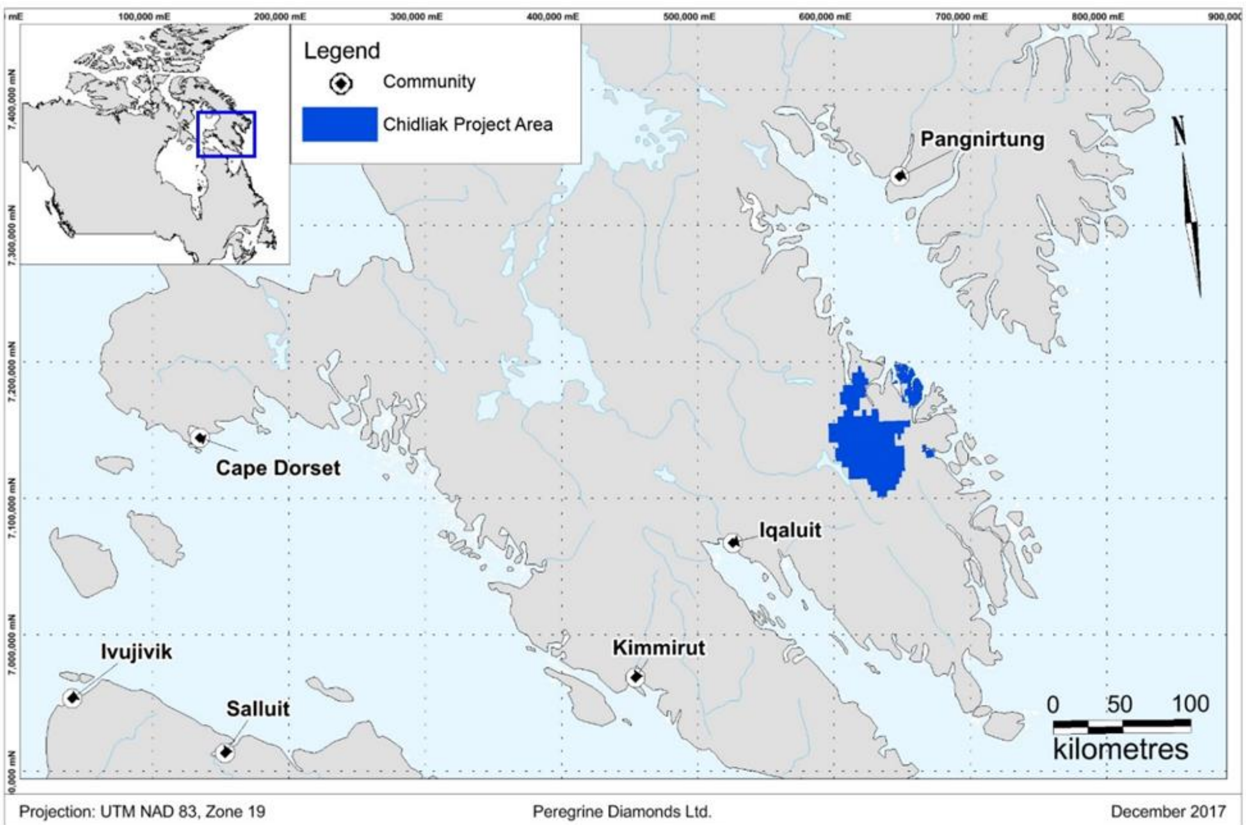
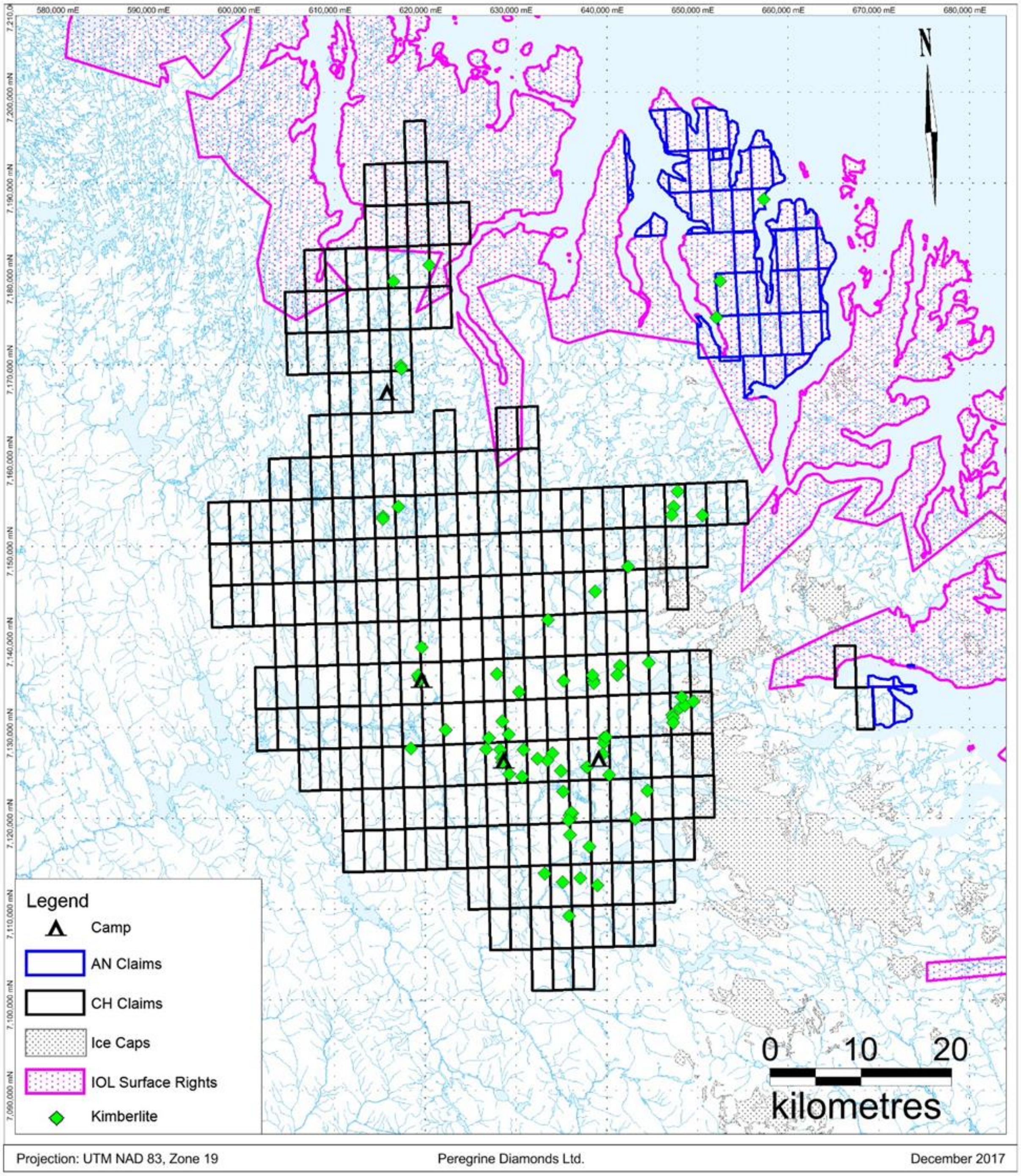


Figure 4-2: Claims Map and Location of Kimberlites on the Chidliak Project



4.2 Mineral Tenure

Nunavut exploration activities are regulated by Indigenous and Northern Affairs Canada (INAC), which is a federal department that ensures compliance with the Nunavut Mining Regulations (NMR) across the territory. Under the Nunavut Land Claim Agreement enacted in 1993, the mineral rights for approximately 2% of the territory have been entrusted to the Inuit and fall into the classification of Inuit-Owned Lands (IOLs). All remaining lands are Crown lands, which are owned by the federal government and fall under their authority and control. There are three main types of mineral interests under the NMR: a mineral claim, a prospecting permit and a mineral lease (also referred to as a mining lease).

The Chidliak Project currently consists of 266 CH claims covering 277,997 ha [or 686,945 acres (ac)] and 53 AN claims covering 37,126 ha (91,740 ac). Approximately 83% of the CH and AN mineral claims are located on Crown lands with the remaining 17% located on IOLs. A complete list of all claims is provided in Appendix 1. The CH and the AN claims are collectively referred to as the Chidliak Project, which comprises a combined total of 319 claims covering 315,123 ha (778,686 ac) (refer to Figure 4-2). Mineral claims are in good standing until at least August 17, 2018, with some claims in good standing until August 16, 2021. All claims are registered to Peregrine Diamonds Ltd.

The Chidliak Project Phase 1 development area is entirely on Crown Land.

4.3 Project Agreements

Peregrine currently has 100% ownership of all claims at Chidliak and is only subject to a Crown Royalty as prescribed in the Nunavut Mining Regulations.

4.4 Environmental Studies, Liabilities and Considerations

Baseline environmental and archaeological studies have been completed at Chidliak since 2009 (see Section 20.2, Table 20-2), with data collected for nine consecutive years. The purpose of the environmental baseline work is to collect enough data to satisfy knowledge gaps and effectively design a baseline program for mine development.

Current environmental liabilities include the four exploration camps and work equipment on-site that Peregrine would be responsible for removing.

4.5 Permit Requirements

4.5.1 Existing Permits

For mineral claims located on Crown lands, Peregrine holds a Class “A” land use permit from INAC that authorizes four field camps and mechanical exploration (drilling, trenching etc.). A land use license from the Qikiqtani Inuit Association (QIA) is required to access mineral claims located on IOLs. Peregrine holds two land use licenses for this privilege, one for the CHI claims and another for the AN claims. Both QIA licenses only permit use of hand tools. Land use permits and water licenses to work on Crown lands and on QIA lands are in place.

4.5.2 Future Permit Requirements

The permitting process for exploration and development of any mineral project is continuous and changes as the Project advances: activities take on a smaller geographic extent as work becomes more intensive.

The permitting framework is set out in both the Nunavut Lands Claim Agreement and the Nunavut Planning and Project Assessment Act. Development projects in Nunavut are subject to a multiphase review process conducted by the Nunavut Impact Review Board (NIRB) that is best summarized by six "benchmark" phases:

- Phase 1: The proponent submits a project proposal describing the mining project to the NIRB;
- Phase 2: The NIRB determines the scope of the Project proposal and identifies significant elements of the proposal that require study and analysis. The NIRB issues project specific guidelines for the production of an environmental impact statement (EIS);
- Phase 3: The proponent collects baseline data according to the NIRB guidelines and assembles this data into a draft environmental impact statement (DEIS);
- Phase 4: The DEIS is submitted to the NIRB at a prehearing conference. The NIRB conducts a technical review of the DEIS and deficiencies and areas of concern are noted and summarized in a pre-hearing conference report (PCR);
- Phase 5: The proponent revises the DEIS based upon the PCR and produces a final environmental impact statement (FEIS);
- Phase 6: The FEIS is submitted to the NIRB and NIRB conducts an administrative and technical review of the FEIS. A hearing is called by the NIRB, which serves as a public forum for the discussion of the proposed project. At the completion of the hearing the NIRB issues a final hearing report with a recommendation to the federal Minister of INAC for the Project to either "proceed" or "not proceed". The Minister has authority to accept or reject the recommendation made by the NIRB; and
- Acceptance to proceed is marked by issuance of a project certificate by the NIRB.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

This section is extracted from Fitzgerald et al., 2018.

5.1 Accessibility

The Chidliak Project is located approximately 120 km northeast of the city of Iqaluit, the capital city of Nunavut (population 7,740 in 2016 Census). There is no permanent road access to the Project and it is accessed primarily by air from Iqaluit via helicopter or fixed-wing aircraft. Work at Chidliak is staged from camps established on the Project, the primary camp being Discovery camp, which has a 570 m long natural gravel landing strip for fixed-wing aircraft and a helicopter pad to support drilling and logistical operations. Access to Iqaluit is by airplane with commercial flights scheduled daily from Ottawa, thrice weekly from Montreal and thrice weekly from Edmonton via Yellowknife.

During winter months when sufficient ice and snow is present, the Project can also sometimes be accessed via the Iqaluit-Chidliak winter trail, which supports the transport of equipment, supplies and samples if required.

5.2 Climate and Physiography

The climate of the area is typical of the Eastern Arctic, being cold in the winter (-25°C to -45°C) and cool to mild in the summer (5°C to 10°C). Precipitation is generally low but snow is possible during all months. Lakes typically have ice until mid-June and freeze up begins in late September. Soil is formed slowly and permafrost extends to at least 540 m depth.

Topography varies from sea level at the coast to 760 masl inland. The topography is rugged near the coast and inland is a rolling upland. The topography at Chidliak in the area where any future mining infrastructure would be located is rolling upland with elevations ranging from 600 masl to 760 masl.

5.3 Flora and Fauna

Flora in the area of Chidliak is characteristic of arctic tundra with discontinuous vegetation, and a short growing season. Sparse vegetation consists primarily of moss, lichen and low shrubs such as purple saxifrage, dryas species, dwarf birch, dwarf willow and various rushes and species of sedges.

Fauna at Chidliak includes mammals such as lemmings, arctic hares, caribou (during migratory season), arctic foxes, wolves, migratory birds such as snow buntings, falcons, ptarmigan, waterfowl, raptors, and other animals such as fish and insects.

5.4 Infrastructure

Peregrine holds authorizations for four field camps at Chidliak: Discovery, Sunrise, Aurora and CH-6 Temporary Camp. The primary camps are Discovery and Sunrise, which are used seasonally, while the Aurora and CH-6 camps are only occasionally used. Discovery camp is the main work staging area and was selected due to the presence of a natural gravel landing strip. Sunrise camp is located on the shore of the 8 km long Sunrise Lake, upon which an ice landing strip can be cleared in winter, as was done in winter 2015 for the CH-7 bulk sample program. All camps are of temporary construction and consist mainly of wooden and/or metal framed tent structures.

Access to the Project is primarily by aircraft. However, during winter months it is sometimes possible to access the field camps by winter trail from Iqaluit. In addition, an inter-camp trail network established during winter facilitates the movement of people, equipment and supplies.

Communications at site comprise a satellite phone system and internet connection.

At present, there is no electrical grid supplying power to the Chidliak Project site. Power on site is currently provided by small (15 kW to 25 kW) diesel generators.

Potable water for current exploration operations is sourced from lakes and intermittent streams proximal to the various camps. Process water for any potential future mining operation may be obtained from one of two lakes located approximately 15 km from two potential processing facility sites. Water would be pumped from one of these lakes through a heat-traced pipe to the processing facility with fresh water augmented by recycled and treated process water.

5.5 Local Resources

Services available in Iqaluit include an international airport, a seasonal shipping port, local bulk fuel, light industry, hotels, groceries, heavy equipment rental, a hospital, hardware supplies and expediting. Additionally, there are fixed-wing aircraft based in Iqaluit available for charter.

Since the inception of substantive exploration activities at the Chidliak Project in 2008, Peregrine has hired local northern employees and has encouraged contractors to hire locally as well. Work on site is seasonal, primarily during the winter and summer months. The majority of local workers originate from the communities of Iqaluit and Pangnirtung, located 190 km to the northwest of the Project. Local hires have an aggregate of 6,219 person days on the Project.

6 History

This section is extracted from Fitzgerald et al., 2018.

6.1 Prior Ownership

There is no record of prior claims in the region of what is now the Chidliak Project, which is currently 100%-owned by Peregrine.

6.2 Exploration History

In 1996 and 1997, an area of Hall Peninsula, including some of the ground now covered by Chidliak, was explored by International Capri Resources Ltd (Larouche, 1997; Lichtblau, 1997). This constitutes the only reported mineral exploration work in the area. They prospected the area for magmatic nickel-copper-platinum group elements (Voisey's Bay and Raglan-type), metamorphosed massive sulphide (SEDEX type) and volcanogenic massive sulphide (VMS) lead-zinc, lead-zinc-copper, and lode gold deposits.

There is no record of exploration for diamonds in the Chidliak Project area, prior to 2005 when BHPB and Peregrine began working in this region.

7 Geological Setting and Mineralization

This section is an abridged version of information contained in Fitzgerald et al., 2018.

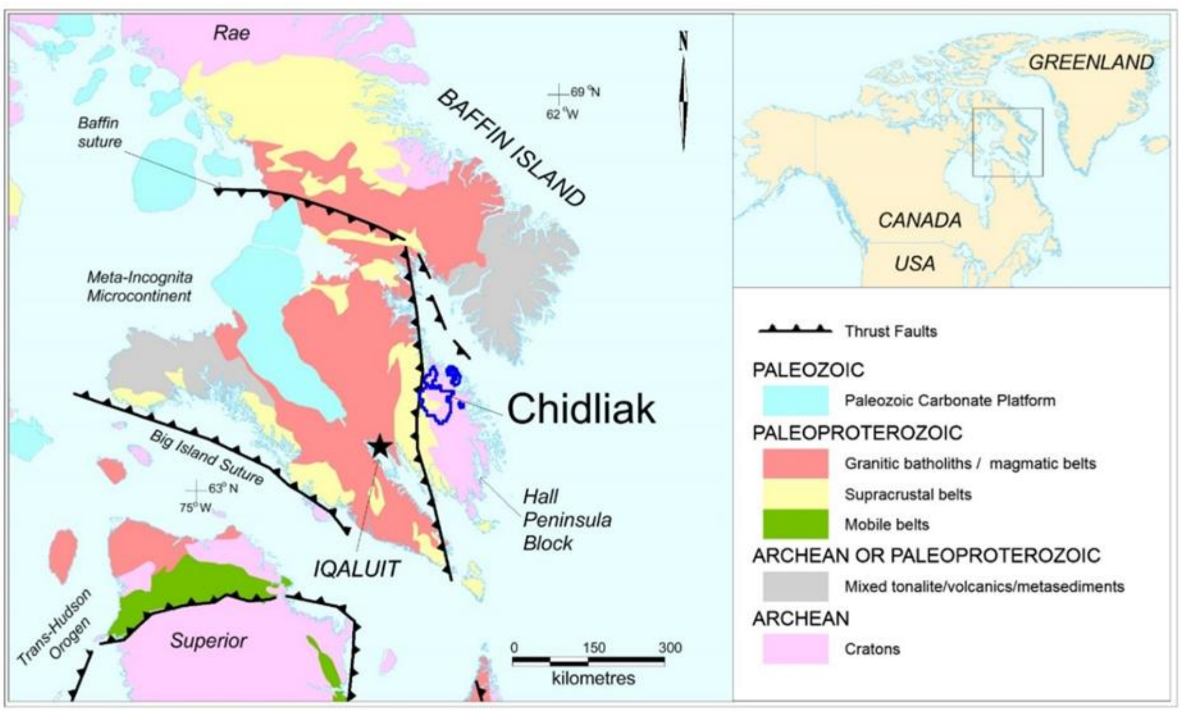
7.1 Regional Geology

7.1.1 Bedrock Geology

The Hall Peninsula is divided into three major crustal entities. In the west ~1.865 Ga to 1.845 Ga granitoids of the Cumberland Batholith (Whalen et al., 2010) crop out. The central belt is comprised of granulite-facies continental margin shelf metasediments with maximum depositional ages of between 2.09 and 1.84 Ga (Rayner, 2015) that is correlated with the Lake Harbour Group on the Meta Incognita Peninsula (St-Onge et al., 2006). In the east, there is an Archean gneissic terrain (Scott, 1996, 1999; St-Onge et al., 2006) termed the Hall Peninsula block (Whalen et al., 2010) which comprises ~2.92 Ga to 2.69 Ga orthogneisses and 1.96 Ga to 2.71 Ga supracrustal rocks (From et al., 2013; 2014; 2016; 2017; Machado et al., 2013; Rayner, 2014; 2015; Scott, 1999; Steenkamp and St-Onge, 2014).

Rocks of the Hall Peninsula block (Figure 7-1) host all of the Chidliak kimberlites.

Figure 7-1: Simplified Geological Map of Southern Baffin Island. Quaternary Geology

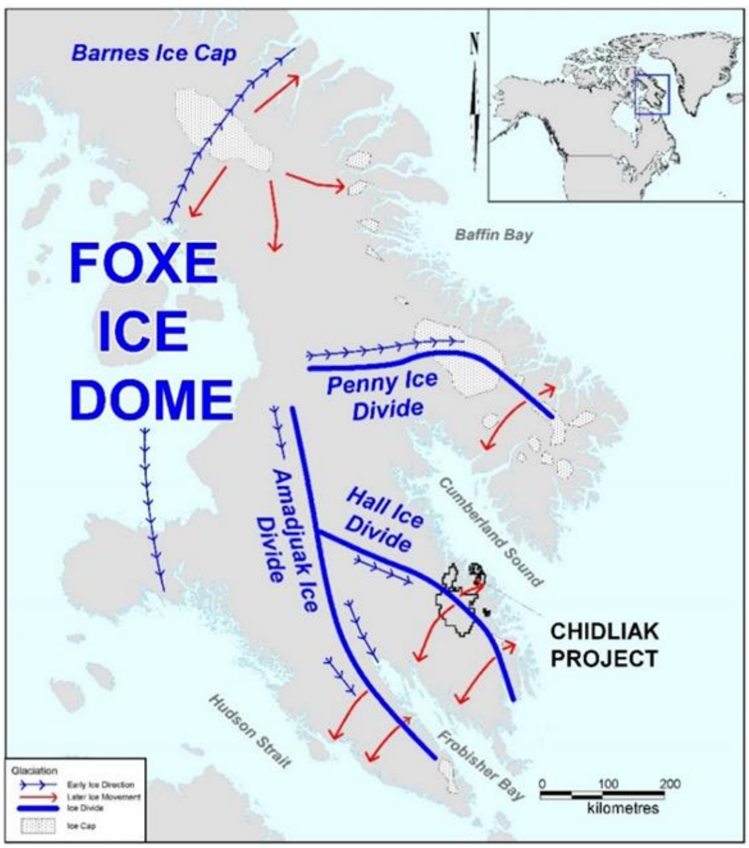


Source: Modified from Pell et al., 2013 after St.Onge et al., 2006 and Whalen et al., 2010.

7.1.2 Quaternary Geology

The majority of the Canadian Arctic was ice covered during much of the Quaternary period by the Laurentide Ice Sheet, with the last glacial maximum (LGM) occurring from approximately 18,000 to 8,000 years ago (Dyke, 2004; Dyke et al., 2002; Dyke and Prest, 1987). In the Baffin area, the manifestation of the Laurentide Ice sheet during the last glaciation was the Foxe Dome, a continental-type ice sheet that was centered over the Foxe Basin (Kaplan et al., 1999; Marsella et al., 2000). The present-day equivalent of the Foxe Dome is the Barnes Ice Cap in north-central Baffin Island, which most likely contains Pleistocene age ice (Andrews, 1989). The dominant ice directions for the Foxe Glaciation (Figure 7-2) radiate out from the Foxe Basin to Baffin Bay (north-north-easterly in central Baffin), to Cumberland Sound (east-south-easterly) and to Frobisher Bay and Hudson Strait (south-easterly to south-south-easterly) in south-eastern and southern Baffin (Kaplan et al., 1999; Marsella et al., 2000). During the waning of Laurentide glaciation, these major directions were, to some extent, modified or overprinted by ice radiating from the smaller Penny, Hall and Amadjuak domes centered over Cumberland Peninsula, Hall Peninsula and southwest Baffin, respectively and by ice radiating out from the remnant Barnes Ice Cap in central Baffin Island.

Figure 7-2: Dominant Ice Flow Directions for the Foxe Glaciation



7.2 Local Geology of Project Area

7.2.1 Bedrock Geology

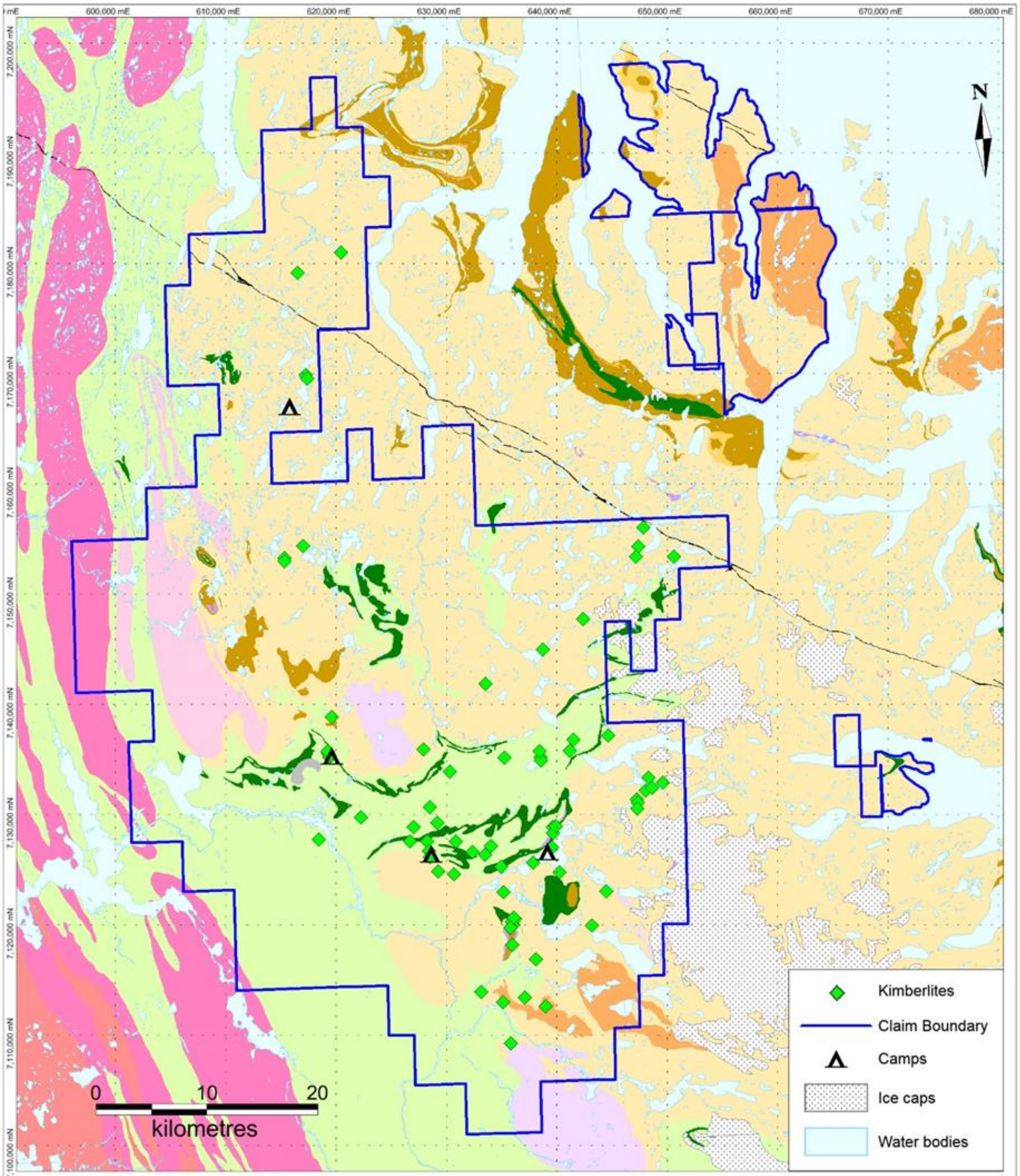
Archean orthogneissic basement and Archean to Paleoproterozoic supracrustal metasedimentary cover rocks of the Hall Peninsula block (Figure 7-3) underlie the majority of the Chidliak Project area and host all kimberlites at Chidliak. The Archean basement orthogneiss complex comprises gneissic to migmatitic tonalite to monzogranite, with local enclaves and pods of amphibolites and crosscutting granite to syenogranite dikes (Machado et al., 2013; From et al., 2014). Several hydrated ultramafic intrusions crosscut the basement orthogneiss and are locally wrapped by the pervasive gneissic foliation (Steenkamp and St-Onge, 2014). The Archean orthogneiss complex is locally disconformably overlain by a variably metamorphosed supracrustal sequence (Mackay et al., 2013; MacKay and Ansdell, 2014; Steenkamp and St-Onge, 2014). Crystallization ages between 2843 Ma and 2687 Ma have been obtained from basement samples in the immediate Chidliak area (From et al., 2017; Rayner, 2014; Ansdell et al., 2015).

On the extreme western margin of the Project area, pelitic to psammitic granulite-facies metasedimentary strata intercalated with garnet and biotite-bearing leucogranite sills and dykes and interleaved with orthopyroxene-bearing diorite to monzogranite crop out (Figure 7-3). Several larger, laterally continuous, tonalitic to quartz dioritic intrusions also cut into the psammitic to pelitic supracrustal strata (Steenkamp and St-Onge, 2014). Rayner (2014) interprets the crystallization age of a compositionally equivalent sample taken from a laterally contiguous panel in the southern field area at ca. 1890 Ma that is consistent with ages from the Cumberland Batholith.

7.2.2 Quaternary Geology

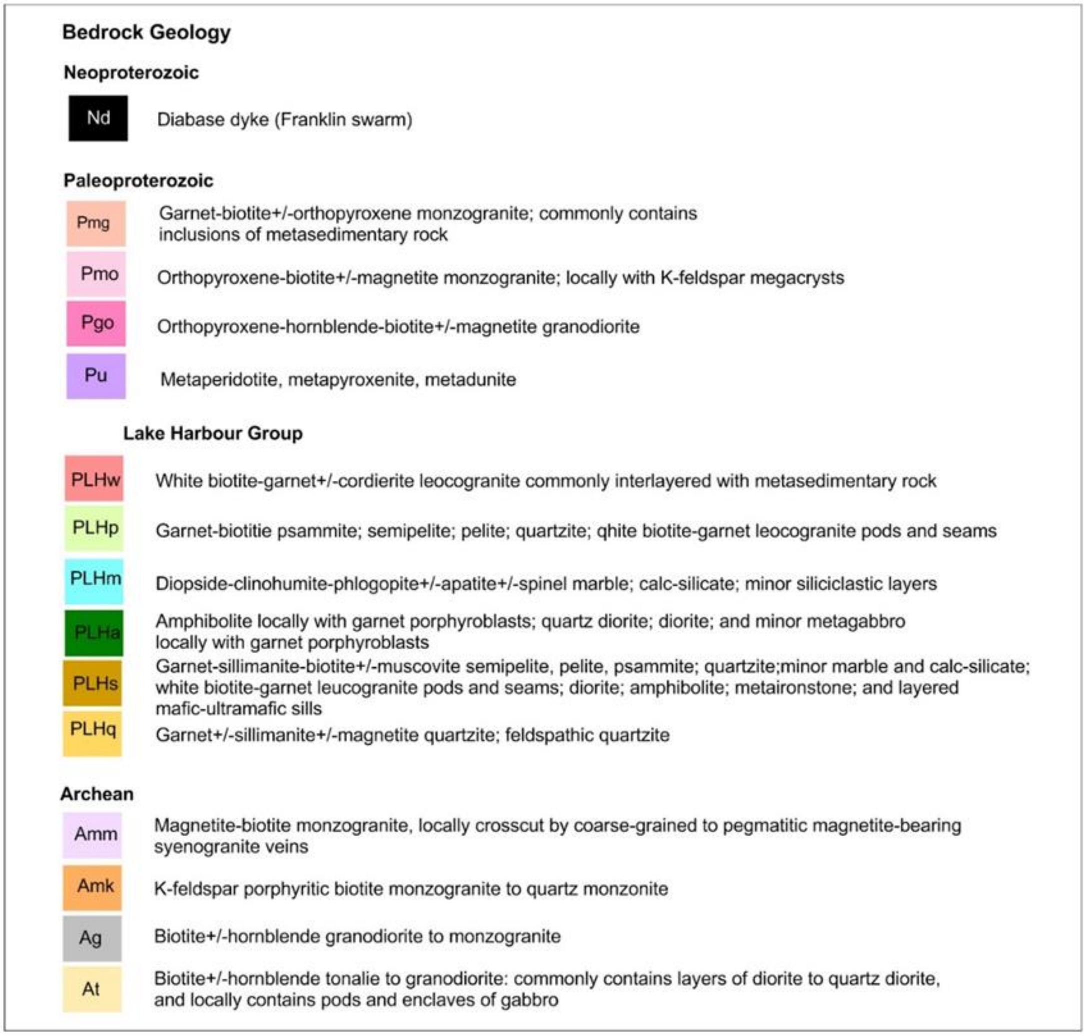
The Chidliak property is on the north side of Hall Peninsula, much of which comprises upland surfaces (Baffin Surface) and stepped plain or dissected upland surfaces (Andrews, 1989). The area was inundated by the Laurentide Ice Sheet during the LGM approximately 18,000 to 8,000 years ago (Dyke, 2004; Dyke et al., 2002), and remnants of this ice sheet persist at Chidliak to the present day, at approximately 700 m above sea level (masl). Glacial till is found throughout the Chidliak area and is generally present as a variable veneer typically 0 m to 3 m thick and locally up to 15 m thick as proven by drilling.

Figure 7-3: Local Geology of the Chidliak Area



Source: Modified from Steenkamp et al., 2016a, 2016b, 2016c, 2016d.

Figure 7-3 (continued): Local Geology of the Chidliak Area, Legend



7.3 Chidliak Kimberlites

7.3.1 General Geology

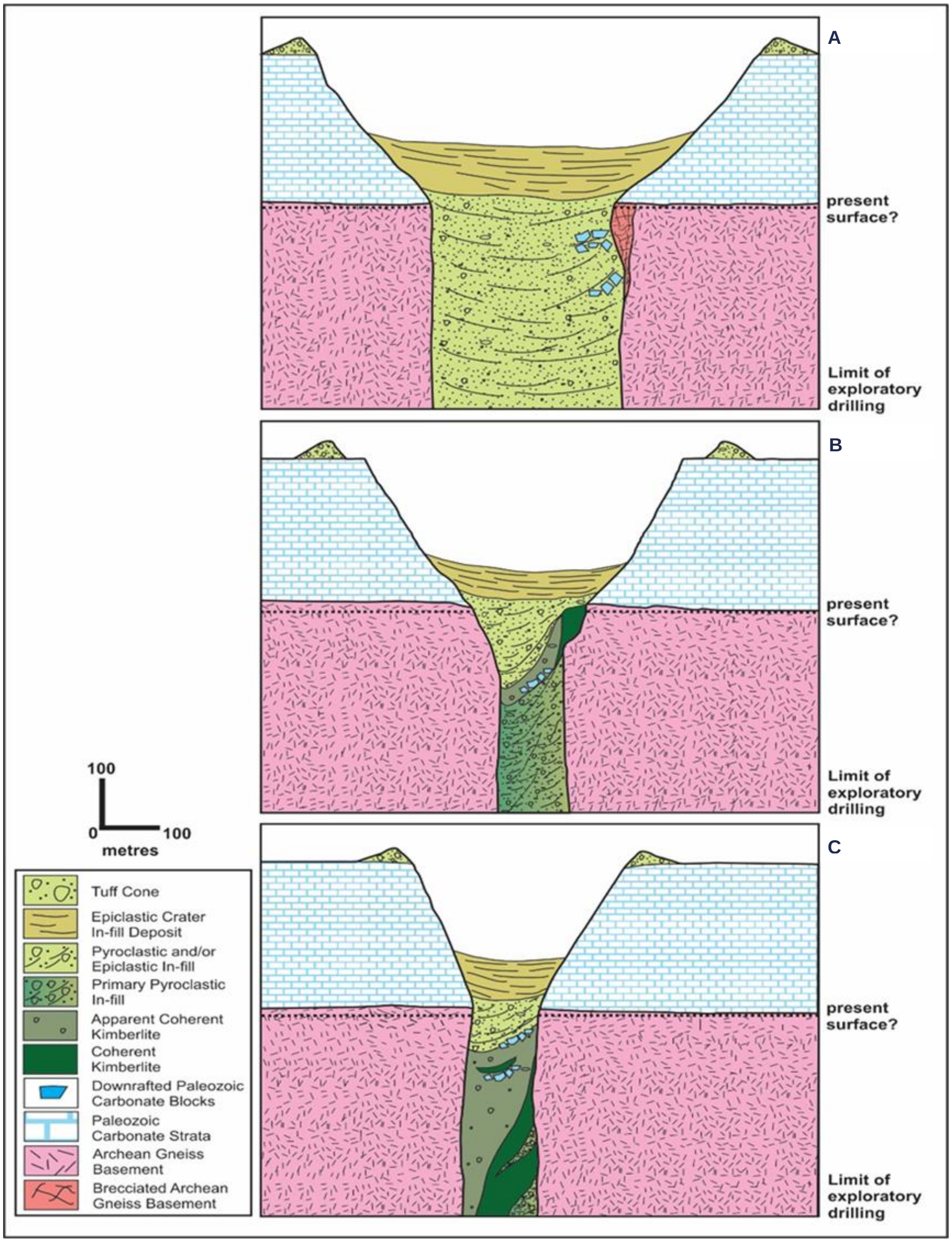
The kimberlites at Chidliak were emplaced during the Jurassic period, between 157 and 139.1 Ma (Heaman et al., 2015). Both steeply dipping sheet-like and larger pipe-like bodies have been discovered at Chidliak (Pell et al., 2013). The sheet-like bodies are mainly coherent, hypabyssal kimberlite (HK) dykes, which may contain basement xenoliths. Most of the pipe-like bodies contain, in addition to basement xenoliths, Late Ordovician to Early Silurian carbonate and clastic rock xenoliths derived from the paleosurface and incorporated into an open vent structure (Zhang and Pell, 2014). The occurrence of these Paleozoic carbonate xenoliths in the Chidliak pipes proves that this part of Hall Peninsula was overlain by Lower Paleozoic sedimentary rocks at the time of kimberlite eruption. The sedimentary succession is estimated to have been 270 to 305 m in thickness and was removed by erosion between the Early Cretaceous and the present (Zhang and Pell, 2013; 2014).

The Chidliak kimberlite pipes have a range of textural types of infill and, broadly, can be assigned to two main types: pipes containing only volcaniclastic kimberlite (VK) infill and pipes infilled by a combination of VK, coherent kimberlite (CK), and welded or agglutinated kimberlite deposits referred to as apparent coherent kimberlite (ACK) (Pell et al., 2013).

The VK-only pipes tend to be larger (125 to 150 m radius) than the combined-infill pipes and are dominated by pyroclastic kimberlite (PK) with lesser resedimented volcaniclastic kimberlite (RVK) (Figure 7-4a). VK-only pipes may have subtle internal variability with respect to olivine content, packing and grain size and commonly contain easily recognized melt-bearing pyroclasts (Scott Smith et al., 2013). Paleozoic carbonate, gneissic basement (also referred to as country rock) and mantle xenoliths, combined, typically comprise up to 5% by volume of the pipe, with local, inhomogeneously distributed zones comprising up to 15% by volume xenoliths. Typically, carbonates are more abundant than gneissic basement, which are, in turn, more abundant than mantle xenoliths. These pipes contain within-vent, PK and RVK deposited during the waning phases of eruption when it is possible for material to accumulate in the conduit from highly explosive gas-rich eruptions.

The combined-infill pipes are commonly 50 to 75 m in radius and can range from VK-dominated, with lesser CK and ACK (Figure 7-4b), to dominantly infilled by ACK, with minor amounts of CK and VK (Figure 7-4c). The VK deposits in the combined-infill pipes are similar to those in the VK-only pipes. The ACK deposits are dark, competent and massive and show some features of CKs (Scott Smith et al., 2013; e.g., lava, dykes or sills) such as a finely crystalline groundmass. However, they lack sharp intrusive contacts and contain well-dispersed Paleozoic carbonate xenoliths. They also exhibit other textural features, including olivine grain size variation, close packing of olivine and other components, occasional broken garnet and olivine grains and diffuse magmaclasts (Scott Smith et al., 2013) suggesting they are products of explosive volcanism (e.g., clastogenic pipe infill) rather than effusive volcanism (e.g., lava) or intrusion (e.g., hypabyssal kimberlite) (Pell et al., 2012. 2013). The VK and ACK deposits in these combined-infill pipes have a lower carbonate and gneissic xenolith content (typically <5% by volume) than the VK-only pipes. The CK rocks typically contain only gneissic basement xenoliths and lack Paleozoic carbonate xenoliths. They represent an extreme, even hotter end-member of the volcanic processes described above and either also have a pyroclastic origin or have formed from unfragmented lavas.

Figure 7-4: Schematic Models of the Chidliak Kimberlite Types



Source: Pell et al. (2015)

7.3.2 CH-6 Geology

The CH-6 kimberlite pipe is a steep-sided, slightly south-west plunging, kidney shaped to elliptical body with a surface area of approximately 1.0 ha. It was emplaced into basement paragneisses and now-eroded Paleozoic carbonate rocks. The body does not outcrop and is overlain by approximately 3 m of overburden, deepening to 25 m in the southeast. It was discovered in 2009 by core drilling the southwestern edge of a magnetically reversed geophysical anomaly.

The pipe infill comprises two volumetrically significant kimberlite units: KIM-L and KIM-C. Generally, the KIM-L and KIM-C units can be distinguished megascopically by the respective presence or paucity of carbonate xenoliths. Sharp contacts between KIM-L and KIM-C have not been observed.

The KIM-L unit is the volumetrically dominant pipe infill, comprising 89% by volume of the pipe and occurring between the base of the overburden at depth of from 3 mbs to 25 mbs and the base of drilling at 540 mbs. The upper portion of KIM-L (to approximately 40 mbs) is weathered and is referred to as wKIM-L in drill core logs. KIM-L is a dark grey to greenish black, competent, texturally variable rock with conspicuous and diagnostic Paleozoic carbonate xenoliths. These xenoliths are generally 2 to 3 cm in size, rarely up to 15 cm, with very rare blocks up to 13 m. Dilution by carbonate and country rock xenoliths is low (< 5% by volume) but locally may exceed 10% by volume. Melt-bearing pyroclasts are present in variable amounts, as are broken melt-free olivine and broken, primarily fresh garnet macrocrysts. With depth, the texture of KIM-L changes from PK to ACK, where it either lacks or contains diffuse melt-bearing pyroclasts. KIM-L is interpreted to have been emplaced by explosive volcanic processes that varied from high to low energy that resulted in pipe infill ranging from pyroclastic to apparently coherent clastogenic deposits, respectively.

The KIM-C unit occurs along the north and northeast margin of the pipe below 80 mbs (600 masl) to a drilled depth of 315 mbs (365 masl) and occupies 11% by volume of the pipe. It is a dark grey to black to greenish black, massive, homogeneous CK, distinguished macroscopically by having few or no Paleozoic carbonate xenoliths and a low gneissic basement xenolith content. No intrusive contacts are observed between KIM-L and KIM-C, and KIM-C is interpreted to be extrusive. The whole rock geochemistry and groundmass chrome spinel signature of KIM-C is identical to that of KIM-L. However, microdiamond sampling of KIM C completed in 2014 was sufficient to establish a clear difference in diamond content compared to KIM-L.

A variety of other minor units occur within CH-6, such as thin intercalated CK intervals of uncertain origin, blocks of carbonate, and rare gneissic blocks. Table 7-1 summarizes geological units encountered in drill core in CH-6.

For more detailed geology of the CH-6 kimberlite, please refer to Nowicki et al. (2016).

Table 7-1: Summary of Attributes and Occurrences of Geological Units in Drill Core at CH-6

Unit	Texture	# Core Holes Rock Type Occurs In	Total Drilled* (m)	Description
KIM-L	PK to ACK	39	5,928.42	Variably textured, locally crudely layered, Paleozoic carbonate xenolith-bearing kimberlite; local diffuse to well-developed melt-bearing pyroclasts & broken, melt-free olivine. Upper 40 metres is weathered and clay altered.
KIM-C	ACK	10	545.73	Massive, homogeneous with few to no Paleozoic carbonate xenoliths, HK-like olivine distribution, well-crystallized groundmass, garnets commonly kelyphitized.
OTHER	CK, ACK	23	388.51	Small intervals of CK that are hypabyssal in origin (KIM-HK), and intervals that require further work to determine their origins.
LSTX (+/- B)	n/a	16	104.19	Competent Paleozoic carbonate block, with or without brecciated texture, occurring as internal blocks to KIM-L.
CR	n/a	37	3,400.76	Competent, fresh to little altered gneiss. Occurs marginal to pipe and as rare internal blocks. Variably broken and weathered near surface.
BCR (+/-K)	n/a	33	712.68	Broken and variably altered gneiss, with or without thin coherent kimberlite veins.
CRB (+/-K)	n/a	11	36.16	Brecciated and variably altered gneiss, with or without coherent kimberlite veins present, occurring marginal to pipe.
OVB (+/- K)/NR	n/a	46	585.47	Overburden, with or without minor amounts of weathered kimberlite mixed in. Primarily OVB was not recovered.

*Intercepts in core. RC holes excluded
NR denotes No Recovery

7.3.3 CH-6 Geological Domains and Three Dimensional Model

The CH-6 geological units identified in drill core define two geological domains for the purposes of 3-D geology modelling and Mineral Resource estimation purposes, with each domain dominated by the geological unit of the same name: KIM-L and KIM-C. The term “geological domain” is applied to modelled portions of kimberlite pipe that are distinct in terms of their geological characteristics and meaningful from a Mineral Resource estimation point of view.

The 3-D geology model of CH-6 has evolved in multiple stages based on the availability of drilling and sampling data over time. Drill core intervals were logged in detail in order to define the kimberlite pipe margins, to describe kimberlite units, and to define internal geological boundaries. Once logged, the kimberlite units were sampled for diamonds, density, whole rock chemistry and petrography, all information that was integrated to interpret the CH-6 pipe shape and internal geology, in order to create a geological model.

Based on completion of 15 core holes in 2017, the CH-6 geological model has been updated from that reported in 2016 (Nowicki et al., 2016) using Dassault Systèmes GEOVIA GEMS™ software version 6.8 (Figure 7-4).

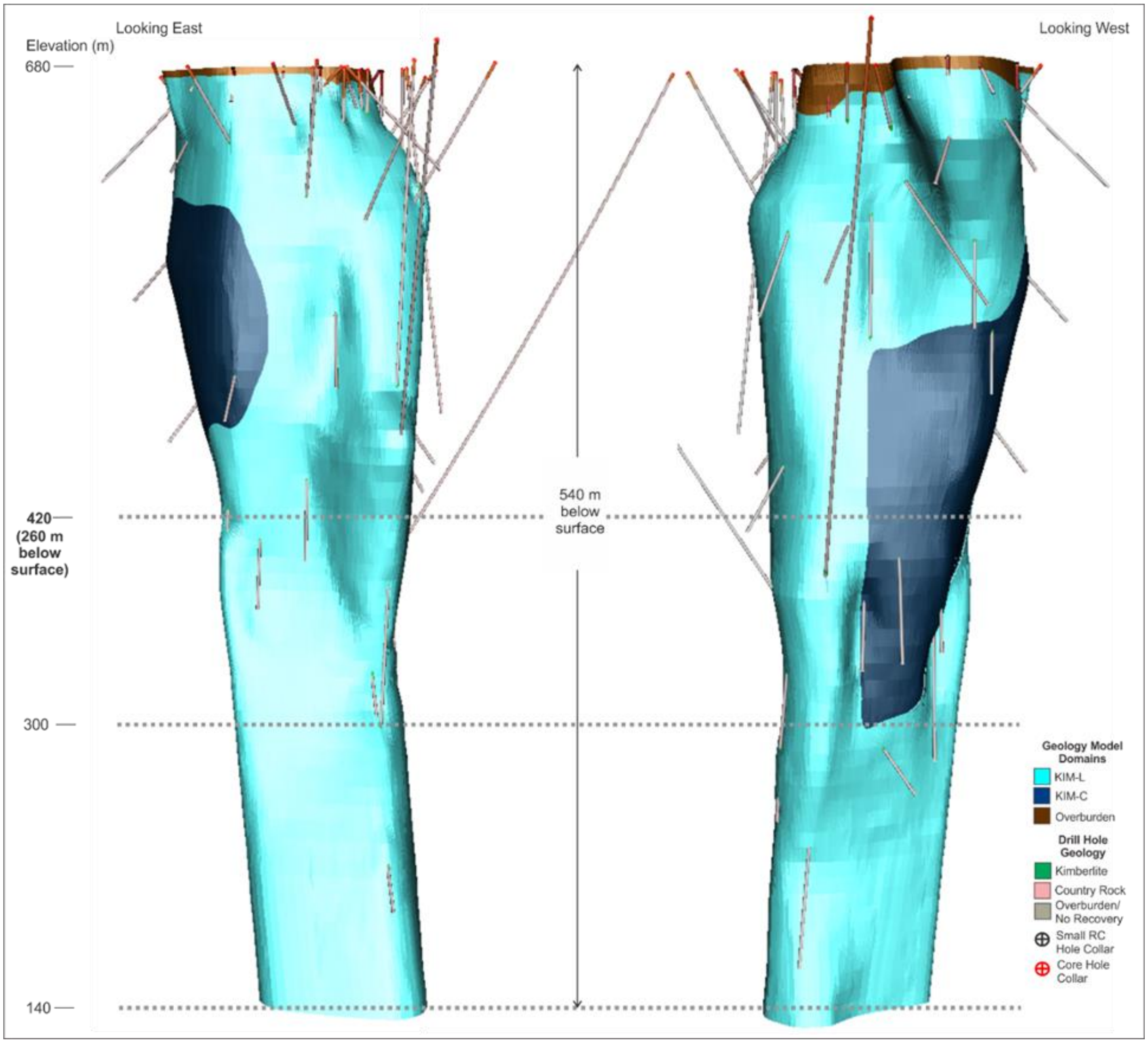
The external pipe shape was modelled utilizing all available country rock / kimberlite contacts recognized in core or small-diameter RC drill holes. Pipe contacts are typically sharp at CH-6 and are defined with ease when logging core. Weathered contacts are sometimes difficult to discern and can be poorly represented in clay-rich material or chips recovered from small-diameter RC drill holes. Possibly ambiguous contact relationships were resolved by trace element geochemistry in these instances.

The topographic surface was modelled from airborne geophysical survey digital elevation data. The overburden basal surface was created using small-diameter RC drill hole intersections of the base of overburden, combined with base of overburden intersects in diamond drill holes where overburden was in contact with country rock. Drill holes where overburden was in contact with kimberlite were not used, as in all cases the weathered kimberlite in these holes was washed away during diamond core drilling and did not define the true base of overburden.

Geology domains were defined based on the distinct geological characteristics and spatial distribution in drill core of their constituent major geological units. Domains were modelled using polylines defining shapes every 10 m of elevation, defined using intersects between geological units in core drill holes, combined with guidelines between contacts at different elevations. The guidelines provide shape control in vertical sections and in 3-D view.

The updated CH-6 geological model has an ellipsoidal shape in plan view with steep-sided walls that dip slightly to the south-west at approximately 77° (Figure 7-5). The surface dimensions are 120 m by 100 m with an approximate area of 1.0 ha. The top of the pipe is covered by 3 m of glacial overburden that thickens to 25 m at the southeast of CH-6. The pipe expands with depth in north-south dimension down to approximately 280 mbs, below which it begins to contract, such that at 300 mbs it measures 70 m by 110 m and occupies an area of 0.5 ha. In areas where the external pipe shape is not well-constrained by drilling, the morphology was interpreted using projections of angles between drill hole contacts from higher and lower elevations, combined with knowledge of kimberlite pipe emplacement models and typical shapes for kimberlites of mixed VK and ACK, and the types of shapes of pipes observed at Chidliak.

Figure 7-5: 3-D Geology Domain Model of CH-6



7.3.4 CH-7 Geology

The CH-7 kimberlite is a steep-sided, southwest-plunging body comprised of at least two coalescing lobes with a combined surface area of approximately 1.0 ha. It was emplaced into basement paragneisses and now-eroded Paleozoic carbonate rocks. The north-eastern part of the body outcrops and elsewhere CH-7 is overlain by an average of 3 m of overburden. CH-7 was discovered in 2009 as outcrop/subcrop during prospecting.

Five main kimberlite units, KIM-1 to KIM-5, with distinct physical, and in some cases geochemical, characteristics, have been identified at CH-7. The KIM-2 unit predominates, and occupies 61% by volume of the CH-7 pipe model. The units were named in the order that they were identified, and the numbering does not have any implications as to the genesis or the order of emplacement of the units (summarized in Table 7-2). In addition to the five main kimberlite units, a variety of other minor units are present, such as a gneiss xenolith-bearing CK and internal blocks of both fresh and brecciated gneiss and carbonate rocks. The upper approximately 50 to 60 m of the kimberlite pipe is weathered, friable and clay-altered.

KIM-1 is the only unit to outcrop at CH-7 and is restricted to the north / north-eastern region of the pipe, extending from surface to approximately 125 mbs in core. It occurs adjacent to KIM-5 to the west and KIM-2 to the south, and is perched above KIM-4 (Figure 7-6). It is a dark green, massive, competent, coarse-grained, olivine-rich macrocrystic CK with rare (<1%), conspicuous gneissic xenoliths that are typically highly altered and rounded. Carbonate xenoliths are very rare to absent in KIM-1, unlike in all other major units at CH-7. No melt-bearing pyroclasts have been observed. A pyroclastic variant of KIM-1 occurs along the southern margin of KIM-1. It has a restricted depth range of between 40 and 175 mbs and occurs in six of 29 core holes.

KIM-2 is the volumetrically dominant geological unit at CH-7 and occupies the central and southern part of the pipe, extending from beneath the overburden at approximately 3 mbs to the limit of drilling at 263 mbs. It is a fine-medium or medium-grained, moderately olivine-rich PK. Near surface (above 60 mbs) it is highly weathered, light green, light to dark grey/olive-grey to pale-olive in colour, texturally variable and friable in nature. Below 60 mbs, KIM-2 is comparatively fresh, greenish black to medium grey to medium-dark grey in colour, massive and often has a waxy appearance. It contains variable amounts of both gneiss and carbonate xenoliths (typically <5% by volume of each), variably altered olivine macrocrysts (generally >25%) and is characterized by common to abundant, grey and ovoid melt-bearing pyroclasts hosted in a serpentine-rich interclast matrix.

KIM-3 is an apparently rootless kimberlite unit found in the central part of the CH-7 pipe adjacent to KIM-2 and beneath varied units near surface, and above KIM-4, between 70 and 200 mbs. It is a bluish-grey to bluish-green-grey, massive to locally bedded, olivine-rich, hard, competent PK containing variable amounts of inhomogeneously distributed gneissic and carbonate xenoliths with carbonates commonly more abundant and larger than gneisses. KIM-3 is texturally variable with respect to olivine content, packing and grain size. Olivine is mostly fresh and commonly broken, and ash layers and recognizable bedding are present locally. KIM-3 is characterized by melt-bearing pyroclasts that are sub-irregular to curvilinear, predominantly uncored and variably amygdaloidal (filled by serpentine/ carbonate), and set within a microcrystalline carbonate ± serpentine interclast matrix. Gneiss blocks up to 4.9 m in core length, carbonate xenoliths up to 1.5 m in core length and intervals of CK are present locally within KIM-3, most notably in the southwestern marginal zone.

KIM-4 is found in the central and northern part of the pipe mostly at depths greater than 145 m, occurring below KIM-3 and to the north of KIM-2. It is a greenish black to dark greenish grey, massive to locally bedded, clast supported, loosely to closely packed, fine- to coarse-grained, olivine-rich, hard PK that contains variable amounts (overall <5%) of inhomogeneously distributed gneissic and carbonate rock xenoliths. Olivine is partially serpentinized and poorly to moderately sorted, melt-bearing pyroclasts are uncommon, and components are hosted in a serpentine-rich interclast matrix. KIM-4 has a relatively high mantle content, which combined with the degree of sorting in the rock and very low proportion of melt, is diagnostic of KIM-4.

KIM-5 occurs close to surface (from ~ 3 mbs to 100 mbs) and dominates the north-western lobe of the pipe, occurring west of KIM-1 and north of KIM-2. It is a light to dark olive-grey, medium grey, medium-bluish-grey to dark greenish grey, massive to thickly bedded unit that ranges from PK to ACK in texture. It also varies from being extremely fresh and competent, even near surface, to completely altered (lateritized) to red mud. The unit has a low gneissic xenolith content (generally <1%) and carbonate rock xenoliths are inhomogeneously distributed and vary from being present in trace amounts to locally comprising nearly 15% of the rock. It is variable with respect to olivine content, degree of sorting (generally very poor), packing (often clast supported) and grain size, and locally can have a grainy appearance with an interclast matrix of serpentine and spinel. Melt-bearing pyroclasts are variably abundant, whereas mantle xenoliths and indicator minerals are notably so.

For more detailed geological descriptions of each unit within CH-7 refer Nowicki et al., 2016.

Table 7-2: Summary of Attributes and Occurrences of Geological Units at CH-7

Unit	Texture	# Core Holes Unit Occurs In	Total Drilled* (m)	Description
KIM-1	CK	14	334.46	Coarse-grained, fresh olivine and a well-crystallized groundmass of monticellite, spinel, phlogopite and perovskite; CRX in low abundance and heavily altered. No LSTX. Locally can be heavily altered (lateritized) and is it denoted KIM-1_RM.
KIM-1A	PK, ACK	6	98.75	Competent to fissile with melt-bearing pyroclasts that have the same groundmass as KIM-1. Contains LSTX & more olivine and KIM's than KIM-1; olivine is commonly medium to coarse-grained and shardy. Interpreted as a PK version of KIM-1.
KIM-2	PK	18	1,588.04	Predominantly serpentinized olivine and abundant, distinct, ovoid melt-bearing pyroclasts easily recognized in core. Serpentine-rich interclast matrix; most serpentine-rich unit at CH-7. Upper ~60 metres are weathered and clay-rich, denoted as wKIM-2.
KIM-3	PK	11	357.15	Bluish-grey to bluish-green-grey, locally bedded, dominantly fresh olivine and common distinct, sub-irregular to curvilinear shape, variably amygdaloidal MPBs and a carbonate-rich interclast matrix.
KIM-4	PK	3	188.72	Locally bedded, moderately sorted. Variably serpentinized olivine, paucity of fine grains; melt-bearing pyroclasts not common. Inter-clast matrix is serpentine ± carbonate. Relatively high mantle content.
KIM-5	PK/ACK	7	289.55	Massive to thickly bedded, extremely texturally variable, poorly sorted; variably fresh, olivine macrocryst-rich to lateritic red-mud (denoted KIM-5_RM). Distinctive cored and uncored melt-bearing pyroclasts. Some horizons contain notably more LSTX on average than other units in CH-7. Relatively high mantle content.
OTHER	CK, ACK, PK	27	369.53	Includes small intervals of a CK unit (KIM-6) and various CK, ACK and PK units.
LSTX (+/-B)	n/a	3	6.65	Competent Paleozoic carbonate block, with or without brecciated texture, occurring as internal blocks.
CR (+/-K)	n/a	24	1,118.30	Competent, fresh to little altered gneiss. Occurs marginal to pipe and as rare internal blocks. Very rare intervals contain thin CK veins, which are denoted as CR+K.
BCR (+/-K)	n/a	15	309.62	Broken and variably altered gneiss, with or without thin CK veins.
CRB (+/-K)	n/a	8	60.33	Brecciated and variably altered gneiss, with or without CK veins present, occurring marginal to pipe.
OVB (+/-K)/NR	n/a	28	239.12	Overburden, with or without minor amounts of weathered kimberlite mixed in. Primarily not recovered.

*Intercepts in core. RC holes excluded

NR denotes No Recovery

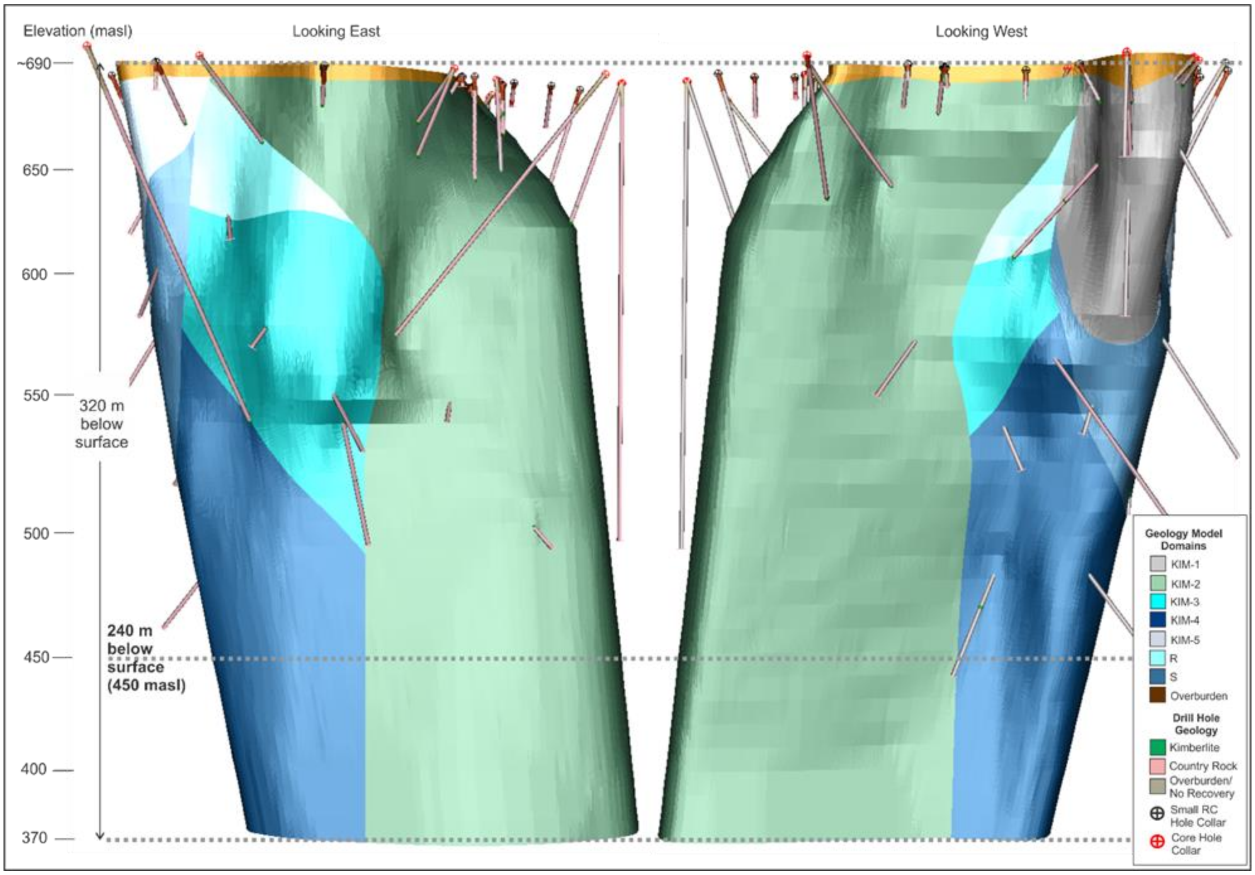
7.3.5 CH-7 Geological Domains and 3-D Model

The CH-7 geological units have been combined into seven geological domains for 3-D modelling, with five of the domains dominated by the geological unit of the same name: KIM-1 through to KIM-5. The remaining two domains, R and S, are not dominated by one particular geological unit but rather comprise several units in roughly equal proportions. The complexity in geology and/or insufficient drilling and geological information precludes subdivision of these portions of the CH-7 pipe at this stage. Domain R represents a zone of geological complexity in the pipe at the junction of the other domains (further complicated by a high degree of weathering) and is distinguished by different LDD sample grade characteristics to those of the KIM-2 and KIM-3 domains (Section 14.5.3). Domain S is a zone of poor to moderate drilling coverage comprised mainly of KIM-1 and possible KIM-5 (low confidence KIM 5).

The KIM-2 domain occupies 61% of the modelled CH-7 pipe volume, primarily in the central and southern portion of the pipe from approximately 3 mbs to the base of the model at 320 mbs (370 masl). The Northern sector of the pipe holds the remaining 39% by volume of CH-7, consisting of domains KIM-1 (3%), KIM-3 (11%), KIM-4 (14%), KIM-5 (3%), R (4%) and S (4%). Figure 7-6 shows the geological model of CH-7. It is unchanged from that reported in 2016 (Nowicki et al., 2016).

The CH-7 pipe model has an elongate and lobate shape in plan view with steep-sided walls that dip to the south-west at approximately 80°. The surface dimensions are approximately 140 m by 80 m with an area of 1.0 ha. The top of the pipe is covered by variably thick (up to 10 m locally, but an average of 3 m) glacial overburden. The pipe is elongate in a north-south direction and contracts in width with depth, such that at 240 mbs it measures approximately 165 m by 45 m and occupies an area of 0.8 ha. In areas where the external pipe shape is not well constrained by drilling, the morphology was interpreted using projections of angles between drill hole contacts from both higher and lower elevations, combined with knowledge of kimberlite pipe emplacement models and typical shapes for kimberlites dominated by mixed VK-ACK infill, and the types of shapes of pipes observed at Chidliak. Best-fit, minimum and maximum pipe shapes were created, primarily to model a range of pipe volumes to depth. The range of shapes were interpreted using the same principles as described above, but utilized different combinations of drill hole contacts that showed varying projections to depth. In areas of extensive drill coverage (i.e. generally above 110 mbs), the variance between the three pipe shapes is very limited.

Figure 7-6: 3-D Geology Domain Model of CH-7



8 Deposit Types

This section is extracted from Fitzgerald et al., 2018.

Kimberlites and lamproites are volcanic and subvolcanic varieties of ultramafic rocks and are the main hosts for terrestrial diamonds. The vast majority of global primary diamond mines are hosted in kimberlite, and this rock type is the target at the Chidliak Project. Kimberlites are mantle-derived, volatile-rich ultramafic magmas that transport diamonds from depths of 150 km to 200 km to the earth's surface, together with fragments of mantle rocks from which the diamonds are directly derived (primarily peridotite and eclogite). Kimberlites occur at surface as volcanic pipes, irregular shaped intrusions, or sheet-like intrusions (dykes or sills). Due to the wide range of settings for kimberlite emplacement, as well as varying properties of the kimberlite magma itself (most notably volatile content), kimberlite volcanoes can take a wide range of forms and be infilled by a variety of deposit types, even within a single kimberlite field, like Chidliak (refer to Figure 7-4).

The Chidliak kimberlites are stratified bodies and different pipes contain different types of infill ranging from VK-only to mixed VK, ACK and CK deposits (referred to as combined-infill pipes). None of the Chidliak pipes contain massive VK-type infills like observed in many southern African kimberlites and in Canadian pipes at Gahcho Kué or Renard (Field and Scott Smith, 1999; Field et al., 2008, Fitzgerald et al., 2009; Hetman et al., 2004). The Chidliak kimberlites also differ from many other Canadian kimberlites, such as those found at Fort à la Corne and Lac de Gras. The Fort à la Corne kimberlites are large, shallow, champagne-glass-shaped pipes infilled entirely with pyroclastic kimberlite. The Lac de Gras pipes are small, steep-sided pipes characterized by an abundance of resedimented volcaniclastic kimberlite (RVK) and associated PK (Field and Scott Smith, 1999; Scott Smith, 2008).

The Chidliak kimberlites do however have similarities to those at Victor in the Attawapiskat region (van Straaten et al., 2009) with respect to their general emplacement and types of pipe infill. The timing of kimberlite magmatism at Chidliak roughly corresponds with that of some of the younger intrusions in the Attawapiskat province (Heaman et al. 2012), which were also intruded into a Paleozoic carbonate-dominated sequence. Unlike at Chidliak, some of the Paleozoic strata are preserved in the Attawapiskat region and the Chidliak bodies may be deeper analogues of Victor-type PKs (Pell et al., 2013).

The diamond content of the Chidliak pipes is controlled by the efficiency of sampling diamondiferous mantle material at depths of 150 km to 200 km, and rapid transport to surface. At Chidliak, any kimberlite with significant total mantle-derived garnet content is assessed as potentially having significant diamond content, especially if eclogitic or websteritic garnets are present (Pell et al., 2013).

9 Exploration

This section is an abridged version of information contained in Fitzgerald et al., 2018.

9.1 Exploration Programs 2005 to 2017

Exploration on the Chidliak Project since 2005 has consisted of:

- Surficial sediment sampling predominantly of glacial till and to a lesser extent stream or esker sediments for heavy minerals;
- Geological, structural and glacial mapping and outcrop sampling;
- Airborne geophysical surveys (DIGHEM® and RESOLVE® magnetic and electromagnetic / resistivity);
- Ground geophysical surveys (magnetic, horizontal-loop electromagnetic and OhmMapper);
- Core drilling for exploration, delineation and sampling;
- Small-diameter RC drilling for exploration and delineation;
- Surface trench excavation for mini-bulk and bulk sampling;
- Microdiamond and commercial-size diamond sampling and analyses;
- Diamond valuation;
- Bulk density and geotechnical measurements; and
- Petrography and whole rock chemistry analyses.

To date, 74 kimberlites have been discovered at Chidliak by surface prospecting, small-diameter RC drilling or by core drilling. Work on the Project completed prior to 2018 is discussed in detail in (Pell, 2008, 2009, 2010a, 2010b, 2011; Farrow et al., 2014, 2015; Nowicki et al., 2016 and Fitzgerald et al., 2018) and described briefly below.

A summary of the work completed at Chidliak between 2005 and 2018 is shown in Table 9-1.

Table 9-1: Summary of Work at Chidliak

Year / Activity	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Till samples collected (#)	166	232	872	221	1290	541	513	413	-	146	-	-	-
Probe confirmed KIM-positive till samples (#)	5	36	294	105	138	176	350	238	-	90	-	-	-
Analyzed KIMs from till (#)	44	460	3811	1798	2678	2374	9918	4265	-	2554	-	-	-
Airborne geophysics (line-km)	-	-	-	11858	-	20442	14872	-	-	-	-	-	-
Ground geophysics (line-km)	-	-	-	157	1096	1855	2188	6428	87	2327	-	-	-
Ground gravity (# of gravity stations)	-	-	-	-	-	-	-	-	2888	-	-	-	-
Anomalies ground-checked (#)	-	-	-	12	63	114	84	52	205	20	-	-	-
Core drilling (m)	-	-	-	-	3951	7798	8869	2379	-	3697	1361	-	5288
Small diameter RC Hornet drilling (m)	-	-	-	-	-	1445	1692	159	-	1544	-	-	-
Large diameter RC drilling (m)	-	-	-	-	-	-	-	-	-	-	1212	-	-
Kimberlite discoveries – prospecting (#)	-	-	-	3	6	17	1	2	6	1	-	-	-
Kimberlite discoveries – core drilling (#)	-	-	-	-	7	11	6	-	-	-	-	-	-
Kimberlite discoveries – small-diameter RC drilling (#)	-	-	-	-	-	9	2	-	-	3	-	-	-
Analyzed KIMs from kimberlites (#)	-	-	-	1659	5497	8751	17406	-	-	1272	-	-	-
Microdiamond sample processed (kg)	-	-	-	899	3404	7473	9826	251	535	3591	3397	-	1936
Samples processed by coarse caustic (t)	-	-	-	2.28	0.91	1.17	-	-	-	-	-	-	-
Mini-bulk samples processed by DMS (dry t)	-	-	-	-	49.6	61.2	32.7	-	-	-	-	-	-
Bulk samples processed by DMS (dry t)	-	-	-	-	-	-	-	-	404.3	-	809.5	-	-

9.2 Surface Samples for Commercial Size Diamond Testing

Due to the limited amount of glacial overburden overlying several kimberlites of interest at Chidliak, it is possible to collect large samples of kimberlite by trenching. Surface trench samples have been collected from the CH-1, CH-28, CH-6 and CH-7 kimberlites for commercial-size diamond testing. Sample collection, transport and results of the 2010 CH-7 surface mini-bulk sample and the 2013 CH-6 surface bulk sample are detailed here, whereas surface sampling of the CH-1 and CH-28 kimberlites are documented in Pell (2010a) and Pell (2011).

9.2.1 CH-7 Mini-Bulk Sample – 2010

In 2010, a 47.2 t (dry) mini-bulk sample was collected from a surface trench in the KIM-1 unit in the northeast of the CH-7 kimberlite, as a test for commercial-size diamonds. Between June 21st and July 16th an estimated 50 t (wet) of in-situ kimberlite was collected. Snow and glacial overburden were stripped from surface using a CAT® multi-terrain loader and once kimberlite was exposed, several controlled blasts were used to fragment the kimberlite. Kimberlite was sampled from between 0.35 and 2.0 m depth by hand digging and collected in 76 double-layered 1 t capacity ore bags. Laser-inscribed diamond tracers were added to some filled ore bags, the inner bag was closed with a uniquely numbered security seal and outer bags were closed and labelled with a unique sample number. Bags were transported directly from the trench site to a secure hangar at the Iqaluit airport with a Bell 212S helicopter, two bags at a time. The bags were palletized and shrink-wrapped in Iqaluit, and shipped via chartered 767 aircraft to the Edmonton airport and onwards to the Saskatchewan Research Council (SRC) in Saskatoon using transport trucks (Holmes, 2010).

The mini-bulk sample was divided into four processing units, though processed at the SRC by DMS as one batch, and returned 356, +1.18 mm sieve size diamonds that weighed 47.29 cts, for a diamond content of 1.00 ct/t (Table 9-2).

Table 9-2: Results of the 2010 CH-7 Mini-Bulk Sample

Unit		KIM-1
Sample Weight (dry tonnes)		47.20
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.850 mm	146
	+1.180 mm	172
	+1.700 mm	111
	+2.360 mm	55
	+3.350 mm	16
	+4.750 mm	2
Total Number of Diamonds		502
Carats >0.850 mm		49.07
Carats >1.180 mm		47.29
Carats / tonne >1.180 mm		1.00

9.2.2 CH-6 Bulk Sample – 2013

In spring 2013, a 404.3 t (dry) bulk sample was collected from a surface trench in the weathered portion of KIM-L in CH-6, as a test for commercial-size diamonds. Between February and May 2013, 507 t (wet) of in-situ kimberlite were collected in 516, 1.5 t capacity ore bags. The trench site, near the northern margin of the CH-6 pipe, was chosen due to the limited overburden depth in the area. The trench was prepared for excavation by placing 203 small drill holes (located by a professional surveyor) loaded with stick dynamite, that were detonated in three controlled blasts, in order to excavate a 6 x 6 x 4 m trench (Pell and O'Connor, 2013).

Overburden depth in the trench varied from 2.8 to 4 m, and the contact between overburden and kimberlite was sharp. Once overburden was removed, the trench floor was broken up and an excavator was used to place the kimberlite in a stockpile adjacent to the trench. Kimberlite was collected from the stockpile via a small loader and placed into double-layer ore bags labelled with a unique sample number. Laser-inscribed diamond tracers were added to some filled ore bags, the inner bag was closed with a uniquely numbered security seal and outer bags were closed. Once all sampling was complete, the excavated trench and overburden stockpile were surveyed and the trench was reclaimed by backfilling with overburden (Pell and O'Connor, 2013).

Of the 516 bags collected, 406 were shipped overland over a one-month period from site to Iqaluit via the Iqaluit-Chidliak trail using Challengers with sleds, 102 were transported using a DC-3T aircraft and eight remaining bags were transported via Twin Otter during the subsequent summer exploration program. All sample bags were stored in Iqaluit in a secure area at the airport until they were shipped south and transported to the processing labs. Ten bags were flown to Winnipeg in late June and transferred to trucks for transport to the SRC. The remainder were shipped to Montreal via sealift in late summer 2013. Once arriving in Montreal, bags were transferred to transport trucks, with security-sealed trailers, for shipping to the De Beers plant in Sudbury, Ontario (Pell and O'Connor, 2013). Moisture content was calculated from samples of head feed taken at the processing plants. In total 43 moisture measurements were made. The moisture content of the samples ranged from 9.99% to 17.40%.

The 404.3 t sample returned a grade of 2.58 ct/t at a +1.18 mm bottom cut off (Table 9-3). The sample included 90 diamonds weighing over 1.0 ct and 270 diamonds weighing over 0.50 cts, with the largest diamond being an 8.87 ct white / colourless octahedron.

Table 9-3: Results of 2013 CH-6 Trench Bulk Sample

Sample		13-1	13-2	13-3	TOTAL 2013 Bulk Sample
Description		Batch B (Test)	Batch A	Batch C	
Sample Weight (dry tonnes)		8.41	213.8	182.1	404.31
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.850 mm	222	2,967	2,899	6,088
	+1.180 mm	135	3,233	2,825	6,193
	+1.700 mm	60	1,436	1,184	2,680
	+2.360 mm	26	595	474	1,095
	+3.350 mm	3	139	125	267
	+4.750 mm	1	32	24	57
	+6.700 mm	0	2	4	6
Total Number of Diamonds		447	8,404	7,535	16,386
Carats >0.850 mm		21.74	578.75	523.46	1,123.95
Carats >1.180 mm		18.85	538.66	484.54	1,042.05
Carats / tonne >1.180 mm		2.26	2.52	2.66	2.58

9.2.3 Microdiamond Sampling Associated with Surface Bulk Samples

In 2010, Peregrine collected microdiamond samples of the CH-7 kimberlite during the trench mini-bulk sampling program, in order to test the reliability of the DMS results and to determine moisture content. One microdiamond sample of approximately 30 kg was collected for every two ore bags filled, such that a 967.9 kg representative sample in 38 buckets was available for further processing (Holmes, 2010). One moisture determination was made from a subsample from each of the 38 buckets. The moisture content of the samples ranged from 1.67% to 11.59%, averaging 5.84%. Approximately half of the representative microdiamond sample (467.25 kg) was processed at the SRC by caustic fusion to recover +0.425 mm diamonds, and 0.68 cts of +1.18 mm diamonds were recovered (Table 9-4). The remaining 500.65 kg portion of the representative sample was introduced to the DMS sample and is included in the head feed weight of the 2010 mini-bulk sample.

Concurrent with collection of the CH-6 2013 surface bulk sample, Peregrine completed microdiamond sampling of the kimberlite material excavated in order to monitor DMS processing efficiency. A 750 kg microdiamond sample of KIM-L was collected from representative locations throughout the trench. A 400 kg “split” portion was retained for potential future work and the remaining 350 kg “split” was sent for caustic fusion assay to the SRC (Pell and O’Connor, 2013). In total, 907 stones +0.106 mm in size were recovered from this sample, including 10 diamonds larger than 1.18 mm weighing 0.39 cts (Table 9-5).

Table 9-4: Results of Caustic Fusion Analyses for Representative KIM-1 from the 2010 CH-7 Mini-Bulk Sample

Unit		KIM-1
Sample Weight (kg)		467.3
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.425 mm	40
	+0.600 mm	19
	+0.850 mm	8
	+1.180 mm	2
	+1.700 mm	1
	+2.360 mm	0
	+3.350 mm	1
	+4.750 mm	0
Total Number of Diamonds		71
Total Carats		0.90
Carats >0.850 mm		0.78
Carats >1.180 mm		0.68

Table 9-5: Results of Caustic Fusion Analyses for Representative wKIM-L from the 2013 CH-6 Bulk Sample

Unit		KIM-L
Sample Weight (kg)		350.0
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.106 mm	317
	+0.150 mm	228
	+0.212 mm	150
	+0.300 mm	99
	+0.425 mm	60
	+0.600 mm	32
	+0.850 mm	11
	+1.180 mm	9
	+1.700 mm	1
Total Number of Diamonds		907
Total Carats		0.79
Carats >0.850 mm		0.52
Carats >1.180 mm		0.39

10 Drilling

This section is an abridged version of information contained in Fitzgerald et al., 2018.

Approximately 33,343 m of core drilling, 4,840 m of small-diameter reverse circulation (RC) drilling and 1,212 m of large-diameter reverse circulation drilling (LDD) has been completed at Chidliak between 2009 and 2017. Further information regarding these programs is documented in Nowicki et al., 2016 and Fitzgerald et al., 2018.

10.1 Summary of Drilling at CH-6 and CH-7

10.1.1 Drilling at CH-6

A total of 11,701.92 m of core drilling (46 holes) and 479.32 m of small-diameter RC drilling (44 holes) has been completed at CH-6 (Table 10-1 and Table 10-2), all of which were used to create the 3-D geological model of the pipe, as discussed in Section 7.3.3. Of the core drilling at CH-6, 9,620.23 m (38 holes) were completed for the purpose of discovery, delineation and obtaining geotechnical information for open-pit design and optimization studies and 2,081.69 m (eight holes) were for commercial-size diamond testing. All small-diameter RC drilling was completed in order to define depth of overburden or near-surface kimberlite margins. Refer to Figure 10-1 for CH-6 core collar locations in plan view, Figure 10-2 for small-diameter RC collar locations in plan view and Figure 10-3 for a 3-D view of drilling.

Table 10-1: Summary of Core Drilling at CH-6

Kimberlite	Purpose	Year	# Holes	Length (m)	Diameter (mm)
CH-6	Delineation	2009	5	843.00	NQ
	Mini-bulk sample	2010	8	2,081.69	HQ
	Delineation	2011	11	1,774.83	HQ+NQ
	Delineation	2014	5	1,183.00	NQ
	Delineation	2015	2	520.40	NQ
	Delineation + Geotechnical	2017	15	5,299.00	HQ+NQ
Total				**11,701.92**	

Table 10-2: Summary of Small-Diameter RC Drilling at CH-6

Kimberlite	Purpose	Year	# Holes	Length (m)
CH-6	Delineation	2012	20	127.07
	Delineation	2014	24	352.25
Total				**479.32**

Figure 10-1: Plan View of Drilling to Date at CH-6

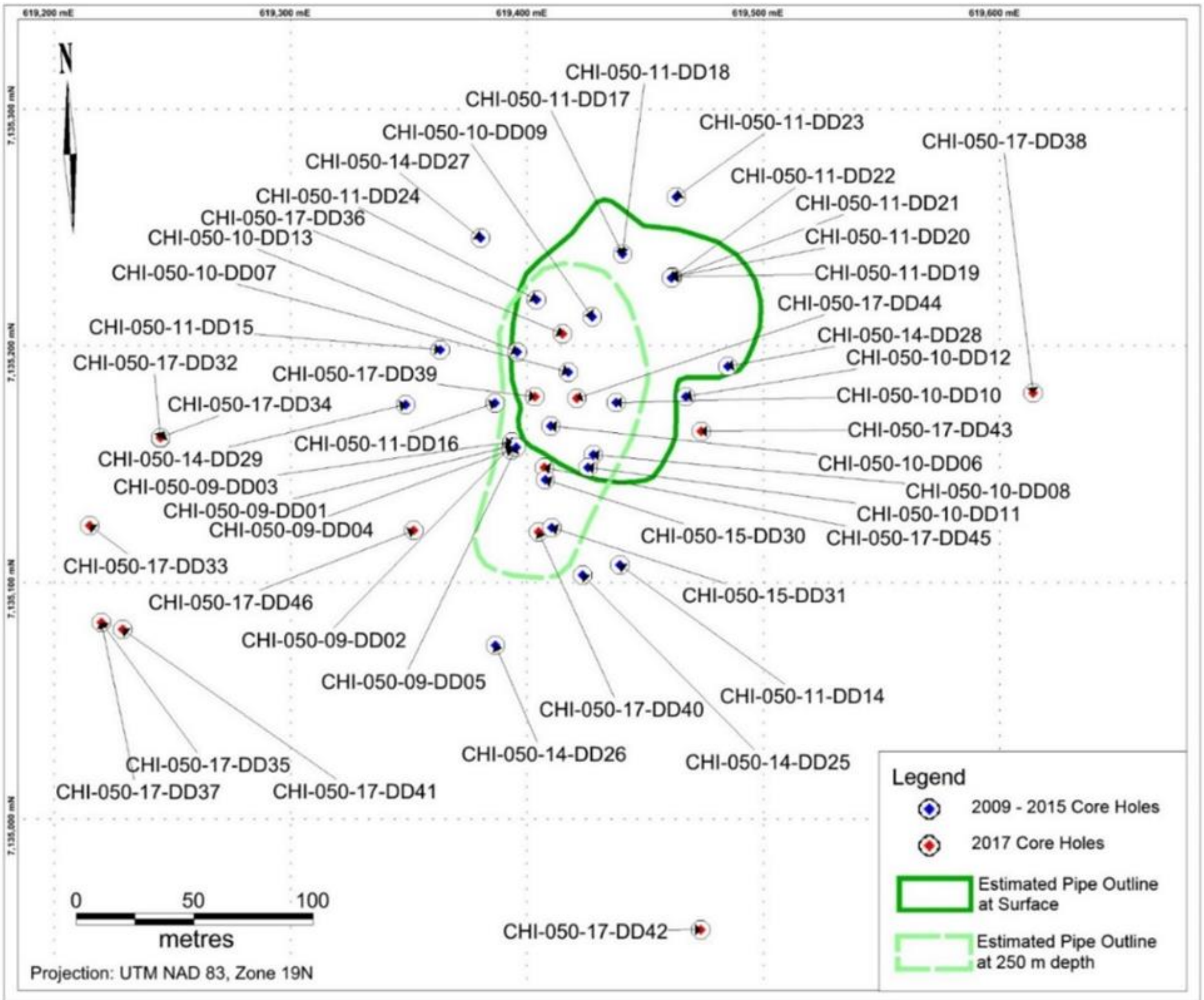


Figure 10-2: Plan View of Small-diameter RC Drilling to Date at CH-6

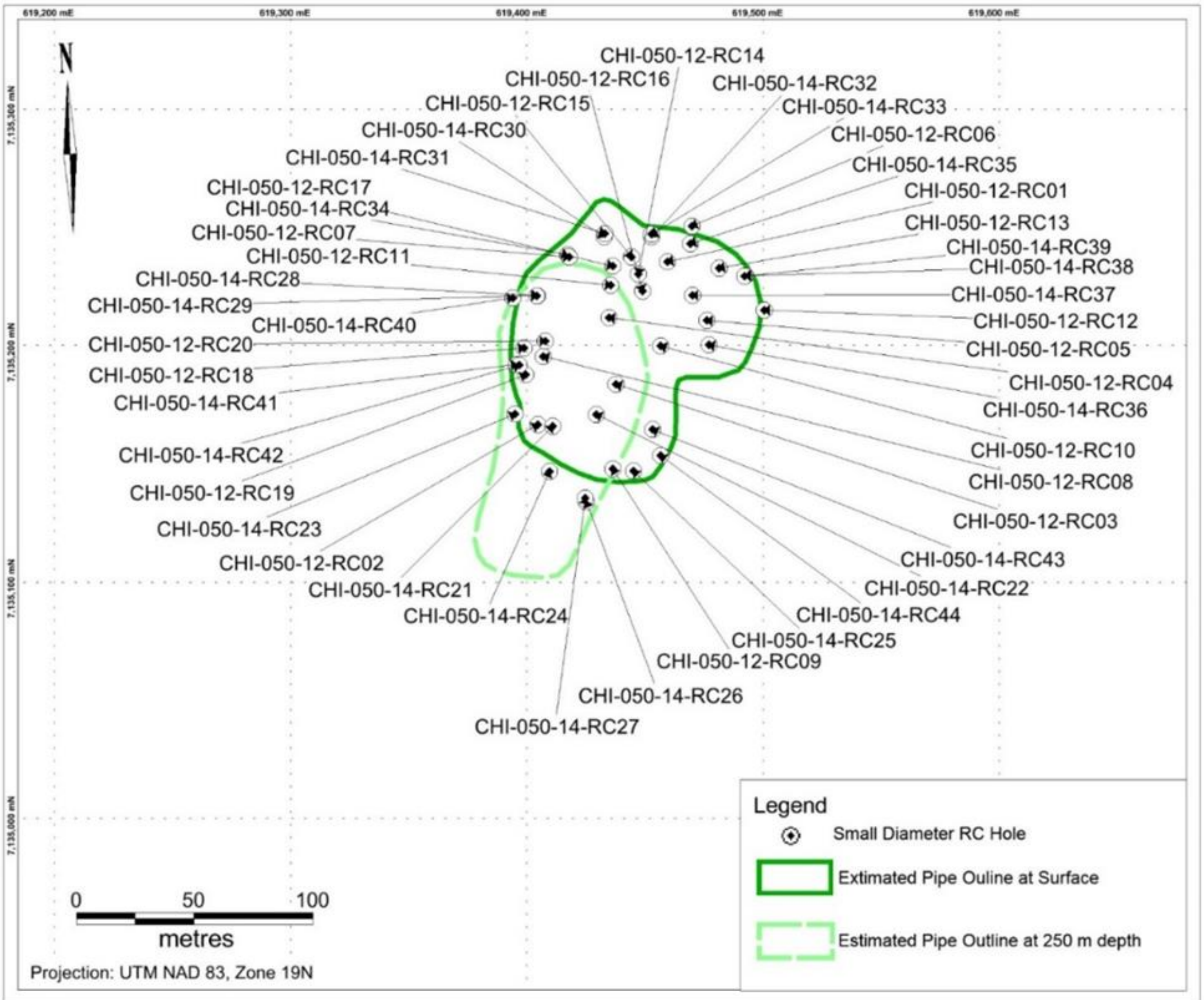
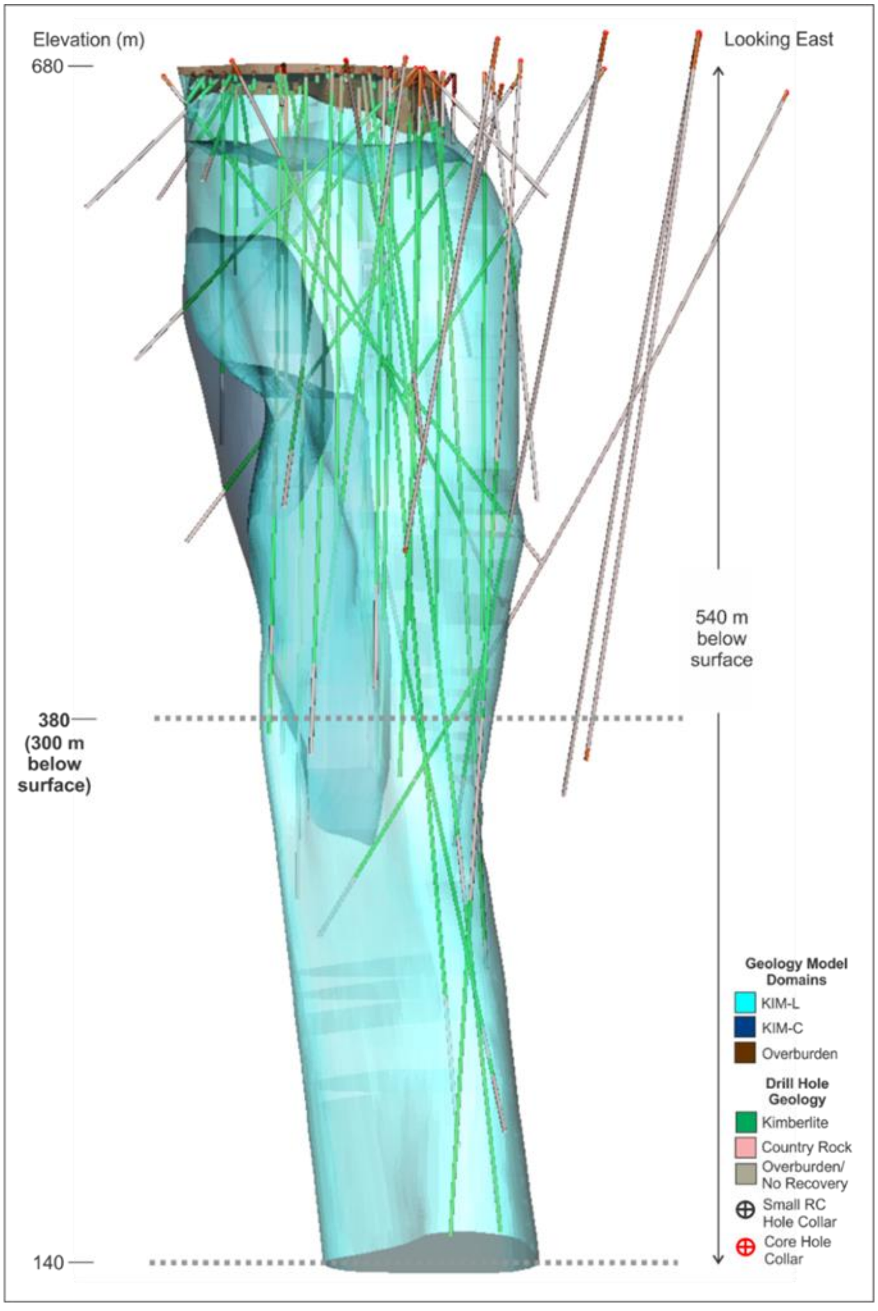


Figure 10-3: Cross-Section of Drilling to Date at CH-6 with Geology Model



10.1.2 Sampling of Drill Core at CH-6

To date, CH-6 drill core has been sampled for commercial-size diamonds (13.84 t), caustic fusion diamond analysis (6,651.38 kg), whole rock chemical analysis (322 samples), bulk density analysis (2,396 samples), representative archival purposes (1,419 samples), petrography (246 samples) and limited early-stage geotechnical analysis (201 samples).

10.1.2.1 Bulk Density Samples from Drill Core

Density has been measured on drill core samples of both country rock and kimberlite at CH-6, since the initial drilling in 2009. A total of 2,396 useable dry bulk density measurements have been made at CH-6, the majority of which were made in the field by Peregrine. A summary of the number of useable bulk density samples by methodology is provided in Table 10-3.

Table 10-3: Summary of Dry Bulk Density Measurements at CH-6

Kimberlite	Analysis Method	No. Samples
CH-6	Air Dried – Displacement	2,364
	Oven Dried – Displacement	7
	Oven Dried – Waxed - Displacement	25

10.1.2.2 Whole Rock Chemistry Samples

The Peregrine whole rock chemistry dataset for CH-6 comprises 286 samples of kimberlite from 35 core holes and one small-diameter RC hole (Table 10-4). The database also includes analyses of mantle xenoliths and various country-rock types that occur in the Project area.

Table 10-4: Summary of Whole Rock Chemistry Samples

Kimberlite	Sample Type				
	Kimberlite	Mantel Xenoliths	Paleozoic Clasts	Country Rock	Other (Till, Mixed)
CH-6	286	21	3	12	2*

One sample collected from RC drilling

10.1.2.3 Microdiamond Samples

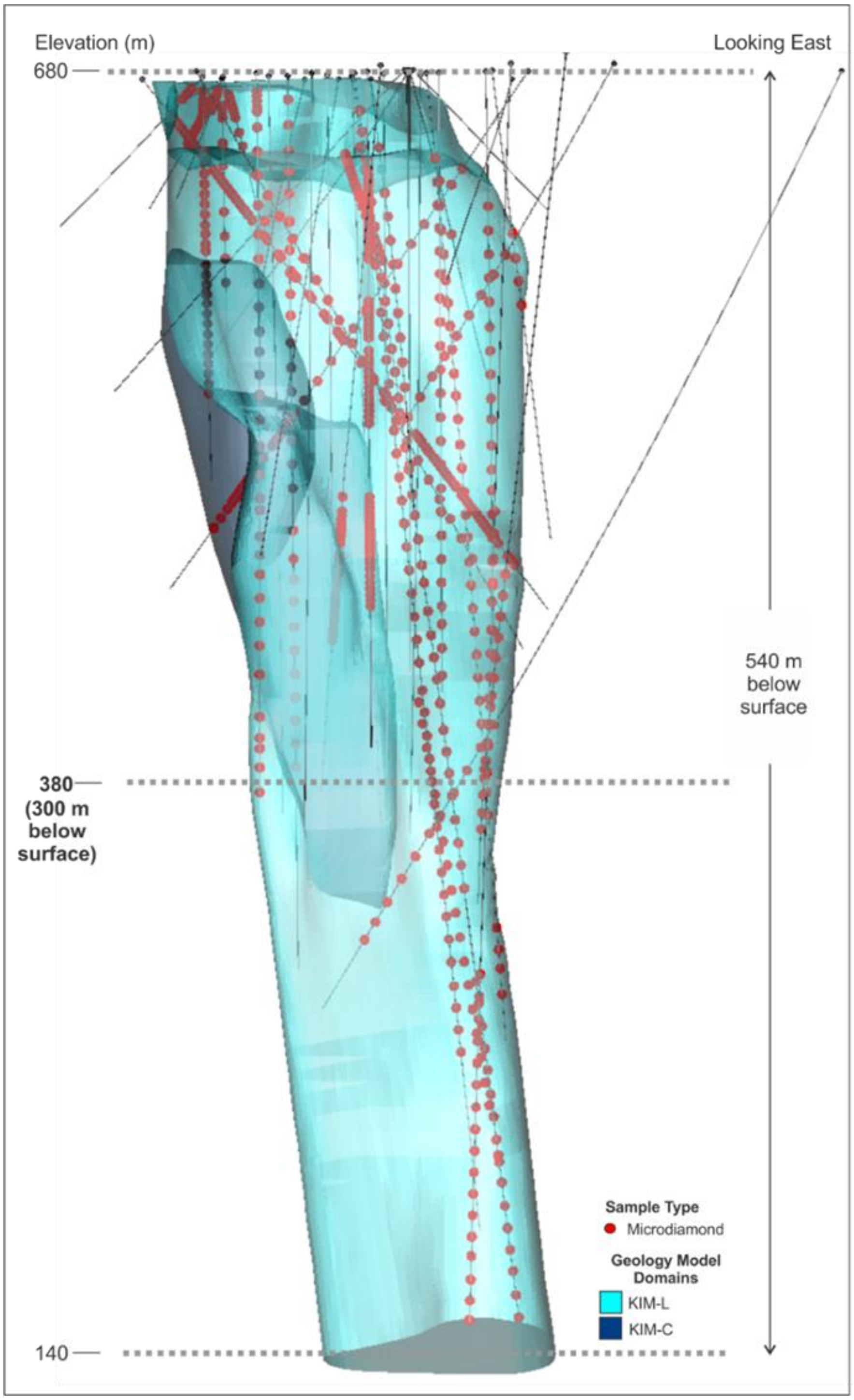
Results from samples collected from drill core and processed by caustic fusion for diamonds at a +0.106 mm bottom cut off are presented in Table 10-5. The nature of these results and implications for resource estimation are discussed in Section 14-4. The distribution of samples with microdiamond results from CH-6 is illustrated in Figure 10-4.

Table 10-5: Summary of Microdiamond Results from Core Samples from CH-6

Kimberlite & Domain		CH-6 KIM-L	CH-6 KIM-C
Sample Weight (kg)		5,543.28	349.49
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.106 mm	6,742	267
	+0.150 mm	4,242	185
	+0.212 mm	2,692	108
	+0.300 mm	1,745	60
	+0.425 mm	1,112	23
	+0.600 mm	685	17
	+0.850 mm	332	12
	+1.180 mm	192	2
	+1.700 mm	63	2
	+2.360 mm	20	1
	+3.350 mm	10	0
Total Number of Diamonds		17,835	677
Carats >0.850 mm		25.31	0.58
Carats >1.180 mm		21.59	0.44

Note: These results are for drill core only and do not include the microdiamond results that accompany the 2013 bulk sample.

Figure 10-4: Distribution of Microdiamond Sample Results for CH-6



10.1.2.4 Commercial-size Diamond Samples from Drill Core

During July 2010, a 13.84 t (dry) mini-bulk sample was established by aggregating CH-6 drill core, to test for commercial-size diamonds. The goal of the program was to sample the kimberlite in a representative fashion both geologically and spatially across the southern two-thirds of the pipe. The mini-bulk sample was aggregated in five processing units from eight HQ-sized core holes drilled in summer 2010 (1,576 m, representing 85% of the sample weight), and augmented by NQ-sized drill core remaining from seven holes drilled in 2009 (representing 15% of the sample). The 2010 holes were all vertical holes drilled 25 to 35 m apart to a maximum depth of 325 mbs, whereas the 2009 holes were primarily inclined near-surface delineation holes. A total of 14.1 t (wet) of kimberlite was sampled from both the KIM-C and KIM-L units, with the majority of material being from KIM-L, the dominant infill of the CH-6 pipe. The locations of drill core intercepts contributing to the overall sample are shown in Figure 10-5. Results show 37.97 cts of diamonds larger than 1.18 mm were recovered in the sample (Table 10-6).

Table 10-6: Results of the 2010 CH-6 Mini-Bulk Sample

Sample		10B-1	10B-2*	10B-3	10B-4	10B-5	10B-Cleanup	TOTAL 2010 Mini-Bulk Sample
Description		KIM-C	KIM-L	KIM-L	wKIM-L	KIM-L	Plant Clean Up	
Sample Weight (dry tonnes)		4.06	1.95	3.46	1.02	3.35		13.84
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.850 mm	12	22	36	16	37	14	137
	+1.180 mm	21	36	49	27	70	13	216
	+1.700 mm	17	20	27	10	32	2	108
	+2.360 mm	7	8	17	5	14	0	51
	+3.350 mm	1	3	3	3	0	0	10
	+4.750 mm	0	0	0	1	0	0	1
Total Number of Diamonds		58	89	132	62	153	29	523
Carats >0.85 mm		4.70	7.44	10.62	7.03	9.53	0.72	40.04
Carats >1.180 mm		4.52	7.14	10.07	6.80	8.95	0.49	37.97
Carats / tonne >1.180 mm		1.12	3.68	2.92	6.71	2.69	-	2.74

Note: Weights here may differ from pre-2016 disclosure, to reflect audited dry tonnage calculations.

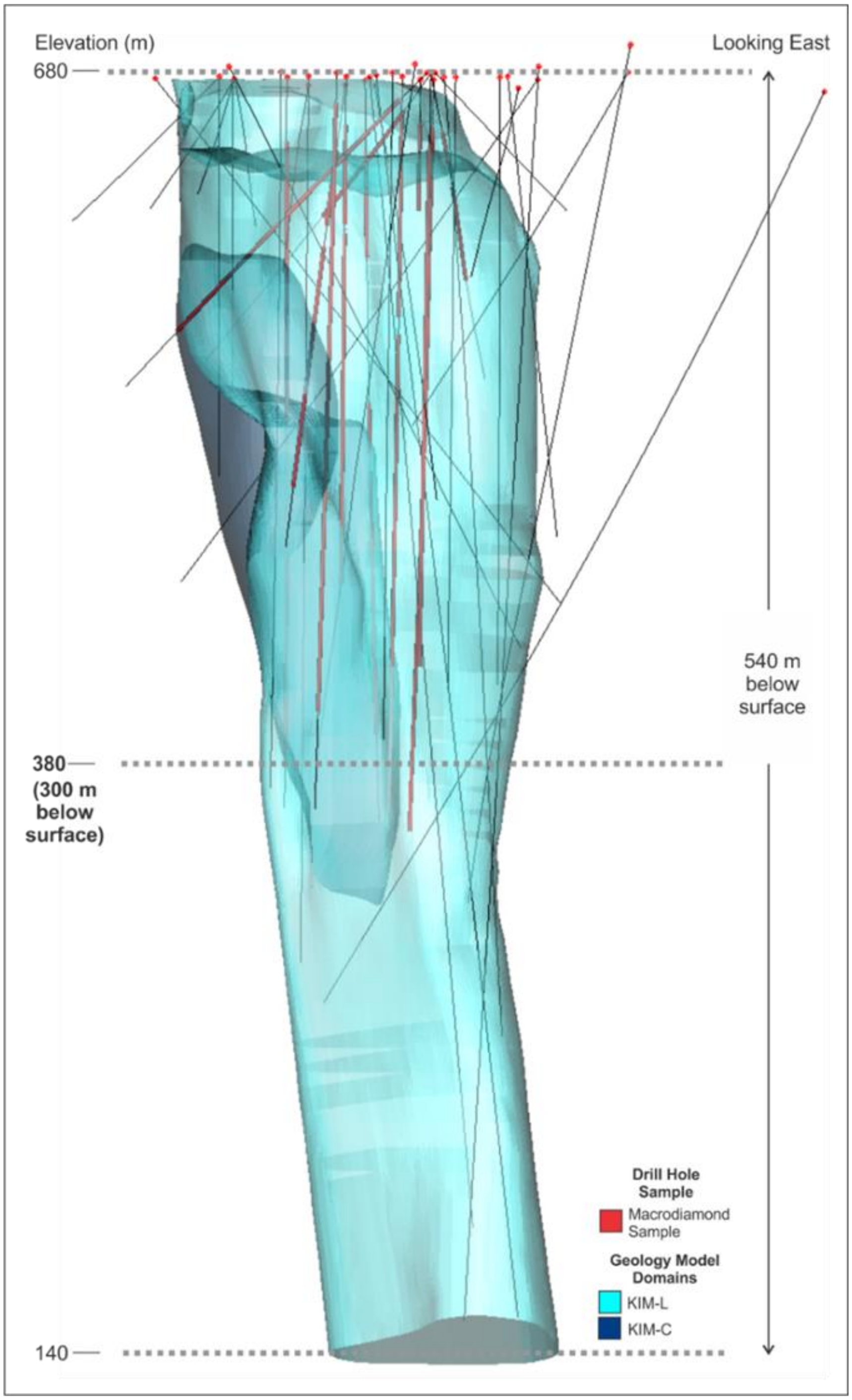
In Sample 10B-2, a 157.8 kg sample of limestone breccia was processed by coarse caustic, and should be added to the total weight to bring the total weight of the whole sample to the previously reported 14 tonnes.

Concurrent with aggregation of the 2010 mini-bulk sample, 465.30 kg of microdiamond samples were also collected from within each of the five processing units, the purpose of which was to assess DMS processing efficiency and to measure moisture content. One moisture determination was made for each processing unit and the moisture contents of the samples were very low, ranging from 0.88% to 1.87%. Of the total sampled, 124 kg was of KIM-C and 341.3 kg was of KIM-L. This sample was processed for diamonds larger than +0.425 mm sieve size at the SRC by caustic fusion in five batches. Results show 1.98 cts of diamond larger than 1.18 mm were recovered from the 465.30 kg processed (Table 10-7).

Table 10-7: Results of Caustic Fusion Assays, 2010 CH-6 Mini-Bulk Sample

Sample		10B-1	10B-2	10B-3	10B-4	10B-5	Total
Description		KIM-C	KIM-L	KIM-L	wKIM-L	KIM-L	
Sample Weight (kg)		124.00	60.95	113.65	44.50	122.20	465.30
% Moisture		0.88	1.77	1.09	1.87	1.53	-
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.425 mm	9	4	10	7	19	49
	+0.600 mm	5	4	7	7	19	42
	+0.850 mm	1	2	6	3	9	21
	+1.180 mm	1	1	5	3	5	15
	+1.700 mm	1	0	0	1	2	4
	+2.360 mm	0	0	0	0	1	1
	+3.350 mm	0	1	0	0	0	1
	+4.750 mm	0	0	0	0	0	0
Total Number of Diamonds		17	12	28	21	55	133
Carats >1.180 mm		0.08	1.04	0.19	0.18	0.49	1.98

Figure 10-5: Distribution of Samples Collected and Processed for Commercial-Size Diamonds (+0.85 mm) from Core in CH-6



10.1.3 Drilling at CH-7

A total of 4,960.22 m of core drilling (29 holes) (Table 10-8), 675.74 m of small-diameter RC drilling (41 holes) (Table 10-9) and 1,212.13 m (six holes) of large-diameter RC drilling (Table 10-10) has been completed at CH-7. Data from all of these holes, with the exception of the LDD holes, were used to create the 3-D geological model of CH-7, as discussed in Section 7.3.5. Refer to Figure 10-6 and Figure 10-7 for CH-7 collar locations in plan view and Figure 10-8 for a 3-D view of drilling.

Table 10-8: Summary of Core Drilling at CH-7

Kimberlite	Purpose	Year	# Holes	Length (m)	Diameter (mm)
CH-7	Delineation	2010	6	812.22	NQ
	Delineation	2011	8	1197.00	HQ
	Delineation	2012	4	983.00	HQ+NQ
	Delineation	2014	7	1127.50	NQ
	Geology	2015	4	840.50	NQ
Total				**4960.22**	

Table 10-9: Summary of Small-Diameter RC drilling at CH-7

Kimberlite	Purpose	Year	# Holes	Length (m)
CH-7	Discovery	2010	2	77.11
	Overburden Depth Determination	2012	3	13.86
	Overburden Depth Determination & Delineation	2014	36	584.77
Total				**675.74**

Table 10-10: Summary of Large-Diameter RC Drilling at CH-7

Hole #	Orientation			Start Hole Diameter (in)	End Hole Diameter (in)	Unit	Bulk Bags Filled
	AZ	Dip	Length (m)				
CHI-251-15-LD01	0	-90	219.10	28	22	KIM-2	127
CHI-251-15-LD02	0	-90	222.00	24	22	KIM-2	122
CHI-251-15-LD03	0	-90	240.00	24	22	KIM-2, 3 & 4	134
CHI-251-15-LD04	0	-90	237.30	24	22	KIM-2, 3 & 4	127
CHI-251-15-LD05	0	-90	74.63	24	22	KIM-5	26
CHI-251-15-LD06	0	-90	219.10	24	22	KIM-2	117
Total			**1212.13**				**653**

Of the core drilling at CH-7, 3,614.22 m (23 holes) were completed for the purpose of delineation and 1,346 m (six holes) acted as geological pilot holes for LDD RC sampling. Of the 41 small-diameter RC drill holes completed, 518.17 m (26 holes) were for delineation and 157.57 m (15 holes) were for defining the depth of overburden.

In 2015, six LDD RC holes were completed to establish a diamond grade profile for geological units at CH-7 and to collect a parcel of commercial-size diamonds for valuation. The LDD hole locations were chosen to maximize the amount of kimberlite collected that was representative of each geological domain and to be volumetrically representative of major geological units within the CH-7 kimberlite. A pilot core hole was associated with each LDD hole. A total of 809.5 t (dry) of kimberlite was sampled, the details of which are documented in Section 10.1.4.4.

Figure 10-6: Plan View of Core and LDD Drilling to Date at CH-7

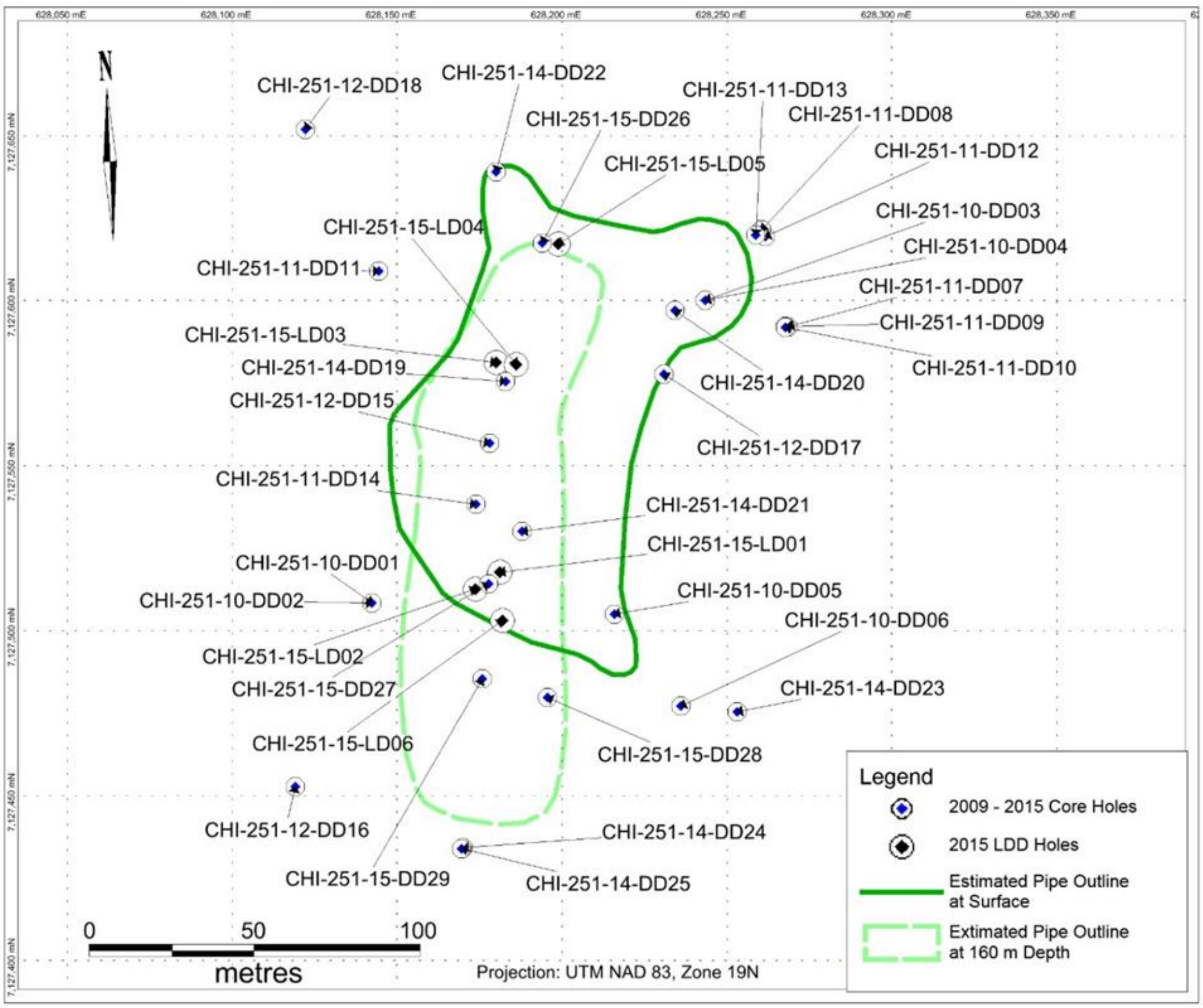


Figure 10-7: Plan View of Small Diameter RC Drilling to Date at CH-7

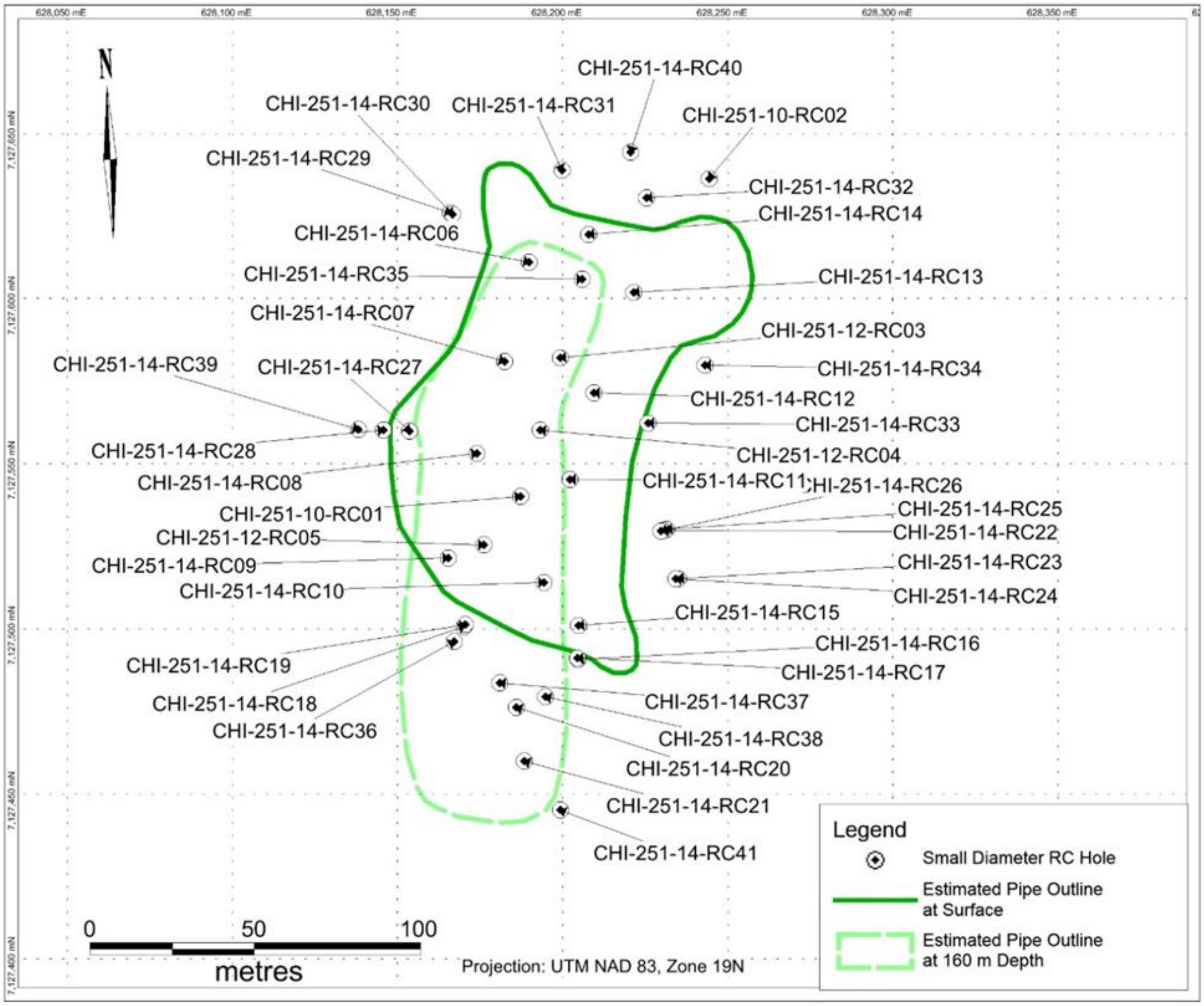
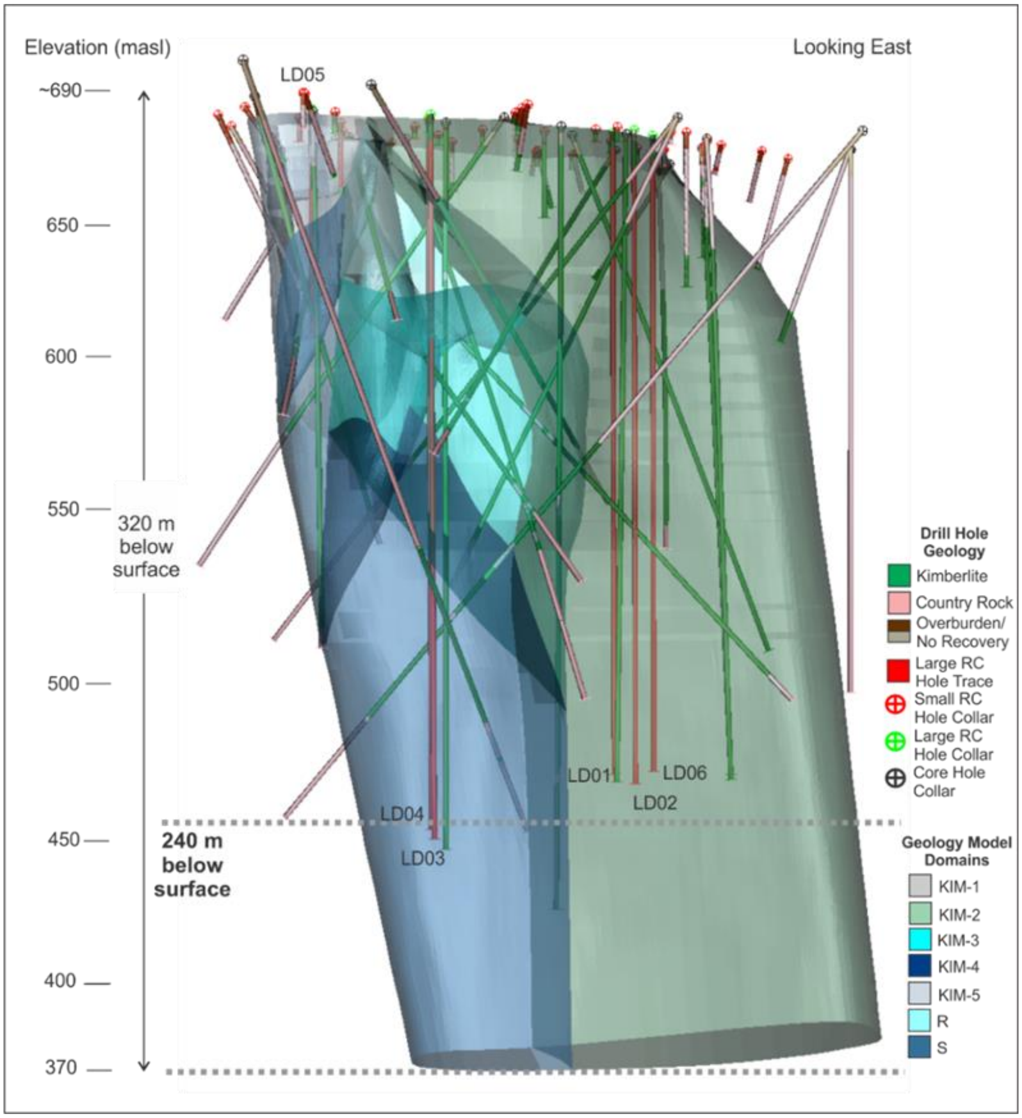


Figure 10-8: 3-D View of Drilling to Date at CH-7 with Geology Model



10.1.4 Sampling of Drill Core at CH-7

To date, CH-7 drill core has been sampled for microdiamonds (4,908.13 kg), whole rock chemical analysis (477 samples), bulk density analysis (988 samples), representative archival purposes (1,300 samples), petrography (283 samples) and limited early-stage geotechnical analysis (29 samples).

10.1.4.1 Bulk Density Samples from Drill Core

Density has been measured on drill core samples of both country rock and kimberlite at CH-7 since the initial drilling in 2010. A total of 990 useable dry bulk density measurements have been collected at CH-7, the majority of which were made in the field by Peregrine. A summary of the number of useable bulk density samples by methodology is provided in Table 10-11.

Table 10-11: Summary of Dry Bulk Density Measurements at CH-7

Kimberlite	Analysis Method	No. Samples
CH-7	Air Dried – Displacement	948
	Oven Dried – Displacement	10
	Oven Dried – Waxed – Displacement	32

10.1.4.2 Whole Rock Chemistry Samples

The Peregrine whole rock chemistry dataset for CH-7 comprises 442 samples of kimberlite from 25 core holes, eight small-diameter RC holes and surface (Table 10-12). The database also includes analyses of mantle xenoliths and various country-rock types that occur in the Project area.

Table 10-12: Summary of Whole Rock Chemistry Samples

Kimberlite	Sample Type				
	Kimberlite	Mantle Xenoliths	Paleozoic Clasts	Country Rock	Other (Till, Mixed)
CH-7	442	13	1	19	2

10.1.4.3 Microdiamond Samples

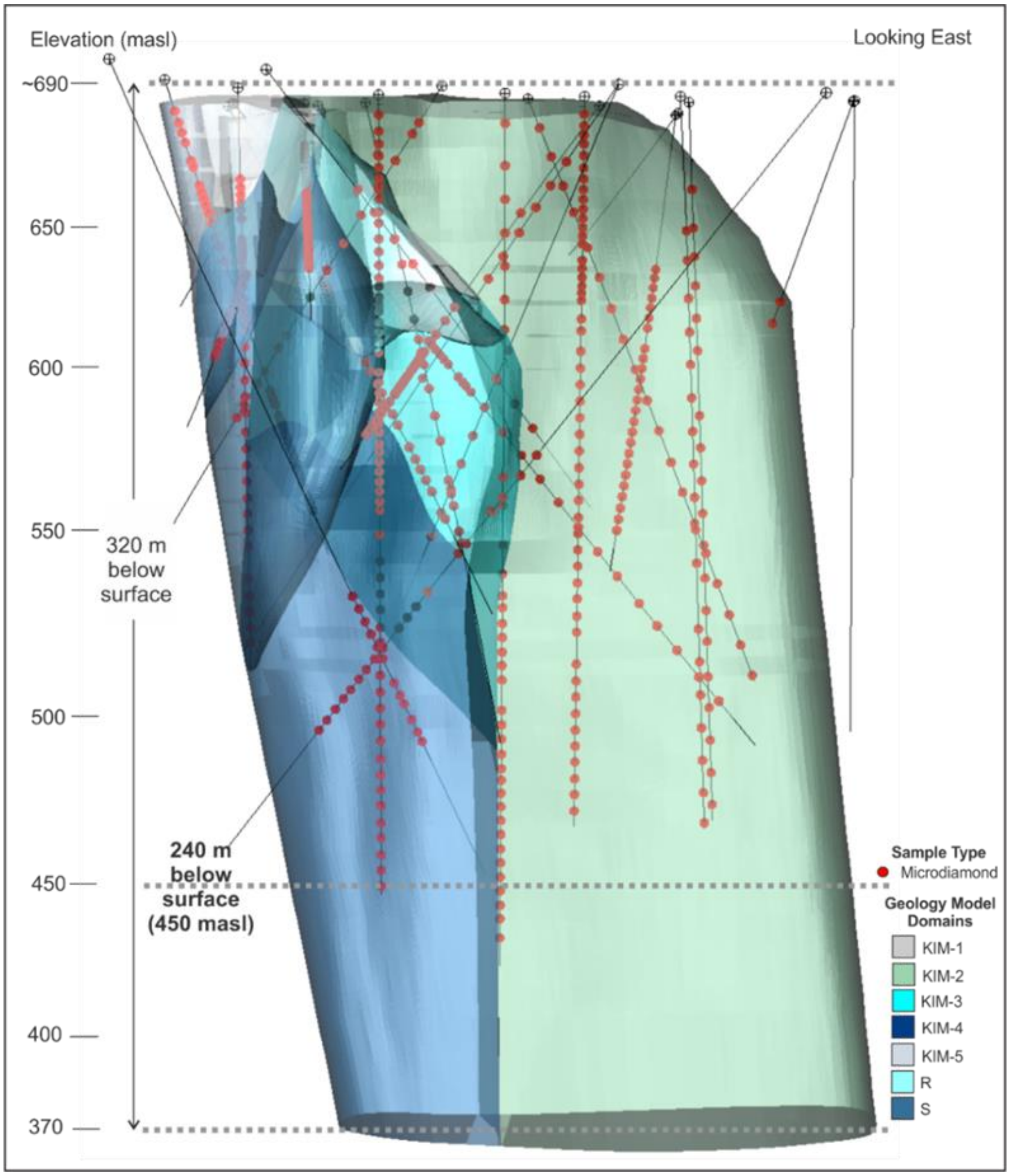
Results for samples collected from drill core and processed by caustic fusion for diamonds at a +0.106 mm bottom cut off are presented in Table 10-13. The nature of these results and implications for resource estimation are discussed in Section 14-5. The distribution of samples with microdiamond results from CH-7 is illustrated in Figure 10-9.

Table 10-13: Summary of Microdiamond Results from Core Samples from CH-7

Kimberlite & Domain		CH-7 KIM-1	CH-7 KIM-2	CH-7 KIM-3	CH-7 KIM-4	CH-7 KIM-5	CH-7 R	CH-7 S	CH-7 OTHER*
Sample Weight (kg)		254.50	1,832.88	786.04	375.91	539.17	455.75	525.44	199.35
Number of Diamonds per Sieve Size (mm square mesh sieve)	+0.106 mm	132	787	550	173	827	291	352	226
	+0.150 mm	104	566	359	138	557	202	292	166
	+0.212 mm	72	389	173	84	349	123	185	123
	+0.300 mm	28	241	130	48	257	79	101	60
	+0.425 mm	22	133	54	38	119	34	49	11
	+0.600 mm	15	73	35	16	56	24	38	19
	+0.850 mm	7	26	17	6	39	5	23	18
	+1.180 mm	2	11	3	3	19	5	11	2
	+1.700 mm	1	3	1	1	4	0	2	0
	+2.360 mm	1	5	3	0	2	0	3	0
	+3.350 mm	0	0	0	0	1	1	0	0
Total Number of Diamonds		384	2234	1,325	507	2,230	764	1,056	625
Carats >0.850 mm		0.30	2.42	1.10	0.30	2.17	1.07	1.32	0.26
Carats >1.180 mm		0.21	2.13	0.90	0.24	1.74	1.02	1.04	0.09

*Includes microdiamond samples comprised of mixed geologic domains

Figure 10-9: Distribution of Microdiamond Sample Results for CH-7



10.1.4.4 Commercial-sized Diamond Samples by Large-diameter RC Drilling

In winter 2015, Peregrine completed a LDD RC bulk sample program at CH-7, which comprised six, 22 in diameter drill holes totalling 1,212.13 m drilled between March 21 and May 8, 2015. The purpose of the program was to collect representative, sufficiently sized parcels of diamonds from five of the seven geological domains in order to assess diamond grade and value for the CH-7 kimberlite. In total, 558.8 t (wet) of kimberlite screened at 1.13 mm square mesh was collected in 653, 1 t ore bags. The six holes were drilled using a Cooper-14 large-diameter RC drill rig contracted through Cooper Drilling LLC of Monte Vista, Colorado.

Downhole caliper surveys on the 2015 LDD holes were performed immediately after completion of each LDD hole by DGI Geosciences Inc. (DGI) in order to determine drill hole diameter. Caliper measurements are provided in 5 cm depth increments for each LDD hole. These measurements were used to calculate the volume of kimberlite sampled (in cubic metres) along the length of the hole. The theoretical volume (V) of each sample is calculated as $V = pi (r^2) x h$, where r is the radius of the drill bit and h is the height interval. The actual volume sampled is based on hole diameters measured by caliper, which accounts for irregular LDD hole dimensions. Sampled volumes for individual process units and an entire LDD hole are determined by summing calipered volumes over the appropriate 5 cm depth increments.

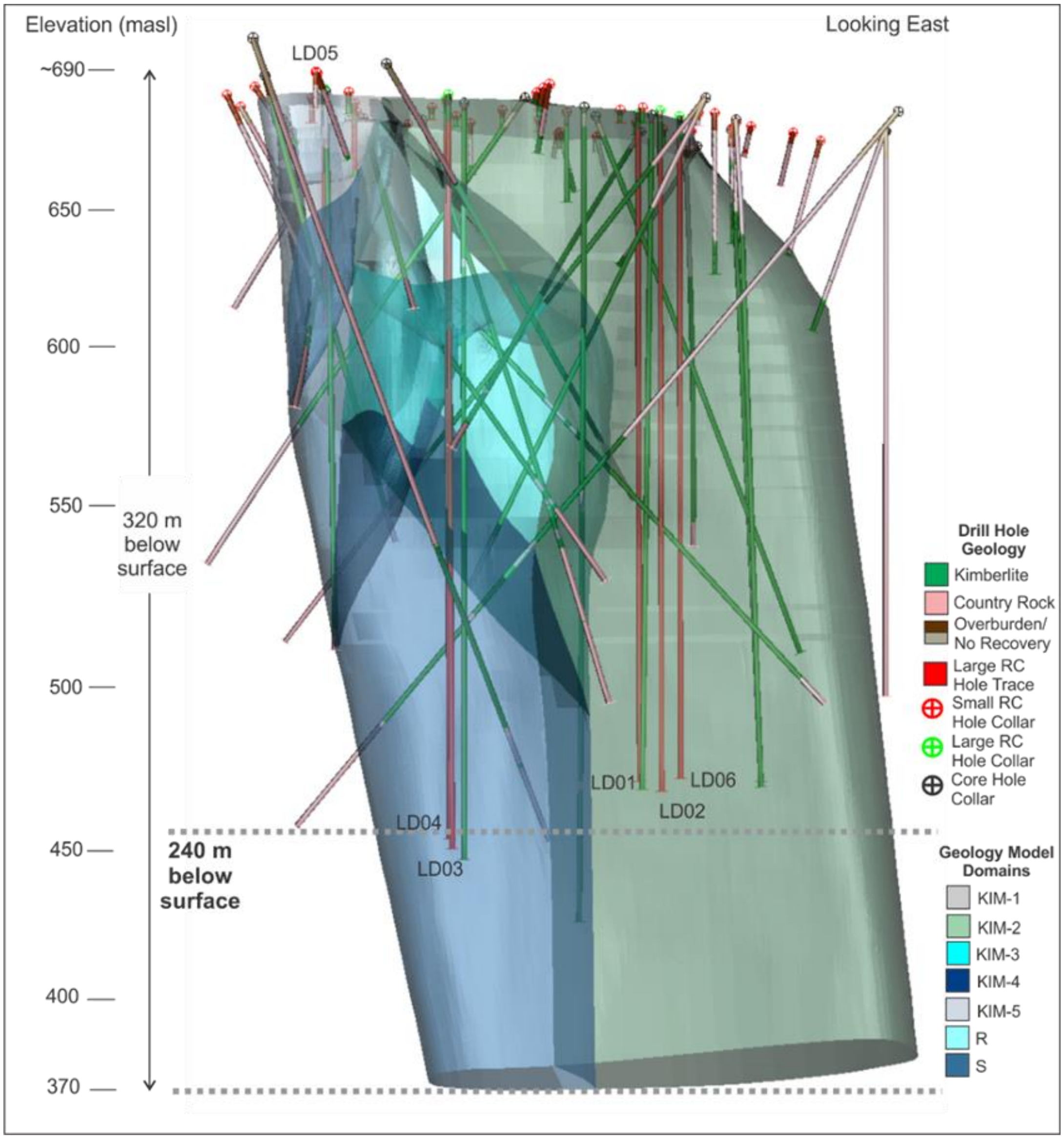
Once the volume was determined, it was combined with detailed density data in order to calculate the weight of kimberlite sampled, which is required to complete the grade estimate. The weight removed from the hole must be calculated by this method because undersize kimberlite material finer than the 1.13 mm shaker screen is not collected in the sample bags. Fine particles may also be lost through fractures in the hole wall and sloughing may also occur during drilling.

It was calculated that a total of 809.5 t (dry) of kimberlite was sampled in six LDD holes from CH-7 during the 2015 bulk sample program, with 717.65 cts of commercial-size (+1.18 mm) diamonds recovered, for an average sample grade of 0.89 ct/t (Table 10-14). The sample tonnage has been revised to 809.5 t (dry) from the 814 t (dry) reported in the Peregrine news release of January 12, 2016, as a result of density models constrained by Mineral Services for various geological domains at CH-7 (see Section 14.5.2, Table 14-15 and Table 14-16 of Nowicki et al., 2016). Refer to Figure 10-10 for a depiction of the LDD hole locations in CH-7.

Table 10-14: Results of the 2015 CH-7 Bulk Sample

Domain		KIM-2	KIM-3	KIM-4	KIM-5	R	TOTAL
Sample Weight (dry tonnes)		476.50	83.10	144.30	45.70	59.90	809.50
Number of Diamonds per Sieve Size (mm square mesh sieve)	+1.180 mm	2200	455	1098	389	286	4428
	+1.700 mm	953	211	473	165	137	1939
	+2.360 mm	393	71	166	62	59	751
	+3.350 mm	118	14	31	12	31	206
	+4.750 mm	13	5	6	5	3	32
	+6.700 mm	3	0	2	0	1	6
Total Number of Diamonds		3680	756	1776	633	517	7362
Carats >1.180 mm		363.66	68.14	157.93	60.08	67.84	717.65
Carats / tonne >1.180 mm		0.76	0.82	1.09	1.31	1.13	0.89

Figure 10-10: Large-Diameter Drill Holes in CH-7 for 2015 Bulk Sample Program



10.2 Drilling Quality Assurance & Quality Control

10.2.1 Core Drilling

The majority of drill collar positions at Chidliak have been obtained post-drilling utilizing a Differential Global Positioning System (DGPS), primarily with a Trimble 5800 RTK DGPS or a Trimble R10 GNSS Receiver with base station, operated by Peregrine staff. From 2013 onwards, most drill collar locations at CH-6, CH-7 and CH-44 were surveyed by a professional surveyor post-drilling with either Topcon Hiper GA or Leica

Viva GPS/GNSS receivers that utilize RTK GPS. In 2013 and again in 2014, permanent horizontal and vertical control points were established at six sites at Chidliak for future surveying use.

Drill holes completed in 2009 and 2011 were surveyed downhole with a single-shot magnetic tool every 50 m. In many cases, a measurement was taken at the end of the hole as well. Drill holes completed in 2010, 2012 and 2014, in addition to being surveyed with the single-shot tool, were also surveyed with a Reflex Gyro multi-shot, non-magnetic downhole gyroscopic survey tool that recorded measurements every 5 m, with few exceptions. In 2015, drill holes were surveyed downhole every 5 m using only the Reflex Gyroscope. In 2017, drill holes were surveyed downhole once drilling was completed using an Axis Mining Technology Inc. Champ Navigator tool, a non-magnetic, north-seeking gyroscope. Holes were surveyed in either single-shot or continuous mode, depending on drilling conditions. No small-diameter RC holes were surveyed downhole and for all un-surveyed holes, whether core or RC, proposed orientations were utilized in the drill database and for 3-D modelling.

Core hole data was verified by Peregrine in the following manner:

- Downhole survey data was checked against original data printouts and/or digital files from downhole survey tools and inconsistent / poor quality survey points were removed from single-shot data;
- Variation in downhole survey data was assessed. For holes that were surveyed with two methods (magnetic and gyroscope) the results were compared and the variation in the location of the end of the hole was never more than 5 m. The amount of variation observed does not materially affect the modelled kimberlite pipe shape;
- End of hole depths were cross-checked using detailed core logs, core photos and driller time sheet;
- Collar locations were confirmed against original data printouts from the DGPS survey tools and with original data and reports from the professional surveyor. Survey tools were calibrated at the time of the survey in the field;
- Meterages downhole were cross-checked with photos and detailed core logs and no inconsistencies were noted; and
- Contacts defined in core logs were cross-checked with core photos.

10.2.2 Small-diameter RC Drilling

The majority of small-diameter RC collar positions at Chidliak were located prior to drilling using a handheld GPS. Some positions were located post-drilling utilizing a DGPS, primarily with a Trimble 5800 RTK DGPS operated by Peregrine staff.

Small-diameter RC hole data was verified by Peregrine in the following manner:

- Visually logged intervals downhole were cross-checked against photos of chips;
- End of hole depths were cross-checked between driller time sheets and core logs; and
- Collar locations were verified against original data printouts from the DGPS survey tools where available. Survey tools were calibrated at the time of the survey in the field.

10.2.3 LDD Drilling

LDD hole locations were inspected in the field by Peregrine geologists in order to assess site suitability for the rig. Once sites were deemed accessible, drill pad requirements were assessed and the position the collar was to be located was surveyed using a Trimble ProXRT DGPS, operating with RTK satellite communication. Once drilling was complete, collar positions were surveyed with the same DGPS.

Downhole caliper surveys on the 2015 LDD holes were performed immediately after completion of each LDD hole by DGI Geosciences Inc. (DGI) in order to determine hole diameter. The caliper surveys were completed using a three-arm caliper tool manufactured by Mount Sopris (model 2CAA-1000) and modified in a proprietary manner by DGI. The system works with a winch and cable and each arm of the tool can extend to a maximum distance of 114 cm (DGI, 2015). In order to conduct the survey, the caliper was lowered to the bottom of the hole, the arms extended until they reached the hole wall and then the instrument is raised at a constant rate with the diameter of the hole recorded every 5 cm. Each hole was surveyed at least two times in order to assess the repeatability of measurements. The casing and casing shoe were used to confirm the accuracy of the tool since the internal dimensions for each were known and could be readily cross-referenced with the downhole data. The data is recorded at site digitally and later processed by DGI before being transferred to Peregrine.

As part of the QA/QC protocol employed by DGI Geosciences for caliper measurements, DGI has established a baseline for all probes and parameters at the Geological Survey of Canada Ottawa Calibration facility, and have developed a calibration data set to create field calibration procedures that supplement manufacturer recommendations. Additionally, they have developed their own calibration drill hole to expand on the quality and breadth of calibration procedures. Each probe has unique measures in place that include:

- On site calibrations to correct for regional variance and/or borehole size;
- Bench tests conducted in the field to ensure probes meet baseline values; and
- Calibration checks, recorded before and after each survey if applicable.

LDD RC data was verified by Peregrine during drilling in the following manner:

- QA/QC of accuracy for sample interval depths and of geological boundaries;
- Recording of changes in drilling conditions or equipment changes;
- Lithological observations;
- Noting sampling consistency and timing;
- Controlling bag movement; and
- Granulometry measurements in order to monitor chip quality.

LDD RC data was verified by Peregrine once drilling was complete in the following manner:

- Collar locations were confirmed against original data printouts from the DGPS survey tool and with original data and reports from the professional surveyor. Survey tools were calibrated at the time of the survey in the field;
- End of hole depths were verified against Pason data and field recordings;

- RC chip logs were cross-checked against core pilot hole geology;
- Caliper data was checked and verified for accuracy by ensuring that hole widths measured were not smaller than the bit diameter and that the known diameter at the bottom of casing was measured accurately. The casing and casing shoe were used to confirm the accuracy of the tool since the internal diameter for each were known and could be readily cross referenced with the downhole caliper data; and
- Depths of sampling intervals were checked for errors and inconsistencies against paper documentation from the field. Very few inconsistencies were found.

11 Sample Preparation, Analyses and Security

This section is abridged and modified from information contained in Fitzgerald et al., 2018.

11.1 Bulk Density Samples

Density has been measured on drill core samples of both country rock and kimberlite since initial drilling in 2009. Distribution of density measurements within drill holes has varied as drill programs progressed. In 2009 (on exploration level holes), measurements downhole were variable but was generally on the order of every 10 m within kimberlite and every 20 m within country rock. Between 2010 and 2015, measurement spacing varied between 3 m to 5 m downhole. In 2017, down hole distance between measurements was standardized to every 5 m downhole.

The majority of density measurements made on Chidliak core were made by Peregrine in the field following the methods of Lipton (2001). Competent, non-porous 10 cm to 15 cm pieces of geologically representative core were selected, their location downhole measured, the density determined and the piece returned to the core box (unless it was sent to a laboratory for testing due to reduced core competency or duplicate core analysis for QA/QC purposes). Density was measured by weighing the sample in air and weighing it again while suspended in water. Bulk density is calculated as:

$$\text{Bulk (wet) density} = [m_{air}/(m_{air}-m_{in\ H2O})]$$

where m_{air} is the sample mass measured on the scale and $m_{in\ H2O}$ is the sample mass measured in water (Lipton, 2001). Chidliak drill core can typically be considered “air-dried” by the time density measurements are recorded within the logging facility, and the data collected are accordingly assumed representative of dry bulk densities. This assumption was confirmed as valid based on results for samples sent for duplicate density assay, and by minimal moisture-content results obtained independently during assay of microdiamond samples.

Density measurements were primarily carried out under the supervision of Chief Geoscientist, Dr. Jennifer Pell.

Core samples that are porous/altered and samples selected for duplicate density analysis (approximately 7% of Peregrine samples) are labelled, packaged and sent to a laboratory. In 2011, these were sent to the SRC, an ISO 9001:2008 certified laboratory for quality assurance. From 2012 onwards, samples were sent to Bureau Veritas of Vancouver, B.C, an ISO 9001:2015 certified laboratory. Specific gravity is measured for these samples using a similar immersion method, except samples are first oven dried at 105 °C to remove all moisture and then allowed to cool. Porous/less-competent samples are wax-coated prior to measurement in order to maintain sample porosity and competency during suspension in water. Samples are weighed in air and then submerged in a container of water, the masses recorded and then specific gravity is calculated taking into consideration the temperature of the water at the time of measurement in order to determine density. The detection limit is 0.01 g/cm^3. Both specific gravity and density were reported by Bureau Veritas.

Due to the nature of the core (non-porous) and the excellent correlation with laboratory measurements, Peregrine densities are utilized as bulk dry densities. All material remaining after analysis was returned to Peregrine and is currently stored in a secure storage facility.

11.2 Microdiamond Samples Processed by Caustic Fusion

11.2.1 Sample Collection

The goal of discovery-stage caustic fusion sampling of a kimberlite is to collect sufficient, spatially representative material from major phases of kimberlite recognized during core logging, mapping or RC drilling to constrain the diamond size-frequency distribution (SFD) of that kimberlite phase. Discovery-stage core and small-diameter RC exploration drilling at Chidliak accordingly aspired to deliver a 200 kg microdiamond sample per major phase, with a 140 kg minimum weight threshold. In later years, it was established that small-diameter RC chip sample weights as low as 40 kg were sufficient to establish whether the sampled kimberlite required follow-up work or additional sampling with a core drill.

During 2009 and 2010, microdiamond samples from drill core were collected over the entire downhole interval of a given geological domain. Spatially representative lengths of core were aggregated from across the entire interval to be sampled, such that a total sample weight of approximately 200 kg was reached. For example, every second or third row of core throughout the interval may have been sampled, depending on the length of the entire interval, and all material within the row chosen was sampled, including gneissic country rock xenoliths and mantle xenoliths, if present. The downhole depths of sampled lengths were not recorded, only the downhole extent of the entire sampled interval. Samples were placed in polyurethane sample bags inside of 20 L plastic pails with tamper-proof lids and sealed with plastic cable ties and a security tag. A single sample number was assigned to the entire sample, typically weighing 200 kg.

In 2011 and subsequent years, microdiamond sampling of drill core proceeded on the same spatially representative basis for a given geological domain, but individual 8 kg aliquots were retrieved from core, individually numbered, and their exact downhole intervals recorded on a per-aliquot basis. An ideal 200 kg weight was made up of 25 spatially representative 8 kg aliquots, each with their own sample number. Gneissic or other xenoliths that fell within the interval sampled were typically included in the microdiamond samples. Rare intersections of avoidable country rock were excluded from the sampled interval as appropriate, and the proportionate weight of the excluded material was recorded so that it would be added to the total sample weight. Each 8 kg aliquot was placed into doubled polyurethane bags and both the inner and outer bags were sealed with plastic cable ties. The outer bag was then sealed with a uniquely numbered metal security tag and a unique sample number written on the outer bag. Bagged aliquots are placed in plastic pails with tamper-proof lids or in a bulk bag that is closed and sealed with a uniquely numbered metal security tag prior to shipping.

11.2.2 Sample Preparation and Analyses

All microdiamond samples have been processed at the SRC Diamond Recovery Laboratory. The SRC laboratory management system operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration laboratories, and is accredited for microdiamond recovery by the Standards Council of Canada under ISO/IEC 17025:2005.

The standard method for processing samples to recover diamonds +0.106 mm by caustic fusion involves:

- Drying and weighing samples, followed by crushing with a 0.50 in gap;
- Adding tracers and fusing the 8 kg aliquots in kilns with NaOH by heating the kilns to 550 °C for 40 hours;
- Screening the hot, liquid NaOH-sample mixture over a +0.075 mm or +0.106 mm square-mesh screen under mild negative-pressure conditions;
- Soaking the screened product in water to remove any remaining caustic and trapped material;
- Water is again poured through the +0.106 mm screen and retained residue is rinsed and treated with acid to dissolve soluble materials;
- Additional tracers (+0.106 mm size) are added;
- The sample is transferred to a zirconium crucible and fused again with NaOH to remove any remaining minerals other than diamond from the sample;
- Remaining residue is wet-screened into microdiamond size classes and sized material stored in plastic vials containing methanol; and
- Trained observers use microscopes to recover and document the natural diamonds and tracers from each size class. From 2010 onwards, Peregrine has chosen not to have the -0.106 mm fraction observed.

For the 2010 CH-6 mini-bulk sample, caustic fusion analysis used the same procedures. However, the processing used a bottom cut off screen size of 0.425 mm. A flow chart of the SRC microdiamond process is depicted in Appendix 2, Figure 1.

11.2.3 Sample Security and Chain of Custody

The sample processing facility at the SRC is a locked facility under 24-hour video surveillance operated and managed by in-house security personnel. In addition, the Diamond Observation Laboratory is also monitored, in part by an outside security agency.

All sample transport was carried out with containers that were locked and secured with uniquely numbered seals. Chain of custody documentation is maintained from the time of sample collection to receipt by the laboratory. Additional chain of custody documentation is employed for receipt of diamonds and residues by Peregrine from the SRC.

11.2.4 Quality Assurance / Quality Control

The SRC monitors the quality of the caustic fusion method by assessing the recoveries of synthetic diamonds added to the sample during the caustic fusion and chemical treatment processes. The method allows for 95% confidence of recoveries of 80% or better. Samples are spiked with up to two sets of synthetic diamonds and results show 22,795 of the 23,005 spikes placed in CH-6, CH-7 and CH-44 samples between 2009 and 2017 were recovered, for a recovery rate of 99.1%.

The method for observing and picking diamonds is based on Canadian Institute of Mining and Metallurgy (CIM) guidelines for reporting diamond results and on documented in-house procedures that ensure that all diamonds have been recovered. The weighing of stones is performed using Ultra Micro Analytical balances, which have scheduled external ISO/IEC 1725:2005 calibrations and daily calibration checks to assure reproducibility to within 0.2 x 10-6 gram.

Once diamond recovery is complete, diamonds and remaining residues are returned to Peregrine using a secure transportation provider and all material is stored securely.

11.3 Commercial-sized Diamond Samples

11.3.1 Commercial-sized Diamond Samples from Drill Core

During July 2010, a 14 t mini-bulk sample of the CH-6 kimberlite was established by aggregation of 63.5 mm diameter (HQ) and some 47.6 mm diameter (NQ) drill core. The mini-bulk sample provided a convenient and permissive means to test for commercial-size diamonds across a significant portion of the CH-6 pipe, and to depths of 325 mbs. Additional details of this program are given in Section 11.6.4.

After drilling of HQ-diameter holes at CH-6 completed in June 2010, the core was transferred from Discovery camp via Twin Otter aircraft to a secure logging facility in Iqaluit. The drill core was logged and sampled as per standard Peregrine protocol and remaining kimberlite core was broken into smaller pieces and collected into 16 double-layered 1 t ore bags. Both inner and outer bags were labelled with unique numbers prior to filling, and once filled both bags were closed and the external bag was sealed with a tamper-proof security seal. Bags were combined into five processing units based on geology and depth in the pipe (Pell, 2010c). The sample was transported to the SRC in Saskatoon via First Air 767 Charter aircraft to be processed by DMS at the SRC. Concurrent with sampling the core for commercial-size diamonds, microdiamond samples were also collected, the purpose of which was to assess DMS processing efficiency and to measure moisture content.

11.3.2 Large-diameter Drilling Samples

Doubled-up ore bulk bags filled during the 2015 LDD RC campaign were sealed with uniquely numbered, tamper-proof security seals under closed-captioned television coverage in the sample collection area at the LDD rig. Relevant records captured in a spreadsheet at the rig by the drill geologist include the drill hole, bag number, weight, depth interval, drilling method, bit type, tracer information, security tag number, date, time and shift geologist identity.

All 653 bags were transported overland on camp winter trails from the CH-7 site to either the airstrip at Discovery camp or the ice airstrip at Sunrise Lake, in order to be transported to Iqaluit. Bag transport from the field was completed primarily via 767 or DC-3T, with a few bags also transported by helicopter. Once received in Iqaluit, bags were immediately loaded into uniquely numbered sea containers at the Iqaluit airport, locked, sealed with a uniquely numbered security tag and then transported to the beach in order to await the summer sealift. Sea containers were placed end-to-end in order to limit access while on the beach. A total of 62 sea containers were filled and shipped to Montreal via two sealifts in August 2015. A Peregrine representative received the containers in Montreal, verified seals and checked the condition of bags before loading them onto 16 trailers for road transport to the SRC in Saskatoon. Once the trailers were full, they were sealed with uniquely numbered security seals.

Throughout the bulk sample program QA/QC protocols were implemented to ensure the integrity of the sample. Procedures at site during sample collection and shipment included:

- Screen inspections and fines tests once per shift;
- Insertion of natural diamonds tracers during sampling in order to monitor DMS processing (discussed further in Section 11.3.4);
- Closed circuit television recorded all activities while the drill was operational;
- Access restrictions to the sampling area of the rig;
- Sealing of full sample bags with uniquely numbered locking cable ties;
- Data verification on site; and
- Chain of custody documentation maintained as sample bags were transferred off site.

The granulometry and production rates observed using the tungsten carbide insert bits was considered excellent. Granulometry was tracked in order to maintain the production of acceptable drilling product. The data for the coarse drill cuttings showed a relatively coarse sample with little evidence of grinding to fines and good proportions of kimberlite fragments greater than 6 mm to 12 mm.

11.3.3 Commercial-size Diamond Processing and Recovery

Commercial-size diamond processing has been completed primarily at the SRC, with the exception of the majority of the 2013 CH-6 bulk sample that was processed at the De Beers DMS facility in Sudbury, Ontario. This facility is a privately owned and operated DMS processing plant and operates under strict internal safety and QA/QC protocols in order to achieve reliable results.

11.3.3.1 CH-7 LDD Bulk Sample Processing

The CH-7 bulk sample was processed at the SRC using their 5 t/h DMS plant in late summer 2015. At the time, the SRC was ISO 17025:2005 accredited for caustic fusion processing but was not accredited for DMS processing and recovery for commercial-size diamonds. However, the SRC and Peregrine both employed QA/QC protocols for diamond processing and recovery and the entire process was under the supervision of QP Howard Coopersmith. The only sample preparation that occurred prior to processing was the screening of the RC chips during sampling in the field to +1.13 mm prior to collection in sample bags and the addition of natural diamond tracers to the sample which is detailed in Section 11.3.4.

Upon arrival at the SRC sample bags were off-loaded, inspected, weighed, reconciled with the extant chain of custody documents and stored in a secure yard. All bags received were in good condition and not in need of repair and all inner bag seals were intact, showed no signs of tampering and matched the shipping manifesto provided by Peregrine (McCubbing and Coopersmith, 2016). Bags were composited together in a pre-determined fashion (Nowicki et al., 2016) to produce separate processing units reflecting geologically relevant intervals.

The bulk sample was processed continuously over two-week periods, one process unit at a time with plant flush cleaning (“soft cleans”) completed between processing units and thorough, invasive plant cleans (“hard cleans”) between geological units. Bags were transferred to the DMS plant building from the secure storage yard using a 6 t forklift and remained sealed until just prior to processing. Bags were weighed, seals recorded and removed, and the bag material was then either loaded into the hopper (if dry) or washed and fed into the scrubber (if wet) to go directly into the plant. The scrubber was 3 m long with a diameter of 1.2

m and rotated constantly at 15 rpm. The material was split on a 12.5 mm punch plate trommel screen with the undersize material dropping to a sump for pumping directly to the feed preparation screen. The +12.5 mm material was fed through a two stage crusher (10 mm gap), and then back to the scrubber (McCubbing and Coopersmith, 2016).

Sized material (+0.85 mm to -12.5 mm) from the scrubber was fed onto a 2440 mm x 915 mm vibrating feed preparation screen that was fitted with 1.0 mm by 17 mm slotted aperture poly screen panels. The sample is then gravity fed into the mixing box and mixed with ferrosilicon, with the mixed dense-media product fed to the 150 mm cyclone. A minimum cut point of 3.00 g/cm^3 was deemed acceptable for the sample treatment based on plant testing, and a cut point of between 3.10 g/cm^3 and 3.20 g/cm^3 was maintained throughout sample treatment. The mixed product is then discharged over 0.6 mm wedge-wire screens with the sinks gravity fed to a can inside a sealed and double-locked rotating cage capable of holding 4 x 20 L securable plastic pails within a double locked glove cage. The float product is gravity fed to a 1,830 mm x 610 mm dewatering screen fitted with a 12 mm x 305 mm poly panel with 6.7 mm square aperture screens. The +6.7 mm float material drops into a feed bin for re-crushing via high pressure grinding rolls (HPGR) with a setting of 4 mm at 55 bar. The -6.7 mm material was collected into a marked double-walled 2 t ore bag for tailings collection, and once filled, was sealed with a uniquely numbered cable seal, weighed and transferred to the secure storage compound (McCubbing and Coopersmith, 2016). A flow chart of the SRC DMS process is depicted in Appendix 2, Figure 2.

Once a concentrate pail was full it was closed and secured with a uniquely numbered security seal internal to the cage via a gloved opening. Pails were then moved from the cage, weighed, numbered and moved to the secure concentrate storage area (McCubbing and Coopersmith, 2016) until sorting was to begin. Concentrates were treated through the recovery circuit as described in Section 11.3.3.5.

The CH-7 material generally treated well with good material flow, scrubbing and crushing and liberation appeared to be excellent (McCubbing and Coopersmith, 2016). Large amounts of clay in certain portions of the sample caused minor processing issues, which, in a production scenario, would be easily addressed by ore blending, sufficient scrubbing and design of slimes handling (Coopersmith, 2016). Sample granulometry showed that the sample treated very well with minimal diamond breakage and quite high diamond liberation. Processing produced 17,201 kg of concentrate, with high DMS yields, often averaging 4% and consisting of large amounts of olivine. Samples from KIM-5 had anomalously high yield of over 10% at times (McCubbing and Coopersmith, 2016). Grease collection for the heavily clay-altered KIM-5 material was hindered due to hydrophilic surfaces on the diamonds in this material. These diamonds were recovered well by the X-ray with only the smallest diamonds (from X-ray tailings) being repelled by grease (Coopersmith, 2016). An extensive audit of the tailings resulted in complete recovery of these refractory diamonds (McCubbing and Coopersmith, 2016).

Coopersmith (2016) noted that the SRC DMS plant operations proceeded normally and the plant was fairly consistent in maintaining a set density and producing quality concentrate with minimal loss of heavy materials. The diamond recovery efficiency appeared to be very high with nominal loss in the small size fraction. Recovery of diamond from the X-ray tails on the grease table may be low, indicating a refractory diamond issue from the surficial weathered kimberlite.

The SRC has QA/QC procedures in place for diamond processing and recovery (refer to Section 11.3.4). Almost the entire process was observed by QP Howard Coopersmith. There were no noted issues with processing, recovery, sorting and reporting (Coopersmith, 2016).

11.3.3.2 CH-6 Bulk Sample Processing – 2013

Of the 404.31 t (dry) of kimberlite collected for the CH-6 bulk sample, 8.41 t were processed as a test sample at the SRC using their 5 t/h DMS (Coopersmith, 2013). The majority of the sample, 395.9 t, was processed at the De Beers 5 t/h DMS facility. At the time, the SRC was ISO 17025 accredited for caustic fusion processing but was not accredited for DMS processing and recovery of commercial-size diamonds. The De Beers facility is privately run and is also not accredited, however, both facilities and Peregrine employed strict QA/QC protocols for diamond processing and recovery. Both facilities are run professionally and the majority of the processing and recovery was under the supervision of Independent QP Howard Coopersmith. No sample preparation occurred prior to processing.

The purpose of the 8.41 t test sample, completed at the SRC in July, 2013, was to determine how well the kimberlite would process in light of its altered nature. The primary goals of the test were to:

- Determine how the trench material fed out of the bulk bags;
- Determine whether primary crushing would be required;
- Ascertain best practices for secondary crushing and re-crushing;
- Observe material liberation characteristics;
- Estimate clay content and optimize handling of clay-rich feed;
- Estimate optimum plant throughputs;
- Obtain moisture contents;
- Estimate heavy mineral concentrate yield ; and
- Design sample treatment protocols for the remaining material to be treated at the De Beers treatment facility.

Upon arrival at the SRC, all bags' conditions and security seals were checked, and bags were weighed. Primary crushing was not required due to the weathered nature of the kimberlite. Material from individual bags was fed into the DMS and subjected to scrubbing and secondary crushing. Any large gneissic pieces were removed by hand by SRC staff and inspected by QP Howard Coopersmith as the sample was fed into the plant. Plant configuration was the same as in 2010 with the exception of having only a single trommel and a single larger cone crusher (10 mm gap) for secondary crushing (Coopersmith, 2013). The final heavy mineral concentrate was treated through the SRC diamond recovery circuit.

The remaining 395.9 t of the sample was treated at the De Beers facility during September through November 2013 using a process identical to that of the SRC (Thomson, 2013). The sample was separated into six processing units and processed one at a time. A primary crushing stage was not required, therefore bags were emptied directly into the scrubber and was then split on a 14 mm woven wire trommel screen with the undersize material dropping to a sump for pumping directly to a feed preparation screen fitted with polyurethane panels with 1.0 mm square aperture openings. The +14.0 mm material dropped through a 4 in x 6 in jaw crusher set at 12 mm with the crushed oversized material recycled back to the scrubber. Oversized country rock was removed by hand from the process at the scrubber feed under the supervision of QP Howard Coopersmith. The material is gravity fed into a ferrosilicon mixing box with the resulting dense-media slurry fed into the 200 mm cyclone. The resulting floats were screened and the -7.1+1.0 mm washed product was discharged into bulk bags for weighing and storage (available for future audits if necessary) while the +7.1 mm oversize was re-crushed via 9 ft by 12 ft double roll crusher set at 5 mm. The sinks (i.e. DMS concentrate) were washed and screened to +1.0 mm, with the +1.0-12.5 mm fraction being gravity fed to 20 L concentrate pails, located within a secure cage (Thomson, 2013). A flow chart of the De Beers DMS process is depicted in Appendix 2, Figure 3. As part of routine sample monitoring, De Beers staff collected small samples for granulometry and moisture-content analysis. The material was removed and returned to the processing circuit under supervision of QP Howard Coopersmith (Thomson, 2013).

DMS concentrates were collected in secure containers in a locked concentrate cage. When cans were full they were closed with locking can rings and uniquely numbered security sealed in the cage through gloved openings. Concentrate cans were moved to a secure area prior to being shipped via Brinks to the SRC for diamond recovery (Thomson, 2013). Treatment of the concentrate through the recovery circuit at SRC is described in Section 11.3.3.5.

The CH-6 material treated very well with good material flow, scrubbing and crushing results and good diamond liberation. A total of 1,920.37 kg of DMS concentrate was produced, representing a 0.41% yield (Thomson, 2013). The weathered nature of the material allowed for a high rate of processing as approximately 75% of the sample by weight reported to undersized tails which required extensive slimes handling but was not problematic. The diamond recovery efficiency was very high with nominal loss at the small size fraction. Recovery of diamond from the X-ray tails on the grease table may have been low, indicating a refractory diamond issue similar to that observed in weathered material from CH-7 (Coopersmith, 2016). However, any unrecovered diamonds would likely be small and of low quality, therefore not affecting revenue.

The SRC and De Beers facilities have QA/QC procedures in place for diamond processing and the SRC for recovery. Almost the entire process was observed by QP Howard Coopersmith.

11.3.3.3 CH-7 Mini-Bulk Sample Processing – 2010

The CH-7 KIM-1 mini-bulk sample was processed at the SRC using their 5 t/h DMS plant in late summer 2010. At the time, the SRC was ISO 17025:2005 accredited for caustic fusion processing but was not accredited for DMS processing and recovery of commercial-size diamonds. However, the SRC and Peregrine both employed QA/QC protocols for diamond processing and recovery and the entire process was under the supervision of QP Howard Coopersmith. No sample preparation occurred prior to crushing and processing other than the addition of natural diamond tracers as discussed in Section 11.3.4.

Upon arrival at the SRC, all bag security seals were checked, bags were weighed and then the material was fed into the 400 mm x 250 mm jaw crusher (set with a 30 mm gap). Crushed material fell into a new ore bag and once full, was closed and sealed with a uniquely numbered security seal and stored on site securely until it was time for further processing (McCubbing, 2011a).

Bags of crushed kimberlite were composited into four processing units simply to mitigate against DMS problems compromising the entire sample. Excess microdiamond material collected but not processed by caustic fusion was crushed and added into the DMS sample. Each processing unit was treated one at a time with plant flushes in between. Material was scrubbed and then split on a 12.5 mm punch plate trommel screen with the undersize material dropping to a sump for pumping directly to a feed preparation screen fitted with 12 mm by 0.85 mm aperture panels. The +12.5 mm material dropped through a two stage crusher (10 mm gap), and was fed back to the scrubber (McCubbing, 2011a).

Sized material (+0.85 mm to -12.50 mm) and scrubbed material was washed and mixed with ferrosilicon, with the mixed dense-media product fed to the 150 mm cyclone. The mixed product is then discharged over 0.6 mm wedge-wire screens with the sinks gravity fed to a can inside a sealed and double-locked cage. The float product is pumped to a tails screen where -6 mm material drops into an ore bag of coarse plant tails, which is sealed, numbered and weighed for storage. The +6 mm float material drops into a feed bin for re-crushing via HPGR with a setting of 4 mm at 65 bar. Re-crushed HPGR product was pumped back to the scrubber for re-processing (McCubbing, 2011a).

DMS concentration was performed on +0.850-12.00 mm feed material and resulting heavy mineral concentrate is fed vial sealed tubes into cans in a locked carousel cage. Once a can is full it is closed and secured with a security seal internal to the cage through gloved opening. Cans are then moved to a secure room prior to diamond recovery (McCubbing, 2011a). The DMS concentrates were treated through the recovery circuit as described in Section 11.3.3.5.

Coopersmith (2011a) noted that the CH-7 material treated easily with good material flow, scrubbing and crushing results with excellent diamond liberation. Processing produced 928.95 kg of concentrate, representing a 1.85% yield. Although the diamond recovery efficiency appears to be sufficiently high, there was nominal loss at the small size fraction.

The SRC has QA/QC procedures in place for diamond processing and recovery. Almost the entire process was observed by QP Howard Coopersmith. There were no noted issues with processing, recovery, sorting and reporting.

11.3.3.4 CH-6 Mini-Bulk Sample Processing – 2010

The CH-6 mini-bulk sample was processed at the SRC using their 5 t/h DMS plant in the fall of 2010 subsequent to the processing of the CH-7 mini-bulk sample. No sample preparation occurred prior to crushing and processing by the SRC and the same process and parameters were used to complete the CH-6 sample processing as was used for CH-7, which is documented in detail in Section 11.3.3.3.

The DMS concentrates were treated through the recovery circuit (as described in Section 11.3.3.5) in the same manner as samples from the CH-7 mini-bulk sample. However, processing units 10B-2 and 10B-4 produced minor concentrate volumes, resulting in a straightforward caustic fusion finish immediately prior to final diamond recovery (McCubbing, 2011b).

Coopersmith (2011b) noted that the CH-6 material treated easily with good material flow, scrubbing and crushing results with excellent diamond liberation. Processing produced 127.15 kg of concentrate, representing a 0.87% yield. Although the diamond recovery efficiency appears to be sufficiently high, there was nominal loss at the small size fraction.

The SRC has QA/QC procedures in place for diamond processing and recovery. Almost the entire process was observed by QP Howard Coopersmith. There were no noted issues with processing, recovery, sorting and reporting.

In addition to the material processed by DMS, a 157.80 kg sample of carbonate breccia collected from within KIM-L was processed by caustic fusion to determine the diamond content of mantle xenoliths that occur within the carbonate breccia. No commercial sized diamonds were recovered from this material although the 157.8 kg sample weight was combined into the total weight of the 10B-2 process unit.

11.3.3.5 Commercial-size Diamond Recovery at SRC

For all commercial-size diamond testing programs, DMS concentrate was processed at the SRC for diamond recovery using an X-ray and grease table recovery circuit. This system consists of a feed hopper, sizing screens, dewatering process, twin stage X-ray unit, secured concentrate unit, grease table and tailings capture bins. Once sealed pails of DMS concentrate are received at the SRC diamond recovery section and the seals verified, seals and lids are removed and the concentrate material is loaded into the primary feed hopper. From there it is wet-screened into the following size classes:

- In 2010, sizing was +0.85 mm, +3.0 mm, +6.0 mm and -0.85 mm square-mesh size (McCubbing, 2011a and 2011b);
- In 2013, sizing was +0.75 mm, +2.0 mm, +4.0 mm, +6.0 mm and -12.5 mm square-mesh size (McCubbing, 2014); and
- In 2015, sizing was +0.50 mm, +2.0 mm, +4.0 mm, +6.0 mm and -12.5 mm square-mesh size (McCubbing and Coopersmith 2016).

Gravity fed sized fractions are then de-watered over a vibrating wedge wire screen (+0.85 mm in 2010, +0.67 mm in 2013 and +0.85 mm in 2015), then passed through two “in-series” X-ray fluorescence units, with X-ray luminescence parameters set according to the size of material being treated. Luminescing diamonds are ejected and gravity fed over a wedge-wire dewatering screen to an infrared dryer, then passed into a secure concentrate pail within a glove box cage. All X-ray feed and X-ray unit controls are controlled by the primary operator through use of a control panel located outside the enclosure of the secured process equipment. Any fines that passed through dewatering screens were captured in a -0.5 mm screened sump trap, processed by caustic fusion and reported as part of the batch cleanup. The +6.0 mm X-ray tailings are fed to an oversize collection pail, which was then dried, and hand sorted. The -6.0 mm X-ray tailings are gravity fed to a grease table, where any captured material is hand-scraped from the deck and placed in 200 mm diameter stainless steel +0.85 mm square mesh sieves. The sieves are then placed in a tray over a locked catch pan in an oven at 80°C overnight to melt the grease. The next day, the sieves are removed from the oven and taken to the secure sorting area for a bath with hot water and a degreasing agent for final cleanup. The solids are placed in a petri dish and then securely sealed inside a polyethylene bag that is locked in the secure sorting area to await the final hand sort. The recovery tails from this process are stored for future auditing, if required.

Final hand sorting consisted of secure transfer of X-ray and grease table concentrates to the sorting lab. Due to the large amount of concentrate in 2013, the -2.00 mm X-ray concentrate was subjected to a caustic fusion finish prior to hand sorting in a sealed glove box (McCubbing, 2014). Grease table solids were hand sorted using a binocular bench top microscope. All concentrates were stored in a double locked cage in the secure sorting area until they were sorted.

A flow chart of the SRC X-ray recovery process is depicted in Appendix 2, Figure 4.

For the 2015 CH-7 bulk sample, average yields of 0.15% for the X-ray concentrate, 0.02% for the grease table concentrate and 2.59% for the +6 mm fraction were recovered from the 59 processing units (McCubbing and Coopersmith, 2016). In general, grease table concentration worked well with little mineral matter other than diamond adhering to the grease (McCubbing and Coopersmith, 2016). Unit KIM-5 did not initially show sufficient grease diamond recovery and the grease tails were subject to an audit as detailed in Section 11.3.4.1.

For the 2013 CH-6 bulk sample, DMS concentrates from both the SRC test and the De Beers plant were processed at the SRC diamond recovery lab (McCubbing, 2014). The recovery of this sample showed good amenability to the recovery technique. Overall, 12% of carats of diamond were recovered by grease, however the grease-recovered stones were small, averaging 0.03 cts in weight (Coopersmith, 2014).

For the 2010 CH-7 mini-bulk sample, the recovery of the sample showed good amenability to the recovery technique, as X-ray ejections and concentrate size were minimal. Overall less than 1% of carats of diamond were recovered by grease, which represents a lower than expected grease recovery (Coopersmith, 2011a).

For the 2010 CH-6 mini-bulk sample, the X-ray ejections and concentrate size were minimal and a good concentration was made. Overall, 18% of carats of diamond were recovered via grease, with most diamonds being small at 0.05 cts in weight. However, one 0.66 ct stone was recovered by this method, showing the importance of this stage in diamond recovery (Coopersmith, 2011b).

11.3.3.6 Commercial-size Diamond Sample Security and Chain of Custody

Measures taken to ensure security and validity of commercial-size diamond samples during DMS processing and diamond recovery include:

- Processing facilities are secured sites with controlled access;
- 24 hour security and video surveillance during diamond recovery;
- Fenced and 24 hour video surveillance of outside sample storage areas;
- Security officer present during processing and sorting;
- Dual custody, comprising a security officer and senior operating personnel, is required at all times for handling of sample material or concentrate, and for seals and locks and for access to concentrate areas and high risk processing equipment;
- Restricted personnel access and records maintained of all visitors and personnel present;
- Uniquely numbered padlock seals used on all metal pails, glove boxes and sample containers;
- Security maintained seal register and log of concentrates with logs reconciled upon completion;
- Independent QP or Peregrine QP periodically on site during processing and diamond recovery to monitor the process;
- Concentrate and diamond handling performed inside locked and sealed glove boxes;
- Concentrate fractions undergo detailed weight reconciliations;
- Weights of concentrate reconciled pre- and post-sorting and must be within 2%;
- Uniquely numbered seals utilized on concentrate containers and verified; and

- Chain of custody documentation maintained throughout sample collection, delivery, processing and recovery.

All shipment of DMS concentrates and diamonds are undertaken by Brinks Canada with strict chain of custody documentation and in containers secured with numbered seals. For shipments of diamonds to Antwerp, Kimberley Process chain of custody documentation was maintained.

11.3.4 Commercial-sized Diamond Sampling Quality Assurance / Quality Control

QA/QC measures undertaken on all commercial-size diamond samples include:

- Adherence to documented processing and handling protocols;
- Addition of identifiable natural diamond tracers to samples prior to processing, both in the field and at the processing plant to determine recovery efficiency;
- Addition of synthetic tracers with a density of 3.53 g/cm^3 to some samples prior to DMS processing in some tests to ensure density cut points maintained during processing;
- Plant inspection prior to and during processing by trained personnel;
- Independent third party process and recovery monitoring and auditing;
- Recording of DMS operating parameters during processing (moisture measurements, screening analysis of head feed, operating medium pressure at the cyclone, medium density, operational time and motion information, ore dressing studies);
- Daily testing the DMS operating efficiency with density tracers and auditing of these tracer tests by an independent third party;
- Audit of representative coarse DMS tailings from select samples as necessary;
- Monitoring of diamond recovery statistics, including size frequency analysis; and
- Review and audit of DMS and diamond data, operating procedures and QA/QC programs.

Peregrine routinely uses natural diamonds to exercise appropriate and efficient sample QA/QC during bulk sample programs. The company has established an inventory of natural diamond tracers, ranging from 0.09 ct to 1.62 ct in weight that are susceptible to X-ray capture and with serial numbers laser-inscribed on a polished face. The diamond tracers were added to random bulk sample bags in the field (CH-7 mini-bulk and bulk samples and CH-6 bulk sample), at a core logging facility (CH-6 mini-bulk sample) and in some cases, additionally at the DMS plant prior to processing. Details regarding tracer addition for each sample are as follows:

- 2015 CH-7 LDD bulk sample: A total of 266 laser-etched diamond tracers ranging in size from 0.09 ct to 1.62 ct were added randomly into sample bags during sample collection by Peregrine. An additional 14 laser-etched diamond tracers were added to sample bags at the SRC DMS plant by QP Howard Coopersmith prior to processing. All 280 tracers were recovered (McCubbing and Coopersmith, 2016) as whole, unbroken stones during sorting at the SRC, a 100 percent recovery rate;

- 2010 CH-7 KIM-1 mini-bulk sample: A total of 111 laser-etched diamond tracers ranging in size from 0.2 ct to 1.62 ct were added to ore bags during sample collection (Holmes, 2010). In addition, 120 blue synthetic tracers with a density of 3.53 g/cm^3 were added to the sample at the SRC prior to DMS processing (Coopersmith, 2011a). All diamond and density tracers were recovered;
- 2013 CH-6 surface bulk sample: A total of 140 laser-etched diamond tracers were added by Peregrine to sample bags in the field during collection (Pell and O'Connor, 2013). An additional 125 laser-etched diamond tracers were added to bags randomly as they were opened at the DMS and ten diamond tracers were added to the X-ray feed by QP Howard Coopersmith (Coopersmith, 2014). Tracers ranged in size from 0.09 ct to 1.62 ct and were previously calibrated to ensure susceptibility to X-ray capture. All except one of the tracers were recovered (0.16 ct stone lost, placed in X-ray feed), for a total recovery of 274 of 275, or 99.6%; and
- 2010 CH-6 core mini-bulk sample: A total of 40 laser-etched diamond tracers ranging in size from 0.26 ct to 1.62 ct were randomly inserted into sample bags by Peregrine at the time of collection (Pell, 2010c). In addition, 35 laser-etched diamond tracers ranging in size from 0.14 to 4.74 cts were added to the scrubber feed by QP Howard Coopersmith at the SRC during processing (Coopersmith, 2011b). Peregrine also added 45 blue synthetic density tracers with a density of 3.53 g/cm^3 to the sample prior to DMS processing. All natural diamond and density tracers were recovered.

The coarse DMS float tails, DMS and recovery concentrate tails and most hand sorted recovery concentrates are stored for periods up to two years, thereby allowing timely audits of diamond recovery efficiency.

Both the SRC and De Beers have QA/QC procedures in place for recovery of diamonds by DMS and related final-recovery processes. Such facilities are governed by a series of detailed procedures that are appropriate to ensure the security and integrity of samples and final results. All samples received in the laboratory are accompanied by chain of custody documentation and with security seals that must be verified prior to sample processing. Upon receipt, the samples are stored in a secure facility with restricted access.

SRC employs strict QA/QC protocols for its diamond recovery process circuit. The X-ray machine is calibrated each day and tested with luminosity index tracers using predetermined settings to determine recovery rate. The temperature of the process water for the grease table is maintained at 25°C automatically. However, no specific grease testing is undertaken by SRC. QP Howard Coopersmith tested the grease with grease specific tracers in 2010 and all were recovered (Coopersmith, 2011a). The diamond recovery circuits are in restricted areas and all samples, concentrates, diamonds and data are locked in safes.

Various measures were implemented during sample processing in order to prevent sample contamination. The DMS circuit was thoroughly cleaned prior to sample processing and after each processing unit was complete. An extra thorough clean was completed between processing of different geological units. Any minor spillage that occurred was collected with security personnel present and reintroduced into the plant with the corresponding processing unit. Screens were cleaned and un-blinded by spraying and scraping. The scrubber was reversed and a corkscrew inside the drum pushed any remaining material forward to the pump box and into the plant. The plant was run for at least thirty minutes without a load to ensure a proper flush of the circuit and to prevent cross-contamination of samples (McCubbing and Coopersmith, 2016). Recovery circuit clean-ups were done between each processing unit also and consisted of de-pegging screens, surging feeders and a wash down of screens and feeders in order to flush the circuit clean. The circuit was run without material for at least 45 minutes before introducing the next processing unit. Once all material had passed through the circuit, a final clean-up was performed (McCubbing and Coopersmith, 2016).

In the SRC diamond recovery laboratory multiple sorts of each sample by separate trained sorters is undertaken to ensure recovery of all diamonds. Each fraction receives at least one clean pass (i.e. no diamonds) by a second sorter. Samples are weighed once they are put into the gloved box for sorting at shift start and again at shift end. Recovered diamonds were sealed in small bags and stored in sealed containers prior to removal from the glove box. For diamond sorting, either QP Howard Coopersmith or Peregrine representatives were present.

11.3.4.1 Audits

Audits of DMS tailings and recovery circuit tailings can be undertaken to benchmark or re-affirm diamond processing and/or recovery efficiencies. Peregrine has repeatedly used the SRC facility and the company is familiar with processing outcomes for six (CH-1, CH-7, CH-6, CH-28, CH-6, CH-7) separate bulk samples treated there since 2010. A comprehensive DMS tailings and recovery circuit tailings audit performed for the CH-1 mini-bulk sample in 2010 established industry-appropriate or better diamond recovery efficiencies for the SRC facility. All but one of subsequent bulk sample processing outcomes fell within diamond recovery parameters expected for this plant and Peregrine's QP for diamond processing endorsed that full audits were not required, nor were any performed, for:

- 2010 CH-6 mini-bulk sample from core;
- 2010 CH-7 mini-bulk sample from surface of KIM-1;
- 2013 CH-6 bulk sample from surface; and
- All, except three of 59 processing units in the 2015 CH-7 LDD bulk sample.

A comprehensive recovery tails audit of all three processing units from the 2015 CH-7 bulk sample hole CHI-251-15-LD05 was ordered after unusually low grease recoveries were noted. The audit comprised three separate, controlled pathways, with all three pathways ending in a caustic fusion finish. The audit revealed that diamonds from LD05 were properly liberated and properly ejected by X-ray fluorescence circuits, and the grease table itself was working properly. However, substantial smaller-sieve diamonds failed to be captured on grease due to the presence on them of a hydrophilic (grease-repellent) coating. The coating is tentatively related to lateritic weathering experienced by the kimberlite in this hole. The post-grease tails audit captured an extra 10.11 carats of diamond, representing a 20% carat-weight uplift over the 49.98 cts recovered prior to the audit (Table 11-1). DMS and recovery circuit tails from the 2015 CH-7 bulk sample have been disposed.

Table 11-1: Results of Caustic Fusion Audit of Hole CHI-251-15-LD05

Processing Unit		15-5A	15-5B	15-5C	TOTAL
Number of Diamonds per Sieve Size (mm square mesh sieve)	+1.180 mm	54	34	63	151
	+1.700 mm	17	6	15	38
	+2.360 mm	6	1	5	12
	+3.350 mm	0	0	0	0
	+4.750 mm	0	0	0	0
	+6.700 mm	0	0	0	0
Total Number of Diamonds		77	41	83	201
Carats >1.180 mm		4.31	1.75	4.05	10.11

11.3.5 Commercial-size Diamond Breakage

11.3.5.1 CH-7 Diamond Reconstruction and Breakage Study

During December 2015, Peregrine commissioned a reconstruction and related breakage study for commercial-sized diamonds obtained during the 2015 LDD RC bulk sampling of the CH-7 kimberlite. The study was triggered by “unusually high” breakage commentary from three informed observers and was completed independently by Dr. Tom McCandless. Dr. McCandless noted damage on 75% to 90% of some 692 cts of diamonds examined, and his efforts to reconstruct whole, +0.66 ct diamonds from visually comparable fragments contained in 54 separate process units were partially successful in 43 of 74 attempts. Estimates of the missing portions of partially reconstructed diamonds support a conclusion that 10% to 40% loss of carat weight occurred due to -1.13 mm diamond fragments reporting to undersize slimes (McCandless, 2016a). The data collected indicated that the diamond breakage occurred predominantly during large-diameter RC drilling. Subsequent cross-correlation of breakage data and drilling parameters suggested the diamond breakage was likely influenced by the competence (or “hardness”) of the kimberlite being drilled (McCandless, 2016b).

11.3.5.2 CH-7 Diamond Breakage Study

The December 2015 reconstruction study by Dr. McCandless was followed up in March 2016 by a conventional (i.e. non-reconstructive) breakage study, again performed by Dr. McCandless (McCandless, 2016c). In this study, a total of 651 carats of diamonds in the +5 DTC size class were examined and categorized by DTC sieve class within each geological domain, such that the incidence of breakage could be compared to industry-standard breakage studies, which are typically recorded and analyzed by sieve class (i.e. not by process unit, and not based on diamond reconstructions). The results showed:

- The CH-7 KIM-1 (surface bulk sample) contains over 21% fragmented diamonds, similar to diamonds from CH-1 (surface bulk sample) examined in 2010 (McCandless, 2010). Mechanical breakage is nearly absent in the fragments;
- More abundant fragments in KIM-2 through KIM-5 (32%) due to their recovery by RC drilling. Mechanical breakage ranges from 7% to 36% with the greatest breakage occurring in KIM-3 (36%) and KIM-4 (28%);

- Fragmented diamond abundance increases with decreasing size class coinciding with a dramatic increase in mechanical breakage surfaces on fragments for RC-collected samples in KIM-2 through KIM-5 (further evidence of breakage from RC drilling); and
- Carat-weight loss related to recovery by LDD drilling (i.e. in excess of that incurred for bulk samples not derived by LDD drilling) is estimated at between 8% and 15% for all diamonds in and larger than the DTC 1 size class.

12 Data Verification

This section is updated from information contained in Fitzgerald et al., 2018.

Peregrine Diamonds maintains in-house databases, software and related reports that capture and describe the growth of geological and related data for the Chidliak Project since exploration work initiated in June 2005. Active or curated data sets related to such work cover:

- Heavy mineral samples and compositions of kimberlitic indicator minerals;
- High-resolution airborne and ground geophysical surveys, utilizing magnetic, electromagnetic and gravimetric methods;
- Core and small-diameter RC drilling for discovery, delineation and sampling of kimberlites;
- Processing and/or assay of samples of drill core for microdiamonds, bulk density, moisture content, geotechnical properties, acid rock drainage (ARD) potential, petrography, whole rock geochemistry and kimberlite groundmass mineral compositions;
- Mini-bulk sampling of kimberlite through surface trenching or core drilling;
- Bulk sampling of kimberlite through surface trenching or large-diameter RC drilling;
- Processing of mini-bulk and bulk samples to recover macrodiamonds; and
- Sorting and valuation of discrete parcels of macrodiamonds.

All data incorporated into the previous Chidliak Technical Reports has been subjected to extensive internal (by Peregrine) and external verification, as documented in Farrow et al. (2014, 2015) and Nowicki et al. (2016). Key datasets used as a basis for geology models and consequently also Mineral Resource and TFFE estimates have been subject to extensive internal verification and/or audit against vendor-issued assay or valuation certificates (e.g. McCubbing and Coopersmith, 2016; Pell, 2016; Grütter and Wilson, 2015; Fitzgerald, 2015a, 2015b). In the case of drilling data, Peregrine conducted extensive internal audit or data collection processes and cross-checked final results, as described in Section 10.2.

The following data sets for the CH-6 and CH-7 kimberlites were verified by independent, external Qualified Persons. They concluded the “data are of high quality and are suitable for use in estimation of Mineral Resources” and “based on the verification work carried out, the authors believe the Project data to be reliable and to meet or exceed the standards of industry best practice” (Nowicki et al., 2016):

- Pre-2017 drill logs, petrography and representative samples for 60 core holes (11,363 m) and 85 small-diameter RC holes (1,155 m);
- Pre-2017 microdiamond sample assay results for 8,770 kg of drill core and surface bulk samples;
- Pre-2017 bulk density assay results for 2,573 samples;
- Drill logs and related data for six large-diameter RC holes (1,212 m) in CH-7 from which a total of 329 m^3 (809.5 t [dry]) of kimberlite was collected and processed for macrodiamonds; and

- Macrodiamond recoveries from 14 t of CH-6 drill core (2010), 47.2 t of CH-7 surface trench material (2010), 404.3 t of CH-6 surface trench material (2013), and 809.5 t of CH-7 RC-drilled material (2015).

All new (2017) data incorporated into the 2018 Mineral Resource Update Report (Fitzgerald et al., 2018) has been subjected to extensive internal verification and/or audit against vendor-issued assay or valuation certificates, in a similar systematic fashion as in previous years. Based on the verification work carried out, the authors of that report believe the Project data to be reliable and to meet or exceed the standards of industry best practice.

13 Mineral Processing and Metallurgical Testing

Specific process and equipment manufacturer metallurgical testing has not been undertaken on material from the CH-6 and CH-7 kimberlites to date. An Ore Dressing Study (ODS) will be delineated at the next engineering stage to refine and optimize the process flowsheet and equipment selection.

Processing procedures of Chidliak kimberlite from the mini-bulk and bulk samples of CH-6 and CH-7 are described in Section 11 and provided preliminary treatability information for the different kimberlite types (e.g. geological units) through the various metallurgical processes i.e. crushing, scrubbing, screening, de-sliming, DMS, final diamond recovery (X-ray and grease), re-crush, de-grit, fines thickening and materials handling. The preliminary treatability information indicates that a conventional DMS-based mineral processing flowsheet for the kimberlite is appropriate to effectively capture diamonds greater than 1.0 mm, as described further in Section 17.

Based on JDS experience from other Northern Canadian diamond operations, a 98% diamond recovery has been assumed for the proposed processing facility.

Initial sample treatment indicated that the amount of clay in the weathered surface kimberlite would not impact the material handling characteristics of the mineralized feed in a production processing plant. The weathered surface kimberlite was observed to be fine grained with low clay content.

13.1 Metallurgical Variability

The Chidliak kimberlites are generally described as hard and competent except for the weathered zones, for CH-6 at 0 to 40 m depth and for CH-7 at 0 to 60 m depth. Some units of CH-7 are described as high, >50%, olivine content. Until confirmed by further bulk sampling, this could have process flowsheet and final diamond recovery plant implications, such as a larger number of X-ray sorters required to treat the higher heavy mineral content and higher DMS concentrate yield.

The CH-6 trench-collected weathered kimberlite which contained a high proportion of granular fines, treated relatively easily. An average of 75% of the sample reported to the -1.18 mm size fraction. Clay was present in the sample but was not problematic. There was good material flow, excellent scrubbing and crushing results and liberation appeared to be excellent. Although this kimberlite had a high olivine content, DMS concentrate volumes were deemed acceptable by the Peregrine independent external QP Howard Coopersmith (Nowicki et al. 2016).

The 2015 CH-7 mini bulk sample Large Diameter Drill (LDD) program recovered 518 t of drill chips for processing at the Saskatchewan Research Council's (SRC) industry standard sample treatment facility in Saskatoon (McCubbing and Coopersmith, 2016). The CH-7 kimberlite drilled sample generally treated well in a standard DMS-X-ray-Grease process circuit. Large amounts of clay in certain upper portions of the pipe caused minor processing issues; in a production scenario these issues can easily be addressed by ore blending and sufficient scrubbing capacity. The CH-7 kimberlite had a high DMS yield (contained olivine) of approximately 4% and locally up to 10% affecting increased recovery throughput; this can be designed for in a mine recovery scenario by DMS density control, high intensity wet magnetic separation and increased X-ray sorter capacity.

Heavy mineral DMS concentrate yields ranged from a low of 0.4% (CH-6 weathered surface kimberlite) to a high of 4% (CH-7 LDD program) of the batch plant feed.

A high DMS concentrate yield can have a significant impact on the design of the production plant final diamond recovery section.

Further bulk sample testing and dressing studies are warranted to improve the understanding of the DMS concentrate yield variability for the future flowsheet development.

13.2 Deleterious Elements

Generally, there are no deleterious elements found in processed kimberlite. Ten kimberlite and eleven country rock samples have been collected from the CH-6 and CH-7 kimberlites and surrounding host rocks to test for metal leaching and acid rock drainage (ML/ARD) on a reconnaissance basis. They were selected to be spatially representative and to reflect the variations in rock types and weathering for both the kimberlite and country rocks. Assessment of the analytical results returned concluded that the ML/ARD potential for the majority of country rock and all the kimberlite is likely to be very low.

However, two samples of country rock, one of which was sulphide-bearing, were classified as potentially ARD-generating and further work is recommended to assess the ML/ARD potential of the waste rock as the Project progresses (Day and Nowicki, 2012).

13.3 Opinion on Mini-Bulk and Bulk Sampling Program

The scale of the Chidliak kimberlite mini-bulk and bulk sampling is appropriate for this stage of the Project evaluation and the sample collection and processing procedures undertaken meet accepted diamond industry standards.

Further bulk sampling, combined with industry standard dressing studies and metallurgical test work, will need to be undertaken during later project development stages to define the final processes and processing equipment for the production plant flowsheet development and plant design.

JDS and its consultant, Mike Rylatt, have reviewed the information provided by Peregrine and based on this review, are not aware of any issues or factors that could materially impact the accuracy and reliability of the mini-bulk and bulk sampling program results.

14 Mineral Resource Estimate

This section has been extracted and abridged from Fitzgerald et al., 2018 and updated with new diamond pricing information provided by WWW (2018a, 2018b).

14.1 Approach to Mineral Resource Estimates

The Mineral Resource estimates for the CH-6 and CH-7 kimberlites are based on four primary components:

1. A geological model for each kimberlite that defines the boundaries of the deposit (pipe shell) as well as the internal geological domains that encompass the volumetrically significant kimberlite units. The geological domains form the basis of the resource domains for which Mineral Resource estimates are being made. The KIM-L geological domain in CH-6 was further subdivided into two resource domains to account for expected grade variation. The geology and resource models are represented as a series of triangulation solids created in Dassault Systèmes GEOVIA GEMS™ version 6.8 (GEMS). Further analysis of density and microdiamond data for CH-6 was performed using Leapfrog™ Geo version 4.2.1.
2. Estimates of bulk density, representing the variation in bulk density within each body and, in combination with volumes derived from the geological models, provided estimates of the tonnage of kimberlite present.
3. Estimates of average diamond grade (in carats per tonne) for each resource domain using diamond size frequency distributions.
4. Estimates of the average value of diamonds within each resource domain based on estimated diamond value distributions (dollar per carat per sieve size class).

14.1.1 CH-6

Microdiamond[3] and macrodiamond[4] data were obtained from a corresponding volume of KIM-L material (the surface bulk sample), allowing for definition of a total content diamond size frequency distribution (SFD) and hence calibration of the ratio of microdiamond stone frequency (stones per kilogram, st/kg) to recoverable macrodiamond grade. Drill core microdiamond results were used, in conjunction with this established calibration, to determine average grade estimates for a high grade KIM-L resource domain (KIM-L.HG), a normal grade KIM-L resource domain (KIM-L.NG), and a KIM-C resource domain. Details of the data and methods used to generate each component of the CH-6 resource estimate are provided Section 14.4.

[3] The term microdiamond is used throughout this report to refer to diamonds recovered through caustic fusion of kimberlite at a bottom screen size cut-off of 105 µm (~0.00002 ct). Rare larger diamonds that may be recovered by a commercial production plant may be recovered through this process but are still referred to as microdiamonds.

[4] The term macrodiamond is used throughout this report to refer to diamonds recovered by commercial diamond production plants, which typically recover diamonds larger than the Diamond Trading Company (DTC) sieve category 1 (~ 0.01 ct). The DTC+1 sieve is roughly equivalent to 0.85 mm square-mesh sieve.

14.1.2 CH-7

The geological model for CH-7 is more complex than that of CH-6 with seven geological and resource domains defined. Estimates of the average grade for these domains were based either directly on the bulk sample LDD sample grades or on a combination of LDD sample grade and distributed microdiamond data in a similar manner to the approach used for CH-6.

Since no new work was performed at the CH-7 kimberlite during 2017, the Mineral Resource previously reported for CH-7 (Nowicki et al., 2016) is being restated as-is in this report. Section 14.5 provides a summary of the data and methods used by Nowicki et al. (2016) to generate each component of the CH-7 resource estimate.

14.2 Previous Mineral Resource Estimates

14.2.1 Previous CH-6 Mineral Resource Estimates

Peregrine commissioned GeoStrat Consulting Services Inc. (GeoStrat) to provide an independent maiden Mineral Resource estimate for the CH-6 kimberlite in 2014 (Farrow et al., 2014). The estimate was prepared in accordance with CIM guidelines and NI 43-101 standards. In 2015, Geostrat updated the Mineral Resource estimate for CH-6 based on additional core drilling, small-diameter RC drilling, and sampling and diamond testing completed to the end of 2014 (Farrow et al., 2015). In 2016, Peregrine commissioned Mineral Services Canada Inc. (MSC) to provide an updated Mineral Resource estimate for CH-6 based on additional core drilling, sampling and diamond testing completed to the end of 2015 (Nowicki et al., 2016). All previous resource statements for CH-6 are summarized in Table 14-1.

Table 14-1: Previous Mineral Resource Statements for the CH-6 Kimberlite

Kimberlite	Resource Classification	Tonnes (Mt)	Carats (Mct)	Average Grade (ct/t) (+1.18 mm)	Depth of Resource (mbs)	Estimate by	Date Released
CH-6	Inferred	2.89	7.47	2.58	250	GeoStrat	2014-05-07
CH-6	Inferred	3.32	8.57	2.58	250	GeoStrat	2015-01-26
CH-6	Inferred	4.64	11.39	2.45	260	MSC	2016-04-07

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

14.2.2 Previous CH-7 Mineral Resource Estimate

Peregrine commissioned MSC in 2016 to provide a maiden Mineral Resource estimate for the CH-7 kimberlite based on core and LDD drilling, sampling and diamond testing completed to the end of 2015 (Nowicki et al., 2016). Results are summarized in Table 14-2.

Table 14-2: Previous Mineral Resource Statement for the CH-7 Kimberlite

Kimberlite	Resource Classification	Tonnes (Mt)	Carats (Mct)	Average Grade (ct/t) (+1.18 mm)	Depth of Resource (mbs)	Estimate by	Date Released
CH-7	Inferred	4.99	4.23	0.85	240	MSC	2016-05-05

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

14.3 Previous Estimates of Targets for Further Exploration

As part of the 2014 Resource definition process (Farrow et al., 2014), Targets for Further Exploration (TFFE) were outlined at CH-6 and CH-7 in the portions of the kimberlite bodies where drilling information was considered insufficient to define a Mineral Resource. New drilling and related data resulted in 2015-era TFFE updates for CH-6 and CH-7 (Farrow et al., 2015) and 2016-era TFFE updates for CH-6 and CH-7 (Nowicki et al., 2016). The TFFEs are summarized in Table 14-3.

Table 14-3: Summary of Targets for Further Exploration at Chidliak

Kimberlite	Year	Tonnage (Mt)		Estimate by	Date Released
		Low	High		
CH-6	2012	3.61	5.73	GeoStrat	2012-04-02
CH-6	2014	2.60	3.47	GeoStrat	2014-05-07
CH-6	2015	3.20	4.38	GeoStrat	2015-01-26
CH-6	2016	2.34	3.74	MSC	2016-04-07
CH-7	2014	2.75	3.97	GeoStrat	2014-05-07
CH-7	2015	3.72	6.01	GeoStrat	2015-01-26
CH-7	2016	0.90	2.36	MSC	2016-06-03 effective

The potential tonnages defined as TFFE are conceptual in nature as there has been insufficient exploration to define Mineral Resources on these targets and it is uncertain if future exploration will result in the tonnage estimates being delineated as Mineral Resources.

14.4 CH-6 Mineral Resource Estimate

14.4.1 Resource and TFFE Domains

The geological domains described in Section 7.3.3 (KIM-L and KIM-C) form the basis of the resource domains (for which Mineral Resource estimates are being made) and TFFE domains (for which no Mineral Resource estimates are being made but for which volume and tonnage ranges are being reported).

Microdiamond data clearly indicate the presence of a zone of elevated microdiamond stone frequency in the southern portion of KIM-L (Figure 14-1). This zone appears continuous throughout the vertical extent of KIM-L down to a depth of at least 470 mbs (210 masl). It is necessary to account for this zone in the CH 6 grade estimates due to the extent of the discrepancy in stone frequency between it and the remainder of KIM-L (Section 14.4.4.3). Substantive characterization of KIM-L has to date not resolved clear geological differences between the high grade and the normal grade portions of KIM-L, though bulk density appears approximately 3% higher on average for samples representing the zone with high diamond grade. Further work is required to establish the basis for the observed difference in diamond grade.

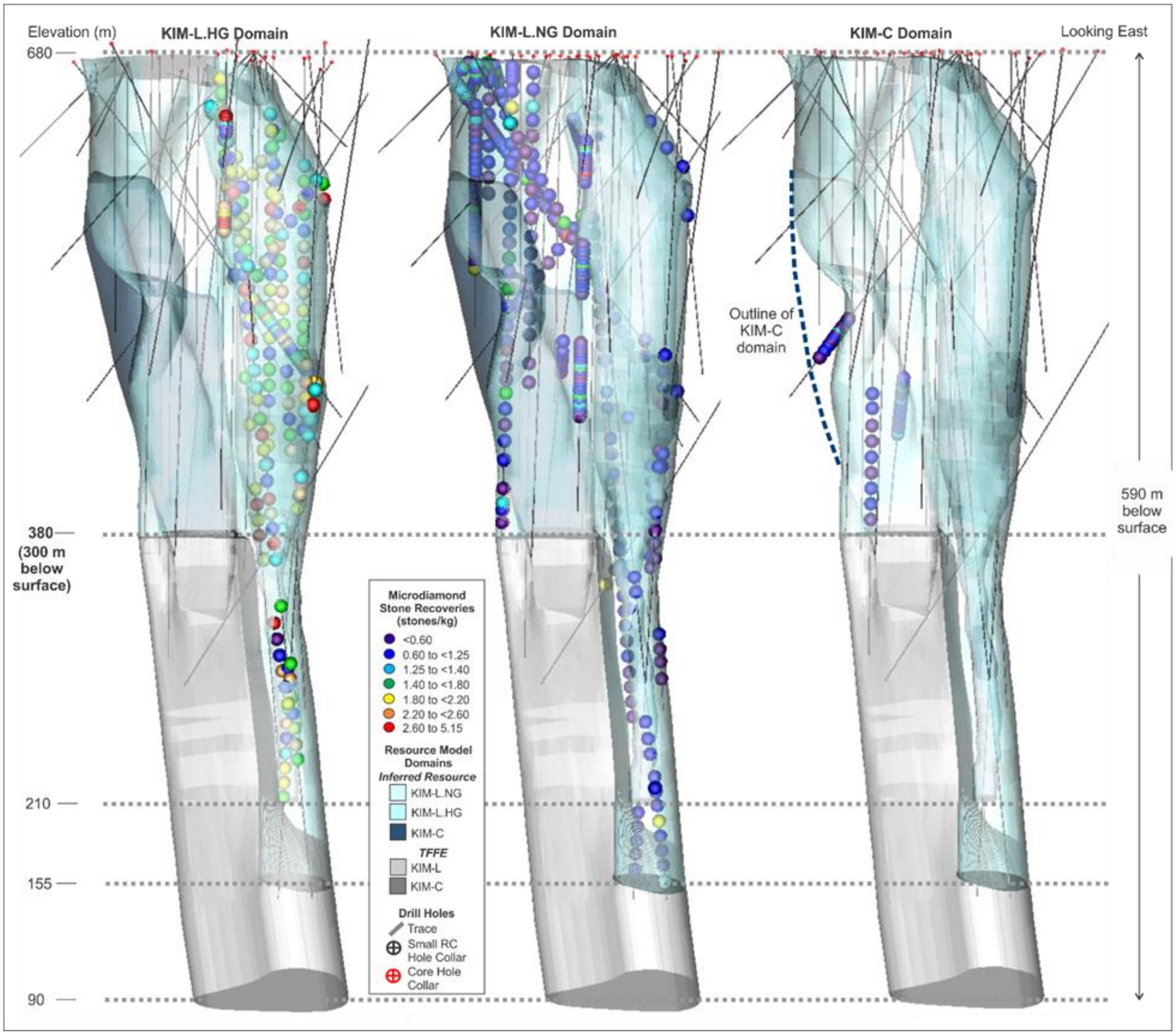
Following the precedent established by Nowicki et al. (2016) to delineate a KIM-L high grade (KIM-L.HG) domain, a 3-D solid was conservatively modelled to encompass microdiamond results above a threshold of 1.25 +212 μm st/kg (Figure 14-1). The updated KIM-L.HG volume does not extend in a material way into areas not sampled, and has expanded laterally and to depth as a result of data acquired during the 2017 drill program. The KIM-L.HG resource domain now extends from surface to a depth of 470 mbs (210 masl). The KIM-L normal grade (KIM-L.NG) resource domain extends from surface to 525 mbs (155 masl) and almost completely surrounds KIM-L.HG. The KIM-C resource domain extends from 80 mbs (600 masl) to 300 mbs (380 masl). Sufficient evaluation data are available to support Mineral Resource estimates for each of these resource domains.

TFFE domains comprise kimberlite for which no Mineral Resource estimates can be made based on data available. The TFFE domains of CH-6 include:

- A portion of the KIM-L geologic domain between 300 mbs (380 masl) and 590 mbs (90 masl), the base of the current model. This portion of KIM-L is not well constrained by drill coverage and is not adequately represented by microdiamond samples.
- A portion of the KIM-C geologic domain between 300 mbs (380 masl) and 360 mbs (320 masl) that is not well constrained by drill coverage nor represented by microdiamond samples.

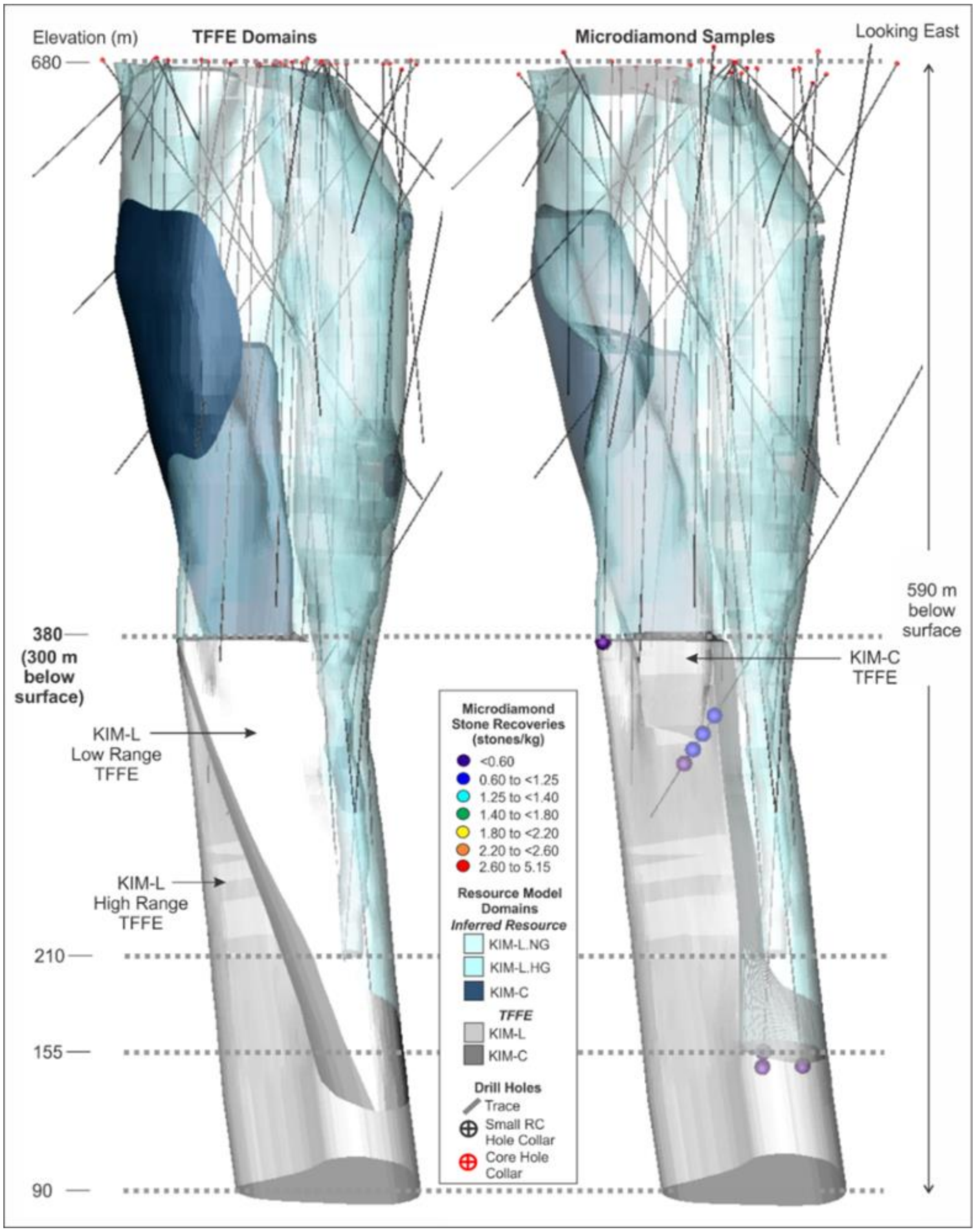
Both a high range and low range shape was modelled for the KIM-L TFFE domain, in order to determine a volume and tonnage range for this portion of the pipe. Refer to Figure 14-2 for illustration of drill holes and microdiamond samples present within the TFFE domains in CH-6.

Figure 14-1: CH-6 Resource Domains and Microdiamond Recoveries



Microdiamond stone recoveries for the KIM-L.HG, KIM-L.NG and KIM-C resource domains in CH-6. Dots illustrate the midpoint location and recoveries (+212 µm st/kg) of each sample aliquot. Some samples appear to fall outside their respective domains – this is an artefact of the 3-D display. Higher stone frequencies are present in a discrete zone in the south of the pipe and were used to model a KIM-L.HG resource domain that spans almost the entire vertical extent of the KIM-L geological domain.

Figure 14-2: CH-6 High Range and Low Range TFFE Domains with Microdiamond Recoveries



Microdiamond stone recoveries for the KIM-L TFFE domain in CH-6. Dots illustrate the midpoint location and recoveries (+212 μm st/kg) of each sample aliquot. Some samples appear to fall outside their respective domains – this is an artefact of the 3-D display.

14.4.2 Block Model

A block modelling approach was used for estimation of volume, tonnage and grade for the KIM-L.HG and KIM-L.NG resource domains, and for estimation of volume and tonnage for the KIM-C resource domain.

The block model comprises 849,600 blocks with dimensions 10 m by 10 m by 10 m in 120 rows, 120 columns and 59 levels. A partial (percent) block modelling approach was applied in order to accommodate estimation of multiple domains using GEMS software. The block model was populated with percentage of each rock type using a vertical needle orientation with a needle density of 3 by 3. Volumes for geology domains were compared to the volumes of the 3-D modelled solids and were accurate within 0.05%.

The block model was then populated with bulk density and grade values as described in Section 14.4.3 and 14.4.4.

14.4.3 Bulk Density and Tonnage

A total of 2,396 bulk density measurements (exclusive of duplicate and repeat QA/QC measurements) were used for bulk density estimation in CH-6. Bulk density displays a clear increase with depth in KIM-L (Figure 14-3). While there is significant overlap in the bulk density ranges for KIM-L within the KIM-L.NG and KIM-L.HG domains, the data indicate a higher overall bulk density for the latter. In the KIM-L.NG domain, density increases with depth to the base of drilling at 540 mbs (90 masl). The KIM-L.HG domain shows a similar trend, but with consistently higher density values for any given elevation and slight inflection point at approximately 280 mbs (400 masl), below which no further increase in bulk density is evident. A possible slight increase in density with depth is observed in KIM-C, however there is less sampling shallower than 140 mbs (540 masl).

Bulk densities averaged for KIM-L.NG and KIM-L.HG across respective blocks in the resource block model (Table 14-4) are consistent with density-depth relationships illustrated in Figure 14-4, and quantify a 3% higher density for KIM-L.HG relative to KIM-L.NG. Checks were performed to ensure there was no density sample overlap between modelled resource volumes.

Table 14-4: Summary Statistics for Bulk Density Data for the CH-6 Resource and TFFE Domains

Resource Category	Resource Domain	No. Samples	Bulk Density (g/cm^3)			
			Average	Minimum	Maximum	Standard Deviation
Inferred	**KIM-L.HG**	**539**	**2.68**	**1.95**	**2.96**	**0.13**
	KIM-L.NG	**900**	**2.60**	**1.83**	**3.04**	**0.16**
	KIM-C	**124**	**2.64**	**2.31**	**2.90**	**0.09**
TFFE	KIM-L	14	2.67	2.58	2.78	0.05
	KIM-C	2	2.65	2.62	2.68	0.04
n/a	CR	817	2.73	2.43	2.98	0.08
	TOTAL	**2396**				

Statistics shown include density data from all rock types that occur within each model domain.

In order to calculate tonnage for resource estimation in CH-6, bulk density data for all rock types sampled within the each resource domain were extracted from drill holes as separate point datasets in GEMS. These were then used to interpolate density locally into the CH-6 block model for the KIM-L.HG and KIM-L.NG resource domains by inverse distance square weighting using a search ellipse of 50 (X) m by 50 (Y) m by 100 (Z) m. In order to avoid over-smoothing of the data, all density interpolated into blocks were informed by a minimum of two and a maximum of four data points. This range ensured that the representative density sampling populated the most blocks possible for the local estimate. GEMS ignores blocks with too few samples and uses the closest samples for blocks with too many samples. This process was chosen for the KIM-L resource domains because bulk density data are spatially representative and comprehensive (refer to Figure 14-3). For the KIM-C resource domain, an average bulk density of 2.64 g/cm^3 was populated into the block model because bulk density data was considered spatially under-represented.

For non-resource domains, average bulk densities were used (refer to Table 14-4). An average bulk density of 2.64 g/cm^3 was used for the KIM-C TFFE domain, as only two data points were available within this modelled volume. No bulk density data exists for overburden from Chidliak, and an estimate of 2.0 g/cm^3 has been used in the 3-D geological model.

Volume and tonnage estimates for the CH-6 resource domains, as determined by block modelling and the procedure described above, are summarized in Table 14-5.

Figure 14-3: Bulk Density Results and Model Averages with Depth for the CH-6 KIM-L.HG, KIM-L.NG and KIM-C Resource Domains

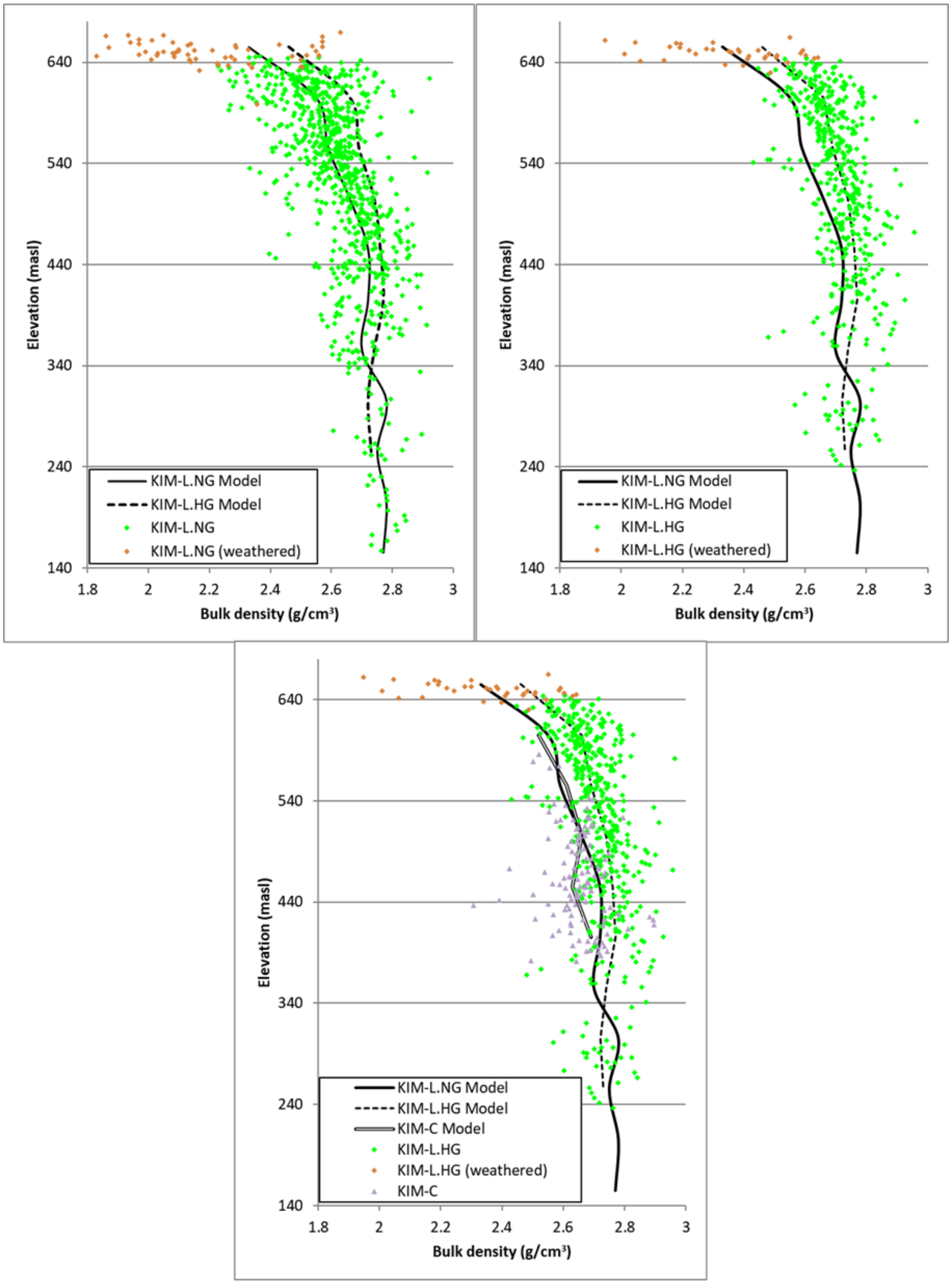


Table 14-5: Volume and Tonnage Estimates for CH-6 Resource Domains

Resource Domain	Volume (Mm^3)	Average Bulk Density (g/cm^3)	Tonnage (Mt)
KIM-L.HG	0.48	2.68	1.29
KIM-L.NG	1.99	2.60	5.18
KIM-C	0.37	2.64	0.99
TOTAL			**7.46**

Inferred Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Volume and tonnage estimates for the TFFE domains are provided in Table 14-6 and were derived by application of average bulk density to the applicable ranges in volume of the modelled solids. Refer to Figure 14-2 for visual representation of these TFFE volumes.

Table 14-6: Volume and Tonnage Estimates for CH-6 TFFE Domains

TFFE Domain	Volume (Mm^3)		Average Bulk Density (g/cm^3)	Tonnage (Mt)	
	Low	High		Low	High
KIM-L	0.38	0.85	2.67	1.01	2.27
KIM-C	0.03	0.03	2.64	0.08	0.08
TOTAL	**0.41**	**0.88**		**1.09**	**2.35**

The potential tonnage defined as TFFE is conceptual in nature as there has been insufficient exploration to define a Mineral Resource. It is uncertain if future exploration will result in the TFFE being delineated as a Mineral Resource.

14.4.4 Diamond Grade

14.4.4.1 Approach to Grade Estimate

The approach adopted for grade estimation follows that of Nowicki et al. (2016) and is based on the concept of using calibrated microdiamond data to estimate diamond grade. Methods to calibrate and implement the concept in practise have been developed over the past few decades and the approach is accepted as a cost-effective industry norm, particularly during resource-development cycles (Davy, 1989; Deakin and Boxer, 1989; Ferreira, 2013; Nowicki et al., 2017; Stiefenhofer et al., 2016; 2017). The successful application of the approach depends on (1) obtaining microdiamond and macrodiamond data that represent the domains for which grades are being estimated, (2) defining a well-constrained, representative geological model with spatially continuous geological units and, (3) obtaining a comprehensive representatively distributed set of microdiamond samples from the resource domain to be evaluated.

For the CH-6 grade estimates, microdiamond data from drill core were used, in conjunction with micro- and macrodiamond data from the 2013 surface bulk sample collected from KIM-L, to estimate diamond grade for each of the resource domains. The principles of this grade estimation approach are as follows:

- Define a representative total content diamond SFD for the KIM-L geological domain. The total content SFD reflects the combined size distribution of microdiamonds and macrodiamonds as established by the 2013 surface bulk sample;

- Define recovery factors that reflect the difference between the total content SFD and the macrodiamonds recovered during sample processing;
- Assess the microdiamond SFD characteristics of KIM-L to confirm significant variation does not occur between or within the defined KIM-L resource domains;
- Assess the microdiamond SFD characteristics of KIM-C to establish its relationship to the total content SFD of KIM-L; and
- If the total content SFD is constant, the relationship between microdiamond stone frequency (stones per kg of kimberlite) and macrodiamond grade is fixed. Thus, microdiamond data from drill core samples can be used in conjunction with total content SFD curves and appropriate recovery factors to estimate recoverable macrodiamond grade.

14.4.4.2 Supporting Data

The diamond datasets generated from the drilling and sampling work discussed in Sections 10.1 and 11.1 were evaluated and outlier samples were excluded through graphical assessments of the results for each kimberlite unit. Outliers were defined as individual 8 kg microdiamond sample aliquots that contain more than 5.15 st/kg (using a bottom cut off stone size of +212 µm), eliminating six of 734 aliquots of KIM-L and one of 44 aliquots of KIM-C from further consideration. The resulting final diamond datasets that were analyzed to estimate and verify grade for KIM-L and to estimate the grade for KIM-C are summarized in Table 14-7. The spatial distribution of samples represented in these datasets is shown in Figure 14-4. The datasets available are as follows:

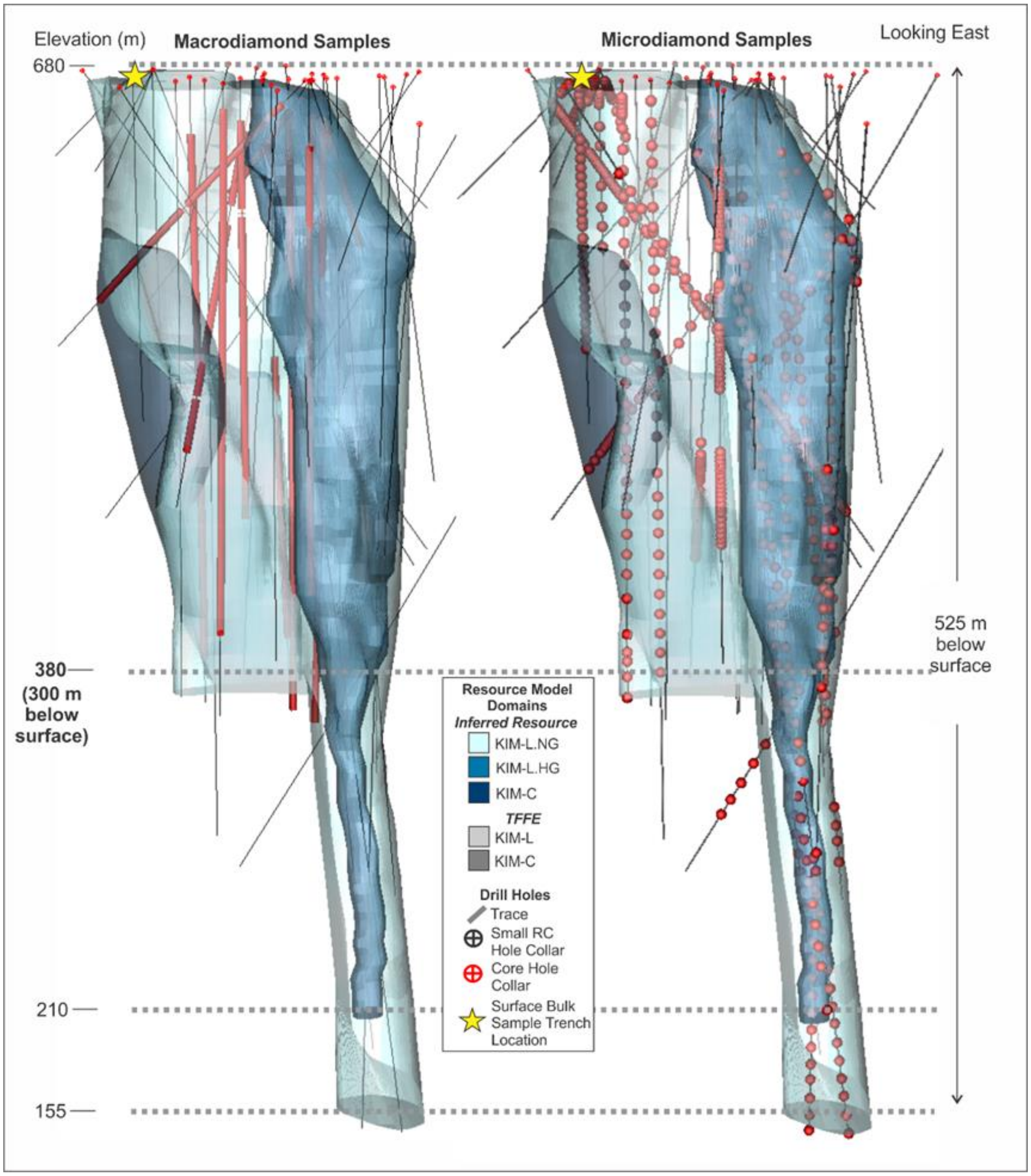
- A surface bulk sample of KIM-L (1,123.95 ct recovered from 404.31 t) and associated representative microdiamond samples (350 kg). These data were used to develop the KIM-L total diamond content SFD model (Section 14.4.4.4), calibrating the relationship between microdiamond stone frequency and recoverable macrodiamond grade;
- Nearly 10 t of KIM-L drill core and approximately 4 t of KIM-C drill core were processed to recover macrodiamonds in 2010, generating parcels of 35.3 ct and 4.7 ct, respectively. Representative samples of the drill core were retained and processed for microdiamonds with a bottom recovery cut off of 425 µm. These smaller datasets were used by Nowicki et al. (2016) to verify the KIM-L grade estimates and the KIM-L total content SFD model. The KIM-C grade estimate reported in this work utilises the 2010-era macrodiamond data as a reference to establish an appropriate SFD model for KIM-C. An independent grade forecasting approach is used to assess the reliability of the KIM-C grade estimate; and
- 5.54 t of drill core from KIM-L and 0.35 t of drill core from KIM-C were processed for microdiamonds. These results have been used to define the extents of the high and normal grade resource domains within KIM-L and to derive the average grade estimates for all resource domains.

Table 14-7: Microdiamond and Macrodiamond Data Used to Estimate and Verify Grade in CH-6

Dataset	Sample Medium	Aliquots (count)	Mass (t)	Process Bottom Cut Off (µm)	Diamonds (+850 µm)	Carats (+850 µm)
Macrodiamond	Surface bulk sample: KIM-L	n/a	404.31	850	16,386	1123.95
	Drill core: KIM-L	n/a	9.78*	850	465	35.34
	Drill core: KIM-C	n/a	4.06*	850	58	4.7
Dataset	**Sample Medium**	**Aliquots (count)**	**Mass (t)**	**Process Bottom Cut Off (µm)**	**Diamonds (+425 / +106 µm)**	**Carats (+425 / +106 µm)**
Microdiamond	Drill core: predominantly KIM-L	53	0.576	425	119	2.37
	Drill core: KIM-C	11	0.124	425	17	0.12
	Surface bulk sample: KIM-L	40	0.35	106	907	0.79
	Drill core: KIM-L	688	5.54	75/106	17,835	31.61
	Drill core: KIM-C	43	0.35	75/106	677	0.76

* Correct total weight is 13.84 t, not 13.76 t as stated in Nowicki et al., 2016

Figure 14-4: CH-6 Resource Model Illustrating Distribution of Macrodiamond and Microdiamond Samples



Three-dimensional view of the KIM-L.HG, KIM-L.NG and KIM-C resource domains showing the spatial distribution of macrodiamond (left) and microdiamond (right) sample coverage in drill core. Red intersections on the left illustrate the entire sampled interval for macrodiamonds. Dots on the right illustrate the midpoint location of each sample aliquot. Some samples appear to fall outside their respective domains – this is an artefact of the 3-D display. The yellow star marks the location of the 2013 surface trench bulk sample from KIM-L.NG.

14.4.4.3 Microdiamond Stone Frequency and SFD Characteristics

Drill core logging and limited petrographic investigation indicate broad scale variability in texture and componentry throughout the KIM-L geologic unit. Microdiamond stone frequency and SFD characteristics were investigated to assess the degree to which the observed geological variability is reflected in the diamond distribution of KIM-L, and no relationship(s) were determined. Drill core logging of KIM-C has outlined broadly similar geological componentry, though with substantially less geological variability than KIM-L. The relationship(s) of KIM-C geological variability and microdiamond sample results have not been investigated further, on account of the null result obtained for similar data sets from KIM-L (see Table 14-7). Summary statistics for microdiamond stone frequency by domain are provided in Table 14-8 and Figure 14-5. The statistics exclude the outliers discussed in Section 14.4.4.2.

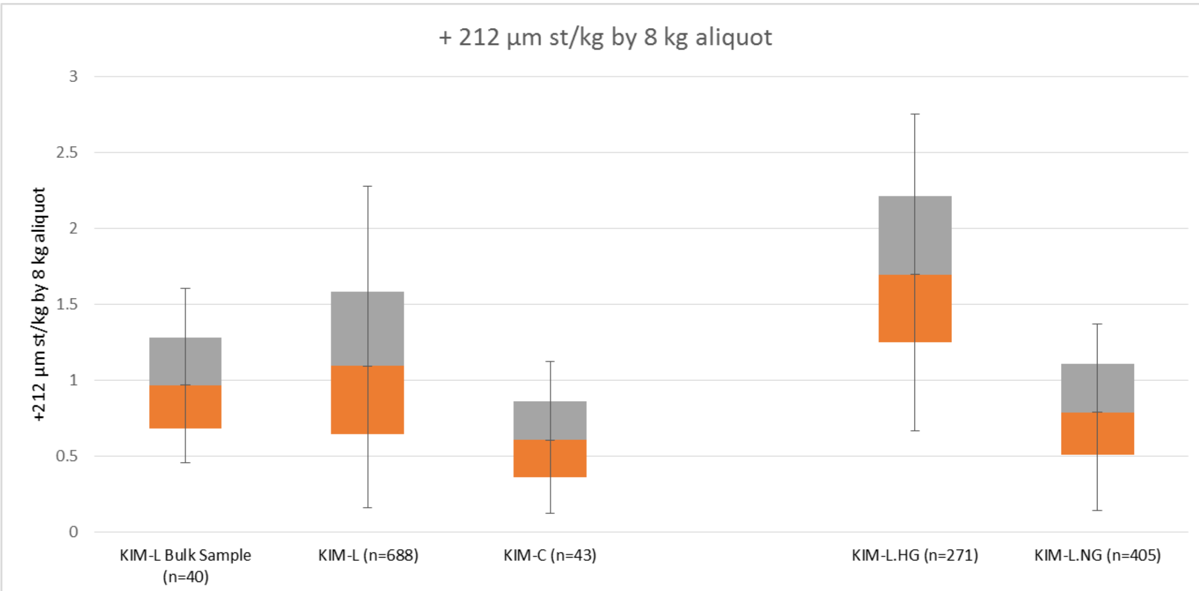
These results show that KIM-C has a lower microdiamond stone frequency than KIM-L and clearly illustrate the difference in microdiamond stone frequency between the KIM-L.HG and KIM-L.NG domains. Variation in stone frequency with depth within the KIM-L resource domains was assessed by grouping microdiamond results for each into depth ranges (Figure 14-6). Holes were sample aliquot depths were not reported were excluded from the depth analysis. No significant changes with depth are evident, and the adoption of average calibrated macrodiamond grades within the KIM-L.HG and KIM-L.NG domains is therefore considered valid.

Table 14-8: Summary Statistics of Microdiamond Stone Frequency (+212 µm st/kg) from CH-6 Drill Core Samples Excluding Seven High-Frequency Outlier Data

Data Type	Descriptor	KIM-L Total	KIM-L.HG	KIM-L.NG	KIM-L TFFE	KIM-C
Sample Information	Count	688	271	405	12	43
	Mass (kg)	5,543	2,204	3,242	98	349
+212 um microdiamond stone frequency statistics	Average (Mean)	1.23	1.83	0.85	0.58	0.64
	Median	1.09	1.70	0.79	0.51	0.61
	Minimum	0.00	0.37	0.00	0.25	0.00
	Maximum	5.15	5.15	3.29	1.32	1.36
	Standard Deviation	0.79	0.82	0.46	0.28	0.35

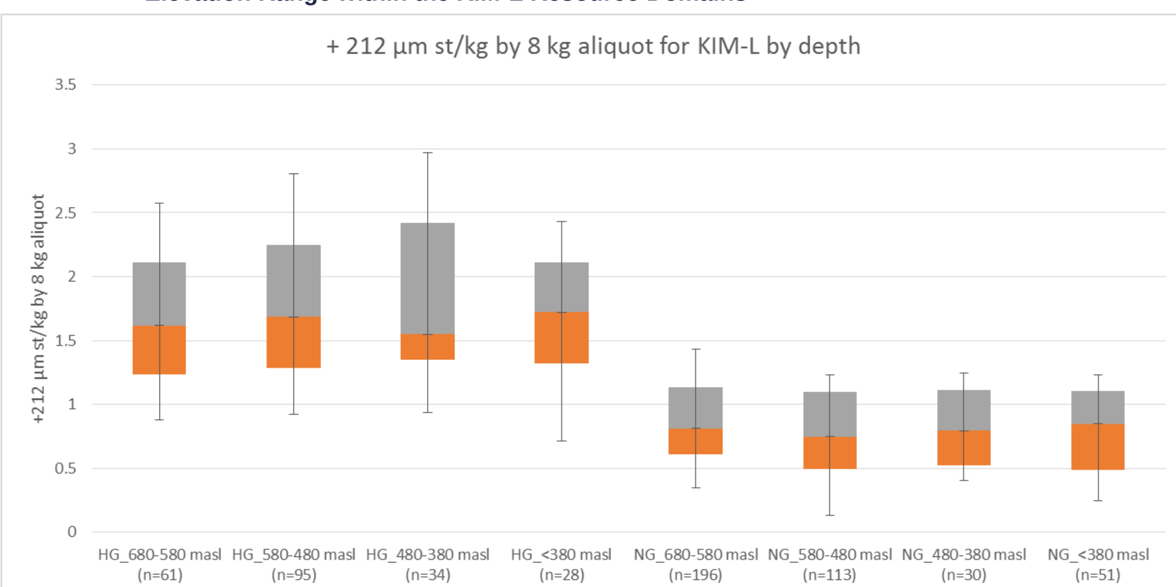
Note: Numbers may not add due to rounding.

Figure 14-5: Microdiamond Stone Frequencies (+212 µm st/kg) from the CH-6 Bulk Sample and Drill Core Samples Grouped by Geological Domain and by Resource Domain



n = number of sample aliquots represented. The orange and grey boxes indicate the 25th to 75th percentile values.

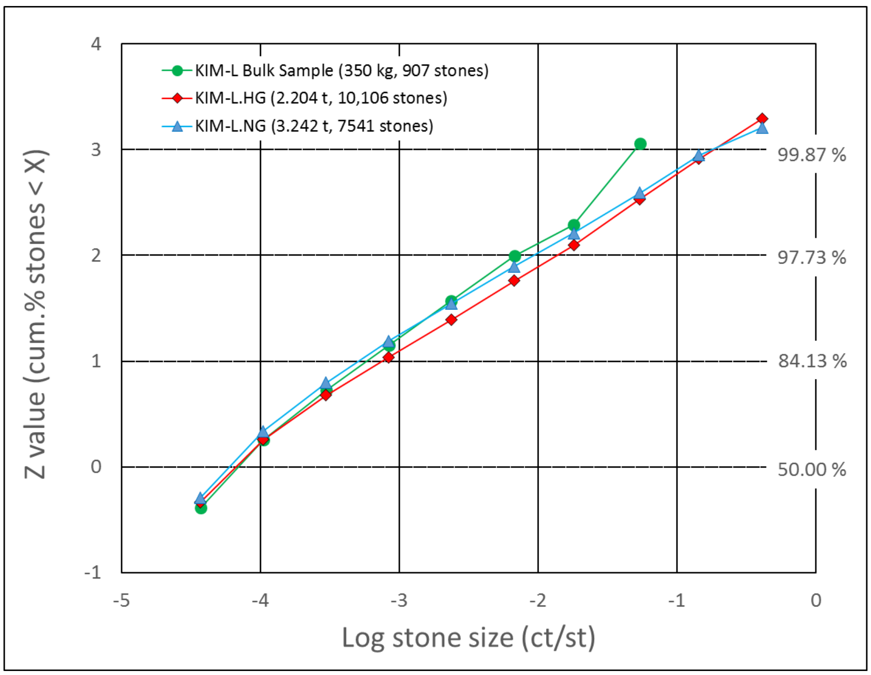
Figure 14-6: Microdiamond Stone Frequencies (+212 µm st/kg) from CH-6 Drill Core Samples Grouped by Elevation Range within the KIM-L Resource Domains



HG = KIM-L.HG, NG = KIM-L.NG, n = number of sample aliquots represented. The orange and grey boxes indicate the 25th to 75th percentile values and the contact between them is the median. Error bars represent the 10th and 90th percentile values.

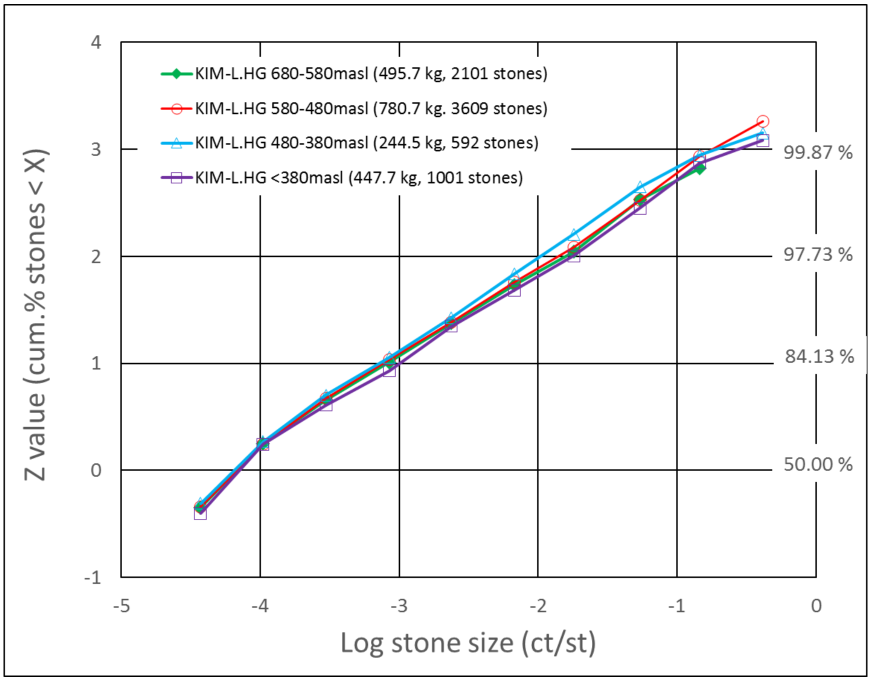
Resource-level diamond grade estimates are made based on the assumption that the SFD remains constant within a resource domain, implying that the SFD as well as stone densities within the CH-6 resource domains need to be assessed. Accordingly Figure 14-7 illustrates microdiamond SFDs for drill core samples from the KIM-L.HG domain (2,204 kg), the KIM-L.NG domain (3,242 kg) and the KIM-L surface bulk sample (350 kg). The SFD's are remarkably congruent, in particular considering the lower-weight 350 kg sample from the KIM-L surface bulk sample, and the (expected) relative variability of coarser-size diamonds in it. The SFD of KIM-L.HG is similar to that of KIM-L.NG, with a subtle indication that KIM-L.HG has a slightly coarser-grained SFD than KIM-L.NG (Figure 14-7). Comparison of SFDs by domain and elevation range (Figure 14-8 and Figure 14-9) indicates no significant differences in SFD within each domain at different elevations.

Figure 14-7: Microdiamond SFDs (+106 µm) for the KIM-L Bulk Sample, and the KIM-L.HG and KIM-L.NG Resource Domains



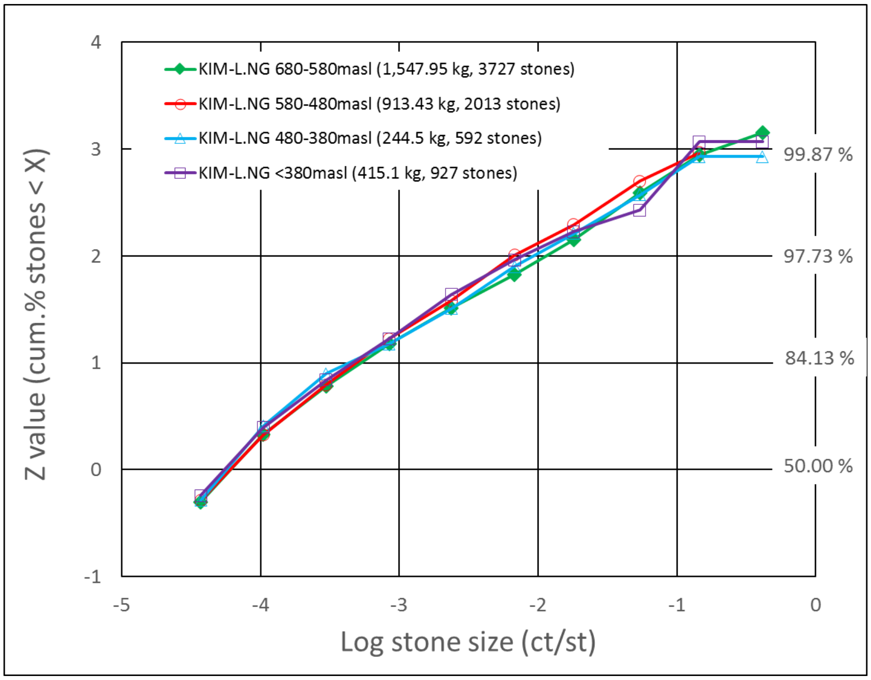
SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size).

Figure 14-8: Microdiamond SFDs (+106 µm) for the KIM-L.HG Domain by Elevation Range



These are the same depth ranges illustrated in Figure 14-6. SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size).

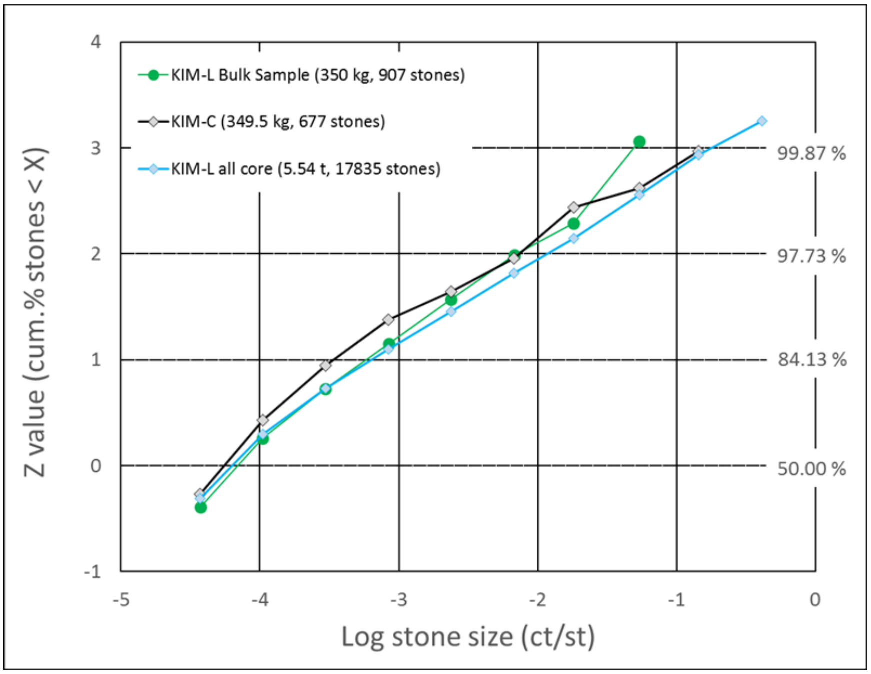
Figure 14-9: Microdiamond SFDs (+106 µm) for the KIM-L.NG Domain by Elevation Range



These are the same depth ranges illustrated in Figure 14-6. SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size).

Since there are no macrodiamond data available for KIM-L.HG to confirm if it indeed has a coarser SFD, the SFD is assumed equivalent to that of the KIM-L.NG domain. This represents a conservative approach to grade estimation, as noted by Nowicki et al. (2016). A 350 kg aggregate sample of KIM-C core defines a microdiamond SFD with a finer distribution than KIM-L at diamond sizes smaller than ~ 0.05 ct (Figure 14-10). At diamond sizes within the commercial size range above ~ 0.05 ct, the SFD for KIM-C closely approaches that of KIM-L, implying that the commercial size range of the KIM-L SFD would serve as a valid proxy to construct a total content SFD for KIM-C. A recoverable macrograde estimate for KIM-C can then be derived by calibrating the domain-specific micro / macrograde relationship and following the same procedures as described in Section 14.4.1.

Figure 14-10: Microdiamond SFDs (+106 µm) for KIM-C Compared to the KIM-L Bulk Sample and all KIM-L Core Samples



SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size).

In summary, a single commercial-sized SFD model, corresponding to that of the 2013 surface bulk sample result, has been used for the purpose of CH-6 resource estimation in this work. Recoverable macrograde estimates then depend primarily on calibration of two micro / macrograde relationships: one for the KIM-L domain and another for the KIM-C domain.

14.4.4.4 Total Diamond Content SFD Models and Recovery Corrections

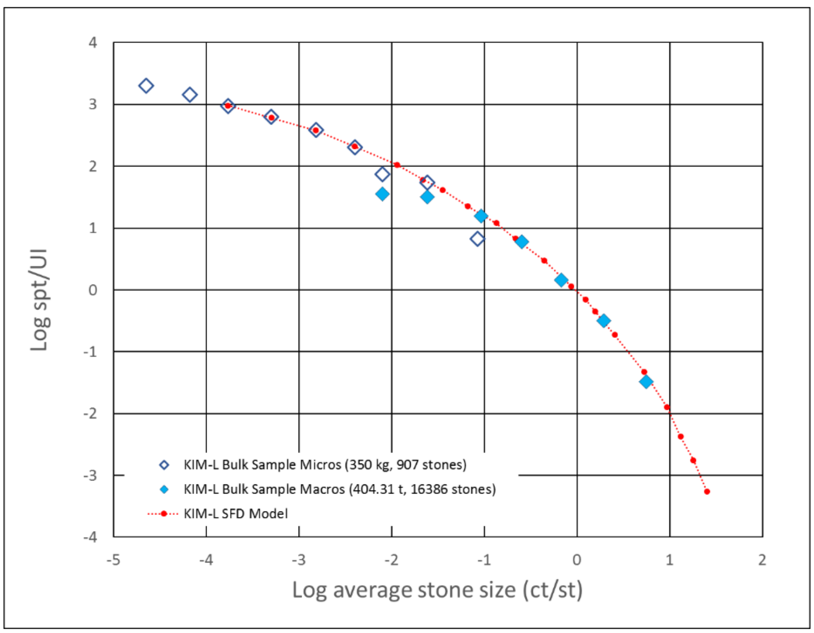
Peregrine QP's have adopted the KIM-L total content SFD model developed by Nowicki et al. (2016) after independently verifying that the SFD model reproduces the KIM-L bulk sample microdiamond and macrodiamond data (Table 14-9). A check was also performed that the recovery factors applied by Nowicki et al. (2016) are reasonable and reproduce the 2.58 ct/t (+1.18 mm) grade obtained for the 2013 KIM-L surface bulk sample. The modified lognormal SFD model of Nowicki et al. (2016) is illustrated in

Figure 14-11 and presented in Table 14-10, which also contains the recovery factors applied.

Table 14-9: KIM-L Bulk Sample Diamond Data Showing Microdiamond and Macrodiamond Parcels used to Define the Total Diamond Content SFD Model for KIM-L

Parcel	Microdiamonds		Macrodiamonds	
Dry Mass	350 kg		404.31 t	
Size Class	St	ct	St	ct
+106 µm	317	0.00624		
+150 µm	228	0.01352		
+212 µm	150	0.02484		
+300 µm	99	0.04683		
+425 µm	60	0.08596		
+600 µm	32	0.10999		
+850 µm	11	0.12563	6,088	81.90
+1180 µm	9	0.28987	6,193	197.63
+1700 µm	1	0.08493	2,680	247.55
+2360 µm			1,095	276.80
+3350 µm			267	179.67
+4750 µm			57	110.44
+6700 µm			6	29.97
Total	**907**	**0.79**	**16,386**	**1,123.95**

Figure 14-11: KIM-L Total Content (+212 µm) Size Frequency Distribution Model



Grade-size plot (spt/UI = stones per tonne per unit interval against the average size of diamonds in each sieve size class, ct/st = carats per stone) showing the final modelled total +212 µm diamond content SFD for the KIM-L geological domain.

Table 14-10: KIM-L Stone Size Frequency Distribution Model and Recovery Factors

Size Class	Total Content (+212 µm) SFD model (% ct)	Recovery Corrections (%)	Recoverable (+1180 µm) SFD Model (% ct)	Recoverable (+1180 µm) SFD Model (ct/t)
+212 µm	1.82	0		
+300 µm	3.36	0		
+425 µm	5.96	0		
+600 µm	9.21	0		
+850 µm	12.12	0		
+1180 µm	4.88	55	4.32	0.111
+3 DTC	9.82	73	11.54	0.298
+5 DTC	11.30	95	17.28	0.446
+7 DTC	7.07	100	11.38	0.294
+9 DTC	8.91	100	14.34	0.370
+11 DTC	10.67	100	17.18	0.443
+13 DTC	5.54	100	8.91	0.230
+15 DTC	1.50	100	2.42	0.062
+17 DTC	2.15	100	3.46	0.089
+19 DTC	3.25	100	5.23	0.135
+21 DTC	1.74	100	2.79	0.072
+23 DTC	0.37	100	0.59	0.015
+10.8 ct	0.19	100	0.30	0.008
+15 ct	0.10	100	0.16	0.004
+20 ct	0.06	100	0.09	0.002
				2.580

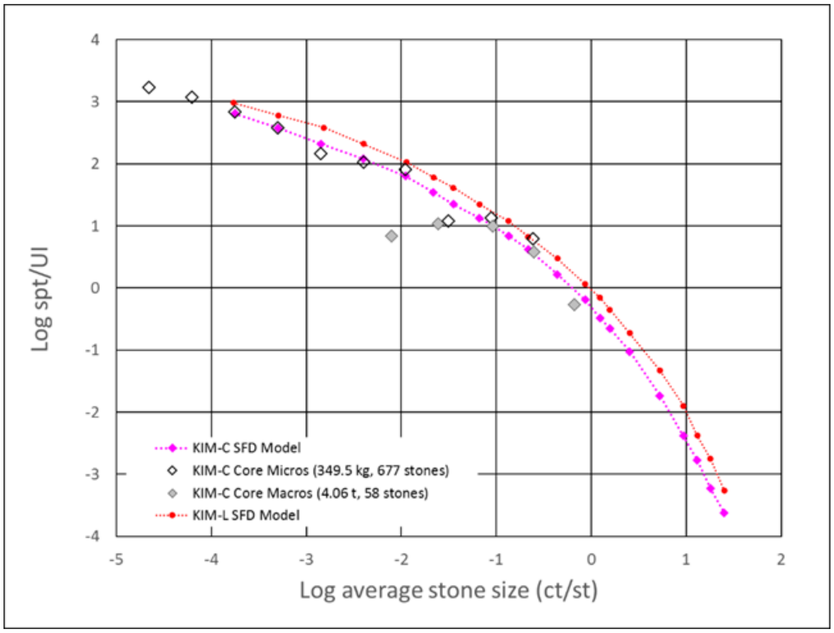
Model of total (+212 µm) and recoverable SFD models (expressed as percent carats (% ct) in each size class) for KIM-L (used to estimate grade of the KIM-L resource domains) (from Nowicki et al., 2016). The model reproduces the 2.58 ct/t +1180 µm grade of the 404.31 t CH-6 bulk sample.

All currently available micro- and macrodiamond data for KIM-C are summarized in Table 14-11. Peregrine QPs adopted the modified lognormal SFD model for KIM-L to fit the limited available commercial-sized diamond data for KIM-C, and to model a KIM-C SFD at larger diamond sizes than recovered by sampling of KIM-C. A total content (+212 µm) KIM-C SFD model was constructed by integrating all available microdiamond data, as illustrated in Figure 14-12 and stated in Table 14-12. The micro / macrograde relationship embodied by the KIM-C model SFD and recovery factors stated in were used as the calibrated basis for the KIM-C grade estimates provided in Section 14.4.4.1.

Table 14-11: Microdiamond and Macrodiamond Results from Core Samples, used to Define the Total Content (+212 µm) SFD Model for KIM-C. One Microdiamond Outlier has been Excluded from the Analysis.

Parcel	Microdiamonds		Macrodiamonds	
Dry mass	349.5 kg		4.06 t	
Size class	St	ct	St	ct
+106 µm	267	0.00573		
+150 µm	185	0.01108		
+212 µm	108	0.01726		
+300 µm	60	0.02828		
+425 µm	23	0.03951		
+600 µm	17	0.07385		
+850 µm	12	0.14068	12	0.18
+1180 µm	2	0.06963	21	0.69
+1700 µm	2	0.14731	17	1.75
+2360 µm	1	0.23	7	1.55
+3350 µm			1	0.53
+4750 µm			0	0
+6700 µm			0	0
Total	**677**	**0.76**	**58**	**4.70**

Figure 14-12: KIM-C Total Content (+212 µm) Size Frequency Distribution Model



Grade-size plot (spt/UI = stones per tonne per unit interval against the average size of diamonds in each sieve size class, ct/st = carats per stone) showing the final modelled total +212 µm diamond content SFD for the KIM-C geological domain.

Table 14-12: KIM-C Stone Size Frequency Distribution Model and Recovery Factors

Size Class	Total Content (+212 µm) SFD Model (% ct)	Recovery Corrections (%)	Recoverable (+1180 µm) SFD Model (% ct)	Recoverable (+1180 µm) SFD Model (ct/t)
+212 µm	2.16	0		
+300 µm	3.57	0		
+425 µm	5.56	0		
+600 µm	9.15	0		
+850 µm	12.47	0		
+1180 µm	4.78	55	4.25	0.063
+3 DTC	9.16	73	10.81	0.160
+5 DTC	11.92	95	18.31	0.271
+7 DTC	7.31	100	11.81	0.175
+9 DTC	9.68	100	15.65	0.231
+11 DTC	10.73	100	17.35	0.256
+13 DTC	5.46	100	8.82	0.130
+15 DTC	1.32	100	2.14	0.032
+17 DTC	1.94	100	3.13	0.046
+19 DTC	2.91	100	4.70	0.069
+21 DTC	1.39	100	2.25	0.033
+23 DTC	0.25	100	0.40	0.006
+10.8 ct	0.13	100	0.21	0.003
+15 ct	0.06	100	0.09	0.001
+20 ct	0.04	100	0.07	0.001
				1.478

Model of total (+212 µm) and recoverable SFD models (expressed as percent carats (% ct) in each size class) for KIM-C (used to estimate grade of the KIM-L resource domains).

14.4.4.5 Grade Estimates

The microdiamond sample coverage for CH-6, while comprehensive, is not strictly spatially representative; certain sampled increments are over-represented (close spaced samples) relative to others (wider spaced samples), and certain portions of the pipe are over-represented relative to others. In order to generate a representative volume-weighted estimate of diamond grade based on microdiamond stone frequency, the data were composited and interpolated into the block model, as described below.

Microdiamond sample results (+212 µm st/kg) were composited at 10 m intervals within the KIM-L.HG and KIM-L.NG domains and were extracted as separate point datasets in GEMS. Long sample increments comprising multiple sub-aliquots for which no spatial (from-to) data were recorded were included with the result centred on the whole sampled interval. Composited sample stone counts were then converted to recoverable grade point data using the established calibration between microdiamond stone frequency and recoverable grade for KIM-L (Section 14.4.4.4). These grade point data were used to interpolate grade into 4,281 blocks in the CH-6 block model for the KIM-L.HG and KIM-L.NG domains using the inverse distance squared method with a search ellipse of 50 (X) m by 50 (Y) m by 100 (Z) m. In order to avoid over-smoothing of the data, all grades interpolated into blocks were informed by a minimum of two and a maximum of four data points. A second pass using a search ellipse of 100 (X) m by 100 (Y) m by 200 (Z) m was used to populate 40 blocks of KIM-L.NG in the southern sector of CH-6 that were not populated in the first pass. For the KIM-C resource domain, each of 703 blocks was populated with a calculated global average recoverable grade of 1.45 ct/t (see Table 14-12), since the microdiamond sampling within this domain is less comprehensive than that within the KIM-L resource domains.

The total tonnage and carats contained within each resource domain was extracted from the block model through volumetric reserves reporting. The results support an average undiluted grade of 4.58 ct/t for the KIM-L.HG domain and 2.11 ct/t for the KIM-L.NG domain.

Dilution within the KIM-L and KIM-C geologic units predominantly takes the form of small and variably distributed xenoliths of carbonate and gneiss that cannot be, and have not been, avoided during normal-course microdiamond sampling. Xenoliths larger than 1 m were accounted for and have been avoided during microdiamond sampling, and these comprise 130 of the 6610 m total length of drill core intercepts within the resource domains (i.e. 1.97%). The drill coverage achieved to date is not of sufficient resolution to demarcate specific zones of elevated dilution, and a downward adjustment of 2% was therefore applied to the average grades to correct for dilution not already represented in the microdiamond results. The dilution-corrected average grades shown in Table 14-13 were populated into the block model by domain as the final average +1.18 mm bottom cut off recoverable grade estimates for CH-6.

Table 14-13: Average Recoverable Grade Estimates for the CH-6 Resource Domains

Resource Domain	Recoverable Grade (+1180 µm ct/t)
KIM-L.HG	4.49
KIM-L.NG	2.07
KIM-C	1.45

Grades are reported on a recoverable basis at an 1180 µm bottom cut off and reflect the recovery efficiency of the sample processing plant used to treat the bulk samples.

14.4.5 Diamond Value

14.4.5.1 Valuation

A diamond parcel of 1,117.09 ct from CH-6 was valued by WWW International Diamond Consultants Ltd (WWW) in February 2014. WWW (2014) describe the parcel as presenting well in terms of quality, colour and shape with a number of yellow diamonds in the smaller size ranges suggesting the possible presence of fancy yellow stones. Based on the 2014 valuation, the average value for all diamonds in and larger than the DTC+3 size category (1,013.54 ct) was 213 US$/ct and an average modelled diamond value of 188 US$/ct was reported (WWW, 2014). This valuation exercise was updated in March 2016 based on the February 1, 2016 WWW price book, yielding an average actual diamond value for the parcel of 162 US$/ct (WWW, 2016a). The valuation exercise was updated a second time in April 2018 using the WWW's March 31, 2018 price book and an average actual diamond value of $164 US$/ct was obtained. The results of this second re-valuation exercise (extracted from WWW, 2018a) are presented in Table 14-14.

Table 14-14: April 2018 Re-pricing Results for a Parcel of 1,013.54 ct from CH-6 in US$/ct

Size Class	Total Carats	Total Stones	Average Carat Per Stone	$/Carat
+9 ct	8.87	1	8.87	2,446.80
+8 ct				
+7 ct				
+6 ct	5.83	1	5.83	2,399.00
+5 ct				
+4 ct	8.73	2	4.37	1,926.40
+3 ct	28.20	9	3.13	546.70
+10 gr	10.64	4	2.66	480.70
+8 gr	40.27	20	2.01	517.95
+6 gr	27.02	17	1.59	315.96
+5 gr	10.52	8	1.32	362.30
+4 gr	48.74	48	1.02	234.13
+3 gr	49.76	66	0.75	150.93
+11 DTC	182.80	436	0.42	93.91
+9 DTC	144.24	664	0.22	57.81
+7 DTC	119.52	887	0.13	44.48
+5 DTC	210.40	3,170	0.07	38.89
+3 DTC	118.00	4,000	0.03	20.16
Total	**1,013.54**	**9,333**	**0.11**	**164.32**

An updated modelled estimate of average diamond value was generated by WWW (2018a) by combining a value distribution model (model of average diamond value by size class) with a single SFD model. The modelling yielded a base case average diamond value of $151 per carat (Table 14-15), with a low modelled average price of $131 per carat and a high modelled average price of $191 per carat (WWW, 2018a). WWW noted (2014) that it is unusual for the modelled average price to be lower than the parcels actual average price for samples of this size. They attributed this to the fact that all the CH-6 diamonds larger than 4 carats per stone were of relatively high value and might not be representative of the all the stones in these size classes if a larger sample were obtained.

This average value reflects the recovery efficiency of the process plant used to treat the CH-6 KIM-L bulk sample and assumes that no diamonds smaller than DTC+3 will be recovered. This recoverable average value corresponds with the recoverable grades reported in Table 14-13. While the bottom cut off of DTC+3 is not precisely the same as the 1180 µm square mesh cut off basis for the grade estimate, these are very similar to each other and for the purpose of an Inferred Mineral Resource estimate, can be considered to be equivalent. In the commercial size ranges, the SFD of KIM-C appears to be very similar to that of KIM-L, so for the purposes of this Mineral Resource estimate, the diamond values from the KIM-L bulk sample are considered applicable to KIM-C.

Table 14-15: April 2018 Average Modelled Diamond Value for CH-6 in US$/ct

Size Class	Model SFD (% ct)	Value Distribution ($/ct per size class)
+10.8 ct	1.31	630
+10 ct	0.15	875
+9 ct	0.18	1,100
+8 ct	0.23	1,290
+7 ct	0.30	1,360
+6 ct	0.40	1,420
+5 ct	0.57	1,395
+4 ct	0.88	1,235
+3 ct	1.51	1,070
+10 gr	0.69	830
+8 gr	2.52	560
+6 gr	2.58	380
+5 gr	1.93	285
+4 gr	4.49	190
+3 gr	6.51	115
+11 DTC	17.87	94
+9 DTC	14.10	58
+7 DTC	11.68	44
+5 DTC	20.57	39
+3 DTC	11.54	20
Average model $/ct		**151**

The average value reflects the recovery efficiency of the plant used to treat the CH-6 bulk sample, but assumes no recovery of diamond smaller than DTC 3. The SFD, value distribution and average $/ct values in this table were extracted from WWW (2018a).

14.4.5.2 QP Comments on CH-6 Diamond Value

Nowicki et al. (2016) reviewed the WWW valuation data and commented: “The WWW (2018a) average value represents their best estimate of diamond value per size class applied to a model of diamond SFD (for diamonds larger than DTC+3) for CH-6. WWW are recognized international leaders in the field of diamond valuation and the QPs for this report believe it is reasonable to rely on the diamond values provided. The model SFD used by WWW differs slightly from that on which the grade estimates for CH-6 are based (Section 14.4.4). However, application of the WWW value distribution model to the CH-6 SFD model did not produce a significantly different average value and no modification to the reported average value for CH-6 is considered to be necessary.” Peregrine’s internal QP’s agree with this statement.

14.4.6 Confidence and Resource Classification

14.4.6.1 Volume and tonnage

The drill coverage and number of external pierce points obtained are considered sufficient to have constrained the overall volume of the CH-6 pipe, from surface to an elevation of 300 mbs (380 masl), to a high level of confidence. The southern portion of the pipe is reasonably constrained by drilling between 300 mbs (380 masl) and 525 mbs (155 masl), though further definition of pipe-wall pierce-points and potential internal boundaries is required in this depth range to establish the same level of confidence as exists at shallower depths. The internal boundaries of the resource domains are based on the logged boundary between the KIM-C and KIM-L geological domains, and the distribution within KIM-L of microdiamond sampling results with stone frequencies higher than 1.25 st/kg that support a distinct high grade domain. The data used to define these domains is considered sufficient to constrain their volumes.

Uncertainty associated with this approach include:

- The KIM-C geological domain appears to represent a remnant of an earlier deposit preserved along the pipe margins. The seemingly complex morphology of this domain has been simplified by 2017 drilling results, although the contact between KIM-C and KIM-L, while informed by a significant number of drill intercepts, still carries some degree of uncertainty. Based on the apparent limited volume of KIM-C in relation to KIM-L, KIM-C does not represent a significant overall potential source of uncertainty in determinations of the volumes of KIM-L resource domains. The 2017 drilling results constrained the depth distribution of KIM-C within the upper 360 m of the northern half of CH-6, though minor additional volumes of KIM-C may be present at the pipe margin, in areas not currently informed by drilling. The drill coverage achieved to date provides a reasonable spatial representation of the pipe and of the KIM-C domain, and it is unlikely that additional unproven KIM-C volumes will impact materially on the resource domain volumes; and
- The microdiamond sample coverage used to model the KIM-L.HG resource domain within the KIM-L geological domain is not sufficiently dense to have constrained this volume to a high level of confidence below 470 mbs (210 masl). The bulk density difference between the KIM-L.HG and KIM-L.NG domains is minor (~ 3%), implying that volume uncertainty pertaining to these domains does not introduce significant tonnage error into the estimate.

Bulk density in CH-6 is considered constrained to a high level of confidence by a large, spatially representative dataset and is not considered to be a potential source of significant uncertainty in the resource estimate. The volume and tonnage estimates for the CH-6 resource domains are considered constrained to a level of confidence acceptable for classification of Indicated Mineral Resources in the depth range shallower than 300 mbs (380 masl), and Inferred Mineral Resources in the depth range 300 to 525 mbs (380 masl to 155 masl). The appropriate overall classification for the resources declared in this report is therefore Inferred Mineral Resources.

14.4.6.2 Diamond Grade Uncertainty

Areas of grade uncertainty and the scale of potential error introduced into the average grade estimates for KIM-L were enunciated by Nowicki et al. (2016). They are duplicated in the discussion points below, with appropriate updates:

- Recoverable macrodiamond grade can be misrepresented by small sample sizes. Extreme value plots (representing cumulative grade with increasing sieve size class) have been assessed and high / low case error range modelling on the KIM-L SFD has been carried out to gauge the scale of this potential error. Due to the relatively large parcel size (1,124 ct) the potential scope of this error is limited (< ±10%);

- The grade estimates for CH-6 are based on a calibration of microdiamond stone frequency to recovered macrodiamond grade in the KIM-L bulk sample. Incorrect calibration of this relationship could occur if the material sampled for microdiamonds is not the same average grade as the overall bulk sample parcel. The microdiamond sample from the bulk sample comprised 40 spatially representative aliquots collected during excavation of the bulk sample. On an individual basis, as expected, these aliquots present variable results. Removal of aliquots with higher and lower recoveries, and generation of multiple random subsets of 20 aliquots from the overall 40 aliquot sample, suggests that the potential error associated with this calibration is less than ±10%;

- The generation of recoverable grade estimates from drill core microdiamond results is based on an assumption of SFD constancy between the 2013 KIM-L surface bulk sample and all KIM-L material comprising the CH-6 resource domains. Despite the geological variability (primarily textural heterogeneity) observed within KIM-L, assessments of microdiamond SFD characteristics (Section 14.4.3) indicate no significant differences between major groupings of results within or between resource domains. Microdiamond data for the KIM-L.HG domain suggest that this material may have a slightly coarser SFD than that of KIM-L.NG (Figure 14-7), implying that the assumption of a constant SFD is a slightly conservative basis for estimation of the grade of the KIM-L.HG domain. The remaining potential for varying SFD relates primarily to the possibility that KIM-L.NG and KIM-L.HG could represent different kimberlite phases. This critical distinction is typically well resolved by routine and special investigations already completed over multiple years at CH-6, and Peregrine QP's consider it a remote and seemingly unresolvable possibility;

- The drill core microdiamond sample database for CH-6 is large, but portions of the pipe at 300 to 540 mbs (380 masl to 140 masl) are under-represented in the available data (Figure 14-4). Based on the level of variability observed in the grade data to date, and based on the grade estimation approach adopted (which mitigates over-clustering of data in certain areas relative to others) it is not likely that artefacts of the sample coverage will introduce error beyond that acceptable for an Inferred Mineral Resource. An assessment of large groupings of drill core microdiamond data with depth (Figure 14-8 and Figure 14-9) suggests that there is not likely to be a significant overall variation in grade with depth. However, the data available imply that local grade variation of up to ±30% from the average may be present on a scale pertinent to monthly mining production and grade reconciliation. This level of variability is considered acceptable for an Inferred Mineral Resource and does not preclude the use of average domain grades; and

- Uncertainty in the relative volumes of the resource domains will also carry an associated grade uncertainty. The volume of the KIM-L.HG domain was conservatively modelled based on the improved resolution of 2017-era drilling and related microdiamond sampling (Figure 14-4). The uncertainty related to the volume of this domain is therefore thought to not present significant potential downside to the current grade estimates.

The scale of potential error related the global average grade estimate for KIM-C has been assessed by application of an exclusive grade forecasting protocol developed by Peregrine QP's. The protocol is based on factoring a test microdiamond data set against micro/macro diamond data for select reference bulk sample results that serve as protocol benchmarks. The protocol is entirely independent of SFD modelling procedures and – importantly – propagates variance parameters that permit assessment of the reliability of grade forecasts. Recoverable grade forecasts based on KIM-C microdiamond data (see Table 14-12) factored against three appropriate benchmarks are:

- 1.78 ± 0.40 ct/t [± 22%] at +1.18 mm;
- 1.40 ± 0.29 ct/t [± 21%] at +1.18 mm; and
- 1.24 ± 0.33 ct/t [± 27%] at +1.18 mm.

Since all three forecasts carry variance less than ± 30% and the average 1.47 ct/t recoverable grade forecast is effectively identical to the 1.45 ct/t estimated resource grade of KIM-C, it is appropriate to consider KIM-C grade supported at an Inferred Resource category.

14.4.6.3 Diamond Value

Uncertainty in average diamond value derives from two main factors: (1) uncertainty in diamond value distribution (dollar per carat per sieve size class), particularly in the less well represented coarse size ranges, due to the limited size of the parcel valued (~1,000 ct); and (2) uncertainty in the diamond size frequency distribution (SFD) to which the value distribution is applied to generate an average recoverable diamond value.

Uncertainty associated with the value distribution model has been assessed by WWW (2018a) through the modelling of high and low value distribution models that represent the range of uncertainty present. These models translate to an uncertainty range of -15 to +30%.

Uncertainty associated with the SFD has been assessed by modelling high and low case SFD models that represent an interpretation of the finest and coarsest SFDs that could potentially be resolved in a production setting. The range of value uncertainty associated with these models is on the order of -20% to +30%.

The use of a single average value for the entire CH-6 Mineral Resource estimate assumes that neither the diamond value distribution nor the SFD will vary materially with depth or laterally in the KIM-L and KIM-C domains. Based on the degree to which microdiamond results display broad scale SFD similarity this assumption is considered valid.

The average diamond value for CH-6 is considered constrained to a level of confidence suitable for reporting of Inferred Mineral Resources.

14.4.6.4 Summary of Confidence and Resource Classification

The level of confidence to which each major component of the CH-6 Mineral Resource estimate is constrained is shown in Table 14-16. The overall resource classification is based on that of the lowest confidence component.

Table 14-16: CH-6 Mineral Resource Estimate Confidence Levels

Body	Volume	Tonnage	Grade	Value	Resource Classification
CH-6	INF	INF	INF	INF	INF

Confidence with which each major component of the CH-6 Mineral Resource estimate is constrained. The overall Mineral Resource is classified at an Inferred level of confidence.

14.5 CH-7 Mineral Resource Estimate

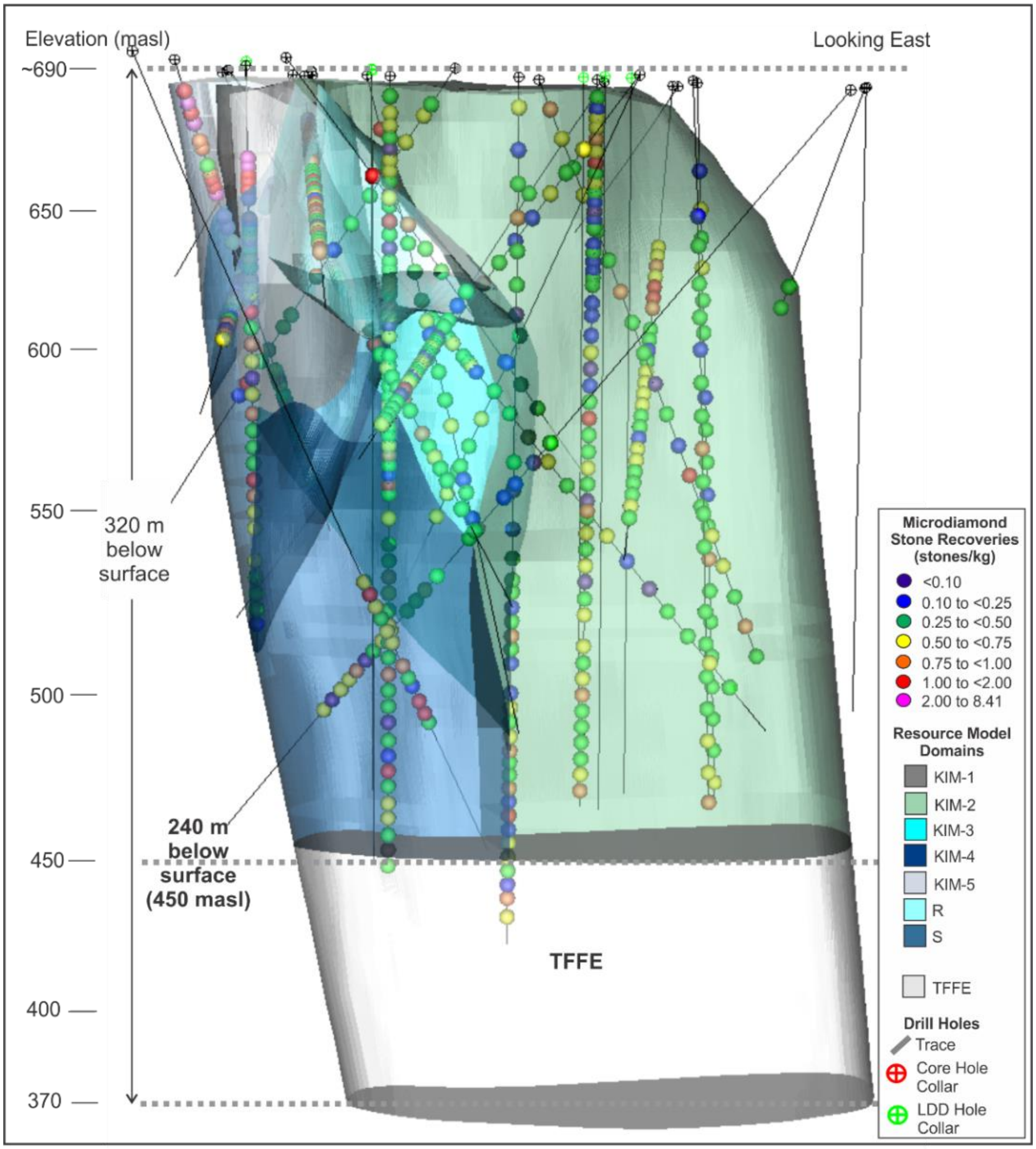
14.5.1 Resource and TFFE Domains

With the exception of a diamond re-valuation exercise by WWW (WWW, 2018b), no new, 2017-era evaluation data for CH-7 are available. All available results were incorporated into an earlier issued technical report for the Chidliak Project (Nowicki et al., 2016). No revision to the reported Resource or TFFE estimates in Nowicki et al. (2016) has been undertaken. Data and results are stated here as derived from Nowicki et al. (2016), with the exception of Section 14.4.5, which has been updated to reflect the new diamond valuation data.

The CH-7 geological domains described in Section 7 form the basis of the resource domains for which Mineral Resource estimates are being made. Further subdivision of these domains was limited to clipping of domains KIM-2 and KIM-4 at an elevation of 450 masl (240 mbs) to exclude underlying material. Different grade and bulk density estimates were made for KIM-5 above and below 620 masl (70 mbs). This domain was not subdivided – the grade and bulk density estimates were populated into the block model by elevation range for the KIM-5 solid.

A single target for further exploration (TFFE) domain corresponds to the entire pipe in the elevation range 450 to 370 masl (240 - 320 mbs). No Mineral Resource estimate can be made for this material due to a lack of evaluation data, and TFFE estimates of volume and tonnage ranges are reported. The geology of this part of the pipe is not sufficiently well constrained to support subdivision into different TFFE domains. Resource and TFFE domains are illustrated in Figure 14-13.

Figure 14-13: CH-7 Resource Domains and Microdiamond Stone Recoveries



14.5.2 Bulk Density and Tonnage

A total of 957 bulk density measurements (exclusive of outlier, duplicate and repeat QA/QC measurements) were used for the CH-7 resource estimate. Summary statistics for results grouped by domain are shown in Table 14-17.

Table 14-17: Summary Statistics for Bulk Density Data from the CH-7 Resource and TFFE Domains

Resource Category	Domain	Samples	Bulk Density (g/cm^3)			
			Average	Minimum	Maximum	Std. Deviation
Mineral Resource	KIM-1	71	2.70	2.32	3.01	0.15
	KIM-2	358	2.56	1.81	2.93	0.14
	KIM-3	162	2.72	2.29	3.00	0.13
	KIM-4	88	2.79	2.56	2.93	0.09
	KIM-5	34	2.70	2.30	3.00	0.19
	R	22	2.47	2.29	2.67	0.11
	S	79	2.90	2.41	3.05	0.11
TFFE	TFFE	7	2.85	2.66	2.97	0.11
n/a	CR above 630 masl	86	2.71	2.36	2.81	0.06
	CR below 630 masl	50	2.67	2.34	2.81	0.08

Bulk density values have been populated into the CH-7 block model for each resource and TFFE domain as outlined in Table 14-18. These values were derived as follows:

- Clear trends of increasing bulk density with depth are present in several domains (KIM-1, KIM-2, and domain S) and were accounted for by averaging the bulk density results within selected elevation ranges. Elevation ranges were selected based on the overall trend for each dataset to adequately represent the degree of variation present.
- In KIM-4 the change in bulk density with depth is very well defined, and bulk density values for selected elevation ranges were derived from a linear regression line fitted to the KIM-4 results. This allowed for estimation of bulk density in shallower portions of KIM-4 that are not represented by sampling.
- The KIM-5 domain shows extreme variability in bulk density values linked to variable degrees of weathering and there are not clear trends with depth for the majority of this domain. However, it is apparent that bulk density samples taken from a small portion of the domain extending below 620 masl yield substantially higher bulk density values than those from higher elevations and hence this portion of the domain has been assigned a higher bulk density (Table 14-18).
- Where no clear trend of bulk density change with depth is present, a single average bulk density was adopted for the entire domain (KIM-3 and domain R).
- The TFFE domain has been assigned an average bulk density based on the data available.

Table 14-18: Resource and TFFE Domain Bulk Density Values Adopted for the CH-7 Estimate

Domain	Elevation (masl)	Bulk density (g/cm^3)
KIM-1	>650	2.65
	<650	2.80
KIM-2	>660	1.98
	660 to 640	2.25
	640 to 620	2.49
	620 to 560	2.58
	<560	2.62
KIM-3	All	2.72
KIM-4	>570	2.63
	570 to 550	2.68
	550 to 530	2.72
	530 to 510	2.76
	510 to 490	2.81
	490 to 470	2.85
	470 to 450	2.89
KIM-5	>620	1.93
	<620	2.87
R	All	2.47
S	>570	2.87
	570 to 550	2.93
	<550	2.99
TFFE	All	2.85

Average values for geological domains (e.g. not Mineral Resource or TFFE estimates) have also been assigned in the CH-7 block model. The approximate base of the country rock weathering horizon in proximity to CH-7 is 630 masl (60 mbs). Average bulk densities of country rock above and below this elevation were therefore adopted for weathered and unweathered country rock. No data are available for overburden material and an assumed average of 2.20 g/cm^3 was adopted. The CR geological domain was assigned the average value of all bulk density samples with a logged geology unit of “weathered country rock” (2.59 g/cm^3).

Resource domain volume and tonnage estimates are provided in Table 14-19. These estimates were derived through volumetric reporting from the CH-7 block model in GEMS. Volume and tonnage estimates for the TFFE domain are provided in Table 14-20.

Table 14-19: Volume and Tonnage Estimates for the CH-7 Resource Domains

Resource Domain	Volume (Mm^3)	Density (g/cm^3)	Tonnage (Mt)
KIM-1	0.08	2.74	0.22
KIM-2	1.13	2.49	2.82
KIM-3	0.25	2.72	0.69
KIM-4	0.22	2.80	0.61
KIM-5	0.05	2.00	0.11
R	0.10	2.47	0.24
S	0.11	2.89	0.31
Total	**1.94**	**2.57**	**4.99**

The bulk density shown for each resource domain, and for the whole of CH-7, is based on the total volume and tonnage extracted from the CH-7 block model in GEMS. Values may not add due to rounding of the reported values to two decimal places.

Table 14-20: Volume and Tonnage Range Estimates for the CH-7 TFFE Domain

TFFE Domain	Volume (Mm^3)		Density	Tonnage (Mt)	
	Low	High	(g/cm^3)	Low	High
CH-7 450 to 370 masl	0.32	0.83	2.85	0.90	2.36

The potential tonnage defined as TFFE is conceptual in nature as there has been insufficient exploration to define a Mineral Resource on this target and it is uncertain if future exploration will result in the tonnage estimate being delineated as a Mineral Resource.

14.5.3 Diamond Grade

14.5.3.1 Approach to Grade Estimation

Micro- and macrodiamond results from each resource domain have been investigated to gauge average grade characteristics, extent of internal grade and SFD continuity and relationships between micro- and macrodiamonds. Two approaches to grade estimation have been applied based on the types of evaluation data available and results of the aforementioned investigation. These include (1) the adoption of recovered LDD grades in KIM-2, KIM-4, KIM-5 and domain R as average grade estimates for these domains, and (2) the use of calibrated microdiamond data to estimate average diamond grade (see Section 14.5.3.1 for a detailed explanation of this approach) in KIM-1, KIM-3 and domain S. Average grades have been adopted for all resource domains.

14.5.3.2 Supporting Data

The diamond datasets generated from the drilling and sampling work discussed in Sections 10 and 11 were evaluated and outlier samples were excluded through graphical assessments of the results for each kimberlite unit. The resulting final diamond datasets that were used to estimate and verify grade in CH-7 are summarized in Table 14-21. The spatial distribution of these datasets is shown in Figure 14-14. The datasets available are as follows:

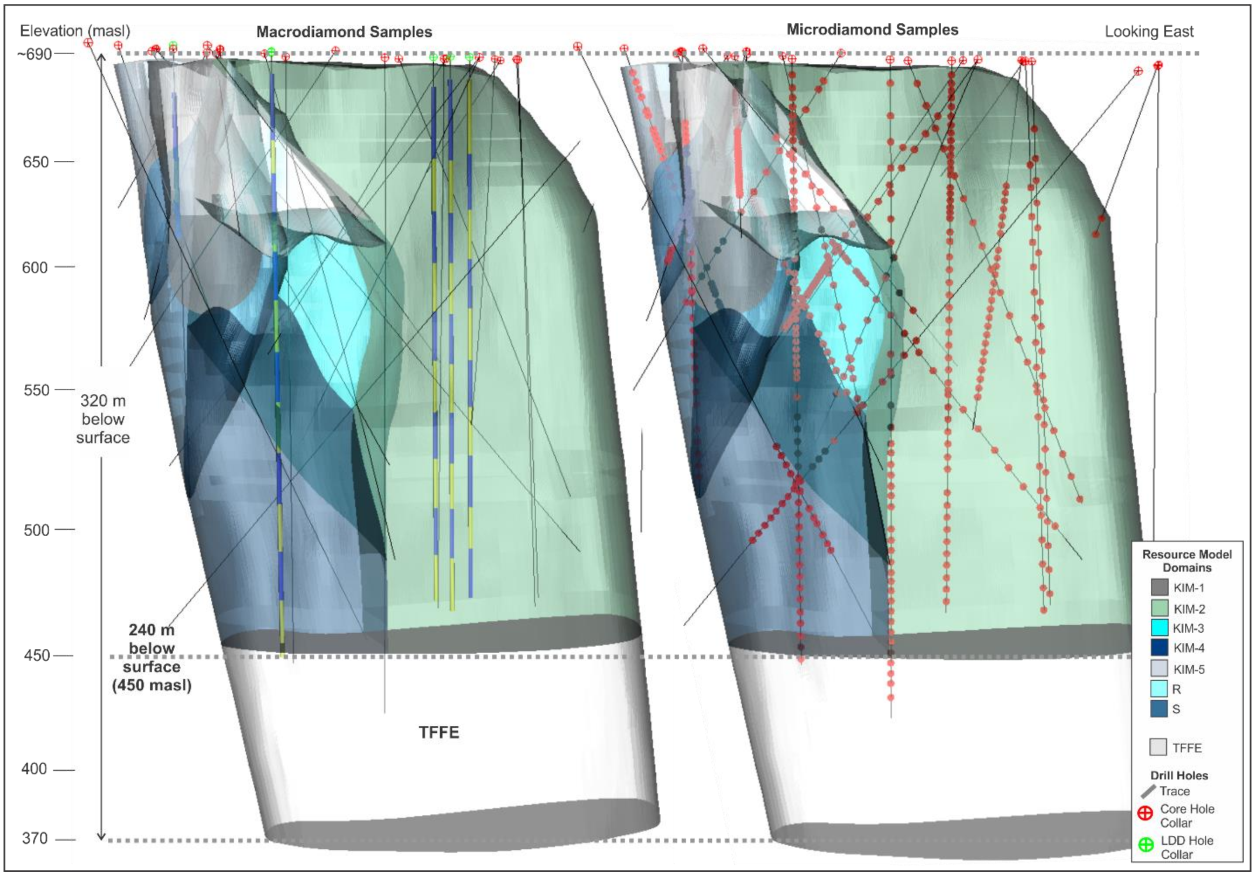
- A surface bulk sample of KIM-1 (49 ct recovered from 47 t) from which a representative microdiamond sample was collected (total of 467 kg). These data were used to develop the KIM-1 total diamond content SFD model.
- Large-diameter drilling of six holes in three locations across the pipe has sampled 809 t in-situ kimberlite from which a combined parcel of 718 ct was recovered from five of the seven resource domains (KIM-2, KIM-3, KIM-4, KIM-5 and R). Microdiamonds have been recovered from 1,183 kg of drill core from pilot holes adjacent to LDD holes. Where possible these results have been used in conjunction with the LDD macrodiamond results to develop total diamond content SFD models.
- An additional 3,616 kg of drill core has been processed for microdiamonds. These results have been used to assess grade and SFD variability and to support grade estimates.
- A surface grab sample of KIM-1 comprising 26 aliquots (205 kg) has been processed for microdiamonds. These results were used to support the development of the KIM-1 total diamond SFD model.

Table 14-21: Microdiamond and Macrodiamond Datasets used to Estimate Grade in CH-7

Dataset	Sample Medium	Aliquots (count)	Mass (t)	Process Bottom Cut Off (µm)	Diamonds	Carats
Macrodiamond	Surface bulk sample	n/a	47.06	850	502	49.07
	Large diameter drill	n/a	809.47	1180	7,362	717.65
Microdiamond	Surface bulk sample	2	0.47	425	71	0.90
	Large diameter drill pilot holes	144	1.18	106	1,800	2.77
	Drill core	451	3.62	106	5,721	7.65
	Surface grab (KIM-1)	26	0.21	75	363	0.79

Only results from within the resource domains are included.

Figure 14-14: CH-7 Resource Model Illustrating Distribution of Macrodiamond and Microdiamond Samples



Three-dimensional view of the CH-7 resource domains showing the spatial distribution of macrodiamond (left) data from LDD holes and microdiamond (right) sample coverage in drill core. The yellow and blue intersections on the left illustrate the intervals sampled for each processing unit. Red dots on the right illustrate the midpoint of individual 8 kg sample aliquots.

14.5.3.3 Microdiamond Stone Frequency and SFD Characteristics

Summary statistics of +212 µm microdiamond stone frequency grouped by resource domain are provided in Table 14-22 and illustrated in

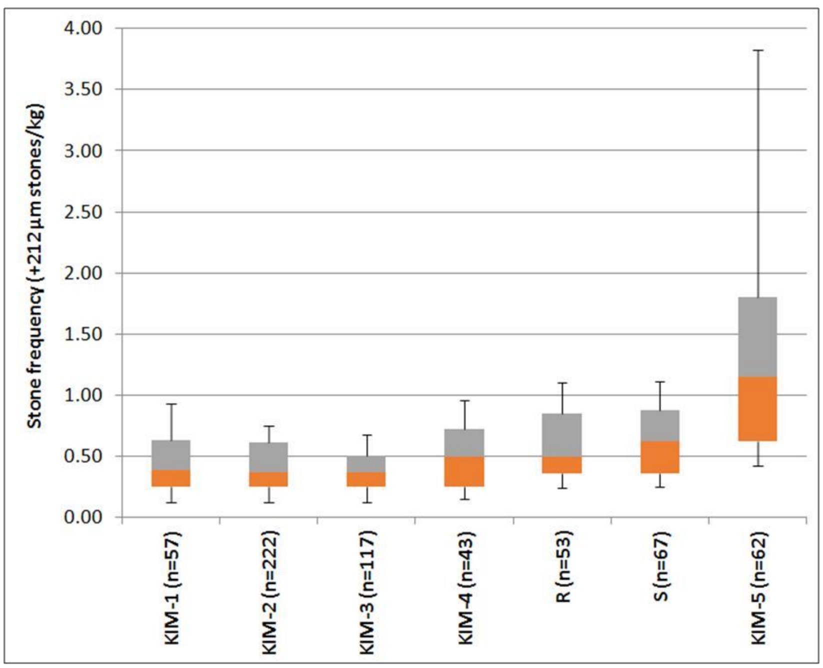
Figure 14-15. The individual domains present varying average stone frequencies, ranging from a low of 0.41 st/kg in KIM-3 to a high of 1.62 st/kg in KIM-5. Stone frequency variation with depth in the domains sufficiently well represented to allow for a meaningful spatial assessment (this excludes Domains KIM-1 and R) is shown in Figure 14-16. Results imply that grade is not likely to vary significantly with depth in any of these domains, with the possible exception of KIM-4 and KIM-5.

Size frequency distributions (SFDs) of microdiamonds from each domain are shown in Figure 14-17. The SFDs of KIM-2, KIM-3, KIM-4, KIM-5 and S are all very similar. The SFDs of KIM-1 and R are similar to each other, but possibly finer grained than those of the remaining domains. Comparison of microdiamond SFDs by elevation range in KIM-2 and KIM-3 (Table 14-22 and Figure 14-18) reveals that SFD does not appear to change meaningfully in KIM-2 or KIM-3 with depth. A spatial analysis of microdiamond SFD characteristics in the other domains is not possible due to their smaller volumes and the more limited microdiamond populations.

Table 14-22: Summary Statistics of Microdiamond Stone Frequency (+212 µm stones/kg) by Domain in CH-7

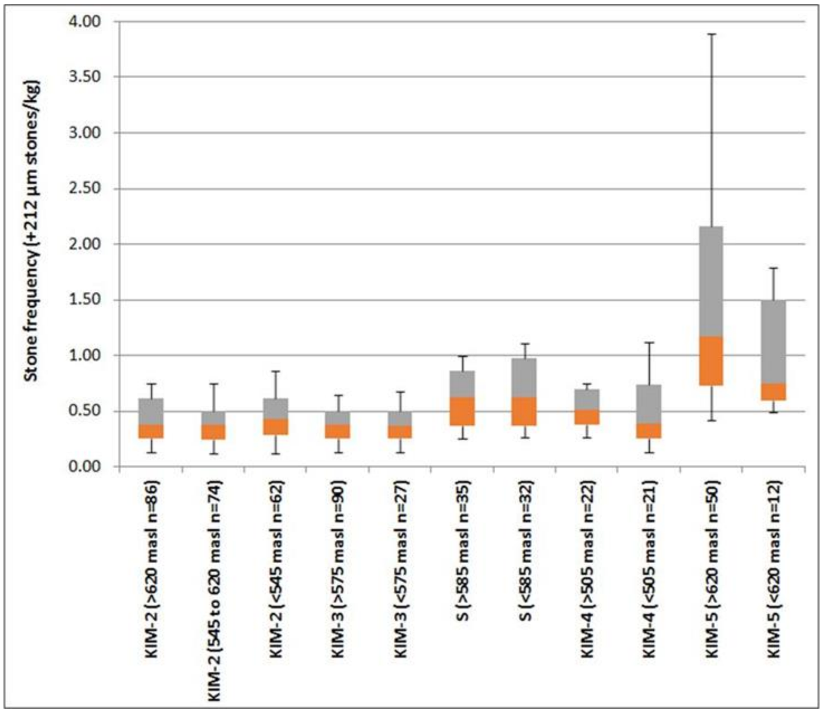
Data Type	Descriptor	KIM-1	KIM-2	KIM-3	KIM-4	KIM-5	R	S	TFFE
Sample information	Count	57	222	117	43	62	53	67	7
	Mass (kg)	452	1,792	937	344	501	431	547	56
+212 µm microdiamond stone frequency statistics	Average	0.50	0.43	0.41	0.51	1.62	0.60	0.66	0.52
	Standard deviation	0.32	0.25	0.24	0.30	1.59	0.39	0.35	0.26
	Median	0.39	0.38	0.37	0.50	1.15	0.50	0.63	0.50
	Maximum	1.38	1.23	1.38	1.14	8.41	1.79	1.62	0.87
	Minimum	0.00	0.00	0.00	0.00	0.13	0.00	0.00	0.12

Figure 14-15: Microdiamond Stone Frequencies (+212 µm st/kg) from Drill Core Samples Grouped by Resource Domain



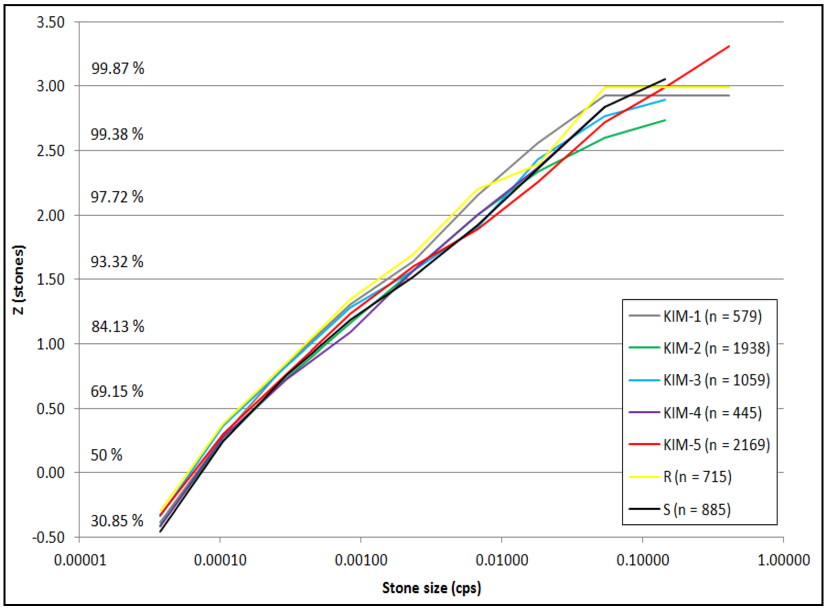
n = number of sample aliquots represented. The orange and grey boxes indicate the 25th to 75th percentile values and the contact between them is the median. Error bars represent the 10th and 90th percentile values.

Figure 14-16: Microdiamond Stone Frequencies (+212 µm st/kg) from Drill Core Samples Grouped by Resource Domain in Selected Elevation Ranges



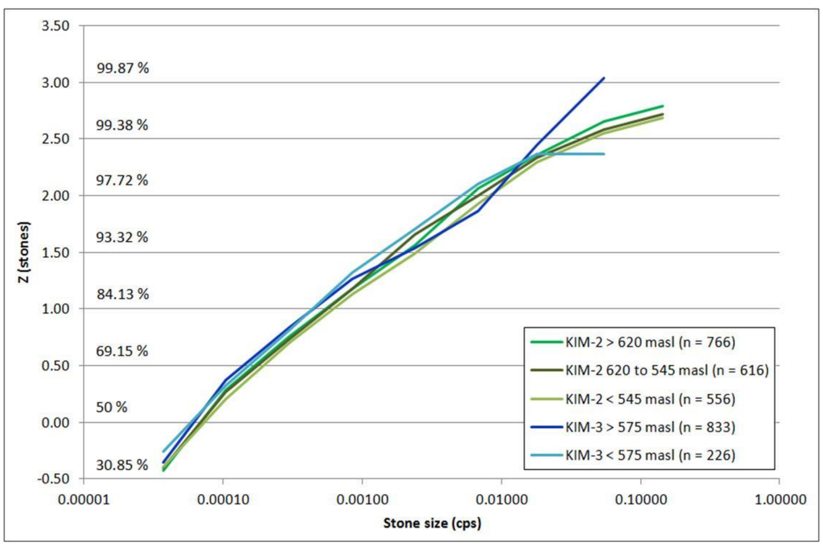
n = number of sample aliquots represented. The orange and grey boxes indicate the 25th to 75th percentile values and the contact between them is the median. Error bars represent the 10th and 90th percentile values.

Figure 14-17: Microdiamond SFDs (+106 µm) for the CH-7 Resource Domains



SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size); cps = carats per stone; n = number of +106 µm stones illustrated.

Figure 14-18: Microdiamond SFDs (+106 µm) for KIM-2 and KIM-3 by Elevation Range



The same elevation ranges illustrated in Figure 14-16 for these two domains are used. SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size); cps = carats per stone; n = number of +106 µm stones illustrated.

14.5.3.4 Macrodiamond Stone Frequency and SFD Characteristics

Large-diameter drill (LDD) sampling macrodiamond results are summarized by domain in Table 14-23 and compared to results from the KIM-1 surface bulk sample. The LDD stone frequency (+1180 µm st/t) results by processing unit and resource domain are shown in Figure 14-19. Stone frequency results for KIM-2 are consistent with depth and do not provide any indication of significant grade variation beyond that expected to be present in the relatively small 10-15 t (dry) processing units (Figure 14-19). Limited sampling results in KIM-3 imply a possible decrease in grade with depth. However, the number and size of samples do not conclusively resolve a grade change, and the scale of the apparent grade change, particularly in the context of this small domain, is insufficient to justify a model of changing grade with depth in KIM-3. The remaining resource domains (KIM-4 and R) are poorly represented spatially and their LDD grade results are highly variable. This variability is very likely controlled by the observed presence of different kimberlite units in both domains (MSC16/010R). The results from KIM-4 in particular clearly imply the presence of higher and lower grade units.

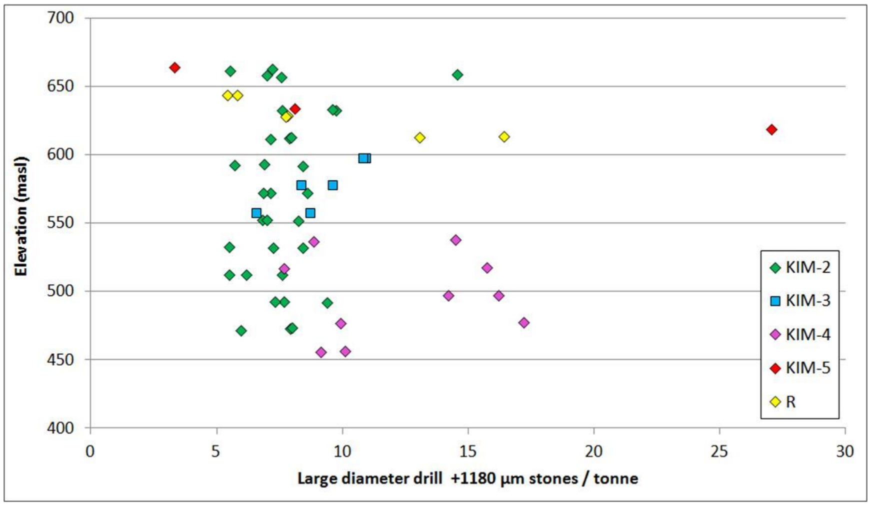
Grade results from the three processing units in KIM-5 are extremely variable. This likely reflects the extensive collapsing of material down hole that occurred during drilling. The overall volume of the KIM-5 LDD sample is considered to be well constrained through volumetric survey (caliper), however the majority of the sloughed volume (added to the upper sample increment) collapsed and was recovered during deeper drilling, and results from the individual processing units are thus not reliable on a singular basis.

Table 14-23: CH-7 Macrodiamond Data from LDD and Surface Bulk Sampling Including Stone Frequency Results (+1180 µm st/t) for Each Domain

Resource Domain	Sampling Method	Process Units	Weight (t)	Total Stones	Total Carats	st/t	ct/t	Minimum st/t	Maximum st/t
KIM-1	Surface trench	1	47.06	354	47.29	7.52	1.00	N/a	N/a
KIM-2	LDD drill	34	476.53	3680	363.66	7.72	0.76	5.47	14.54
KIM-3	LDD drill	6	83.04	756	68.14	9.10	0.82	6.55	10.91
KIM-4	LDD drill	10	144.29	1776	157.93	12.31	1.09	7.67	17.19
KIM-5	LDD drill	3	45.68	432	49.98	9.46	1.09	3.29	27.02
R	LDD drill	6	59.92	517	67.84	8.63	1.13	5.43	16.41
Audit*	LDD drill	N/a	0.00	201	10.11				

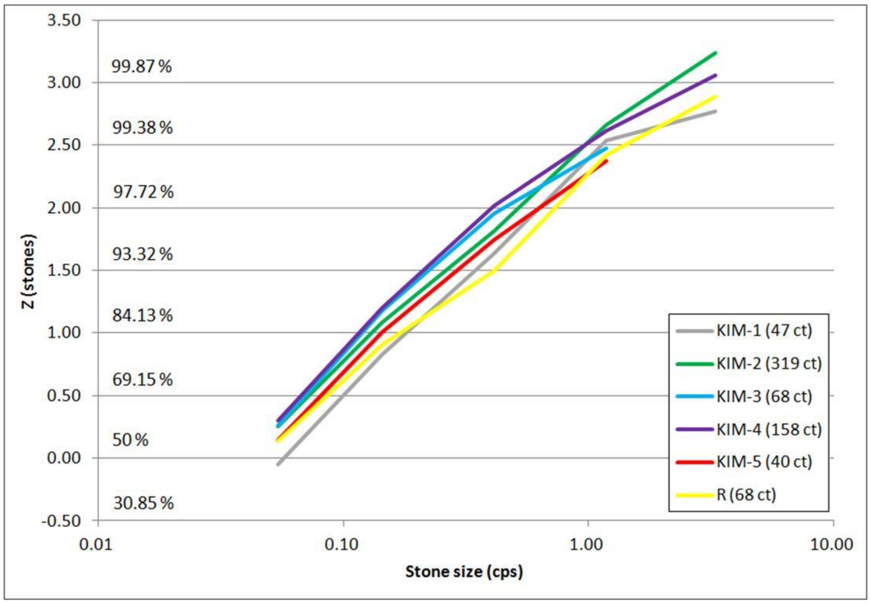
*Audit work carried out on hole LD05 (KIM-5) recovery tailings, as discussed in Section 11.6.8.
KIM-1 data excludes 850 µm results.

Figure 14-19: LDD Stone Frequency (+1180 µm st/t) Results from Discrete Processing Units by Domain



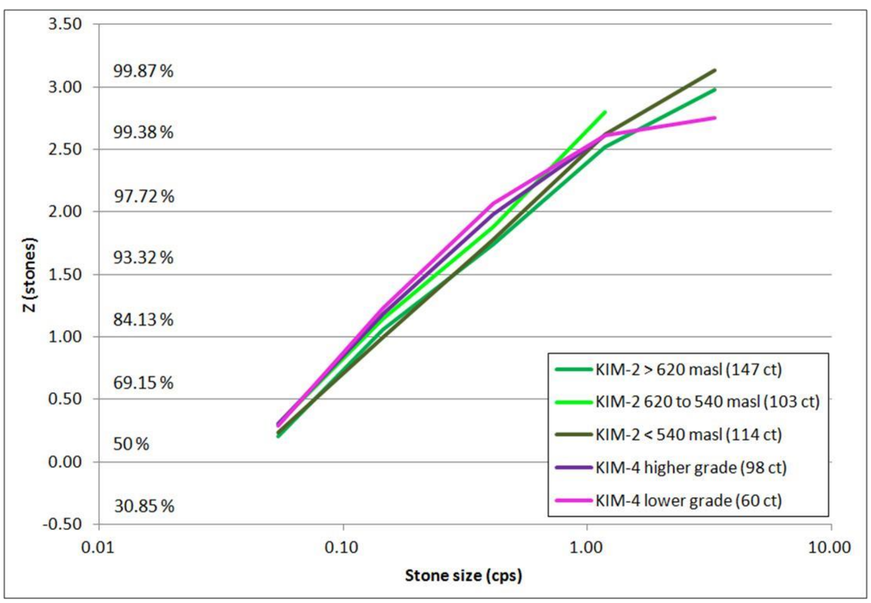
The SFDs for each macrodiamond parcel (including surface bulk sample results for KIM-1) are shown in Figure 14-20. Differences between the domains are present, however it is unclear if these represent true SFD variations or if they are artefacts of the generally small parcel sizes. KIM-1 presents a markedly different SFD in the finer size ranges. This is potentially related to the sampling method (KIM-1 was excavated at surface while remaining domains were sampled through LDD drilling). The SFD within KIM-2 by elevation range and within the higher and lower grade parcels derived from KIM-4 displays encouraging similarity despite the small parcel sizes (Figure 14-21), suggesting continuity in SFD within these domains.

Figure 14-20: Macrodiamond SFDs (+1180 µm) for the CH-7 Resource Domains



SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size); cps = carats per stone.

Figure 14-21: Macrodiamond SFDs (+1180 µm) for KIM-2 by Elevation Range and for Groupings of Higher and Lower Grade Results in KIM-4



Results indicate good internal SFD continuity for these domains. SFD is shown on a cumulative log probability plot (showing the proportion of diamonds below a given stone size); cps = carats per stone.

14.5.3.5 Total Diamond Content SFD Models and Recovery Corrections

Total (+212 µm) diamond content size frequency distribution (SFD) models were constructed for the KIM-1, KIM-2 and KIM-3 domains where this was supported by the available LDD sample data and associated microdiamond data from pilot drill cores. In the case of KIM-1, the SFD model was based on macrodiamond data from the KIM-1 surface bulk sample and microdiamond data from the KIM-1 surface grab sample (Table 14-24). For KIM-2 and KIM-3, SFD models were based on macrodiamond data from the LDD bulk samples and microdiamond data for equivalent volumes of kimberlite as sampled by pilot core drill holes adjacent to the LDD holes. The micro- and macrodiamond data used to define these SFD models are provided in Table 14-25. The SFD models were used, in conjunction with more spatially representative drill core microdiamond data, to estimate average grades for the KIM-1, KIM-3 and S domains. Drill core microdiamond data in KIM-1 and KIM-3 suggest the overall domain grades will differ from their respective bulk sample grades. Average grade for the S domain (for which no macrodiamond parcel is available) has been estimated using its domain-wide drill core microdiamond data in conjunction with the KIM-2 total content SFD. This approach is considered justified based on the similarity between the KIM-2 and the S microdiamond SFD. The total content SFD models that were used as a basis for grade estimation are provided in Table 14-22, along with the recovery corrections applied (based on recovery efficiency achieved during sample processing) to convert total (+212 µm) diamond content to that recoverable at a 1180 µm bottom cut off.

Table 14-24: Microdiamond and Macrodiamond Parcels Used to Define the Total Diamond Content SFD Models for KIM-1, KIM-2 and KIM-3

Domain	KIM-1				KIM-2				KIM-3			
Parcel	Microdiamonds		Macrodiamonds		Microdiamonds		Macrodiamonds*		Microdiamonds		Macrodiamonds	
Dry Mass	205.1 kg		47.06 t		403.6 kg		429.25 t		129.5 kg		83.04 t	
Size Class	St	ct	St	ct	St	ct	St	ct	St	ct	St	ct
+75 µm	94	0.00090										
+106 µm	94	0.00223			144	0.00328			45	0.00115		
+150 µm	68	0.00458			124	0.00822			40	0.00262		
+212 µm	51	0.00983			60	0.01092			32	0.00593		
+300 µm	35	0.01960			43	0.02023			16	0.00852		
+425 µm	11	0.01635			24	0.03474			7	0.00782		
+600 µm	8	0.03666			23	0.09156			3	0.01185		
+850 µm	0	0.00000	148	1.78	7	0.08048			2	0.01690		
+1180 µm	1	0.05899	170	5.49	2	0.03441	2,008	67.83			455	15.26
+1700 µm	0	0.00000	112	10.09	1	0.09478	868	73.81			211	17.68
+2360 µm	0	0.00000	54	12.23	3	0.90511	349	84.11			71	17.10
+3350 µm	1	0.64033	16	10.77			104	67.29			14	9.24
+4750 µm			1	2.18			11	18.40			5	8.86
+6700 µm			1	6.53			2	7.39			0	0.00
Total	**363**	**0.78946**	**502**	**49.07**	**431**	**1.28373**	**3,342**	**318.82**	**145**	**0.05477**	**756**	**68.14**

*Only diamonds from LD01, LD02 and LD06 were used to define the KIM-2 total diamond content SFD model

Table 14-25: SFD Models for CH-7 KIM-1, KIM-2 and KIM-3

Size Class	Total Content (+212 µm) SFD Models (% ct)			Recovery Corrections (%)	Recoverable (+1180 µm) SFD Models (% ct)		
	KIM-1	KIM-2	KIM-3		KIM-1	KIM-2	KIM-3
+212 µm	2.57	2.41	2.37	0			
+300 µm	4.29	4.27	4.48	0			
+425 µm	6.32	6.86	7.11	0			
+600 µm	8.80	10.48	10.80	0			
+850 µm	10.53	13.10	13.31	0			
+1180 µm	4.25	5.02	5.10	55	3.84	5.00	5.17
+3 DTC	8.32	9.86	9.80	72	9.85	12.85	13.02
+5 DTC	10.66	11.60	12.03	82	14.38	17.22	18.20
+7 DTC	6.39	6.84	6.68	92	9.66	11.39	11.33
+9 DTC	8.27	8.30	7.99	100	13.60	15.03	14.74
+11 DTC	10.31	9.70	8.71	100	16.95	17.56	16.06
+13 DTC	6.04	4.74	4.23	100	9.94	8.57	7.80
+15 DTC	1.73	1.20	1.15	100	2.84	2.17	2.12
+17 DTC	2.63	1.64	1.68	100	4.32	2.97	3.11
+19 DTC	4.30	2.22	2.50	100	7.07	4.02	4.61
+21 DTC	2.92	1.19	1.44	100	4.81	2.16	2.66
+23 DTC	0.74	0.27	0.32	100	1.21	0.48	0.59
+10.8 ct	0.46	0.16	0.17	100	0.75	0.29	0.31
+15 ct	0.26	0.09	0.09	100	0.44	0.16	0.16
+20 ct	0.21	0.08	0.06	100	0.35	0.14	0.12

Models of total content (+ 212 µm) and recoverable SFD (expressed as percent carats in each size class) used as the basis for average grade estimates in domains KIM-1, KIM-3 and S (adopted KIM-2 model). The recovery corrections used to convert the total content to recoverable SFD models are shown, and are based on the actual +1180 µm recovery efficiency achieved during sample processing.

14.5.3.6 Grade Estimates

Grade estimates for the CH-7 resource domains are provided in Table 14-26. All grades are reported as those recoverable at an 1180 µm bottom cut off, and reflect the actual recovery efficiency achieved during sample processing. These estimates will therefore need to be adjusted to reflect the expected recovery efficiency of the planned production processing plant. In all cases, average grades have been adopted. The basis for each grade estimate is as follows:

- KIM-1: microdiamond sample coverage from drill core in this domain is limited but is more spatially representative than the single bulk sample at surface. Drill core results were therefore applied to the calibrated ratio between microdiamond stone frequency and recoverable grade for KIM-1 to estimate an average grade slightly lower than that recovered in the bulk sample at surface;
- KIM-2: the large and spatially representative drill core microdiamond dataset, as well as the macrodiamond data available, imply good internal SFD and grade continuity for the whole KIM-2 domain. The recovered LDD bulk sample grade was therefore adopted as the average grade for the domain;
- KIM-3: microdiamond data suggest that the overall grade of the KIM-3 domain will be lower than that of the LDD bulk sample (the average stone frequency for drill core samples throughout the domain is lower than was recovered from the core pilot hole adjacent to the KIM-3 LDD samples). Core drilling indicates that dilution is elevated towards the base of KIM-3, mainly in proximity to the domain boundaries, and the average dilution in the LDD pilot core hole is lower than that of the whole domain. Core drilling has also revealed the presence of minor amounts of different kimberlite units (e.g. KIM-6) within the domain. The effect of these units on grade is not constrained, and it is thus not clear if the discrepancy in average microdiamond stone frequency between the domain as a whole and the LDD pilot hole is controlled by varying proportions of different kimberlite units or by varying degrees of dilution, or if it simply reflects the small sample size derived from the pilot hole. Average microdiamond stone frequencies from the KIM-3 domain were applied to the calibrated ratio between microdiamond stone frequency and recoverable grade for KIM-3 to generate an average grade estimate slightly lower than that recovered by LDD sampling;
- KIM-4: the recovered LDD grade has been adopted as the average grade estimate for the KIM-4 domain. Results from the two LDD holes intersecting KIM-4 are very different (average sample grades of 1.32 and 0.85 ct/t, respectively), despite them being drilled in close proximity to each other. This is thought to reflect the presence of different kimberlite units with different diamond grade. Comparison of microdiamond results from the domain as a whole to those derived from the pilot hole adjacent to the KIM-4 LDD holes suggests that on average the grade of KIM-4 may be higher than that reflected in the LDD sample, suggesting that the proportion of lower grade material in the domain as whole is lower than that reflected in the LDD sampling. Due to the poorly constrained geology of KIM-4 and the variable nature of the LDD sample results it is not possible to incorporate this potential upside into the estimate for KIM-4 (see Section 14.5.5.2 for further discussion on grade uncertainty) and the average sample grade has been adopted as a conservative best estimate of the domain grade;
- KIM-5: the recovered LDD grade has been adopted as the average grade estimate for the KIM-5 domain above 620 masl (70 mbs). KIM-5 grade and bulk density data suggest that mass loss in kimberlite related to alteration / weathering has a significant control on grade per unit mass. The

distribution of "red-mud" intervals within KIM-5 simultaneously decreases bulk density and increases grade, suggesting that on a volume basis the grade may be more consistent than is apparent in microdiamond results, which are on a grade per unit mass basis. No "red-mud" units were intersected in drill core below 620 masl and this small portion of KIM-5 was not intersected by the LDD holes. Due to the lack of red mud, a significantly higher average bulk density was assigned to KIM-5 material below 620 masl (Section 14.5.2). The average grade for this zone was based on the recovered LDD grade for KIM-5 proportionally corrected downwards by the ratio between microdiamond stone frequency in KIM-5 above 620 masl relative to that below 620 masl;

- R: comparison of microdiamond results from the R domain as a whole to those derived from the pilot hole adjacent to the LDD samples of this domain suggests that on average the grade may be higher than that reflected in the LDD sample. The geology of R is complex, encompassing multiple different kimberlite units, and the variable grade results appear to reflect this complexity. Based on the results available, it is not possible to define a total content SFD model that is sufficiently constrained for use in grade estimation. The (potentially conservative) recovered LDD grade was therefore adopted as the average grade estimate for the R domain; and
- S: no macrodiamond data are available for the S domain. Based on the similarity in microdiamond SFD between the S and KIM-2 domains (refer to Figure 14-17), the total content SFD model for KIM-2 has been used, in conjunction with S domain drill core microdiamond stone frequencies, to estimate average recoverable grade for the S domain.

Table 14-26: Average Grade Estimates for CH-7 Resource Domains

Resource Domain	Bulk Sample Results (+1180 µm)			Domain Recoverable Grade (+1180 µm ct/t)
	Weight (t)	Diamonds (ct)	Grade (ct/t)	
KIM-1	47.06	47.29	1.00	0.94
KIM-2	476.53	363.66	0.76	0.76
KIM-3	83.04	68.14	0.82	0.71
KIM-4	144.29	157.93	1.09	1.09
KIM-5 > 620 masl	45.68	49.98	1.09	1.09
KIM-5 < 620 masl				0.90
R	59.92	67.84	1.13	1.13
S	N/a	N/a	N/a	1.12

Grades are reported on a recoverable basis at an 1180 µm bottom cut off and reflect the recovery efficiency of the sample processing plant used to treat the bulk samples.

14.5.4 Diamond Value

14.5.4.1 Valuation

A diamond parcel of 735.75 ct from CH-7 was valued by WWW International Diamond Consultants Ltd (WWW) in February 2016. WWW (2016b) describes the parcel valued as presenting well in terms of quality, colour and shape. Approximately 25-30% of the diamonds are classified as white gems and the proportion of boart diamond is low. Brown diamonds make up 25-30% of the parcel. Very few stones display strong fluorescence.

The valuation exercise was updated in April 2018 using the WWW's March 31, 2018 price book and an average actual diamond value of $103 US$/ct was obtained. The results of this valuation exercise (extracted from WWW, 2018b) are presented in Table 14-27. The parcel was delivered and assessed as six sub-parcels derived from resource domains KIM-1 to KIM-5, inclusively, with diamonds derived from weathered KIM-2 material also assessed separately. The average diamond values for these smaller parcels (ranging from 44 to 306 ct) varied from 82 US$/ct to 142 US$/ct with an overall parcel average value of 103 US$/ct. No significant differences in terms of diamond characteristics or value distribution between domains were noted, however the parcels were considered too small to confirm a consistent value distribution between domains. While there is no clear indication that the samples have different SFDs, the parcels are too small to reliably confirm this, and further sampling may resolve SFD differences between domains (WWW, 2018b).

Table 14-27: April 2018 Re-pricing Results for a Parcel of 735.75 ct from CH-7 in US$/ct

Size Class	Carats	Stones	Average Carats per Stone	$/ct
+6 ct	6.50	1	6.50	31
+5 ct	15.68	3	5.23	1,434
+4 ct	8.59	2	4.30	89
+3 ct	6.39	2	3.20	128
+10 gr	7.79	3	2.60	21
+8 gr	17.37	8	2.17	722
+6 gr	24.08	15	1.61	220
+5 gr	10.23	8	1.28	203
+4 gr	28.27	28	1.01	130
+3 gr	37.03	48	0.77	114
+11 DTC	145.55	356	0.41	65
+9 DTC	113.45	548	0.21	48
+7 DTC	73.37	591	0.12	35
+5 DTC	155.36	2,449	0.06	27
+3 DTC	86.09	2,719	0.03	20
Total	**735.75**	**6,781**	**0.11**	**103**

Source: Extracted from WWW, 2018b

WWW does not ordinarily quantify or specifically track breakage during valuations and in this instance did not notice any significant difference in the amount of fresh diamond damage when compared to previous samples valued (WWW, 2016b). An independent assessment of diamond breakage (McCandless, 2016a) has however noted that diamond fragmentation in the CH-7 diamond parcel is significant, with only 10-15% of the diamonds presenting no breakage. Breakage was characterized as aggressive, with percussion marks and abrasion present, and is considered to be primarily related to the LDD sample collection method.

A modelled estimate of average diamond value was generated by WWW (2018b) based on the parcel valuation data by combining a value distribution model (model of average diamond value by size class) with a single SFD model representing the proportion of diamond (by weight) expected in each size class. These models yield an estimated average diamond value of 114 US$/ct (Table 14-28). This average value reflects the recovery efficiency of the process plant used to treat the CH-7 bulk samples and assumes that no diamonds smaller than DTC+3 will be recovered. This recoverable average value corresponds with the recoverable grades reported in Table 14-25. While the bottom cut off of DTC+3 is not precisely the same as the 1180 µm square mesh cut off basis for the grade estimate, these are very similar to each other and for the purpose of an Inferred Mineral Resource estimate, have been considered equivalent.

Table 14-28: April 2018 Average Modelled Diamond Value for CH-7

Size Class	Model SFD (% ct)	Value Distribution ($/ct per size class)
+10.8 ct	0.99	550
+10 ct	0.15	1,020
+9 ct	0.19	1,020
+8 ct	0.24	1,020
+7 ct	0.32	1,065
+6 ct	0.44	1,065
+5 ct	0.64	1,050
+4 ct	0.99	875
+3 ct	1.71	770
+10 gr	0.77	590
+8 gr	2.74	390
+6 gr	2.66	265
+5 gr	1.91	205
+4 gr	4.17	135
+3 gr	5.43	80
+11 DTC	16.75	65
+9 DTC	15.87	48
+7 DTC	10.26	35
+5 DTC	21.73	27
+3 DTC	12.04	20
Average Model $/ct		**114**

Values are reported in US$/ct. The average value reflects the process efficiency of the plant used to treat the CH-7 bulk samples, and assumes no recovery of diamond smaller than DTC+3. The SFD, value distribution and average $/ct values in this table were extracted from WWW (2018b).

Source: WWW (2018b)

14.5.4.2 Comment on 2016 CH-7 Diamond Value (from Nowicki et al., 2016)

The WWW (2016b) average value represents their best estimate of diamond value per size class applied to a model of diamond SFD (for diamonds larger than DTC+3) for CH-7. WWW are recognised international leaders in the field of diamond valuation and the QPs for the previous technical report (Nowicki et al., 2016) believe it is reasonable to rely on the diamond values provided.

Bulk sample results suggest that the resource domains will present different SFDs when resolved with larger diamond parcels. The impact of varying SFD was assessed by modelling SFD for individual domains (where possible, and in all cases at low levels of confidence) and comparing the average values derived in this way with the declared average value for the Mineral Resource estimate. The range of values suggests that the use of a single SFD to generate average value for all resource domains is valid based on the resolution of the data available. See Section 14.5.5.3 for more discussion on value uncertainty.

14.5.5 Confidence and Resource Classification

14.5.5.1 Volume and Tonnage

The drill coverage and number of external pierce points obtained are considered sufficient to have constrained the overall volume of the CH-7 pipe, from surface to an elevation of 450 masl (240 mbs), to a high level of confidence. This represents the portion of the pipe for which Mineral Resource estimates are being made. Confidence in volume below 450 masl to the base of the geological model at 370 masl (320 mbs; the portion of the pipe classified as TFFE) decreases substantially.

The internal boundaries of the resource domains are based on visually logged boundaries between the geological domains, supported by petrographic analysis of core slabs and thin sections, and whole rock chemistry. On an individual basis the smaller geological domain boundaries (e.g. KIM-1, KIM-5) are less well constrained – it is possible that increased drill resolution could result in significant adjustments to the volumes of these small domains. Several of the currently defined resource domains (KIM-4, R and S) are geologically complex and may be further refined into additional domains with improved drill resolution. Uncertainty in the relative volumes of these small domains is only relevant to tonnage estimates if the different domains have significantly different bulk densities. While there are localized instances where uncertainty in boundaries between domains could introduce uncertainties in overall tonnage estimates of up to ±30% (e.g. the boundary between KIM-1 and KIM-2 in shallow portions of the pipe), due to the small volumes of these units, the extent of this error will likely not be relevant on a scale pertinent to mining and monthly / quarterly resource reconciliations. Uncertainty in the volumes of the internal domains is also relevant to grade uncertainty. This is discussed further in Section 14.5.5.2.

Bulk density in CH-7 is considered to be constrained to a high level of confidence in all domains with the exception of KIM-5, which represents <3% of the CH-7 Mineral Resource by volume. Even in KIM-5, where substantial small-scale bulk density variation is likely to manifest as a result of localized alteration, it is unlikely that bulk density variation will result in tonnage estimate inaccuracies on a scale pertinent to mining and resource reconciliation.

The overall volumes and tonnages for the CH-7 Mineral Resource estimate are considered to be constrained to a level of confidence acceptable for classification of Indicated Mineral Resources.

14.5.5.2 Diamond Grade

The macrodiamond parcels obtained from the resource domains of CH-7, which form the basis for all grade estimates, are generally small (refer to Figure 14-21). The small sample sizes introduce two aspects of grade uncertainty. The first is that a small sample may not be spatially or geologically representative of a domain and may misrepresent the average grade. The impact of this on the CH-7 grade estimates was assessed through review of the geological nature and grade information (micro- and macrodiamond) for each domain, to assess the overall continuity and the extent to which grade is likely to vary internally. The second is that coarse diamonds, which can contribute significantly to overall grade, are usually not adequately represented in small diamond parcels. Extreme value plots (representing cumulative grade with increasing sieve size class) were assessed and high / low case error range modelling of the macrodiamond SFD (where possible based on the data available) was carried out to gauge the scale of this potential error. In both instances, the degree of uncertainty introduced by small sample size is considered to be within acceptable levels for an Inferred Mineral Resource estimate.

Diamonds recovered during LDD drilling at CH-7 have been subjected to a degree of breakage that is apparently higher than would typically be incurred during conventional mining. The grade estimates for all CH-7 domains other than KIM-1 are based on the diamonds recovered through LDD drilling. The impact of excessive breakage on diamond populations would be to reduce the proportions of larger stones present and to reduce grade through loss of diamond fragments smaller than the recovery bottom cut off size. McCandless (2016c) has assessed the breakage characteristics of diamonds recovered by LDD drilling in comparison with those from the KIM-1 surface bulk sample and has estimated that excess breakage related to LDD drilling could have resulted in a grade loss of 8% to 15%. This potential grade upside (for all domains other than KIM-1) has not been factored into the Mineral Resource grade estimates.

The grade estimates for KIM-1, KIM-3 and S are based on a calibration of microdiamond stone frequency to recovered macrodiamond grade, and on an assumption of SFD continuity within these domains. Incorrect calibration of this relationship could occur if the material sampled for microdiamonds is not the same average grade as that comprising the bulk sample. The microdiamond sample results used to calibrate total content SFD curves (where they were used as a basis for grade estimation) have been assessed and the potential error associated with this calibration is considered to be less than ± 20%. Based on the degree of variability observed in the overall deposit in terms of the relationships between micro- and macrodiamonds and SFD variation within domains (where this can be properly assessed), it is considered unlikely that varying SFD (within domains) will introduce a degree of grade uncertainty beyond that acceptable for an Inferred Mineral Resource.

Aspects of grade uncertainty relevant to specific resource domains of CH-7 are discussed in the points below.

- KIM-1: The grade estimate for KIM-1 is based on a small microdiamond dataset derived from drill core with limited spatial coverage of the domain. These recoveries are however more spatially representative than the surface bulk sample from a single location at surface. Based on the degree of variation observed in the sample aliquot data it is not considered likely that this will introduce a significant degree of uncertainty into the KIM-1 grade estimate. The KIM-1 total content SFD model is based on a microdiamond dataset (surface grab sample) that is not fully spatially representative of the macrodiamond sample. The representative microdiamond sample collected during excavation of the macrodiamond sample was processed at a bottom cut off of 425 µm, and could therefore not be used to model a total +212 µm distribution. The +425 µm results from the representative microdiamond sample were however found to correlate very closely with the total content model established, and it is considered unlikely that any significant error would be introduced by this approach.

- KIM-2: The adoption of the KIM-2 recovered LDD grade as the average grade estimate for KIM-2 assumes grade (and SFD) constancy within this large domain. KIM-2 is well represented by microdiamond data, which display no significant variability in grade or SFD internally and are comparable with microdiamond data from pilot core holes drilled directly adjacent to LDD holes. LDD macrodiamond results display good grade and SFD constancy with depth. The assumption of grade and SFD continuity within KIM-2 is considered well supported;
- KIM-3: Available grade and geological information suggests that grade variation in KIM-3 will be controlled by varying dilution and/or the presence of minor amounts of different kimberlite units. The estimate for KIM-3 is based on a large and well distributed microdiamond parcel, and potential grade variation is thought to be adequately represented on an overall basis;
- KIM-4: Highly variable LDD grades and varying geology in the pilot core hole directly adjacent to the LDD holes suggest that grade variation in KIM-4 is likely controlled by the presence of different kimberlite units. The current drill coverage is not adequate to allow these units to be resolved into separate domains, and the grade information available is not at sufficient resolution to constrain the grade of individual units. The average LDD grade adopted is considered to be of low confidence, but with possible upside as microdiamond stone frequencies from all drill cores in KIM-4 are higher on average than those from the pilot core hole;
- KIM-5: The limited and highly variable grade (micro- and macrodiamond) results available for KIM-5 did not permit a conventional assessment of grade uncertainty. Two alternative approaches to estimation of grade in KIM-5 were applied to gauge confidence levels in the current estimate. In the first alternative approach, LDD average grade by volume was converted to grade per unit mass using bulk density for KIM-5 above and below 620 masl. In the second alternative approach, a total content SFD model for KIM-5 was defined (as best possible based on the available data) and was used in conjunction with drill core microdiamond stone frequency data from above and below 620 masl to estimate grade. Results did not vary by more than ±30% from the current estimate;
- R: This geologically complex domain is comprised of different kimberlite units that cannot be spatially resolved with the current drill coverage. Adoption of the LDD grade as the average grade for R is possibly conservative, as the average microdiamond content for the domain (from all drill core samples) is higher than that in the pilot core hole directly adjacent to the LDD holes; and
- S: The grade estimate for this domain is based on an assumption of SFD continuity between it and KIM-2. Based on the range of SFDs (where adequately constrained) in other domains and considering that the main kimberlite units present in S display similar components and textures to other CH-7 kimberlite units (e.g. KIM-5), it is unlikely that an SFD different to that assumed could introduce error beyond the range of -20 / +30%.

The average grade estimate for KIM-2 is constrained to a higher level of confidence than the other domains. KIM-2 comprises ~60% by volume of the CH-7 Mineral Resource estimate. The estimates for the remaining domains are subject to higher degrees of uncertainty. However, individually these domains represent very limited volumes within CH-7, and the impact of this uncertainty is therefore largely mitigated in the overall context of the Mineral Resource estimate.

As discussed in Section 14.5.5.1, the individual volumes of most of the smaller domains (KIM-1, KIM-5, R and S) are poorly constrained. However, the grade estimates for these domains, which are all located in the north of CH-7, are all very similar and range from 0.9 ct/t to 1.13 ct/t. The impact of this volume uncertainty on grade is therefore very limited, and is less than the uncertainty on the grade and value estimates themselves.

All grade estimates for CH-7 are domain average estimates. The data suggest that grade in geologically complex / poorly resolved domains (KIM-4, R and, to a lesser degree, KIM-3) could vary significantly (up to ± 30%) locally on a scale pertinent to mining and grade reconciliation. Forward planning and economic studies should take cognisance of this probable local grade variation. Grade variation is likely to be less pronounced in the more geologically uniform domains, particularly in KIM-2.

The grade estimates for CH-7 are constrained to a level of confidence appropriate for Inferred Mineral Resources.

14.5.5.3 Diamond Value

Uncertainty in the average diamond value for CH-7 derives from two main factors: (1) uncertainty in value distribution (value per size class), particularly the less well represented coarser size ranges, due to the limited size of the parcel valued; and (2) uncertainty in the diamond size frequency distribution (SFD) to which the value distribution has been applied to generate an average recoverable diamond value for all CH-7 resource domains.

Uncertainty associated with the value distribution model has been assessed by WWW (2018b) through the modelling of high and low value distributions that represent the range of uncertainty present. These models translate to an uncertainty range of -15 to +40%. Uncertainty associated with the degree of accuracy with which the SFD model used to generate average value has been constrained is considered to be on a similar or lower scale, based on the average values generated by high (coarse) and low (fine) SFD models that represent the range of uncertainty present in the SFD.

Bulk sample results imply that the resource domains may yield different SFDs when resolved with larger diamond parcels. The uncertainty associated with the assumption of a single SFD as a basis for all value estimates was assessed by modelling SFD for individual domains (where possible, and in all cases at low levels of confidence) and comparing the average values derived in this way with the declared average value for the Mineral Resource estimate. The range of values suggests that the use of a single SFD to generate average value for all resource domains is acceptable based on the resolution of the data available.

Breakage of diamond during LDD sampling is likely to have negatively impacted the valuation of diamonds from CH-7. However, it is not possible to quantify to what extent diamond breakage during LDD drilling may have exceeded that expected to occur during processing of CH-7 ore through a conventional DMS plant. No attempt has been made to correct the average value for diamond breakage.

The average diamond value reported for CH-7 remains subject to significant uncertainty and it is possible that additional sampling may resolve different SFDs and hence different average values for at least some of the resource domains. Average value is however still constrained to a level of confidence appropriate for the reporting of Inferred Mineral Resources.

14.5.5.4 Summary of Confidence and Resource Classification

The level of confidence to which each major component of the CH-7 Mineral Resource estimate is constrained is shown in Table 14-29. The overall resource classification is based on that of the lowest confidence component.

Table 14-29: CH-7 Mineral Resource Estimate Confidence Levels

Body	Volume	Tonnage	Grade	Value	Resource Classification
CH-7	IND	IND	INF	INF	INF

Confidence with which each major component of the CH-7 Mineral Resource estimate is constrained. The overall Mineral Resource is classified at an Inferred level of confidence.

14.6 Mineral Resource Statement

A Mineral Resource statement for the Chidliak Project that includes all currently defined Mineral Resources is presented in Table 14-30. All grades are reported as those recoverable above a 1.18 mm bottom cut off and assume the recovery efficiency achieved in the sample process plants used to treat Chidliak kimberlite and recover diamonds from surface excavation and large-diameter drill (LDD) samples. The recoverable grade estimates would typically be adjusted for the expected recovery efficiency of the planned production processing plant. Average US$/ct values have been derived by applying best estimate value distribution models to models of recoverable diamond SFD, and therefore also represent "recoverable" values that correlate with the +1.18 mm grades reported. Changing process plant efficiency (relative liberation and recovery of diamonds) would typically also require an adjustment to these values. The resource estimates for CH-6 and CH-7 extend to depths of 525 mbs (155 masl) and 240 mbs (450 masl), respectively.

Table 14-30: Mineral Resource Statement for the Chidliak Project

Body	Resource Classification	Depth Range	Volume (Mm^3)	Density (g/cm^3)	Tonnage (Mt)	Grade (ct/t)	Carats (Mct)	Value (US $/ct)
CH-6	Inferred	0 to 525 mbs	2.85	2.62	7.46	2.41	17.96	151
CH-7	Inferred	0 to 240 mbs	1.94	2.57	4.99	0.85	4.23	114
All	**Inferred**		**4.79**	**2.60**	**12.45**	**1.78**	**22.19**	**132**

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

14.7 Reasonable Prospects for Economic Extraction

The CIM Definition Standards for Mineral Resources and Mineral Reserves states that in order to be classified as a Mineral Resource there should be a reasonable prospect for the eventual economic extraction of the specified ore. JDS and collaborating QP's previously determined Mineral Resources at CH-6 and CH-7 to possess reasonable prospects of eventual economic extraction by completing a 2016-era PEA that supports an estimated after-tax net present value (NPV) of C$471 million (Doerksen et al., 2016). First-order economic and related engineering assumptions made in the 2016 PEA were re-assessed by JDS during February 2018 in view of the addition to the open pit potential of the potential underground mining methods required to extract mineral resources at CH-6 to depths near 550 mbs (130 masl). JDS concluded that the updated CH-6 resource presented in Fitzgerald et al. (2018) and duplicated in this report satisfies reasonable prospects of economic extraction to a depth of 525 mbs (155 masl) based on the first-order parameters summarized in Table 14-31.

Table 14-31: First-order Whittle™ Open Pit and conceptual Underground Mining Optimization Parameter Values Used to Demonstrate Reasonable Prospects for Economic Extraction of the CH-6 Mineral Resource in Fitzgerald et al. (2018)

Parameter	CH-6 Open Pit	CH-6 Underground
Process and G&A Cost	C$60/t processed	C$60/t processed
Nunavut Royalty	C$10/t processed	C$10/t processed
Mining Cost	C$4.00/t mined	C$105/t mined
Selling Costs	4% of carat price	4% of carat price
Mining Recoveries	100%	100%
Exchange Rate	1.28C$:US$	1.28C$:US$
Overall Pit Slope	50 degrees	

15 Mineral Reserve Estimate

No Mineral Reserve has been established at the Chidliak Project to date.

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. This PEA does not support an estimate of Mineral Reserves, since a pre-feasibility or feasibility study is required for reporting of Mineral Reserve estimates. This report is based on potentially mineable material (mineable tonnes and/or mineable resources).

Mineable tonnages were derived from the resource model described in the previous section. Inferred resources were used to establish mineable tonnes.

Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that all or any part of the Mineral Resources or mineable tonnes would be converted into Mineral Reserves.

16 Mining Methods

16.1 Geotechnical Considerations

16.1.1 Geotechnical Characterization

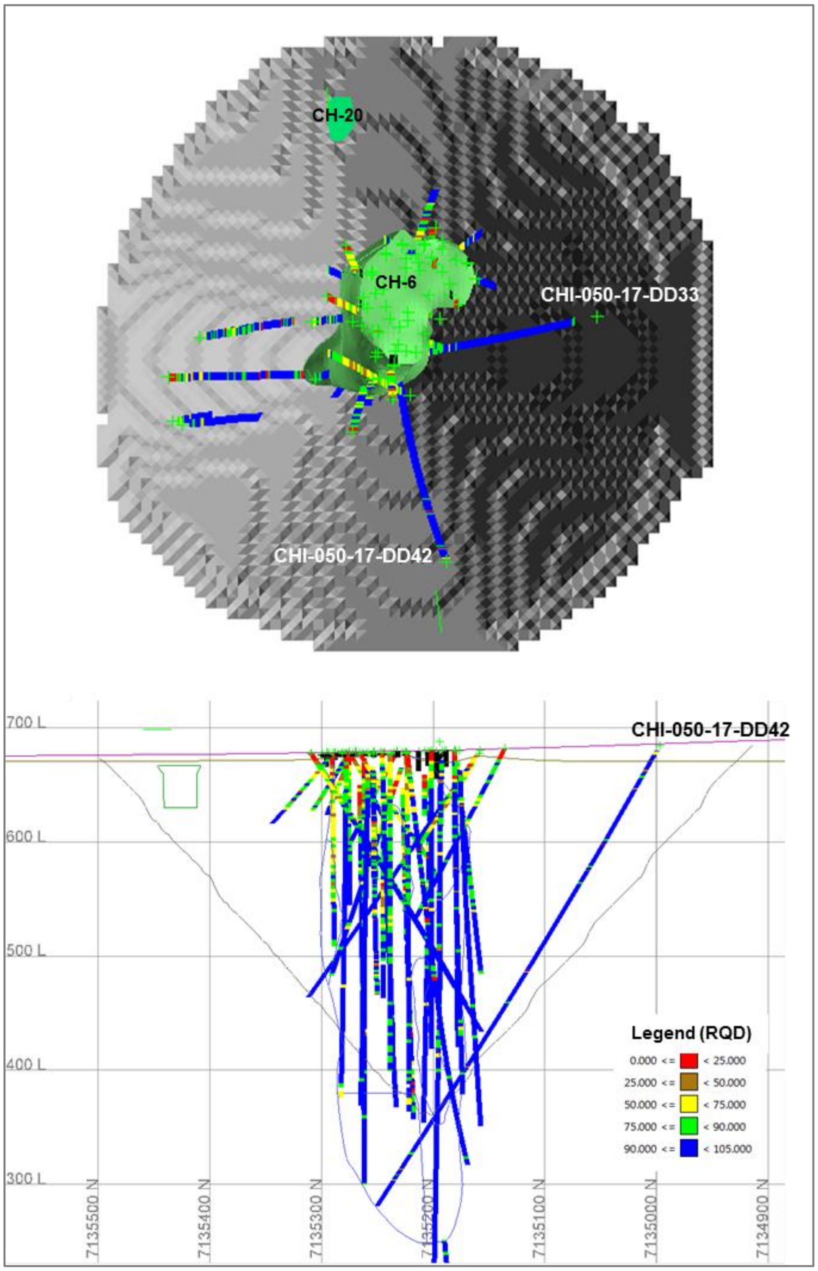
Historically drill coverage at the CH-6 and CH-7 deposits has focused in and immediately around the kimberlite orebody, with limited geotechnical information relating to the proposed pit slope walls in the surrounding paragneiss country rock. However, as part of the 2017 drill campaign, 11 drill holes were geotechnically logged and point load tested with 6 holes characterizing the country rock near planned pit slopes (Figure 16-1). Two of the holes drilled in the vicinity of the west pit (CHI-050-17-DD33) and south (CHI-050-17-DD42) pit walls were oriented to understand discontinuity orientations and sampled for laboratory strength testing. The location of the two geotechnical holes is shown on Figure 16-1.

The CH-6 geotechnical data was largely collected by Peregrine with on-site guidance at the beginning of the program by SRK Consulting (Canada) Inc. (SRK), and subsequently guided by SRK office-based support. In general, the data was collected to a standard required for pre-feasibility level geotechnical study. A thorough discussion regarding the details and results of the program are contained in a geotechnical factual report (SRK, 2018). The report was reviewed and served as the basis for the CH-6 pit slope design recommendations contained herein.

No additional drilling was conducted in 2017 at the smaller, CH-7 deposit. Two historic drill holes (CHI-251-12-DD16 and 251-12-DD18 are collared west of the kimberlite and provide reasonable insight into the country rock in the area of the proposed western slope. (Figure 16-2). Core photographs for CH-7 were reviewed by SRK as part of the 2016 PEA (Doerksen et al., 2016) with the results summarized in presentation format. The presentation was reviewed and served as the basis for pit slope design recommendations for CH-7.

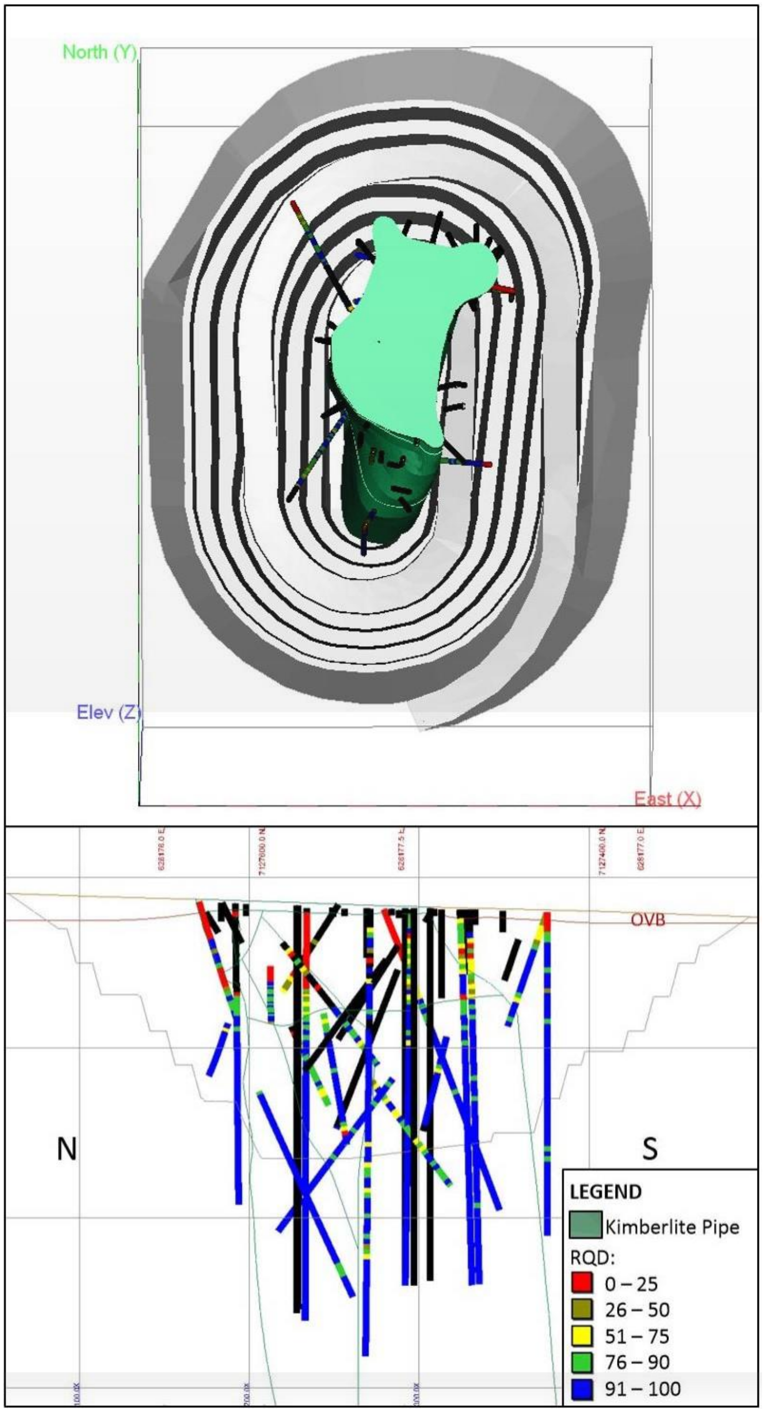
Benchmarking of recommended slope angles against other known, similar context kimberlite pipe open pits was also conducted by SRK as part of the 2016 PEA to ensure that the design recommendations were realistic. Results of the benchmarking are included in Section 16.1.1.4.

Figure 16-1: Plan and Cross-Section of CH-6 Pit Shell



Source: SRK

Figure 16-2: Plan and Section Schematic of CH-7 Pit Shell



Note: Pit geometry shown is final pit from the 2016 PEA (Doerksen et al., 2016)
Source: SRK

16.1.1.1 CH-6 Pipe Rock Mass Conditions

Near Surface Weathered Rock and Overburden

The depth to bedrock is based on information gained from previous RC drilling programs carried out across the CH-6 deposit. The program was designed by Peregrine to define the kimberlite extents at surface, beneath the glacial till. Drill holes were terminated at reasonably competent bedrock. Drill hole traces are shown as black lines near the ground surface in Figure 16-1. The overburden is approximately 2 m deep in the northwest pit area and deepens to approximately 25 m at the south and east pit extents. Rock mass weathering is typically deeper in the kimberlite than the surrounding host rock.

Country Rock

The CH-6 pit walls are located mostly within paragneiss that has a distinct sub-vertical biotite fabric striking NW-SE. Based on the SRK (2018) geotechnical investigation, the paragneiss country rock generally classifies as 'Good' rock mass quality according to the Bieniawski (1989) rock mass rating (RMR) system. Pit walls are anticipated to consist of rock with low fracture frequency, high Rock Quality Designation (RQD), and moderate intact rock strength. Table 16-1 contains a summary of the country rock geotechnical logging information from the SRK (2018) investigation. The potential occurrence of weaker, foliation planes through the rock mass are not anticipated to adversely influence the performance of overall and inter-ramp slopes due to their steep dip but may impact bench-scale stability.

Kimberlite

With the exception of discrete structures observed near the contact, the kimberlite rock mass generally appears of similarly good rock quality as the country rock. Table 16-2 contains a summary of the geotechnical logging information from the SRK (2018) investigation for kimberlite. Based on the final PEA pit shell for CH-6, kimberlite is anticipated to comprise a very small percentage of the bottom pit wall bench.

Table 16-1: Summary of Geotechnical Logging and UCS Testing Statistics for CH-6 Country Rock

Statistic	RQD (%)	Fracture Frequency per metre (F/m)	ISRM Est. Field Strength (MPa)	UCS (MPa)	Joint Condition Rating	RMR89	Barton Jn	Barton Jr	Barton Ja	Barton Q'
Minimum	0	0.0	1	40	7	36	1	1	1	0
25th %	91	1.0	38	61	14	62	2	1	1	7
Mean	91	4.5	58	81	16	66	6	1	2	43
Median	98	1.7	61	74	17	67	3	1	1	17
75th %	100	3.3	75	99	19	72	6	2	3	46
Maximum	100	54.0	157	127	25	87	20	4	10	400
Std. Dev	18	9.0	28	27	3	9	5	1	1	74

Source: SRK (2018)

Table 16-2: Summary of Geotechnical Logging and UCS Testing Statistics for CH-6 Kimberlite

Statistic	RQD (%)	Fracture Frequency per metre (F/m)	ISRM est. Field Strength (MPa)	UCS (MPa)	Joint Condition Rating	RMR89	Barton Jn	Barton Jr	Barton Ja	Barton Q'
Minimum	0	0.0	0	0	9	35	1	1	1	0
25th %	93	0.3	61	65	15	64	1	1	1	8
Mean	91	2.9	77	72	17	71	4	2	2	99
Median	99	1.0	75	69	18	73	2	1	3	25
75th %	100	2.3	75	92	19	0	6	2	3	100
Maximum	100	59.8	150	101	24	89	20	4	6	400
Std. Dev	21	6.7	39	27	3	12	5	1	1	145

Source: SRK (2018)

Structurally, four dominant discontinuity sets are evident from the oriented core and surface mapping data. Three sets are well defined including steep SE and SW dipping sets and a flat to shallow NW dipping set. The two steeply dipping sets are also evident in surface mapping and are believed to have the highest frequency of occurrence. The fourth set evident dips moderately SW and is potentially related to folding within the deposit, based on SRK's (2018) interpretation. The impact of the various discontinuity sets will need to be investigated with respect to bench performance as the Project advances.

No major geological structures with potential to impact inter-ramp or overall slopes have been identified to date. Major geologic structures were observed in core adjacent to and within the kimberlite-country rock contact but at this stage are not expected to influence the overall pit slope design.

A second kimberlite pipe, CH-20 exists to the north of the main CH-6 kimberlite body (Figure 16-1) and will daylight in the CH-6 north pit wall, as currently designed. Drilling in this area is scarce and any relationship or potential connections between the two pipes is unknown at this time. A connection between the two pipes may result in weaker, kimberlite materials comprising a portion of the north pit wall. Consequently, pit wall angles in this sector may need to be reduced once the relationship between the two kimberlite pipes is better understood. Alternatively, the pit wall could be pushed back further to encompass CH-20 if it is found to contain economic grades.

16.1.1.2 CH-7 Pipe Rock Mass Conditions

Near Surface Weathered Rock and Overburden

The depth to competent bedrock is based on information gained from previous RC drilling programs carried out across the CH-7 deposit. The program was designed by Peregrine to define the kimberlite extents at surface, beneath the glacial till. Drill holes were terminated in reasonably competent bedrock. Drill hole traces are shown as black lines near the ground surface in Figure 16-2. The overburden is approximately 2 m deep in the area of the south pit wall crest and deepens to approximately 20 m at the NE pit extent. Rock mass weathering typically extends deeper in the kimberlite materials than the surrounding host rock.

Country Rock

Based on the final PEA pit shell, the CH-7 pit walls will be situated almost entirely in paragneiss country rock with a small amount of kimberlite in the lower bench. The country rock is anticipated to be of 'Good' overall quality according to the Bieniawski (1989) RMR system, consisting of low fracture frequencies, high RQD, and moderate intact rock strengths. No major geological structures have been identified to date with potential to adversely affect inter-ramp or global stability conditions.

Kimberlite

The contact between kimberlite and country rock appears largely unaltered, un-brecciated, and competent based on reviews completed by SRK for the 2016 PEA. The current PEA pit shell indicates kimberlite will make up a very small percentage of the overall final pit walls, comprising only portions of the lowest pit bench.

16.1.1.3 PEA Slope Design Recommendations

Pit slope design criteria for the current PEA are summarized in Table 16-3 and

Table 16-4 for the CH-6 and CH-7 pits, respectively. No adjustments have been made to bench heights in potentially weaker rock mass areas such as the northern sector of the CH-6 pit area or within the weaker kimberlite areas potentially at the toe of the slopes of both pit pits. Pit walls in these areas may need further optimization and allowances made for potential impacts on ramp performance in weaker kimberlite units as the Project advances and additional geotechnical data is collected. The location of potential kimberlite exposures in pit walls should be reviewed once detailed pit designs are developed from the PEA shells.

Pit slope design parameters for the CH-6 pit are considered aggressive but within the range used by other kimberlite pits in Canada. The recommendation for 80° bench face angles along the eastern wall is based on the dominant, steep foliation dip (approximately 80°) and apparent lack of other, moderately dipping joint sets. Bench face and inter-ramp slope angles recommended will require specialized wall control blasting and thorough scaling.

Table 16-3: CH-6 Pit Slope Design Criteria

Sector	Rock Mass	Inter-Ramp (°)	Bench Face Angle (°)	Berm Width Height (m)	Bench Height (m)
160 - 340°	Overburden	35	35	-	-
	Country Rock	59	80*	8.5	20
340 - 160°	Overburden	35	35	-	-
	Country Rock	55	75	8.5	20

Note: * High quality pre-split blasting program required
Source: JDS

Table 16-4: CH-7 Pit Slope Design Criteria

Sector	Rock Mass	Inter-Ramp (°)	Bench Face Angle (°)	Berm Width (m)	Bench Height (m)
0 - 360°	Overburden	35	35	-	-
	Kimberlite	48	70	10.5	20
	Country Rock	53	75*	9.5	20

Note: * High quality pre-split blasting program required
Source: JDS

16.1.1.4 Benchmarking

The 2016 PEA pit design recommendations were compared by SRK to similar operating open pits in the Canadian Arctic (see Doerksen et al., 2016). The results of the benchmarking study are summarized in Table 16-5 and confirm that the Chidliak pit design recommendations compare reasonably well with other operating open-pit diamond mines in similar environments (Table 16-5).

Table 16-5: Benchmarked Slope Design Parameters for Similar Arctic Kimberlite Pits

	IRA* (°)	OSA (°)	Pit Depth (m)
Pit A	53 - 62	48 - 54	210 - 225
		42	210 - 225
Pit B	51 - 63	47 - 51	285 - 305

*IRA = Inter-Ramp Angle, OSA = Overall Slope Angles
Source: SRK

16.2 Open Pit Optimization

16.2.1 Input Parameters

The 3D Mineral Resource block models for CH-6 and CH-7, as provided by Peregrine, were used as the basis for deriving the economic shell limits for the Chidliak Project.

Estimates were made for diamond price, mining dilution, process recovery, off-site costs and non-governmental royalties. Mining, processing, and general administration operating cost estimates (OPEX) were also calculated based on calculated processing throughput and, along with geotechnical parameters, formed the basis for OP optimization (see Table 16-6). The OP mining costs were estimated for both plant feed material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate.

Estimated external dilution factors of 5% and 6% for CH-6 and CH-7 respectively were estimated by digitizing a 1.0 m waste halo around the geological boundary of the kimberlite resource. The volumes of the geological solids were then compared against these expanded solids to estimate an average mining dilution percentage to apply to the individual pipes. A mining recovery factor of 95% was used to account for mining inefficiencies and loss of mineralized material during mining.

Table 16-6: Input Parameters Used in the LOM OP Optimization

Parameter	Unit	Values	
		CH-6	CH-7
Average LOM Diamond Price	US$/carat	173	147
Exchange Rate	C$:US$	1.28	
Selling Cost	% of price	4	
Selling Cost	US$/carat	6.92	5.88
Net Average LOM Diamond Price	C$/carat	213	181
Waste Mining Cost	C$/t waste mined	3.80	5.20
Mineralized Material Mining Cost	C$/t processed	4.40	7.90
Strip Ratio (Estimated tonnes waste : tonnes processed)	t:t	10	4
Mining Cost	C$/t processed	39.90	27.04
Processing	C$/t processed	18.00	
Freight	C$/t processed	16.75	
Coarse PK haulage	C$/t processed	5.00	
General and Administrative (G&A)	C$/t processed	18.10	
Site Services	C$/t processed	9.92	
Government Royalty	C$/t processed	10.00	7.00
Subtotal: Processing/G&A (Excluding Waste Mining)	C$/t processed	78.37	77.47
Total Operating Cost w/ Mining	C$/t processed	118.27	104.51
Process Recovery	%	98	
Mining Dilution	%	5	6
Mining Recovery	%	95	
Process Throughput Rate	t/d	2,000	
Overall Pit Slope Angle	°	35 - 52	35 - 48

*The values in this table vary slightly from those used in the economic model as parameters were further refined in the economic model. The differences are not considered material to the pit shape definition.

The mineral inventory block models for the Chidliak CH-6 and CH-7 deposits were then used with NPV Scheduler pit optimization software to determine optimal mining shells. This evaluation included the aforementioned parameters. The economic shell limits for both CH-6 and CH-7 include only Inferred Mineral Resources. Any TFFE that falls within the CH-6 OP limits is treated as waste.

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the Inferred Resources would be upgraded to a higher resource category.

16.2.2 Optimization Results

A series of optimized shells were generated for the Chidliak deposits based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary pit stage selection. Refer to Table 16-7 and Figure 16-3 for a summary of the CH-6 optimization results and Table 16-8 and Figure 16-4 for CH-7 results.

NPV Scheduler produces both "best case" (i.e. mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and "worst case" (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range of possible outcomes. The shells were produced based on varying revenue factors (0.1 through to 1.2 of base case) to produce a series of nested shells with the NPV results shown. The NPV values noted here do not include initial or sustaining capital cost requirements and were used only to determine the basic mining shapes. The actual NPV of the Project is summarized in Section 23 of this report.

Table 16-7: Overall Optimization Results CH-6 (excluding Capital Costs)

Pit (#)	RevFac (value)	Life (yrs)	Plant Feed (Mt)	Grade (ct/t)	Grade (Mct)	Waste (Mt)	Strip R (t waste: t plant feed)	Total (Mt)	NPV Worst (C$ M)
Pit 1	0.10	0.0	0.02	4.27	0.1	0.01	0.3	0.03	16
Pit 2	0.12	0.3	0.21	3.38	0.7	0.2	1.2	0.5	130
Pit 3	0.14	0.8	0.58	3.08	1.8	1.0	1.7	1.6	311
Pit 4	0.16	1.4	1.03	2.96	3.0	2.5	2.4	3.5	508
Pit 5	0.18	1.6	1.15	2.90	3.3	2.7	2.4	3.9	553
Pit 6	0.20	2.2	1.59	2.74	4.4	4.0	2.5	5.5	700
Pit 7	0.22	3.0	2.21	2.56	5.7	5.4	2.5	7.6	871
Pit 8	0.24	4.0	2.93	2.50	7.3	10.8	3.7	13.8	1,072
Pit 9	0.26	4.7	3.42	2.47	8.4	15.5	4.5	18.9	1,190
Pit 10	0.28	5.2	3.80	2.46	9.4	20.6	5.4	24.4	1,283
Pit 11	0.30	6.3	4.57	2.43	11.1	31.2	6.8	35.8	1,437
Pit 12	0.32	6.6	4.83	2.41	11.6	34.6	7.2	39.4	1,479
Pit 13	0.34	7.0	5.08	2.39	12.2	38.8	7.6	43.9	1,519
Pit 14	0.36	7.2	5.24	2.38	12.5	40.9	7.8	46.1	1,540
Pit 15	0.38	7.5	5.48	2.36	13.0	45.7	8.3	51.2	1,571
Pit 16	0.40	7.8	5.71	2.35	13.4	50.0	8.8	55.7	1,597
Pit 17	0.42	8.0	5.84	2.35	13.7	54.4	9.3	60.2	1,615
Pit 18	0.44	8.1	5.95	2.34	13.9	56.4	9.5	62.3	1,624
Pit 19	0.46	8.3	6.03	2.34	14.1	59.9	9.9	65.9	1,634
Pit 20	0.48	8.3	6.09	2.33	14.2	61.0	10.0	67.1	1,638
Pit 21	0.50	8.4	6.16	2.33	14.3	63.2	10.3	69.4	1,643
Pit 22	**0.54**	**8.5**	**6.18**	**2.33**	**14.4**	**63.6**	**10.3**	**69.8**	**1,644**
Pit 23	0.56	8.6	6.24	2.32	14.5	66.4	10.6	72.7	1,648
Pit 24	0.58	8.6	6.29	2.32	14.6	68.5	10.9	74.8	1,650
Pit 25	0.60	8.6	6.29	2.32	14.6	68.6	10.9	74.9	1,651
Pit 26	0.62	8.7	6.32	2.32	14.7	69.9	11.1	76.2	1,652
Pit 27	0.66	8.7	6.32	2.32	14.7	70.0	11.1	76.3	1,652
Pit 28	0.72	8.7	6.38	2.32	14.8	73.3	11.5	79.6	1,652
Pit 29	0.78	8.8	6.42	2.32	14.9	75.6	11.8	82.0	1,651
Pit 30	0.80	8.8	6.43	2.32	14.9	76.6	11.9	83.1	1,651
Pit 31	0.82	8.8	6.43	2.32	14.9	76.6	11.9	83.1	1,651
Pit 32	0.98	8.9	6.50	2.31	15.0	82.6	12.7	89.1	1,645
Pit 33	1.00	8.9	6.51	2.31	15.1	83.7	12.9	90.2	1,644
Pit 34	1.02	8.9	6.52	2.31	15.1	84.8	13.0	91.3	1,642
Pit 35	1.14	9.0	6.57	2.31	15.2	90.7	13.8	97.3	1,635

Note: Highlighted values denote optimized pit selected for the PEA.

Figure 16-3: CH-6 Pit Optimization Results

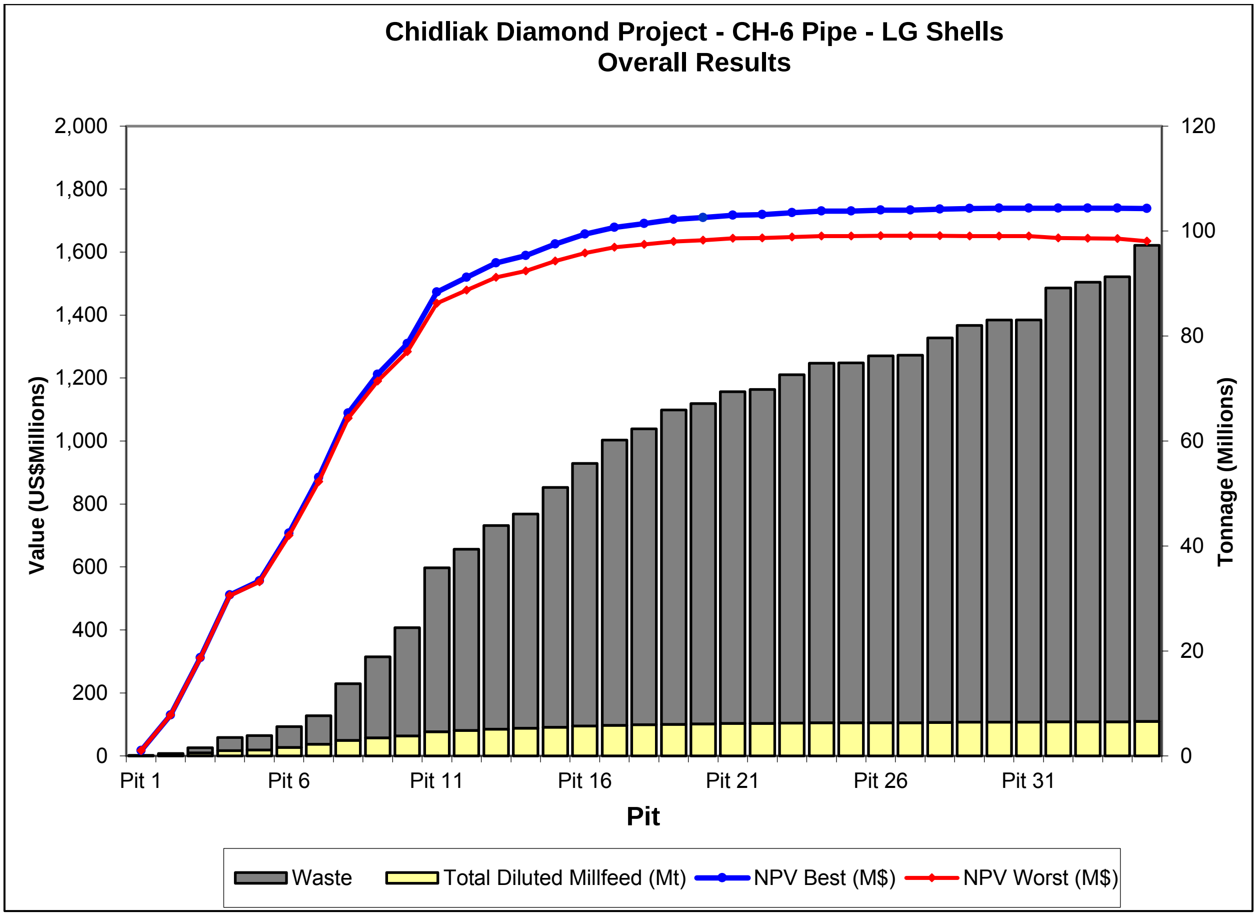
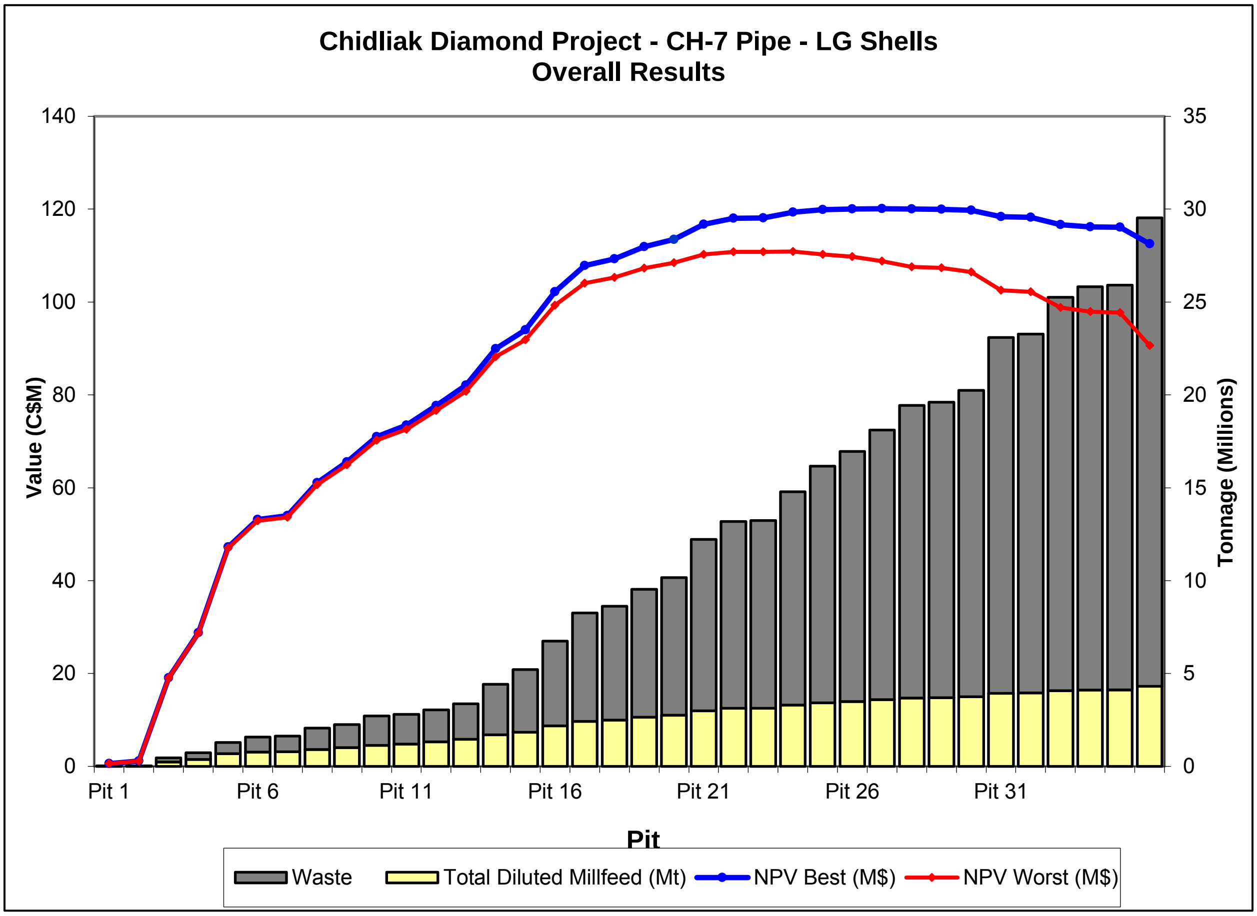


Table 16-8: Overall Optimization Results CH-7 (excluding Capital Costs)

Pit (#)	RevFac (value)	Life (yrs)	Plant Feed (Mt)	Grade (ct/t)	Carats (k ct)	Waste (Mt)	Strip R (t waste : t feed)	Total (Mt)	NPV Worst ($M)
Pit 1	0.48	0.0	0.01	1.03	6	0.0	0.6	0.0	0.6
Pit 2	0.50	0.0	0.01	1.01	12	0.0	0.7	0.0	1.1
Pit 3	0.52	0.3	0.24	0.94	228	0.2	0.9	0.5	19.0
Pit 4	0.54	0.5	0.38	0.94	351	0.4	0.9	0.7	28.7
Pit 5	0.56	0.9	0.68	0.90	609	0.6	0.9	1.3	47.0
Pit 6	0.58	1.1	0.77	0.90	694	0.8	1.0	1.6	52.9
Pit 7	0.60	1.1	0.78	0.90	706	0.8	1.1	1.6	53.7
Pit 8	0.62	1.2	0.91	0.90	819	1.1	1.3	2.1	60.6
Pit 9	0.64	1.4	1.01	0.89	895	1.2	1.2	2.3	65.0
Pit 10	0.66	1.5	1.13	0.88	994	1.6	1.4	2.7	70.2
Pit 11	0.68	1.6	1.19	0.87	1,041	1.6	1.4	2.8	72.6
Pit 12	0.70	1.8	1.31	0.86	1,128	1.7	1.3	3.0	76.6
Pit 13	0.72	2.0	1.45	0.85	1,226	1.9	1.3	3.4	80.8
Pit 14	0.74	2.3	1.70	0.83	1,419	2.7	1.6	4.4	88.2
Pit 15	0.76	2.5	1.83	0.83	1,527	3.4	1.8	5.2	91.9
Pit 16	0.78	3.0	2.18	0.82	1,780	4.6	2.1	6.7	99.3
Pit 17	0.80	3.3	2.42	0.82	1,970	5.8	2.4	8.3	104.0
Pit 18	0.82	3.4	2.50	0.81	2,025	6.1	2.5	8.6	105.3
Pit 19	0.84	3.6	2.65	0.81	2,138	6.9	2.6	9.5	107.3
Pit 20	0.86	3.8	2.76	0.80	2,216	7.4	2.7	10.2	108.4
Pit 21	0.88	4.1	2.99	0.80	2,405	9.2	3.1	12.2	110.2
Pit 22	0.90	4.3	3.13	0.80	2,502	10.1	3.2	13.2	110.8
Pit 23	0.92	4.3	3.13	0.80	2,506	10.1	3.2	13.2	110.8
Pit 24	0.94	4.5	3.30	0.80	2,636	11.5	3.5	14.8	110.9
Pit 25	0.96	4.7	3.42	0.80	2,731	12.7	3.7	16.2	110.2
Pit 26	0.98	4.8	3.49	0.80	2,787	13.5	3.9	17.0	109.8
Pit 27	1.00	4.9	3.59	0.80	2,868	14.5	4.0	18.1	108.8

Note: Highlighted values denote optimized pit selected for the PEA.
Source: JDS (2018)

Figure 16-4: CH-7 Pit Optimization Results



Source: JDS (2018)

For the CH-6 and CH-7 deposits, shells beyond a certain point add mineralized rock and waste tonnages to the overall pits, but have higher incremental strip ratios with minimal positive impact on the NPV. To better determine the optimum shell on which to base the scheduling, and to gain a better understanding of the deposit, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum processing rate of 0.73 Mt/a. No stockpiles were used in the analysis and no CAPEX was added.

Based on the analysis of the shells and preliminary schedule, Pit Shell 22 was chosen as the base case shell for CH-6 and Pit Shell 24 was selected for CH-7.

16.3 Open Pit Stages

The pit shells for Chidliak were further analyzed and optimizations were conducted in order to better define the possible stage shapes within the ultimate pit limits. It was decided to divide the pit sequence into three stages at CH-6 and two stages at CH-7 for the mine plan development to maximize the grade in the early years, reduce the pre stripping requirements, and to maintain the process facility at full production capacity.

The pit tonnages, grades, and contained carats of the stages for both CH-6 and CH-7 are summarized in Table 16-9.

Table 16-9: Chidliak CH-6 and CH-7 Pit Stage Tonnages and Grades

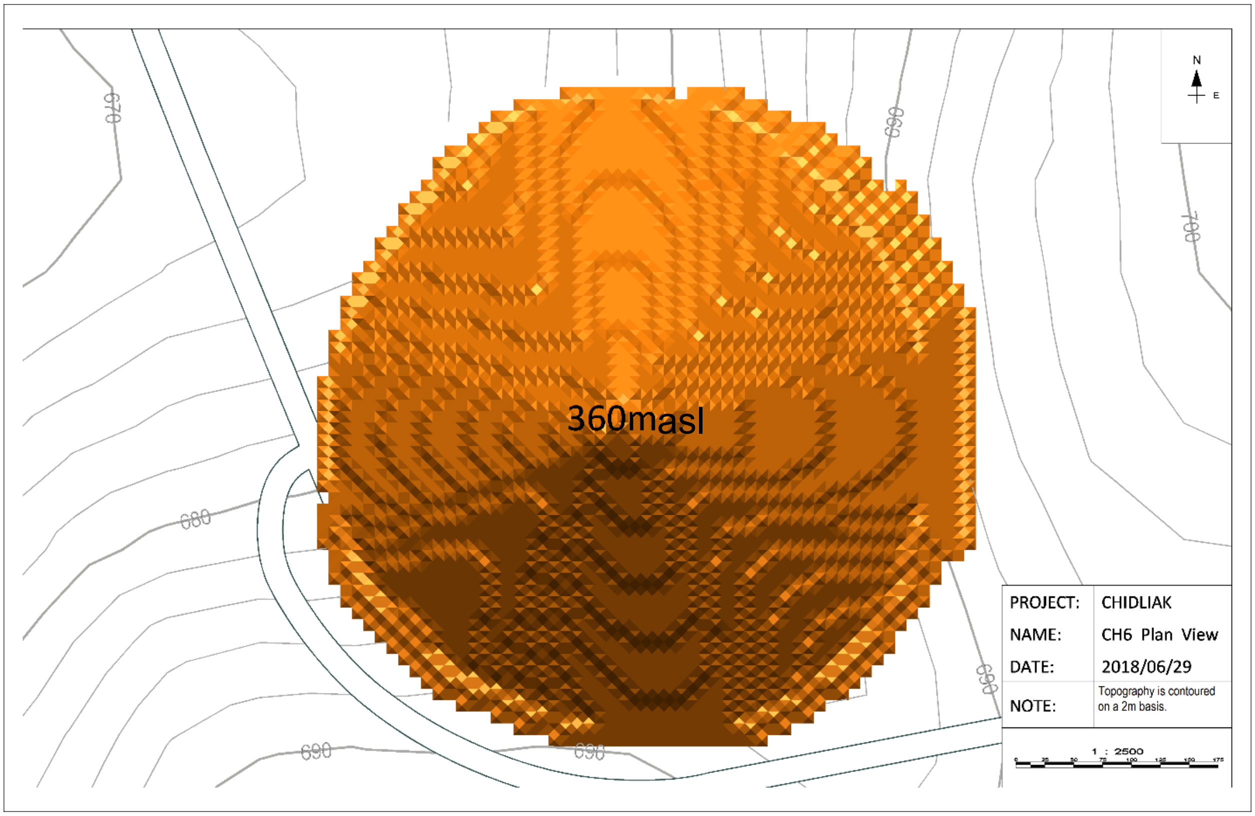
Description	Unit	2018 PEA							
		CH-6				CH-7			CH-6 / CH-7
		Stg 1	Stg 2	Stg 3	Total	Stg 1	Stg 2	Total	Total
Pit Shell	#	6	10	22	**-**	14	24	**-**	**-**
Plant Feed	(Mt)	1.6	2.2	2.4	**6.2**	1.7	1.6	**3.3**	**9.5**
Grade	(ct/t)	2.74	2.26	2.10	**2.33**	0.83	0.76	**0.80**	**1.79**
Contained Carats	(Mcts)	4.4	5.0	5.0	**14.4**	1.4	1.2	**2.6**	**17.0**
Waste	(Mt)	4.0	16.7	43.0	**63.6**	2.7	8.8	**11.5**	**75.1**
Strip Ratio	(wt:ot)	2.5	7.5	18.1	**10.3**	1.6	5.5	**3.5**	**7.9**
Total Material	(Mt)	5.5	18.9	45.4	**69.8**	4.4	10.4	**14.8**	**84.6**

Source: JDS (2018)

Figure 16-5, Figure 16-6, Figure 16-7 and Figure 16-8 represent plan and section views of the planned ultimate pit shapes.

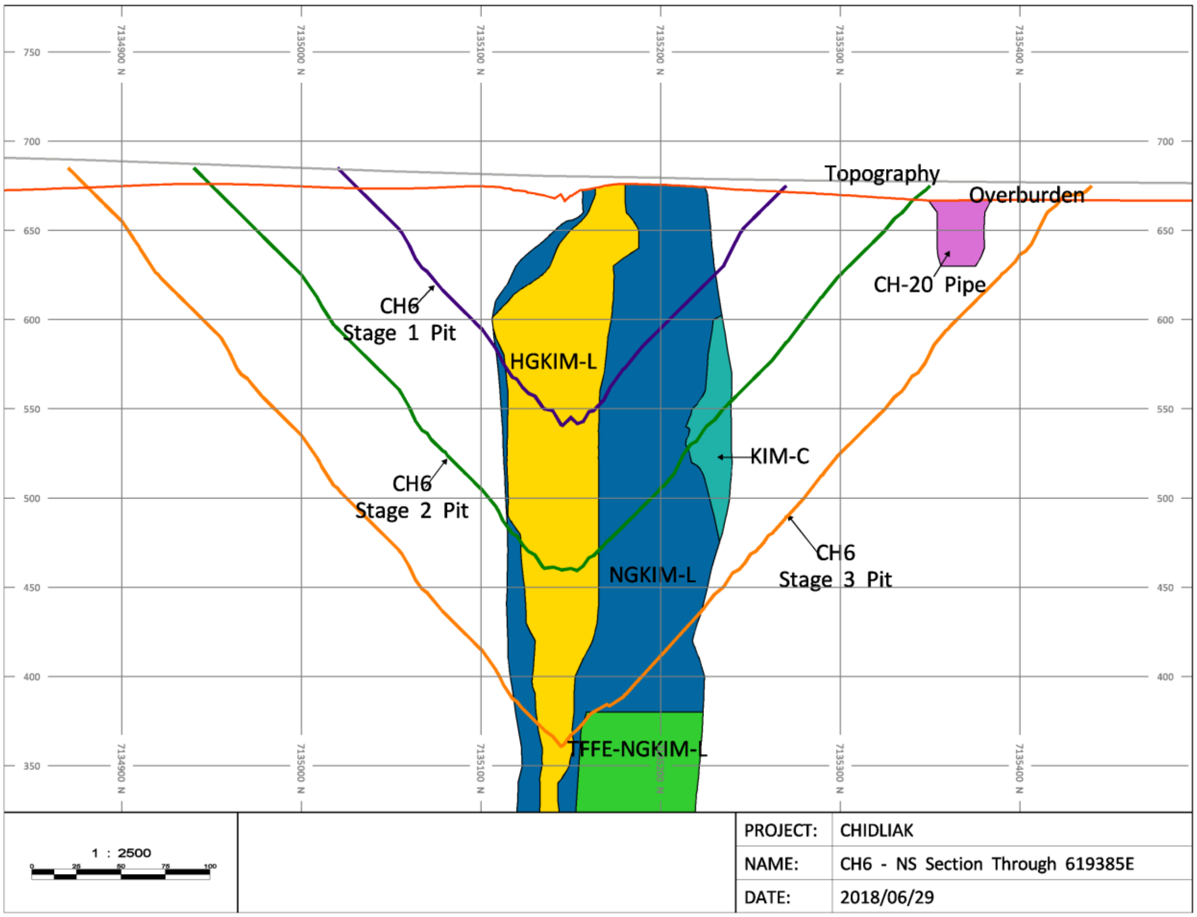
Figure 16-9 further illustrates the planned stage designs for Chidliak, with tonnes, grades, and strip ratios shown.

Figure 16-5: Plan View of CH-6 Pit Shell



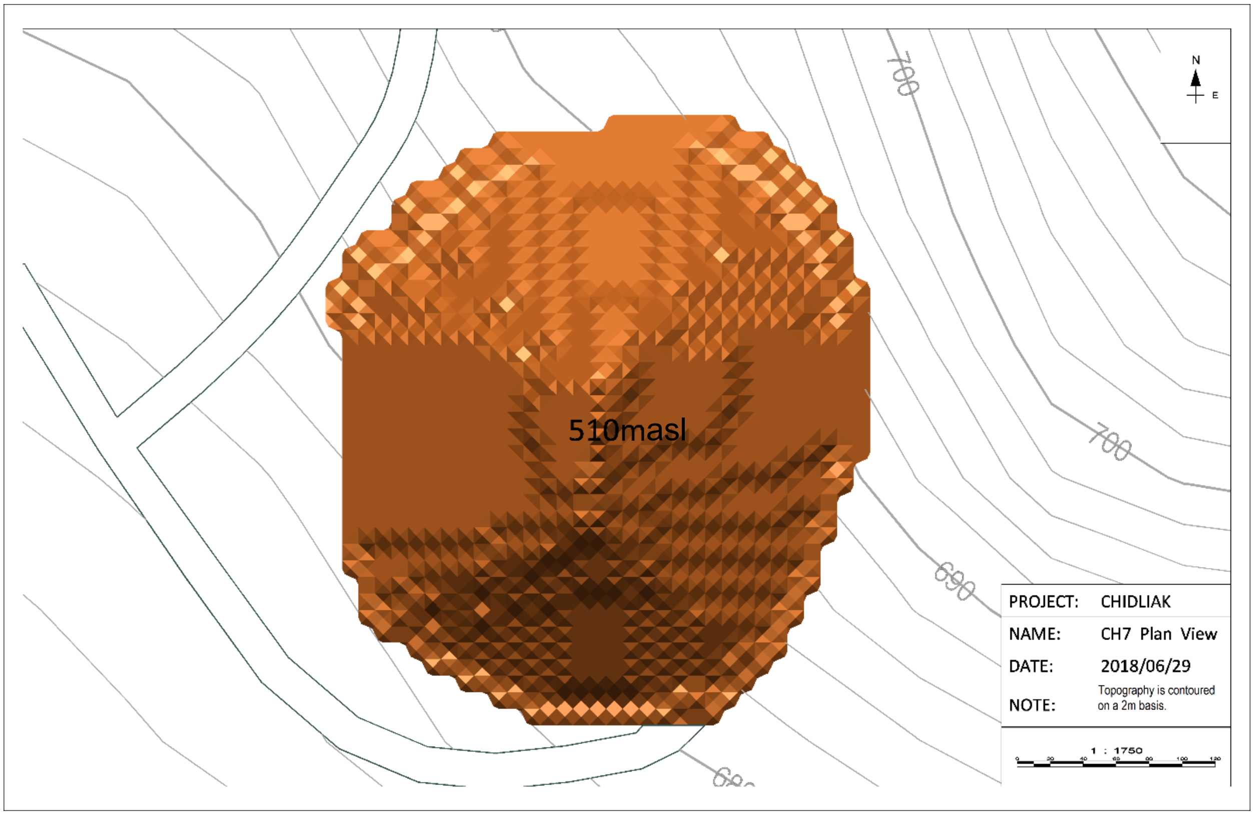
Source: JDS (2018)

Figure 16-6: Section View of CH-6 Pit Shell Showing Final and Intermediate Pit Stages



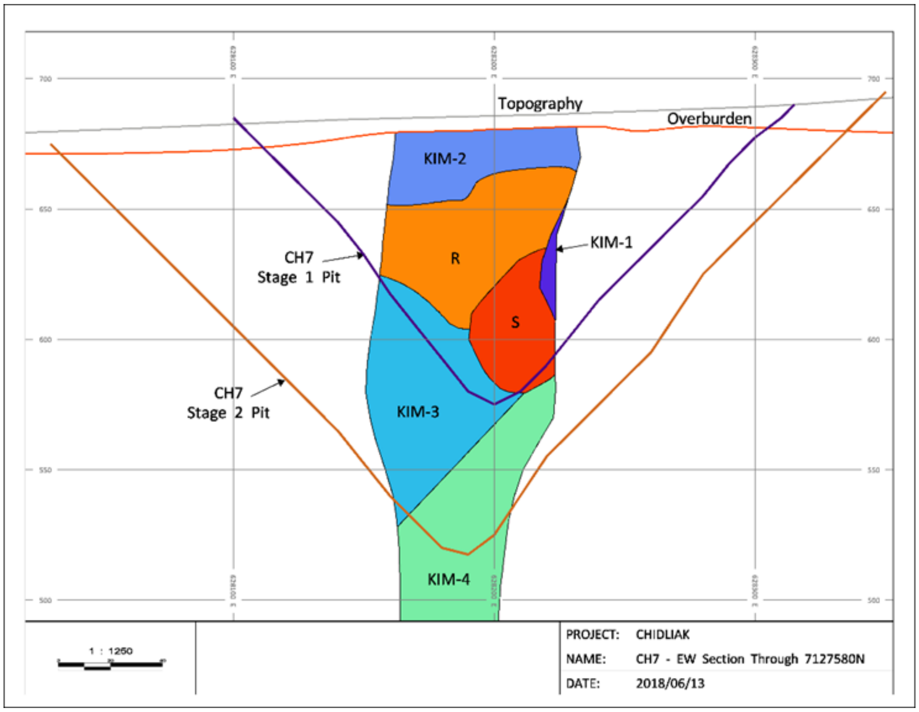
Source: JDS (2018)

Figure 16-7: Plan View of CH-7 Pit Shell



Source: JDS (2018)

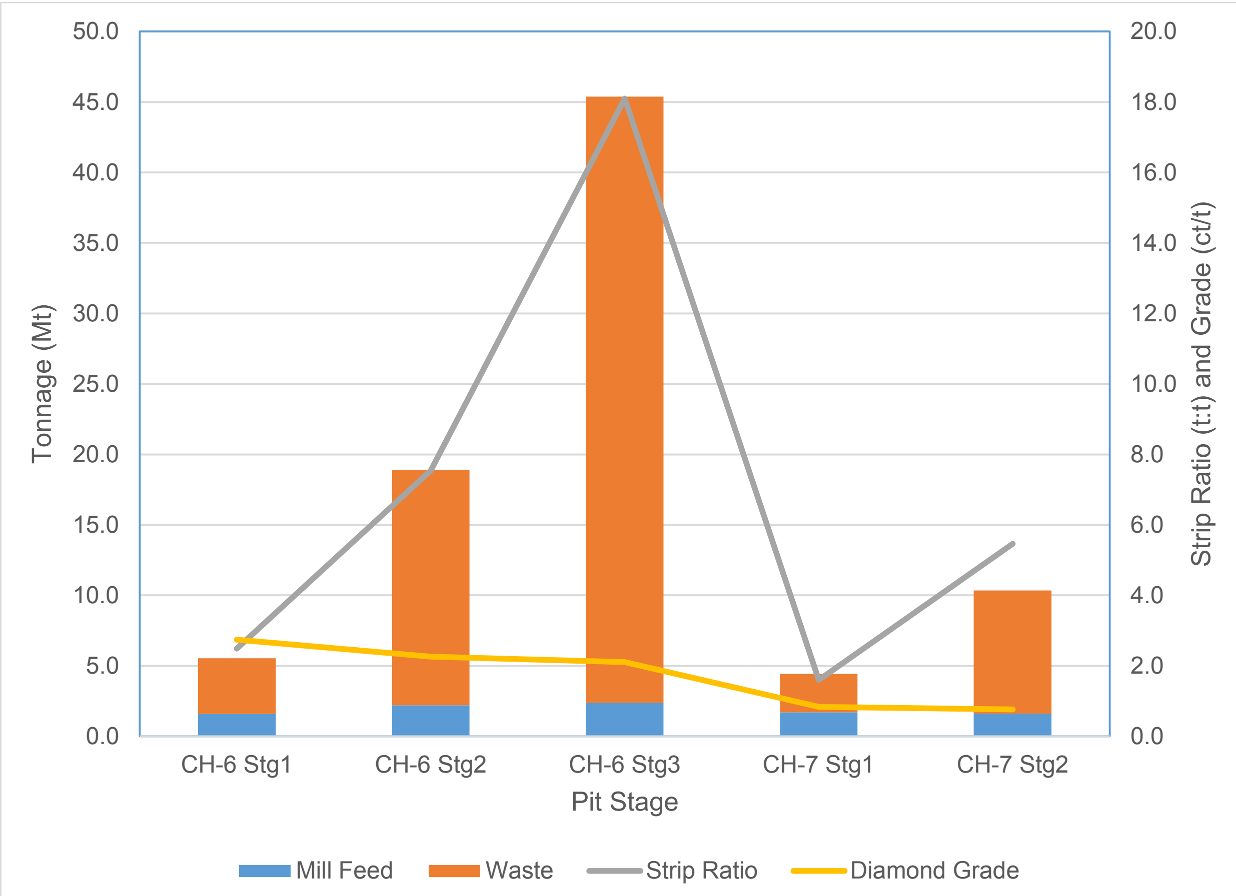
Figure 16-8: Section View of CH-7 Pit Shell Showing Final and Intermediate Pit Stages



Source: JDS (2018)

Figure 16-9 further illustrates the planned pit stages for Chidliak, with tonnes, grades, and strip ratios shown. The planned stages provide reasonable pushback widths with mining starting in the higher grade mineralized zone and progressing outwards from the initial stage.

Figure 16-9: Chidliak Project – Pit Stage Summary



Source: JDS (2018)

16.4 Mine Schedule

The Chidliak deposits are planned to produce a total of 9.5 Mt of mineralized process plant feed rock and 75.1 Mt of waste (7.9:1 overall strip ratio) over a thirteen-year mine operating life. The current LOM plan focuses on achieving consistent processing feed production rates, mining of higher value material early in the schedule, balancing grade and strip ratios, while trying to maximize NPV. Mining would commence at CH-6 and then moves onto CH-7 later in the mine life.

Year -1 represents the planned commencement of pre-stripping at CH-6. The average mining rate over the LOM will be 16,500 t/d, reaching a maximum of 28,200 t/d from Year 1 to Year 6 (Figure 16-10 summarizes mineralized tonnage, waste tonnages, and mined grade by period. Figure 16-11 illustrates the feed tonnage by stage and period, as well as total carats mined. During full production, the mine, on average, is estimated to produce 1.3 Mct/a with a LOM average mining head grade of 1.8 ct/t.

The process plant site will be located near the CH-6 deposit and plant feed from both CH-6 and CH-7 will be hauled to the plant with the mine truck fleet. Waste material from CH-6 will be used to construct the TMF containment facility and also be placed in a waste rock storage facility (WRSF) adjacent to the pit. Waste material from CH-7 will be placed in a separate WRSF near the CH-7 deposit.

Stockpiling of run-of-mine ore is intentionally limited, with small amounts of plant feed stockpiled adjacent to the process plant in the pre-strip period (111 kt).

Figure 16-10). Table 16-10 is a summary of total material movement by year for the LOM production schedule (both as totals, as well as by each stage) and includes the proposed processing schedule.

Table 16-10: Proposed LOM Production Schedule

Description	Unit	Total	Year													
			-1	1	2	3	4	5	6	7	8	9	10	11	12	13
MINE SCHEDULE																
CH-6 Stg1																
Plant Feed	Mt	1.6	0.1	0.5	0.5	0.5	0.0	-	-	-	-	-	-	-	-	-
Grade	ct/t	2.74	2.65	2.50	2.71	3.06	4.28	-	-	-	-	-	-	-	-	-
Contained Carats	Mct	4.4	0.3	1.4	1.2	1.5	0.0	-	-	-	-	-	-	-	-	-
Waste	Mt	4.0	1.0	2.0	0.8	0.2	0.0	-	-	-	-	-	-	-	-	-
Strip Ratio	t:t	2.5	10.0	3.6	1.9	0.3	0.0	-	-	-	-	-	-	-	-	-
Total Material	Mt	5.5	1.1	2.5	1.3	0.7	0.0	-	-	-	-	-	-	-	-	-
CH-6 Stg2																
Plant Feed	Mt	2.2	0.0	0.1	0.3	0.2	0.7	0.7	0.2	-	-	-	-	-	-	-
Grade	ct/t	2.26	1.97	1.97	1.97	1.90	2.10	2.45	3.28	-	-	-	-	-	-	-
Contained Carats	Mct	5.0	0.0	0.1	0.5	0.5	1.5	1.7	0.6	-	-	-	-	-	-	-
Waste	Mt	16.7	2.7	4.2	5.1	2.0	2.0	0.6	0.0	-	-	-	-	-	-	-
Strip Ratio	t:t	7.5	220.1	59.2	18.6	8.2	2.8	0.8	0.0	-	-	-	-	-	-	-
Total Material	Mt	18.9	2.7	4.3	5.4	2.3	2.7	1.3	0.2	-	-	-	-	-	-	-
CH-6 Stg3																
Plant Feed	Mt	2.4	-	-	-	-	-	0.0	0.6	0.7	0.7	0.3	-	-	-	-
Grade	ct/t	2.10	-	-	-	-	-	1.54	1.65	2.21	2.18	2.50	-	-	-	-
Contained Carats	Mct	5.0	-	-	-	-	-	0.0	0.9	1.6	1.6	0.8	-	-	-	-
Waste	Mt	43.0	-	3.5	3.5	7.4	7.6	8.3	10.1	1.9	0.7	0.0	-	-	-	-
Strip Ratio	t:t	18.1	-	-	-	-	-	334.8	18.3	2.6	0.9	0.1	-	-	-	-
Total Material	Mt	45.4	-	3.5	3.5	7.4	7.6	8.4	10.7	2.6	1.4	0.4	-	-	-	-

Table 16-10: Proposed LOM Production Schedule (cont.)

Description	Unit	Total	Year													
			-1	1	2	3	4	5	6	7	8	9	10	11	12	13
CH-6 Total Pit																
Plant Feed	Mt	6.2	0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.3	-	-	-	-
Grade	ct/t	2.33	2.57	2.44	2.43	2.67	2.12	2.42	2.05	2.21	2.18	2.50	-	-	-	-
Contained Carats	Mct	14.4	0.3	1.5	1.8	1.9	1.6	1.8	1.5	1.6	1.6	0.8	-	-	-	-
Waste	Mt	63.6	3.7	9.7	9.5	9.6	9.6	8.9	10.1	1.9	0.7	0.0	-	-	-	-
Strip Ratio	t:t	10.3	33.3	15.6	13.1	13.1	13.1	12.2	13.9	2.6	0.9	0.1	-	-	-	-
Total Material	Mt	69.8	3.8	10.3	10.3	10.3	10.3	9.6	10.8	2.6	1.4	0.4	-	-	-	-
CH-7 Stg1																
Plant Feed	Mt	1.7	-	-	-	-	-	-	-	-	-	0.4	0.7	0.6	-	-
Grade	ct/t	0.83	-	-	-	-	-	-	-	-	-	0.81	0.85	0.83	-	-
Contained Carats	Mct	1.4	-	-	-	-	-	-	-	-	-	0.3	0.6	0.5	-	-
Waste	Mt	2.7	-	-	-	-	-	-	-	-	-	1.8	0.8	0.1	-	-
Strip Ratio	t:t	1.6	-	-	-	-	-	-	-	-	-	4.7	1.1	0.2	-	-
Total Material	Mt	4.4	-	-	-	-	-	-	-	-	-	2.2	1.5	0.7	-	-
CH-7 Stg2																
Plant Feed	Mt	1.6	-	-	-	-	-	-	-	-	-	-	0.0	0.2	0.7	0.7
Grade	ct/t	0.76	-	-	-	-	-	-	-	-	-	-	0.72	0.71	0.74	0.79
Contained Carats	Mct	1.2	-	-	-	-	-	-	-	-	-	-	0.0	0.1	0.5	0.6
Waste	Mt	8.8	-	-	-	-	-	-	-	-	0.0	1.5	2.2	3.0	1.7	0.2
Strip Ratio	t:t	5.5	-	-	-	-	-	-	-	-	-	-	6,046.4	20.2	2.4	0.2
Total Material	Mt	10.4	-	-	-	-	-	-	-	-	0.0	1.5	2.2	3.2	2.5	0.9
CH-7 Total Pit																
Plant Feed	Mt	3.3	-	-	-	-	-	-	-	-	-	0.4	0.7	0.7	0.7	0.7

Table 16-10: Proposed LOM Production Schedule (cont.)

Description	Unit	Total	Year													
			-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Grade	ct/t	0.80	-	-	-	-	-	-	-	-	-	0.81	0.85	0.80	0.74	0.79
Contained Carats	Mct	2.6	-	-	-	-	-	-	-	-	-	0.3	0.6	0.6	0.5	0.6
Waste	Mt	11.5	-	-	-	-	-	-	-	-	0.0	3.4	3.0	3.1	1.7	0.2
Strip Ratio	t:t	3.5	-	-	-	-	-	-	-	-	-	8.6	4.2	4.3	2.4	0.2
Total Material	Mt	14.8	-	-	-	-	-	-	-	-	0.0	3.8	3.8	3.9	2.5	0.9
Total Open Pits																
Plant Feed	Mt	9.5	0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Grade	ct/t	1.79	2.57	2.44	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.80	0.74	0.79
Contained Carats	Mct	17.0	0.3	1.5	1.8	1.9	1.6	1.8	1.5	1.6	1.6	1.2	0.6	0.6	0.5	0.6
Waste	Mt	75.1	3.7	9.7	9.5	9.6	9.6	8.9	10.1	1.9	0.7	3.4	3.0	3.1	1.7	0.2
Strip Ratio	t:t	7.9	33.3	15.6	13.1	13.1	13.1	12.2	13.9	2.6	1.0	4.6	4.2	4.3	2.4	0.2
Total Material	Mt	84.6	3.8	10.3	10.3	10.3	10.3	9.6	10.8	2.6	1.4	4.1	3.8	3.9	2.5	0.9
PLANT SCHEDULE																
Processed	Mt	9.5		0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Processed Grade	ct/t	1.79		2.46	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.80	0.74	0.79
Recovered Carats	Mct	16.7		1.8	1.7	1.9	1.5	1.7	1.5	1.6	1.6	1.1	0.6	0.6	0.5	0.6

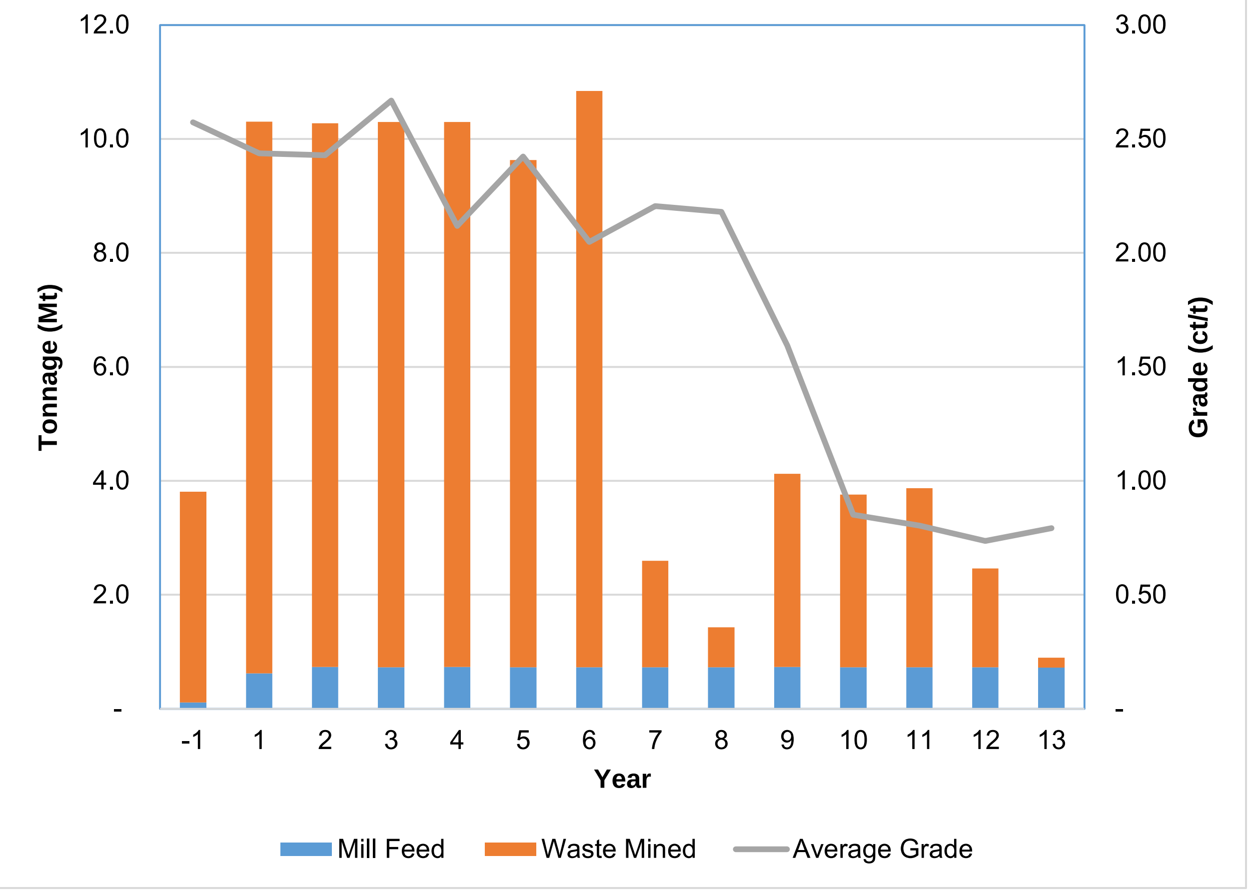
Source: JDS (2018)

Figure 16-10 summarizes mineralized tonnage, waste tonnages, and mined grade by period. Figure 16-11 illustrates the feed tonnage by stage and period, as well as total carats mined. During full production, the mine, on average, is estimated to produce 1.3 Mct/a with a LOM average mining head grade of 1.8 ct/t.

The process plant site will be located near the CH-6 deposit and plant feed from both CH-6 and CH-7 will be hauled to the plant with the mine truck fleet. Waste material from CH-6 will be used to construct the TMF containment facility and also be placed in a waste rock storage facility (WRSF) adjacent to the pit. Waste material from CH-7 will be placed in a separate WRSF near the CH-7 deposit.

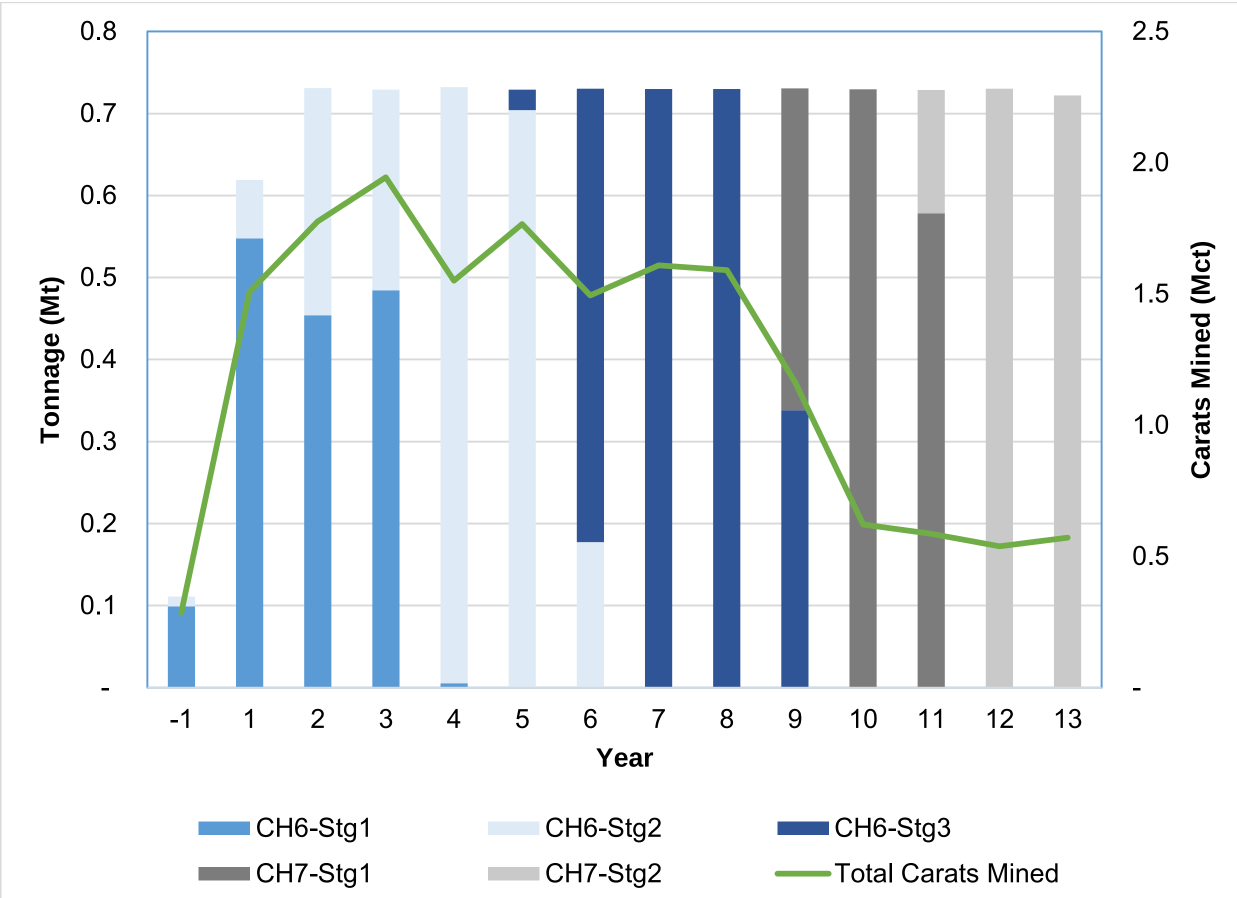
Stockpiling of run-of-mine ore is intentionally limited, with small amounts of plant feed stockpiled adjacent to the process plant in the pre-strip period (111 kt).

Figure 16-10: Mineralized Tonnes, Waste Tonnes, and Grade by Period



Source: JDS (2018)

Figure 16-11: Mineralized Tonnes, Grade and Carats by Stage and Period

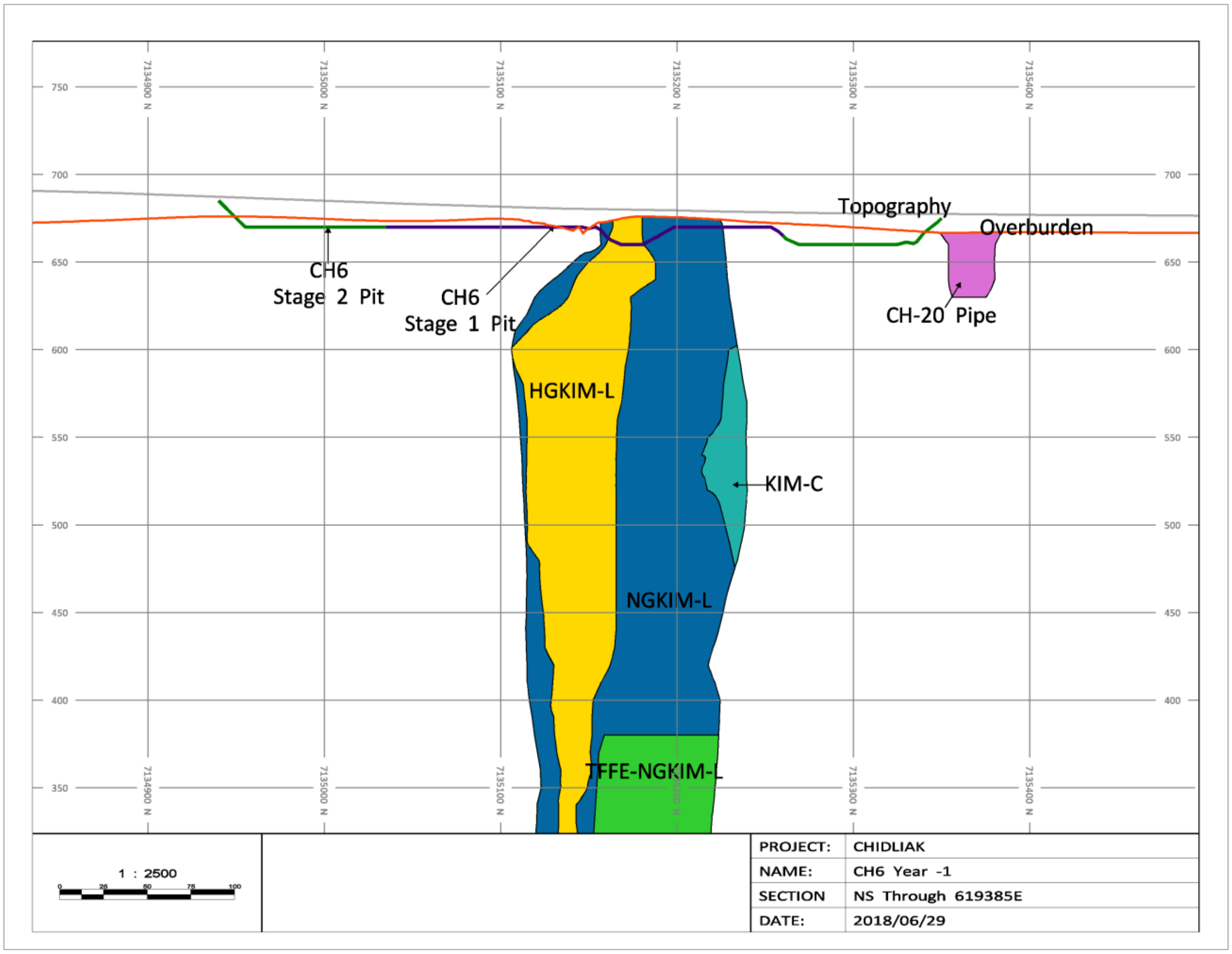


Source: JDS

To further illustrate the progression of mining, Figure 16-12 to Figure 16-20 are cross-sections through the deposits illustrating the progression of mining during various periods.

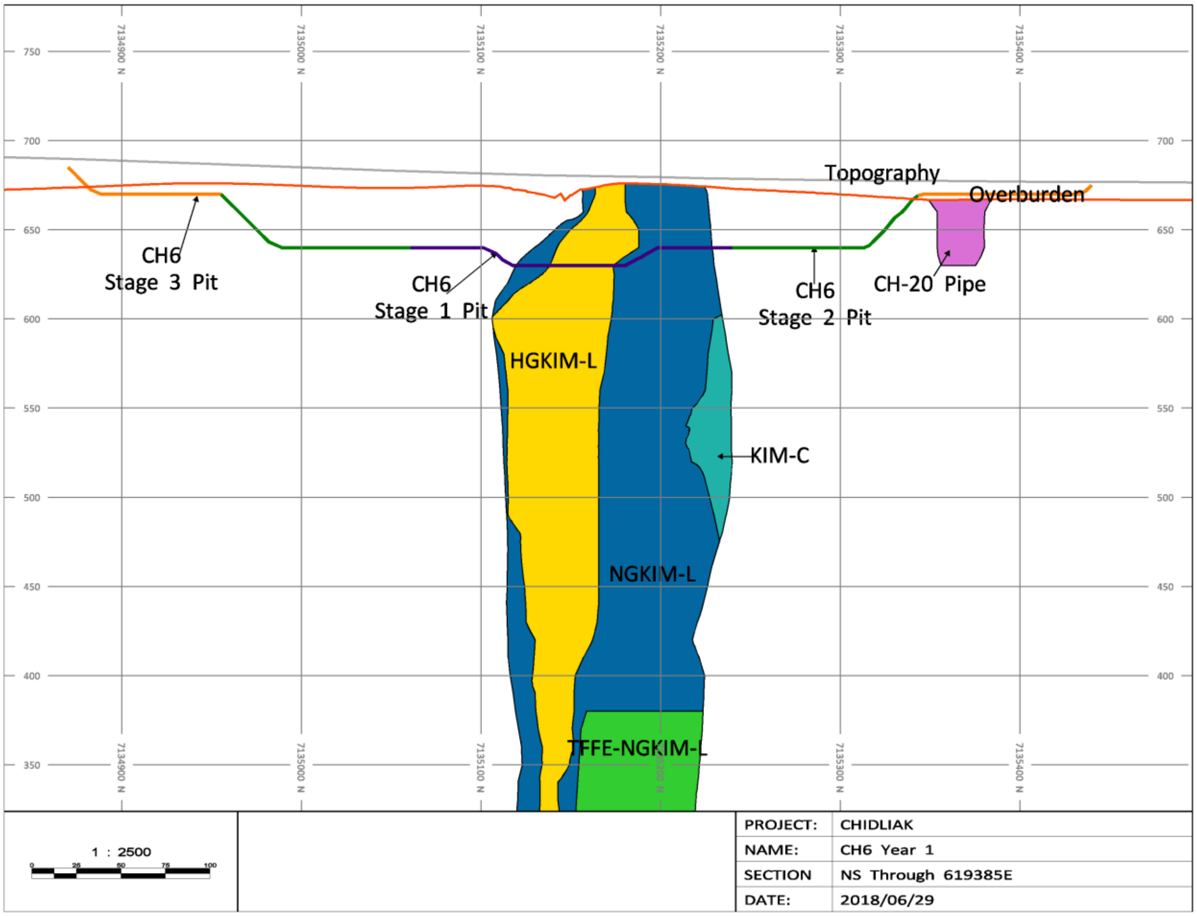
The Chidliak deposit provides maximum returns when the various stages are mined concurrently (commencing with CH-6). The pits are scheduled to be mined out in a couple of push-backs in order to achieve the required process feed, while trying to maximize the NPV of the Project.

Figure 16-12: End of Period Plot – Year -1 CH-6



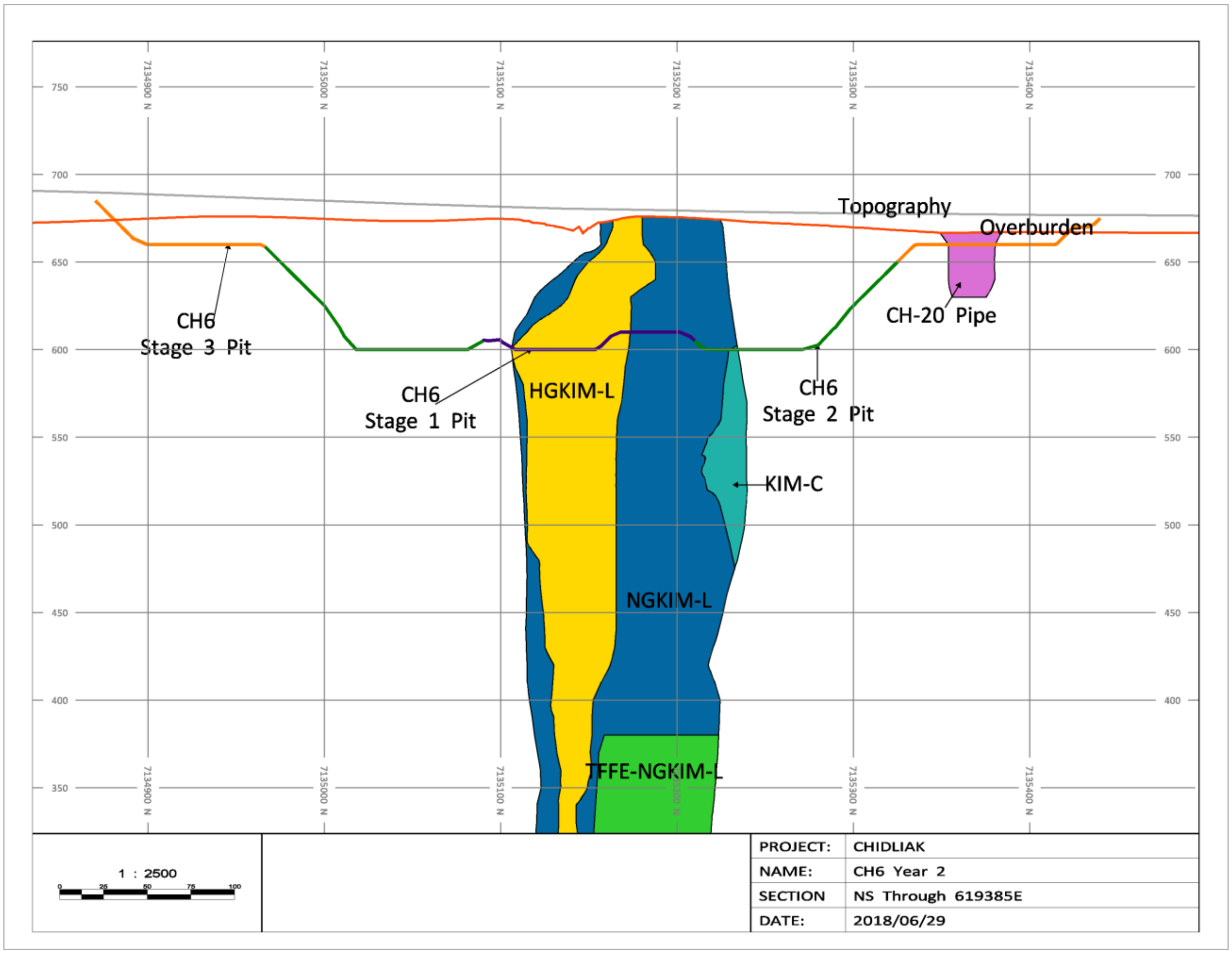
Source: JDS (2018)

Figure 16-13: End of Period Plot – Year 1 CH-6



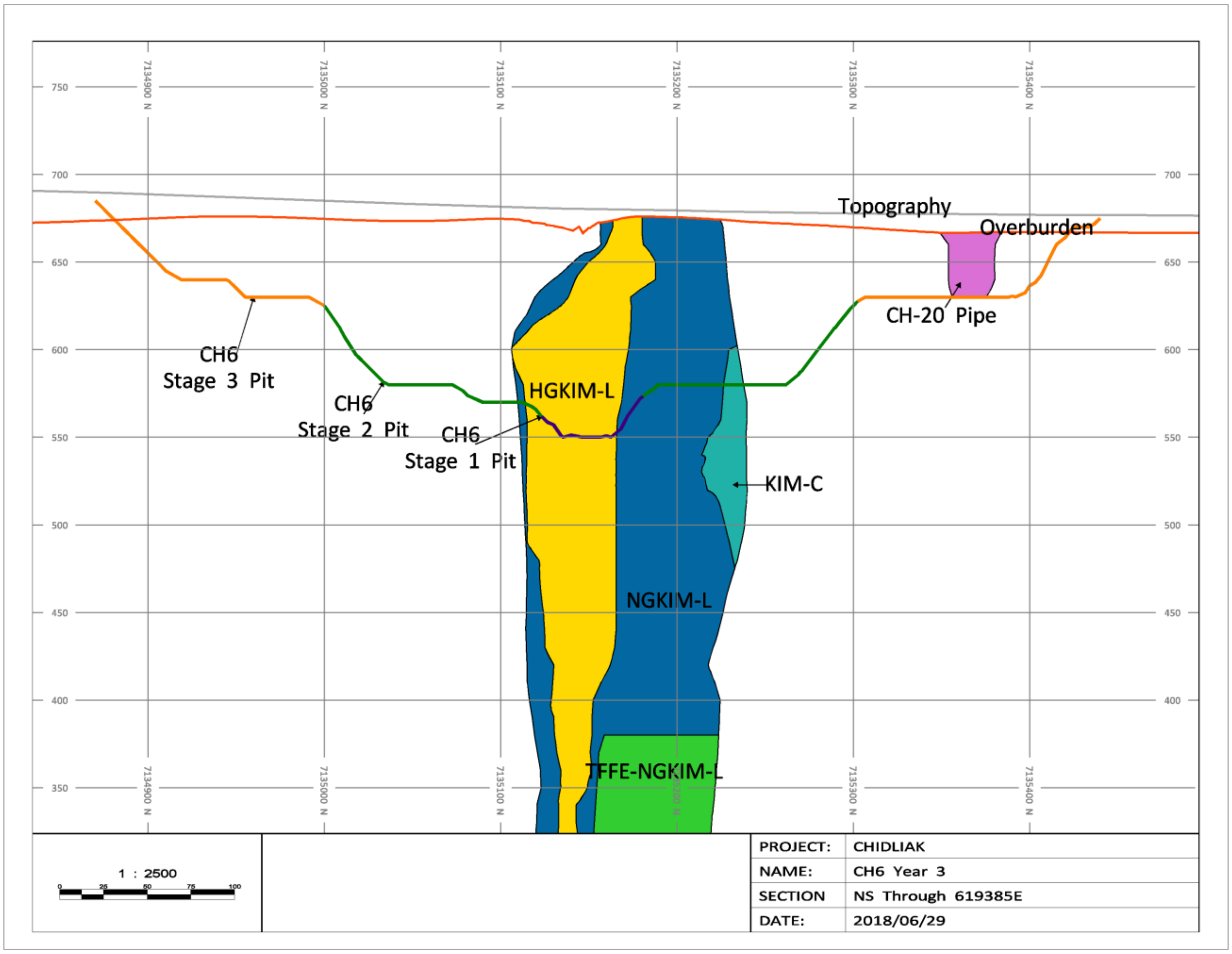
Source: JDS (2018)

Figure 16-14: End of Period Plot – Year 2 CH-6



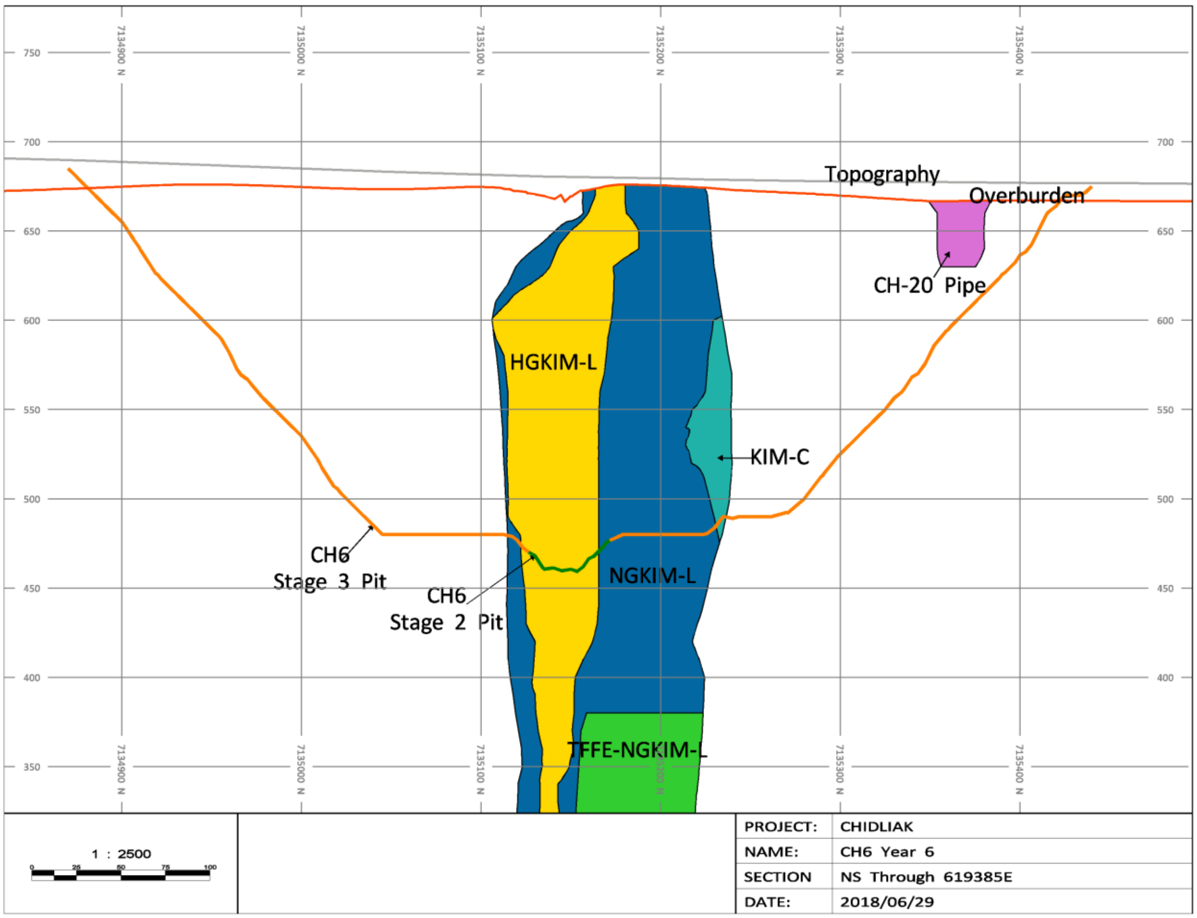
Source: JDS (2018)

Figure 16-15: End of Period Plot – Year 3 CH-6



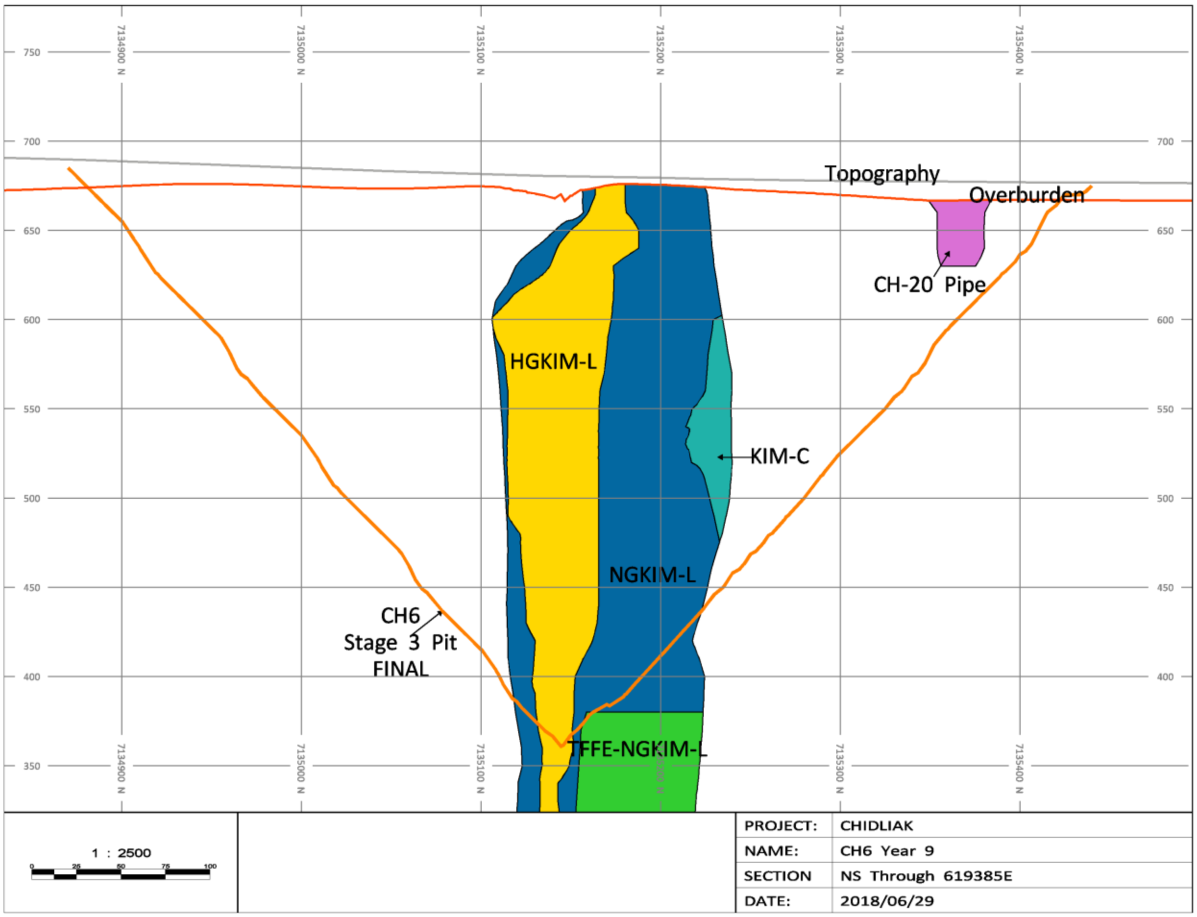
Source: JDS (2018)

Figure 16-16: End of Period Plot – Year 6 CH-6



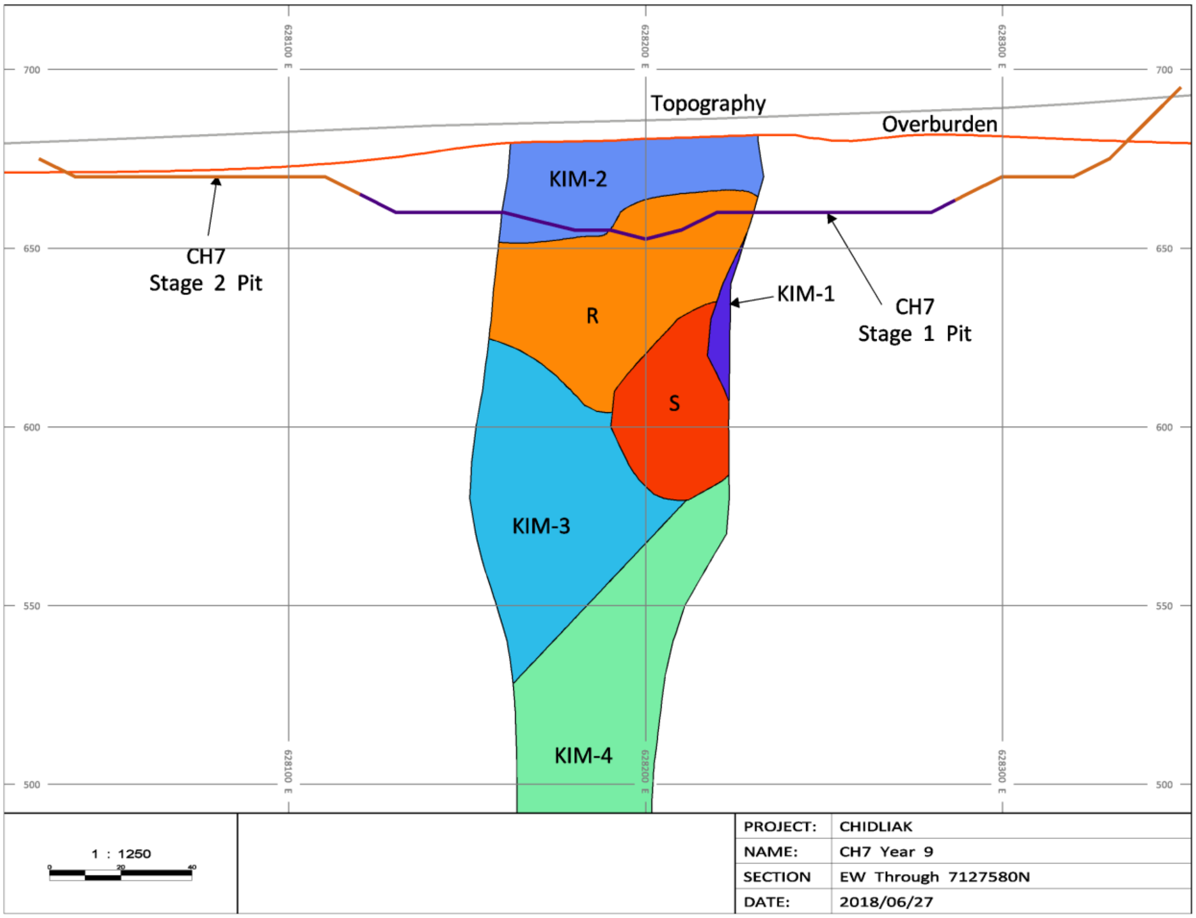
Source: JDS (2018)

Figure 16-17: End of Period Plot – Year 9 CH-6



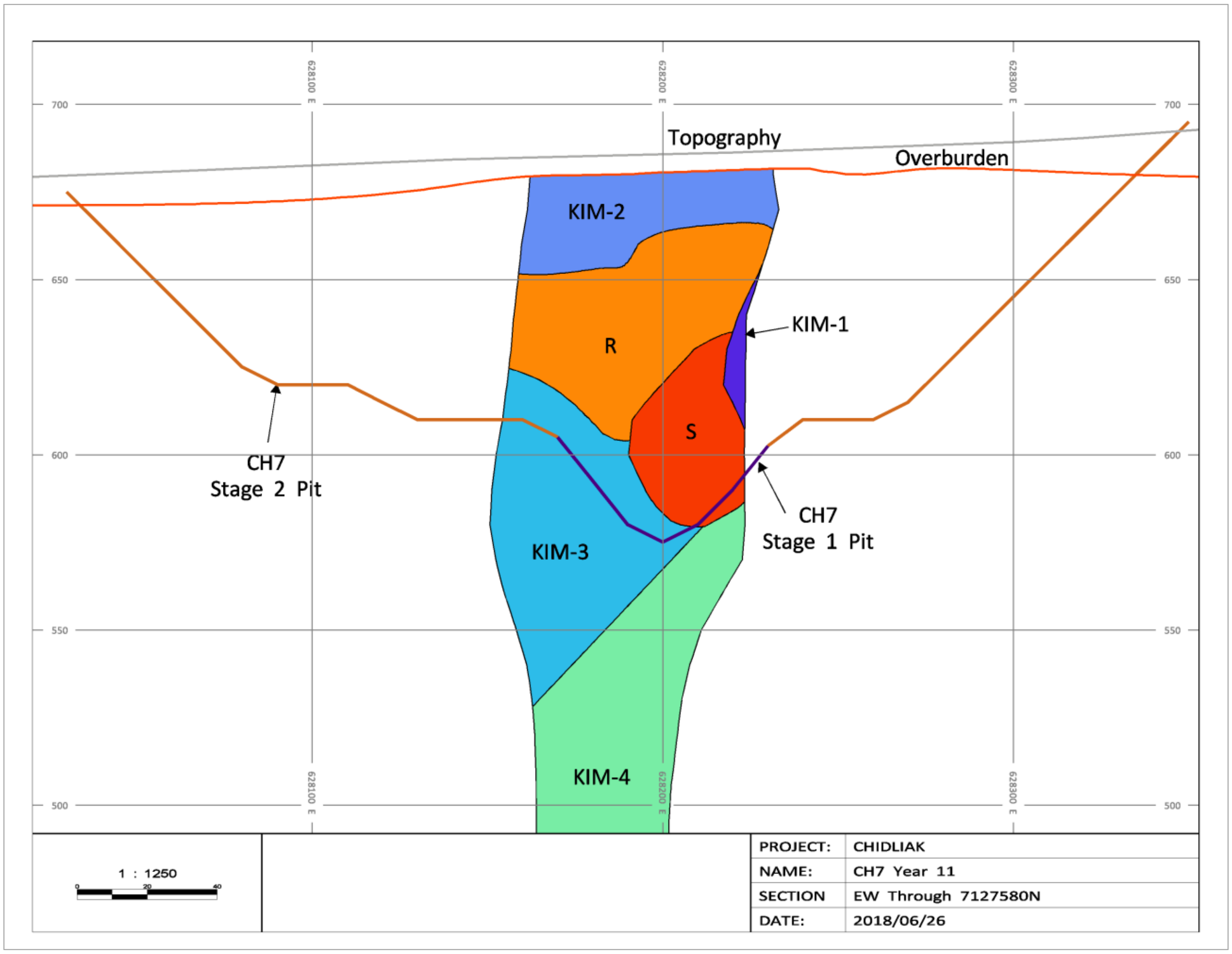
Source: JDS (2018)

Figure 16-18: End of Period Plot – Year 9 CH-7



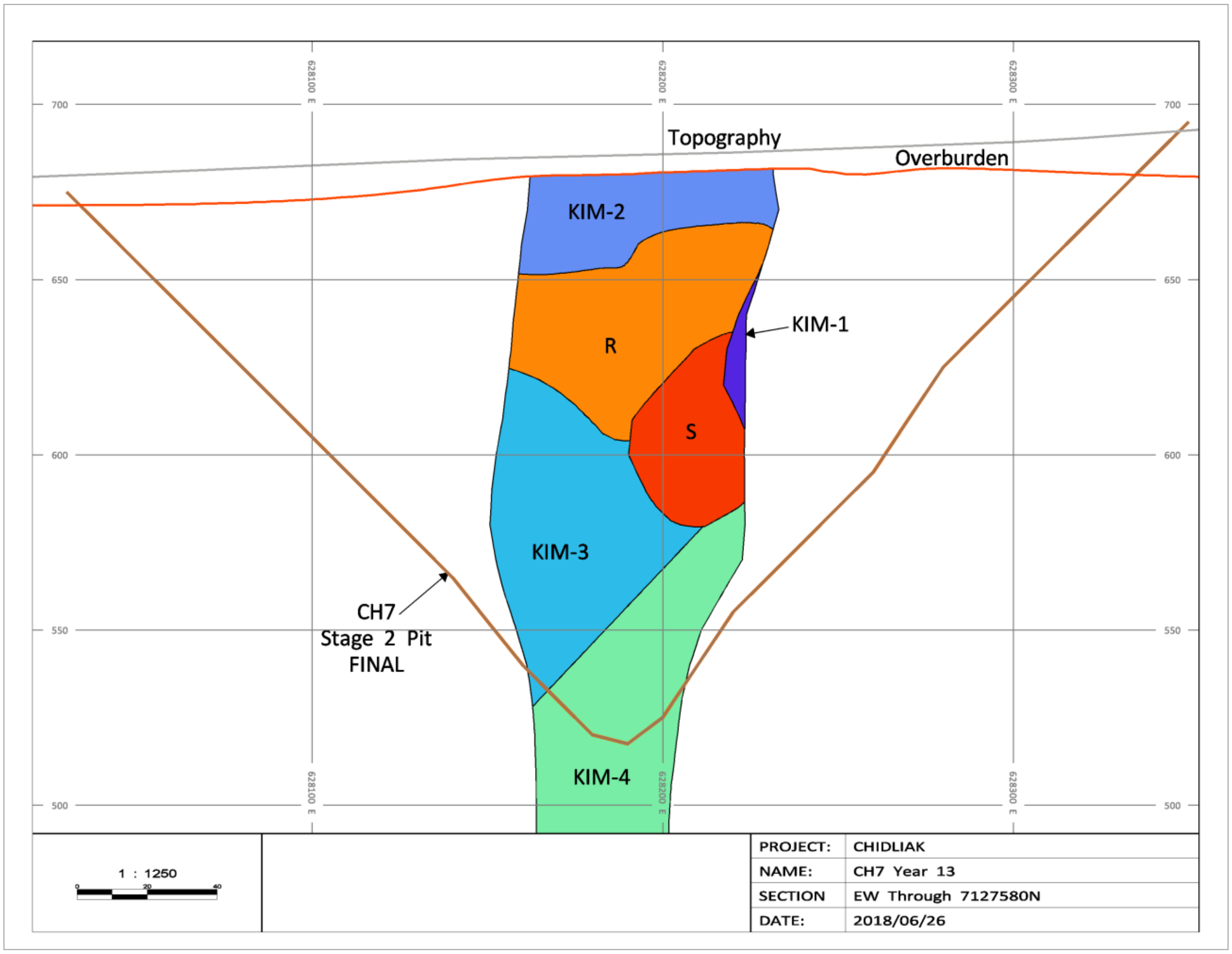
Source: JDS (2018)

Figure 16-19: End of Period Plot - Year 11 CH-7



Source: JDS (2018)

Figure 16-20: End of Period Plot – Year 13 CH-7



Source: JDS (2018)

16.5 Open Pit Mine Equipment

The PEA envisions the use of conventional open pit truck and shovel mining methods. Mining rates will average 2,000 t/d of mineralized material and will range from 4,000 t/d to 28,200 t/d of total material mined over the life of the mine.

The primary owner-operated diesel mining fleet will consist of 64-tonne haul trucks, 7.0 m^3 front shovels, 7.0 m^3 front end loader (FEL) and 150 mm diameter drills. The ancillary open pit mining fleet will consist of typical track dozers, graders, wheel dozers and water trucks. Table 16-11 summarizes the annual open pit mobile equipment requirements for the Project. The fleet shown in Year -3 and Year -2 is for pre-development works and site access.

Table 16-11: Mine Fleet Requirements by Year

Equipment Type	Y -3	Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y 10	Y 11	Y 12	Y 13
Drill (127-203 mm)	1	1	1	1	2	2	2	2	2	1	1	1	1	1	1	1
Shovel (7.0 m^3, Hydraulic)	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
FEL (7.0 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (64 t)	8	8	8	8	9	10	10	10	10	6	6	6	6	6	5	3
Track Dozer (D275 class)	4	4	4	4	4	4	4	4	4	2	2	2	2	2	1	1
Wheel Dozer (824 class)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader (14H class)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Water Truck (55 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Source: JDS (2018)

16.6 Open Pit Mine Personnel

The mine personnel requirements include mine operations equipment operators, mine maintenance, supervision and technical services. Table 16-12 summarizes annual total manpower estimates, where:

- Mine General – front line supervision, superintendents;
- Drill & Blast – drillers, blasters, blaster helpers;
- Load & Haul – shovel/loader operators, haul truck drivers, ancillary equipment operators, labourers;
- Maintenance – heavy equipment mechanics, electricians and welders, maintenance supervisors, tiremen, labourers; and
- Technical Services - mine engineering staff including superintendents, engineers, geologists, surveyors and clerks.

Table 16-12: Mine Operations Personnel by Year

Position	Y -3	Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y 10	Y 11	Y 12	Y 13
Mine General	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Drill & Blast	13	13	13	13	15	16	16	16	16	8	8	8	8	8	6	6
Load & Haul	80	80	80	80	84	84	92	92	92	56	56	56	52	52	46	32
Maintenance	38	38	38	38	38	38	42	42	46	28	28	28	24	24	24	16
Technical Services	13	13	13	13	13	13	13	13	13	9	9	9	9	9	9	6

Source: JDS (2018)

16.7 Mine Development Schedule

- **Year -1:** Development of the Chidliak Project is planned to commence with pre-stripping and mine production of the CH-6 open pit. A total of 3.8 Mt of material is scheduled to be mined, including a small amount of plant feed (0.1 Mt) that will be placed in a stockpile. The pre-stripping is necessary to provide the necessary waste rock (0.6 Mt) to develop the TMF storage facility. The remaining 3.2 Mt is made up of overburden material which was deemed not suitable for TMF construction purposes.
- **Year 1:** The 0.7 Mt/a target plant feed is achieved with mining 0.6 Mt from CH-6 plus rehandling the 0.1 Mt from the stockpile. The average mine grade in Year 1 is estimated to be 2.5 ct/t (from Stage 1 and 2). A total of 9.7 Mt of waste rock is scheduled for an average strip ratio of 15.6:1 (waste tonnes to processed tonnes). Mine production rates are envisioned to average 28,200 t/d total material for the next six years.
- **Year 2:** Process plant feed production is scheduled to be maintained at the target of 0.7 Mt/a (or 2.0 kt/d). Mining continues at CH-6. Total waste planned to be mined from the active stages is 9.5 Mt. The average diamond grade is estimated to be 2.4 ct/t and an overall strip ratio of 13.1:1 for the period.
- **Year 3:** Mining continues in the various Stages at CH-6. Waste tonnage of 9.6 Mt is planned to be mined at an overall strip ratio of 13.1:1. Diamond grades average 2.7 ct/t.
- **Years 4 to 8:** Mining in the Stages at CH-6 are scheduled to be completed over this time frame, while mining at CH-7 is scheduled to commence in Year 8. Average annual 7.0 Mt of total material mined (mineralized material plus waste) with an average strip ratio of 8.5:1. The average diamond grade is expected to be 2.2 ct/t.
- **Years 9 to 13:** Mining will be concentrated at CH-7 in Stages 1 and 2 over the final years of the LOM plan. The average annual strip ratio over this period is expected to decrease to 3.2:1 with an average of 2.3 Mt of waste and 0.7 Mt of process plant feed planned to be mined in each period. Diamond grades decrease to 0.96 ct/t with an average mining rate decreasing to 8,300 t/d.

17 Process Description / Recovery Methods

17.1 Introduction

Due to the lack of ore dressing studies and the limited metallurgical data on CH-6 and CH-7 kimberlite material, the Chidliak Project preliminary process flowsheet is based on other information such as:

- Current Northern Canadian kimberlite processing techniques and equipment applicable to an Arctic operating environment;
- A simplified diamond processing flowsheet to maximize diamond liberation and economic diamond value recovery;
- Maximize comminution and fines generation prior to DMS separation to reduce DMS treatment rates and downstream energy requirements;
- Minimize number of process treatment sections, e.g. single size fraction DMS;
- Proven processes and equipment in Northern Canadian diamond processing plants to utilize current operating and maintenance personnel experience base, this will:
    - Reduce training and operations/maintenance learning requirements;
    - Improve start-up and production ramp-up times;
    - Allow use of existing vendor support services; and
    - Allow a faster operational stability:
        - Standardized equipment types and sizes to reduce on-site spares holdings;
        - Spillage control to reduce plant clean-up labour requirements, e.g. conveyors to be conservatively sized to reduce spillage and full length conveyor underpans to allow conveyor return belt spillage to be collected at defined collection points;
        - Minimize building footprint to reduce operating heating, ventilation and capital construction costs; and
        - Maximize process control, automation and CCTV process operations coverage;
- Northern Canadian operations diamond value management principles to maximize diamond revenue:
    - Minimize material: impact breakage, e.g. crusher gap settings >30 mm, drop height and transport velocity;
    - Maximize value recovery, upper and lower treatment sizes, e.g. nominal 30 mm and 1 mm;
    - Maximize diamond liberation;
    - Maximize diamond recovery using known and proven processes and equipment, e.g. HPGR, DMS, X-ray and grease diamond recovery; and

    - Minimize access and provide secure recovery processes and adequate product protection applicable to the Canadian operating environment;
- Northern construction and process operations cost drivers:
    - Maximize off-site plant fabrication, e.g. primary and secondary crushing plant, conveyor galleries, DMS and recovery plants;
    - Minimize electrical energy requirements; and
    - Minimize plant operations labour requirements.

17.2 Plant Design Criteria

Current mine optimization information shows a potential mineable resource of 9.5 Mt. For preliminary assessment process work, a treatment rate of 100 t/h (2,000 t/d) has been used giving a LOM of up to 13 years. The study process design criteria are shown in Table 17-1.

Table 17-1: Plant Design Criteria

Criteria	Units	Design	Source
Plant Throughput	t/d	2,000	Peregrine / JDS
	t/a	730,000	Peregrine / JDS
Crushing Section Utilization	%	65	JDS
Crushing Section Throughput	t/h	130	JDS
Kimberlite UCS : CH-6	MPa	60-140	Peregrine
Kimberlite UCS : CH-7	MPa	40-145	Peregrine
Process Plant Utilization	%	85	JDS
Plant Throughput	t/h	100	JDS
Processing Power Consumption	kWh/t	14	JDS
DMS Feed Rate	t/h	150	JDS
DMS Treatment Top Size	mm	30	JDS
DMS Treatment Lower Size	mm	1	JDS
DMS Yield : CH-6	%	0.4 to 4.0	Peregrine
DMS Yield : CH-7	%	0.4 to 4.0	Peregrine
Coarse Tailings % Head Feed	%	80	JDS
Fine Tailings % Head Feed	%	20	JDS
Tailings Thickener Underflow Density	% w/w	30	JDS

Source: JDS / Peregrine

17.3 Plant Design

17.3.1 Process Flowsheet Design Features

A proven low-risk Canadian kimberlite processing flowsheet incorporating operations and design elements is based on JDS experience from similar Canadian diamond production plants (Ekati, Diavik, Snap Lake and Jericho). Design features for the planned Chidliak Project process plant will include:

- Conventional primary jaw and secondary cone crushing for ROM material;
- Crushed mil feed stockpile with a plant feed capacity of two days;
- Reduced internal plant storage and surge capacities to minimize the building footprint;
- HPGR comminution before heavy medium separation to maximize fines generation and reduce downstream treatment capacity requirements;
- Single size fraction DMS plant treatment to minimize the building footprint;
- Simplified recovery plant flowsheet consisting of wet high intensity magnetic separation, wet X ray double pass and dry single particle sorter X-ray machines for the +2 mm and grease tables for -2 mm concentrates; and
- High process energy efficiency, material elevations kept to a minimum, reduced circulating loads, high efficiency comminution (e.g. HPGR), wet X-ray treatment (to reduce drying costs), no process water heating.

17.3.2 Mechanical Equipment Design Features

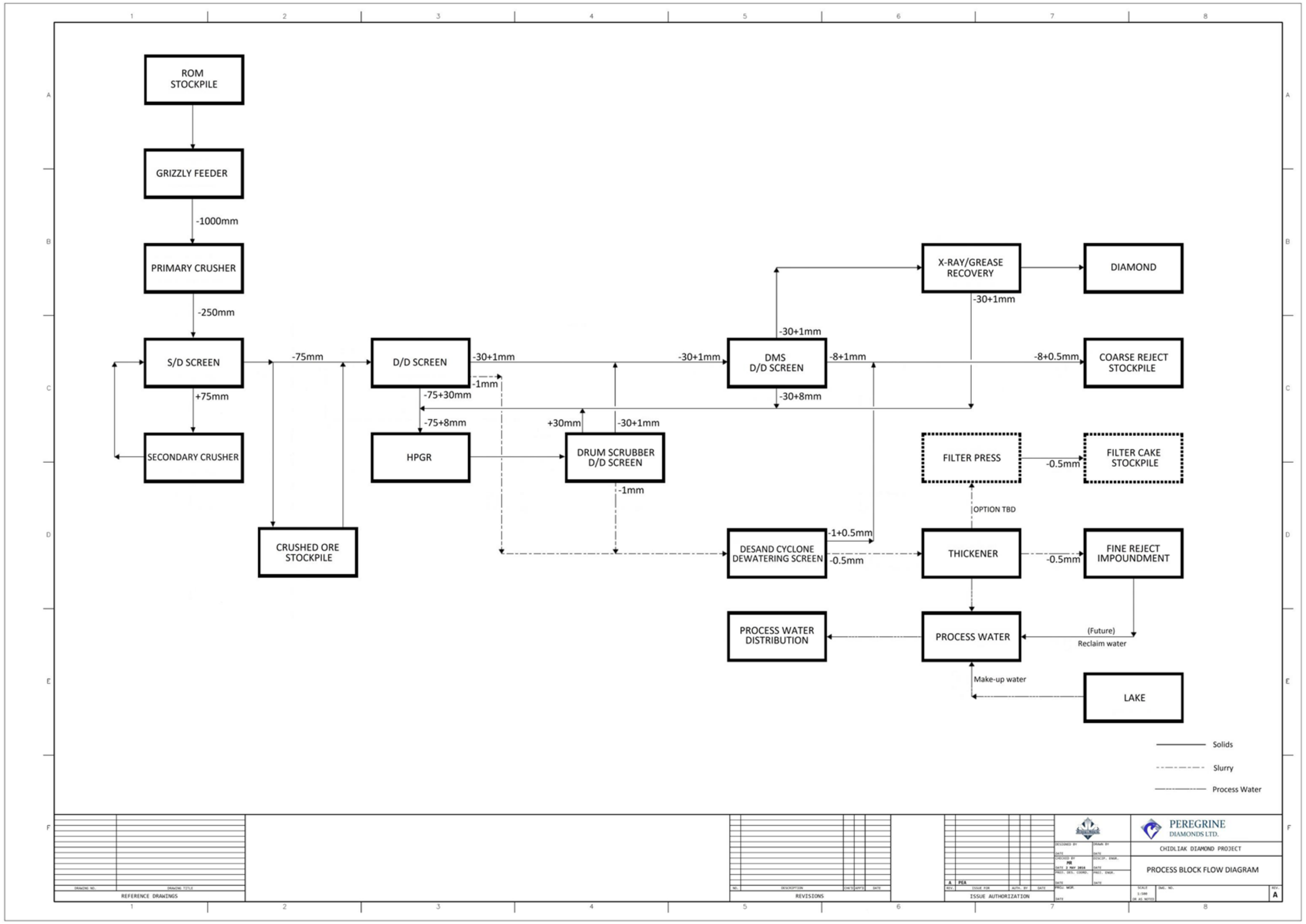
Proven Northern diamond plant equipment will be used to minimize operations risk and exposure in this area. Design features for the planned Chidliak Project process plant include:

- Conventional belt conveyors to minimize spillage;
- Uninsulated, fabric type building for the crushed plant feed stockpile to minimize capital and on-site installation costs;
- Minimized number of different equipment sizes to reduce spares holding requirements, e.g. screens, pumps, conveyor belts;
- Standardize equipment as used at operating Northern diamond mines to reduce spares and vendor support risks;
- High mechanical availability and extended unit wear life, e.g. slurry abrasion resistance (rubber lining of screen wetted parts), tungsten carbide studded HPGR wear segments to reduce wear liner change-out complexity and change-out time; and
- High energy efficiency equipment.

17.4 Process Plant Description

The process plant is designed to treat 100 t/h, 2,000 t/d, 730,000 t/a. The preliminary process flowsheet can be summarized as follows below (see Figure 17-1 for the block flow diagram).

Figure 17-1: Chidliak Project Preliminary Process Block Flow Diagram



17.4.1 ROM Primary and Secondary Crushing

Run of Mine (ROM) crushing to prepare sized process plant feed will consist of a conventional primary jaw and secondary cone crushing circuit with closed circuit screen. The CH-6 and CH-7 kimberlites are classified as medium-hard plant feed containing little clay, even in the upper weathered kimberlite zones.

To reduce on-site installation and construction costs, portable jaw and cone crushing with screening plants are proposed. These will be enclosed within a building structure to provide equipment and personnel protection from the Arctic climate. Heating control will maintain the crushing section operating temperature at approximately 0°C during the sub-zero temperature winter months.

The crushing plant will be fed by a front-end loader from the ROM stockpiles. This will allow plant feed to be pre-sorted for the removal of large oversize material and waste rock and also for plant feed blending capabilities ahead of the crushing plant.

The crushing plant's major equipment consist of:

- Static oversize protection grizzly;
- Feed hopper;
- Vibrating grizzly feeder;
- Primary jaw crusher;
- Conveyor #1;
- -65/70 mm Sizing screen;
- Conveyor #2;
- Secondary cone crusher; and
- Conveyor #3.

17.4.2 Crushed Plant Feed Handling and Stockpile

Crushed plant feed from the primary and secondary crushing section product conveyor discharges into a 50 t process plant feed surge bin equipped with belt weigh feeder to control feed to the process plant feed conveyor. Excess plant feed from the bin discharges via a bypass chute onto the crushed plant feed stockpile stacking conveyor. Sufficient covered storage will be provided for approximately 4,000 t or 2-days plant feed capacity.

Crushed stockpile material will be fed to the process plant by front-end loader (FEL) via a static grizzly, feed hopper and belt weigh feeder onto the process plant feed conveyor. Ferrous tramp metal removal will be allowed for at the discharge of the plant feed conveyor.

17.4.3 Feed Preparation

Initial mini-bulk and bulk sample processing data (refer to Section 11) indicates that the amount of clay, even in weathered kimberlite, to be expected in the plant feed is very low. When additional metallurgical and equipment vendor data is available the flowsheet location of the drum scrubber(s) will be determined at a later engineering stage.

The plant feed material will be sized and washed on horizontal deck and banana type vibrating screens. The -1 mm fines report to the fines pumpbox and the -75 + 30 mm size fraction will be conveyed to a HPGR feed bin equipped with a belt feeder fitted with metal detection to feed the HPGR.

HPGR product, loosely compacted flake, will be conveyed to a rotary drum scrubber for deagglomeration. Scrubber discharge will be sized on horizontal deck and banana type vibrating screens, +30 mm material reporting to the HPGR feed bin, -30 +1 mm to the DMS feed bin and -1 mm to the fines effluent pumpbox.

The HPGR has been the main comminution technology used in Canadian diamond plant designs since the mid-1990s and has considerably improved process efficiencies and diamond recoveries, when compared to previous flowsheet designs, due to the increased diamond liberation and reduced diamond breakage.

17.4.4 Dense Media Separation

A single size fraction DMS plant has been proposed to minimize the overall plant footprint, reduce DMS plant capacity redundancy and section over sizing requirements.

The proposed 150 t/h DMS plant will use three 420 mm diameter pump fed cyclones to reduce the material conveying and elevation required for gravity fed cyclones.

Material will be fed at a controlled rate from the DMS feed bin to the dense medium cyclone feed mixing box where the kimberlite material is mixed with ferrosilicon (FeSi) medium at the correct density. The cyclone feed pump transports the kimberlite material and medium slurry from the mixing box to the dense medium cyclones, where particles will be separated based on their density.

The higher density particles move down the inner wall of the cyclone and discharge through the apex. The lower density particles move towards the central axis of the cyclone and exit from the vortex finder. Both cyclone products report to medium drain and rinse screens to recover the ferrosilicon, the drain section to the correct medium circuit and the rinse section to the dilute medium circuit where the FeSi is recovered by a wet drum magnetic separator and returned to the correct medium circuit.

Correct medium densification, non-nuclear density measurement with Programmable Logic Controller Proportional-Integral-Derivative (PLC PID) control will be used to provide an operating medium density to within 0.01 SG units of the medium set-point density.

DMS concentrate will discharge into a Recovery Plant feed surge bin, DMS concentrate areas will be secured to limit access and will be CCTV monitored.

Dense medium cyclone float material will be sized on a double deck drain and rinse screen, -30 + 8 mm oversize returning to the HPGR feed bin.

17.4.5 Recovery Plant

The recovery plant flowsheet and supply requirements need further development when additional metallurgical and equipment vendor data is available. Recovery plant supply options will include:

- A dry X-ray plant fabricated off-site;
- A wet X-ray plant fabricated off-site; and

- A simplified wet X-ray and grease table plant constructed on-site. This option was used in the order of magnitude cost estimate and based on Northern diamond mine recovery plant costs.

17.4.6 Tailings Deposit

17.4.6.1 Coarse Tailings

The -8 +1 mm coarse DMS tailings will be conveyed to a stockpile outside of the main plant building from where the tailings will be transported to the waste dump by FEL and haul truck.

17.4.6.2 Fine Tailings

The -1 mm solids will be pumped to de-sand cyclones, cyclone overflow gravitating to a deep bed type thickener. De-sand cyclone underflow will be dewatered on inclined deck screen, the -1.0 +0.5 mm oversize discharging to the DMS coarse rejects conveyor.

Thickener overflow will gravitate to the process water tank for reuse and to reduce capital and operating costs, the process water will not be heated, as existing Northern operations have shown there is no advantage in this. Thickener underflow, 30-35% solids by weight, will be pumped to the fine tailings impoundment area.

17.4.7 Water Management

After the initial water fill in the fine tailings impoundment area, the plant make-up (raw) water and plant process water supply will become a ‘closed’ circuit. A reclaim water pump located in the fine tailings impoundment will return raw water to the plant process water tank.

The plant make-up water requirement is estimated at 50 to 80 m^3/h which is 15% of the total process water flowrate.

17.5 Diamond Valuation

Prior to the shipment of diamond parcels from site, diamond cleaning and sorting for government valuation purposes will be provided in a dedicated secure facility located in the recovery plant area.

Sort house operations personnel will assist Peregrine’s contract diamond valuation company in the preparation of the diamonds for government valuation. Diamond valuation cycles are expected to be on a four to six-week basis.

18 Project Infrastructure and Services

18.1 Summary

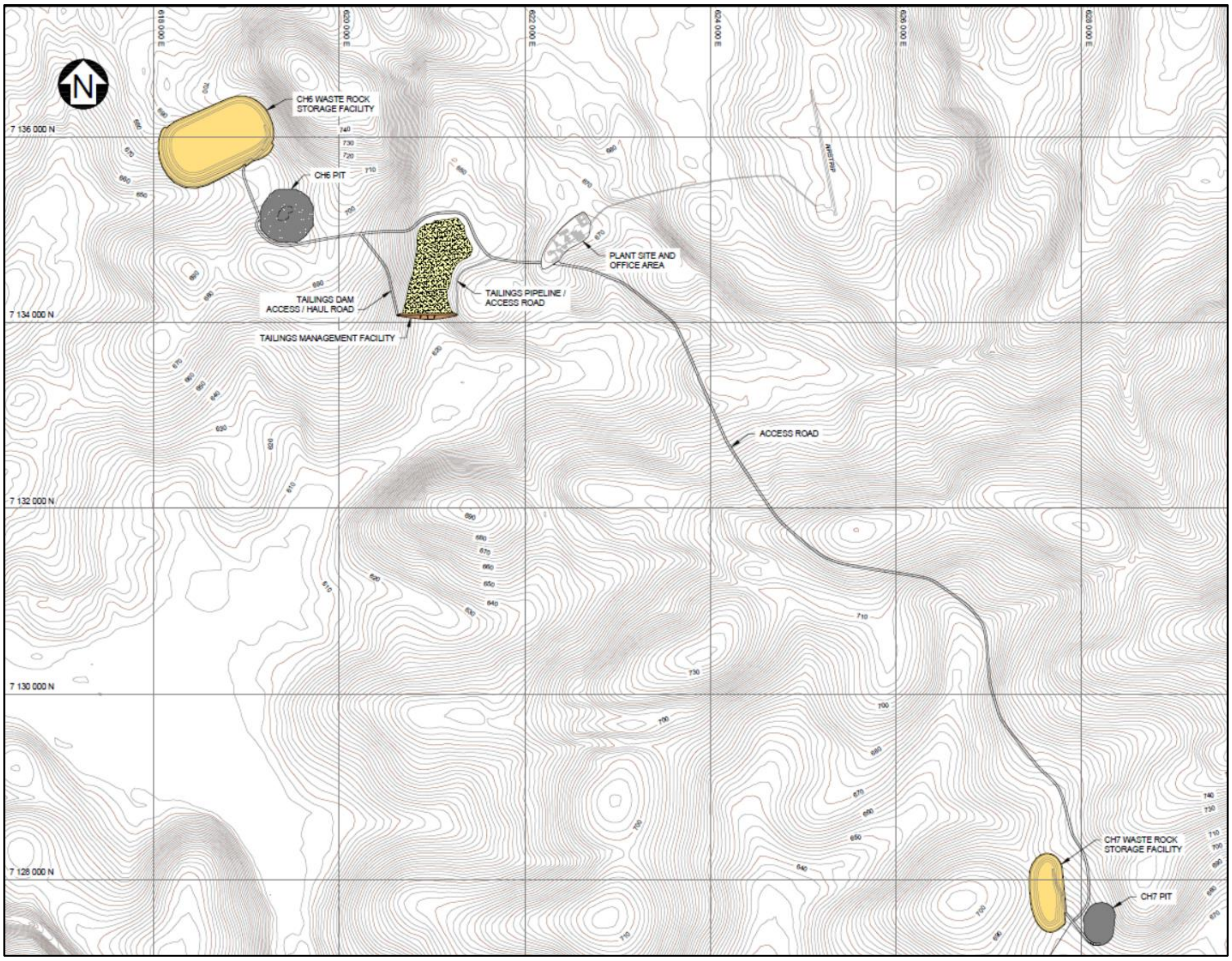
The Project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 157 km of all-seasonal access road from Iqaluit, NU to the plant site location;
- Crusher plant;
- Process facilities;
- Diesel power plant;
- TMF and WRSF;
- Permanent camp (established for the construction stage);
- Administration and mine dry building;
- Truck shop and warehouse;
- Mine dry and office complex;
- 20 ML of fuel storage and containment in Iqaluit, NU;
- 300,000 L of on-site fuel storage and distribution;
- Industrial waste management facilities such as the incinerator; and
- Site water management facilities.

18.2 General Site Arrangement

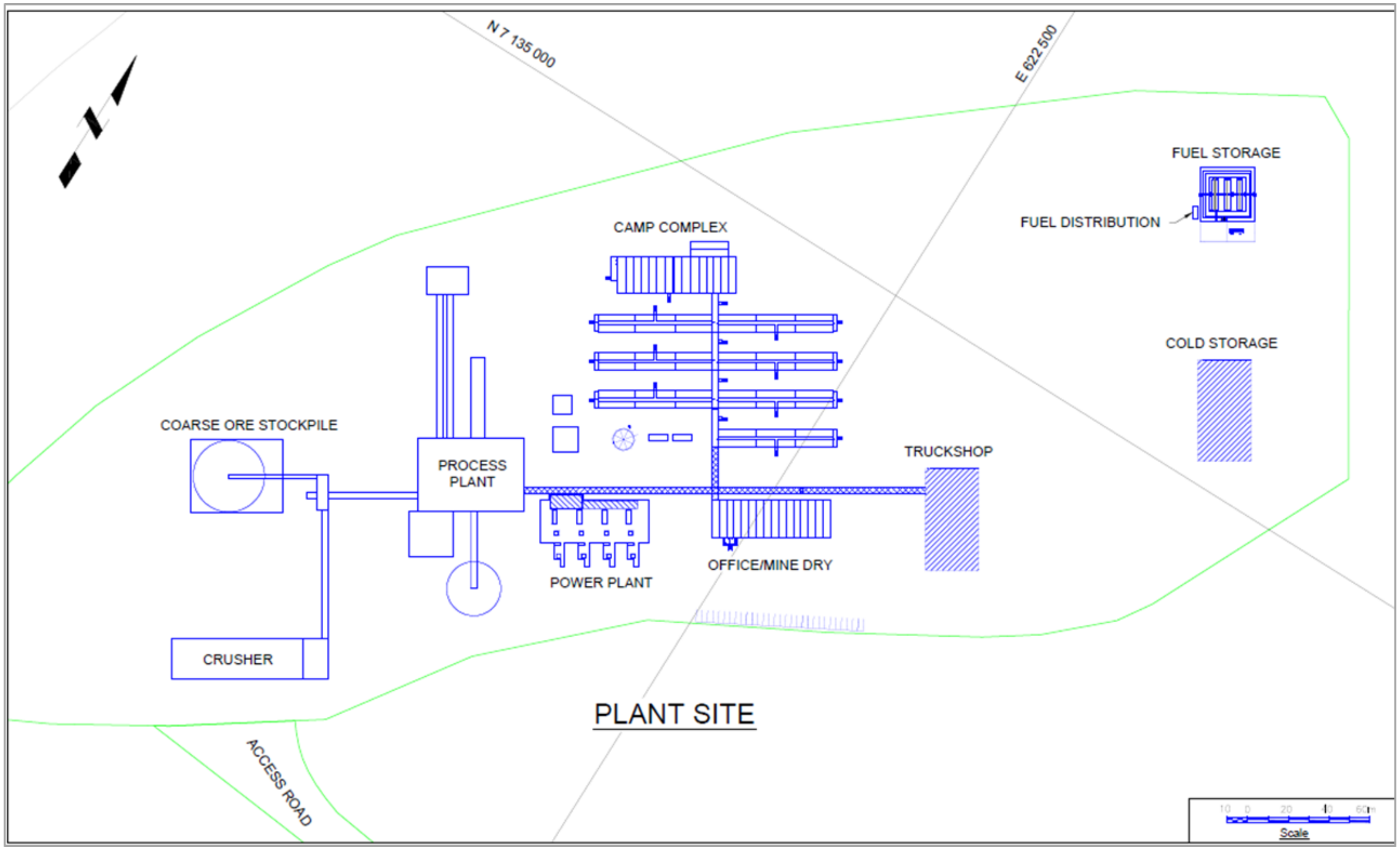
The general location of the planned plant site and open pits are shown in Figure 18-1. A layout for the envisioned plant site area is shown in Figure 18-2. The plant facilities are planned to be located on elevated ground, where bedrock is expected to be shallow. Foundation conditions and drainage requirements are expected to be better in this location as opposed to a lower lying area. The TMF will be located in the cirque valley north of the plant complex. The proposed location minimizes the dam footprint, construction earthwork volume and catchment area, while maximizing the storage capacity of the impoundment.

Figure 18-1: General Site Arrangement



Source: KP (2018)

Figure 18-2: Plant Site Layout



Source: JDS

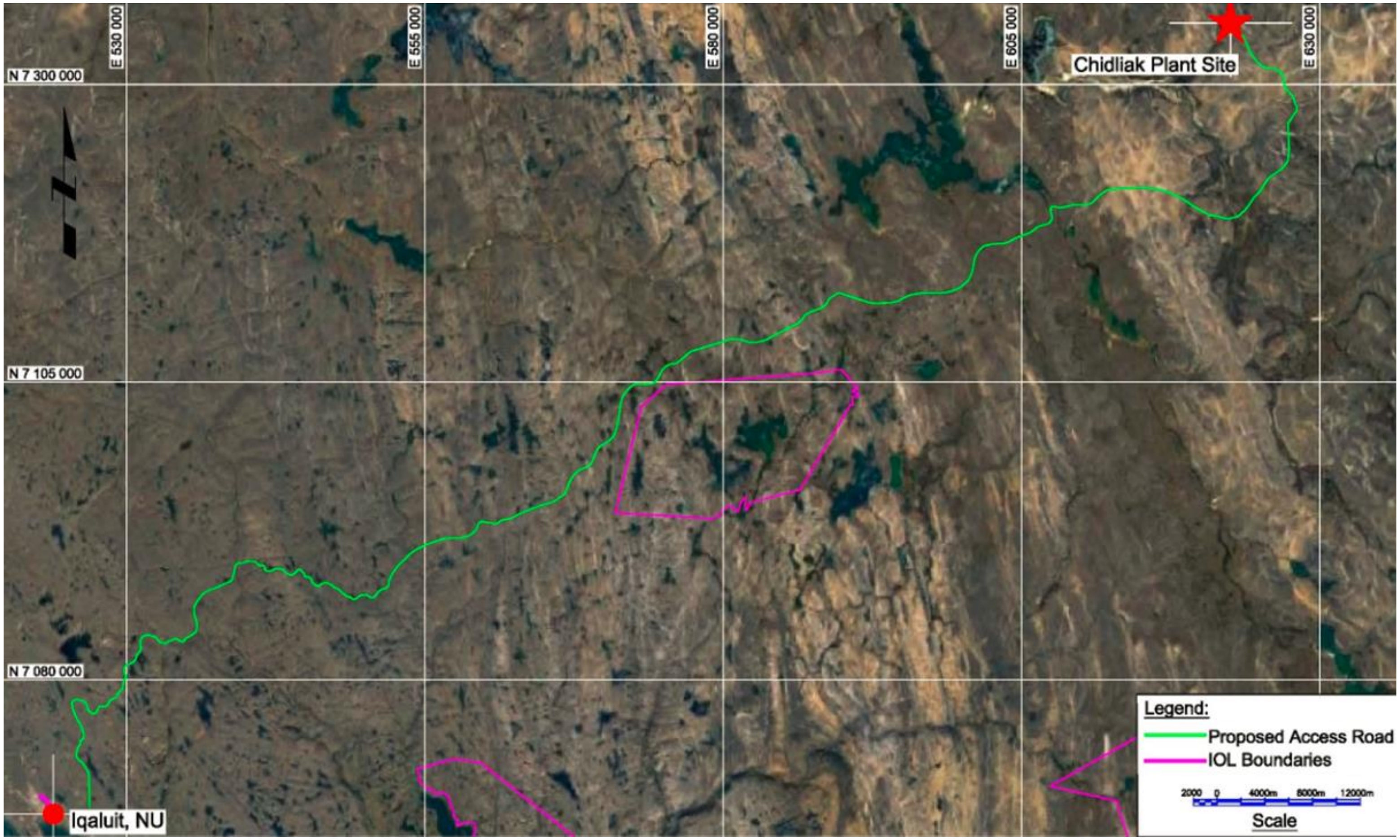
18.3 Site Access

Approximately 157 km of all-season access road will be required to connect the planned plant site to Iqaluit. The road will be a radio assist, single-lane, gravel road with inter-visible turnouts. The access road will be constructed in a permafrost environment and therefore the road will consist exclusively of fill with no cuts except in areas of exposed bedrock where cuts can be constructed. The road will be approximately 15 m wide with a maximum designed grade of 10%. The alignment of the proposed access road is shown in Figure 18-3, and is based on road option 1 of the preliminary road lay-out studies by Tetra Tech EBA (2011b). The access road will be constructed prior to site construction activities with the Owner's mining equipment and contractor labour. The owner’s equipment fleet will be purchased earlier, prior to operations start up, so as to be available for the road construction.

Personnel will either reside in Iqaluit or be flown from southern cities to Iqaluit via chartered aircraft. Personnel will be transported to and from the site by bus. Since personnel and freight will utilize the all-weather road, a separate airstrip will not be constructed. A segment of the internal roads will be designed for and designated as airstrip emergency flights, such as medi-vacs.

Freight will be transported from Eastern Canada to Iqaluit annually via a sealift. Freight will be offloaded at a deep-water port that is assumed to be built by a third party prior to the construction of the Project.

Figure 18-3: Proposed Access Road



Source: Google Maps & JDS

18.4 Buildings and Structures

18.4.1 Crushing and Process Building

The crushing plant will be located in a pre-engineered structural steel building with dimensions of 40 m long by 25 m wide. The crushing building will be heated in the winter to 0°C by glycol air handlers and unit heaters.

The process plant will be located in a pre-engineered structural steel building with dimensions of 52 m long by 36 m wide. Overhead cranes will be provided for equipment maintenance. The building will be heated to 5°C by glycol air handlers and unit heaters.

18.4.2 Truck Shop and Warehouse Building

The truck shop complex at the Chidliak site will consist of a 50 m long by 26 m wide structural steel, pre-engineered building designed to accommodate facilities for repair and maintenance of mining equipment and light vehicles. The building will also house warehouse storage space for spare parts, consumables and other materials and equipment. Additional covered cold storage will be provided with a 50 m long x 30 m wide pre-engineered fabric building.

18.4.3 Mine Dry and Office Complex

The 1,070 m^2 mine dry and office complex will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will rest on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will be provided to connect the mine dry and office complex with the camp core facilities and rooms.

The mine dry facility will service construction and operations staff during the life of the Project. It will be capable of servicing 200 workers during shift change and contain the following:

- Male and female clean and dirty lockers; and
- Showers and washroom facilities with separate male and female sections.

The site office facility will contain the following items:

- Private offices;
- Main boardroom; and
- Mine operations line-up area.

18.4.4 Camp

The camp will comprise single-occupancy rooms with central washrooms. It will be used during the construction stage and throughout the operations stage. There will be seven dormitory wings, each capable of housing 42 people for a total of 294 beds.

The kitchen / dining / recreation complex will include the following:

- Kitchen complete with cooking, preparation and baking areas, dry food storage and walk-in freezer / cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- First aid room;
- Mudroom complete with coat and boot racks, benches and male-female washrooms;
- Housekeeping facilities;
- Reception desk and lobby; and
- Recreation area.

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the main camp complex and dormitory wings.

18.4.5 Fuel Storage

Off-site fuel storage at the Iqaluit tank farm area will be designed with capacity for a 1- year supply of diesel fuel required for operations of the Chidliak sites. A total of two 10 ML field-erected fuel tanks will be constructed in Iqaluit. The fuel tanks will be filled directly from annual fuel supply ships through the shore manifold.

On-site fuel storage is designed with a one week supply capacity. A total of three 100,000 L tanks will be installed within a lined containment berm. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the process plant.

Fuel will be transported by a contractor from Iqaluit to the Chidliak site on a daily basis via the main access road.

18.5 Power Supply

A single captive power plant will be used to meet the electrical power demand necessary to support the Chidliak operation. The power plant comprises four diesel-fired reciprocating engine generator sets (gensets) in an N+2 (2+2) arrangement. Each generator will be prime rated for 1,566 kW running at 1,800 rpm and generating power of 4,160 V. The peak gross power is 3.1 MW (two 1.57 MW operating gensets @ 100%). The average estimated electrical loads is 1,925 kW with the average annual power consumption estimated to be approximately 17,000,000 kWh/a.

To maximize the overall efficiency, this power plant will operate as a combined heat and power plant (CHP Plant).

The power plant will be modular with all gensets interconnected. Each genset will be packaged in a walk-in, sound-attenuated enclosure that is constructed, assembled and tested prior to shipment to site.

18.6 Freight

The logistics and freight transportation of operational supplies is a critical component of the Chidliak Project. The majority of dry freight and fuel will need to be sourced, procured annually and then transported via an annual sealift to Iqaluit where it will be stored until it can be transported to site. Dry freight will be consolidated annually at a marshaling port location in eastern Canada, such as Becancour or Montreal, QC, and then shipped via ocean vessels to Iqaluit. A local contractor will be responsible for offloading the ocean vessels and storing the material until it is required on-site. The majority of the operating supplies and consumables will be transported to site and stored in 20 ft ISO containers (sea containers). A contractor will also be used to transport the freight from Iqaluit to site. Freight will be delivered to site on the access road and off-loaded at the warehouse or another designated area.

18.7 Explosives Storage

Explosive storage at the Chidliak site will consist of the following three main components:

- Bulk ammonium nitrate storage and loading facility;
- Emulsion manufacturing facility; and
- Explosive storage magazines.

Bulk ammonium nitrate prill will be shipped to site in one-tonne tote bags within 20 ft ISO containers, with a storage area is sized to allow for a one-week supply.

Bulk emulsion explosives will be manufactured on-site at the emulsion facility and then pumped into the bulk explosive truck.

Packaged explosives and explosive detonators will be stored in approved explosive magazines located on separate pads. The powder magazine will be a 40 ft container magazine capable of holding 32 t of explosives, and the cap magazine will be a 20 ft container magazine capable of holding approximately 600 cases of detonators.

The design of all storage facilities will meet government regulations and will be located according to required separation distances as regulated by the Explosives Regulatory Division (ERD) of Natural Resources Canada (NRC).

The location of the explosives storage site will be determined as part of future pre-feasibility or feasibility studies.

18.8 Waste Rock Management

The Waste Rock Storage Facilities (WRSF) are proposed to be located immediately adjacent to the final pit limits for the CH-6 and CH-7 deposits. Given the deposit configuration, extraction sequence and possible future resources, no backfilling into previously mined out areas is currently planned for Chidliak.

In addition, the coarse fraction of processed kimberlite (80% of total kimberlite processed) will be co-disposed at the CH-6 WRSF. The CH-6 WRSF is designed to hold a total of 64 Mt of material, while the CH-7 WRSF has a design capacity of 14 Mt.

The WRSFs will be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals giving an overall operational slope of 2.5:1.

18.9 Tailings Management Facility

18.9.1 General

Two tailings streams will be generated from processing kimberlite in the mill:

- Coarse tailings stream (grits)
- Fine tailings stream (processed fine kimberlite or PFK)

The grits, which comprise approximately 80% of the total processed tailings (by weight), will be free of excess moisture. They will be discharged using a conveyor and stockpiled immediately adjacent to the process plant. The grits will be loaded and trucked to the CH-6 Waste Rock Storage Facility (WRSF) for co-disposal with mine waste rock. The PFK, comprising the remaining 20% of the total processed tailings (by weight), will be discharged as a low solids content slurry to the PFK Tailings Management Facility (TMF).

18.9.2 Design Assumptions

The inputs and design assumptions used for this study are included in Table 18-1.

Table 18-1: Design Inputs and Assumptions for the TMF Design

Input	Value	Source / Comment
Mine Production Parameters		
LOM ore tonnes	CH-6 Pit = 6.2 Mt	Provided by JDS (May, 2018)
	CH-7 Pit = 3.3 Mt	
	Total = 9.5 Mt	
Process throughput	Nominal 2,000 t/d	Provided by JDS (May, 2018)
Mine life	13 years	Provided by JDS (May, 2018)
Tailings Disposal		
PFK Tailings Management Facility	CH-6 PFK = 1.2 Mt	20% of CH-6 Ore (NI-43-101 PEA, JDS 2016)
	CH-7 PFK = 0.7 Mt	20% of CH-7 Ore (NI-43-101 PEA, JDS 2016)
	Total = 1.9 Mt	
Waste Properties		
Tailings split	Grits = 80%	NI-43-101 PEA (JDS, 2016)
	PFK = 20%	
Tailings dry density	Grits = 1.35 t/m^3	NI-43-101 PEA (JDS, 2016)
	PFK = 0.95 t/m^3	
Tailings beach slope	0%	NI-43-101 PEA (JDS, 2016)
PKF TMF Storage Requirements		
Water storage allowance	160,000 m^3	KP Update (May, 2018)
Ice cap thickness	Maximum of 2 m	NI-43-101 PEA (JDS, 2016)
Inflow design flood (IDF) and freeboard allowance	2 m	KP Update (May, 2018)
Total storage volume (PFK tailings)	2.0 Mm^3	KP Update (May, 2018). Includes water storage allowance. 2 m ice cap, IDF and freeboard is in addition to the total storage volume.
Site Conditions		
Mean annual precipitation (MAP)	489 mm	NI-43-101 PEA (JDS, 2016)
Mean annual evaporation (MAE)	138 mm	
Active layer thickness (ALT)	Maximum of 2 m	NI-43-101 PEA (JDS, 2016)

Source: Knight Piesold (2018)

The following information is also relevant to the TMF design:

- This is a desktop study, with site information taken from the document library provided by Peregrine and JDS;

- The Project is located in the continuous permafrost region. Permafrost is extensive with thicknesses likely exceeding several hundred metres, and the active layer thickness (ALT) generally less than 2 m, and often less than 1 m (Tetra Tech EBA, 2015);
- No specific geotechnical site characterization studies have been carried out in the vicinity of the proposed TMF site. For preliminary design purposes, it has been assumed that there may be up to 20 m of overburden soils of unknown origin overlying competent bedrock;
- In accordance with the National Building Code of Canada, the Project site is in the "stable" seismic Region of Canada. As a result, seismic stability of the TMF is not expected to be a concern;
- There has not been any geotechnical or geochemical characterization carried out on any of the tailings streams. The assumed tailings densities listed in Table 18-1 are the same as the previous version of the PEA. Additionally, it has been assumed that neither the grits nor the PFK will be Potentially Acid Generating (PAG), and will exhibit neutral drainage metal leaching characteristics;
- The Project site has a net positive climatic water balance, i.e. mean annual precipitation (MAP) exceeds mean annual evaporation (MAE) (Tetra Tech EBA, 2011a);
- A suitable reliable supply of process make-up water for the mill is estimated to be at least 15 km from the site (Tetra Tech EBA, 2011b) and internal water recycle should be optimized to the greatest extent; and
- TMF embankment and storage volume calculations are based on 2 m contour interval topographical mapping for the site provided by JDS.

18.9.3 Tailings Alternatives Assessment

18.9.3.1 Approach

Processed kimberlite tailings management strategies adopted in the Canadian Arctic include both separate grits and PFK management, as well as combined management of the tailings streams. Storage needs, assuming both these scenarios, were previously assessed for the Project:

- Scenario 1 – storage of both grits and PFK in a single TMF; and
- Scenario 2 – storage of PFK only. For Scenario 2, it is assumed that the grits will be co-disposed with waste rock in the CH-6 WRSF.

18.9.3.2 Deposition Sites

Tailings deposition sites were assessed in the 2016 PEA and that assessment forms the basis for the updated design. The initial assessment included a general search radius of approximately 5 km from the CH-6 and CH-7 pits, with sites that will have sufficient capacity to store both grits and PFK tailings (Scenario 1). However, the grits will be co-disposed with mine waste rock and the PFK only (Scenario 2) TMF storage volume is significantly less. A smaller tailings storage site is required, and the PFK TMF site was subsequently selected to provide a larger natural catchment, which is advantageous from a water supply perspective. Site TMF-7 was selected as the preferred PFK TMF site.

18.9.3.3 Tailings Deposition Methods

Results from previous studies are presented herein. Under the assumption that grits will be deposited with mine waste rock, four methods were considered for PFK tailings deposition:

1. Low solids content slurry: conventional slurry tailings with solids content around 30%
2. Thickened tailings slurry: tailings solids content increased to around 60% but still pumpable with centrifugal pumps
3. Paste tailings: additional tailings thickening to the point of requiring pumping with a positive displacement pump. Typically, the slurry solids content is in excess of 60%.
4. Filtered tailings: dewatering tailings that can be handled using conveyors or trucks and shovels. The tailings are "stacked" by placing them in a dedicated facility in compacted lifts.

Comments on the tailings deposition methods are provided below.

- Low solids content slurry deposition is the simplest method, and may result in lower settled tailings density, with the greatest amount of water delivered to the TMF with the slurry. This is a typical tailings management approach.
- Thickened tailings slurry may result in slightly increased settled density and reduced water losses, but requires additional capital and operating costs. Given the level of accuracy of this study, thickened slurry was not considered further for evaluation
- Paste tailings may result in increased settled density, but require positive displacement pumps, at a substantial capital cost increase. Therefore, paste tailings deposition was not considered for this study.
- Filtered tailings are possible if the material is sufficiently coarse and low in clay content to allow efficient dewatering. A filtered tailings stack will result in the smallest tailings footprint, with reduced risks. However, filtering is more complex and has considerable increased capital costs (filter equipment), as well as added operating costs for handling and placing tailings. The PFK material will be fine and, combined with operational complexity and costs, filtered tailings management was not considered for this study.

Considering the level of study, it was decided to proceed with a low solids content slurry TMF for the PFK material. This will yield the lowest upfront capital costs and operating costs. In addition, the PFK TMF will serve as a water storage reservoir, which reduces the need for a separate make-up water supply and storage facility.

Two alternative deposition strategies were considered for the low solids content slurry TMF:

- Sub-aqueous deposition; and
- Sub-aerial deposition.

Given the low settled dry density typically observed for PFK, and to reduce long-term risks associated with the TMF, sub-aerial deposition is preferred. This will enhance consolidation, which will allow the TMF to attain a stable closure condition in a shorter amount of time. In addition, since the PFK is assumed to be geochemically benign, there is no long-term closure advantage to sub-aqueous deposition.

18.9.4 TMF Design

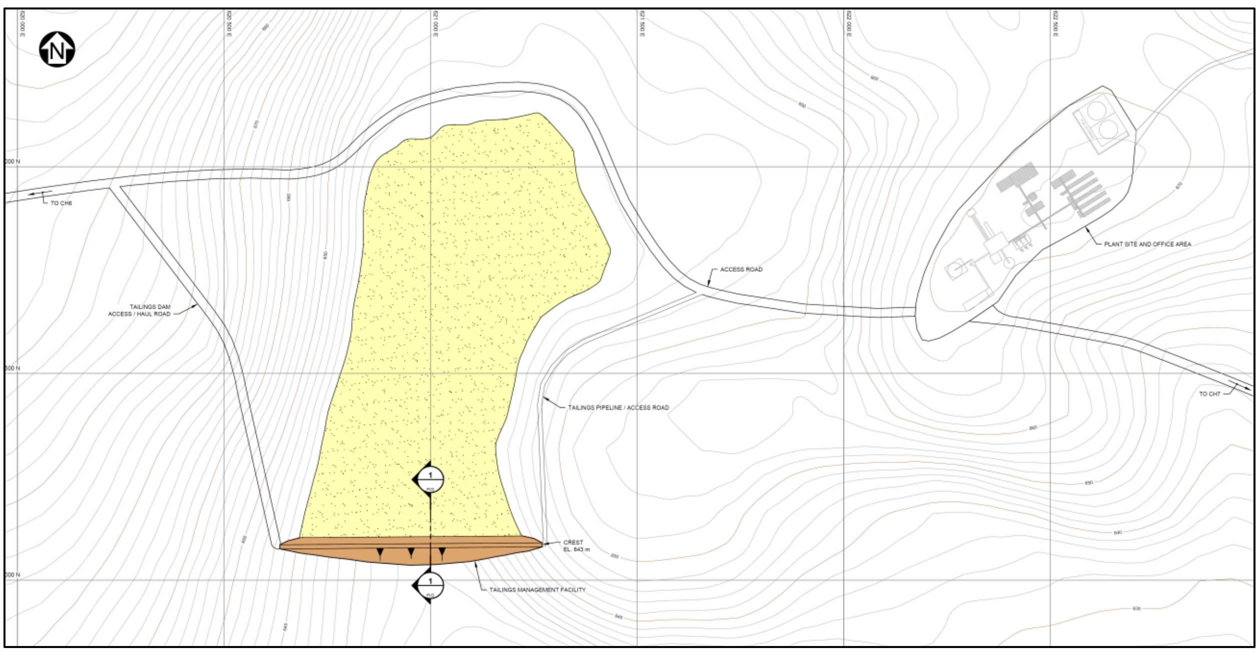
18.9.4.1 Construction Materials

It has been assumed that there are no nearby suitable natural borrow sources for use as construction materials. The embankment will therefore have to be constructed from quarried rock (or pre-development mine waste rock) with associated crushing and screening.

18.9.4.2 Embankment

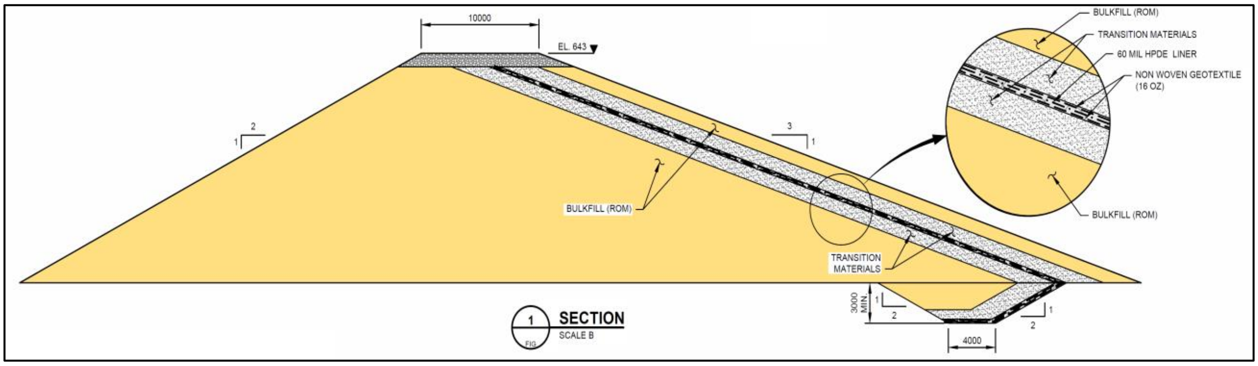
The embankment for the PFK TMF will have a frozen foundation. No analyses to confirm the stability, seepage or thermal integrity of the structure have been completed. The general site arrangement for the PFK TMF is presented in Figure 18-1. The embankment is shown in Figure 18-4 and a typical cross-section is presented in Figure 18-5.

Figure 18-4: PFK TMF Plan View



Source: KP (2018)

Figure 18-5: PFK TMF Typical Cross Section



Source: KP (2018)

The foundation conditions are not known but are assumed to consist of deep overburden overlying bedrock. A key trench will be excavated to a depth of 1 m below the active layer, which is assumed to be 2 m thick at the site (Table 18-1). A geosynthetic liner (assumed to be a 60 mm high density polyethylene (HDPE) liner), sandwiched between two 16 oz. nonwoven geotextile layers, will be placed in the base of the key trench, and will extend along the upstream key trench slope between 0.3 m thick gravel bedding layers.

The embankment is to be constructed with geochemically suitable ROM waste rock or quarried rock and will have an upstream slope of 3H:1V, and a 2H:1V downstream slope. The upstream slope is flatter to facilitate liner installation, which will extend from the key trench along the upstream slope of the embankment. There will be a 1 m thick transition layer of 150 mm minus screened geochemically suitable rockfill on either side of the liner bedding layer.

18.9.4.3 Construction Timing

The embankment key trench is to be constructed during the winter to ensure that there is no deepening of the active layer during excavation. Once the key trench has been backfilled, construction of the remaining above ground structure could continue into summer months.

If mine waste rock is to be used as opposed to quarried rock for construction of the embankment, pre-stripping may be required to ensure that the dam can be completed at least one year prior to the start of operations. This will be necessary to provide at least one season to develop a water supply for mill start-up.

18.9.4.4 Tailings Deposition Plan

To ensure thermal performance of the embankment, the base of the key trench will have to remain frozen, and to maintain that, it will be necessary to develop a tailings beach along the upstream embankment face and push the pond as far away from the key trench as possible. Tailings deposition should therefore initially be from the crest of the embankment. Given the low production rate, and the small volume of PFK tailings, a single spigot discharge is assumed. Therefore, to achieve proper distribution of the tailings beach, the spigot location will need to be moved along the embankment crest over the life of the mine.

18.9.4.5 Construction Quantities

Embankment quantities, accurate to ±30%, are presented in Table 18-2. These are neat line quantities, i.e. they do not account for bulking or shrinkage, and exclude overlap requirements for the liner and geotextile.

Table 18-2: TMF Embankment Schedule of Quantities

Description	Units	Quantity
Key Trench Excavation & Clearing		
Clearing and Grubbing	sq m	11,000
Removal of Cut Off Trench Material	cu m	22,000
Bulkfill (Run Of Mine)		
Over Haul / Spread Bulk Fill	cu m	280,000
Transition Material		
Load / Haul / Spread / Compact Transition Material	cu m	60,000
Liner Bedding		
Load / Haul / Spread / Compact Transition Material	cu m	9,000
HDPE Liner		
Supply, Deliver and Install	sq m	30,000
Geotextile Liner		
Supply, Deliver and Install	sq m	60,000

Source: KP (2018)

18.9.4.6 TMF Closure

Considering that the tailings are geochemically benign, the facility will be closed by removing excess water from the facility before placing a nominal 0.3 m thick rockfill cover over the tailings surface. The cover will have to be constructed during the winter when the tailings surface is frozen to ensure trafficability. An overflow spillway capable of handling storm events will be constructed to manage surface run-off. A dam classification assessment will be conducted in future studies to determine the design storm event.

18.10 Water Management

18.10.1 Make-up Water Sources

Potential make up water supply sources were previously evaluated for the Project (Tetra Tech EBA, 2011a). Two water supply areas (A and D) were deemed potentially suitable, however these sites are located 15 to 20 km from the Project. Consequently, the decision is to use the PFK TMF for water supply to minimize costs and maximize recycle. All water except for fresh water requirements will therefore be sourced from the TMF, as discussed below.

18.10.2 Preliminary Water Balance

18.10.2.1 General

A high-level water balance model was developed for the TMF. The model was used to estimate the magnitude and extent of any water surplus and/or deficit conditions for the duration of mine operations based on average annual climatic conditions. The model did not consider the seasonal (or monthly) variability in climatic inputs and future water balance studies should be conducted on a monthly time-step.

18.10.2.2 Input Parameters and Assumptions

Input parameters to the water balance were taken from the PEA (JDS, 2016) where available. Other parameters were selected by KP based on experience and engineering judgement. Key water balance input parameters are summarized in Table 18-3.

Table 18-3: Water Balance Input Parameters

Component	Assumption
Mean Annual Precipitation (mm)	489
Mean Annual Lake Evaporation (mm)	138
Mine Production Rate (dry metric tonnes)	
Grit Tailings (tpd) (80%)	1,600
PFK Tailings (tpd) (20%)	400
Total Tailings Production, Grit + PFK (Mt)	9.5
Estimated Mine Life (years)	13
Estimated Water in Ore (%)	3.0
Grit Tailings Solids Content by Mass Leaving Mill to CH-6 WRSF (%)	80
PFK Tailings Solids Content by Mass in Slurry to TMF (%)	30
Mill Freshwater Requirement (%)	5.0
Tailings Production	
Annual Grit Tailings to CH-6 WRSF (kt)	584
Annual PFK Tailings to Tailings Management Facility (kt)	146
Grit Tailings	
Tailings dry density (tonnes/m^3)	1.35
Tailings specific gravity	2.6
PFK Tailings	
Tailings dry density (tonnes/m^3)	0.95
Tailings specific gravity	2.6
Tailings Management Facility (TMF)	
Minimum operating pond volume to sustain operations through winter (months)	5.0
Minimum Ice Cover (m)	2.0
Minimum operating freeboard above ice cover for storm and wave run-up (m)	2.0
Embankment Seepage Losses (as % of water in slurry)	1.0
Groundwater inflows from Open Pit (L/s) (see Note 1)	0

Note:

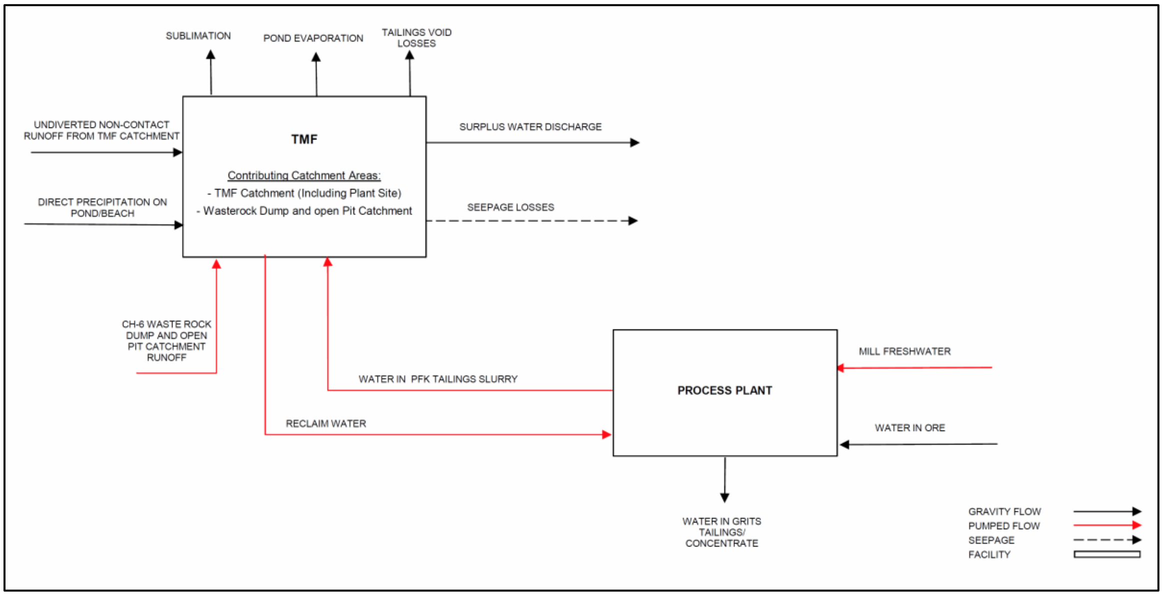
1. Groundwater inflows to the CH-6 Open Pit are assumed to be nil due to the deep permafrost conditions at site.

Source: KP (2018)

Catchment areas for the various elements of the water balance model were delineated by reference to the PEA layout (JDS, 2016), using 2 m interval contour data provided by JDS. Runoff coefficients (RC) were estimated based on the PEA and similar projects.

A schematic illustration of the water balance is presented on Figure 18-6.

Figure 18-6: Water Balance Flow Schematic



The following notes summarize the major components of the water balance as shown above:

- Runoff to the TMF will include rainfall and snowmelt from the area upslope of the TMF, the plant site, and the CH-6 Open Pit, WRSF, and upslope areas.
- Runoff coefficients for the surface areas contributing to the water balance account for sublimation losses and reduction of yield due to ponding and absorption
- The process plant water balance accounts for freshwater make-up supply and water losses to the grit tailings and concentrate. All other water supply to the process plant is assumed to be sourced from the TMF.

18.10.2.3 Results

The water balance model indicates that the TMF will be in an annual water surplus during operations under annual average conditions. For a typical average year (Year 5) the following values were obtained:

- Sum of all TMF inflows: 2.6 Mm^3;
- Sum of al TMF losses: 0.5 Mm^3;
- Water accumulation for ice cap and operating pond: 0.6 Mm^3;
- Net Surplus within the TMF: 1.5 Mm^3; and
- Potential water discharge from the TMF: 0.9 Mm^3.

A large contributor to this surplus is water generated from the undiverted catchment area upslope of the TMF, and diversion of this catchment area is a potential opportunity to control water accumulation and the requirement for surplus water discharge from the TMF

18.10.2.4 Water Storage Requirement

During winter months, the ice cap on the TMF pond could be up to 2 m thick and hence that water will not be available for reclaim to the plant. It will therefore be necessary to have sufficient water storage available under the ice to supply the plant with make-up water during the winter period, which is assumed to be five months of the year. Although there may be ice cover for up to eight months of the year, the ice thickness is considerably less during the shoulder periods and the melted ice will contribute to the operating pond volume.

It is assumed that the tailings pond and runoff water will be frozen and unavailable for return to the plant for a five-month period; however, slurry water could be returned. The total plant demand deficit was estimated to be approximately 32,000 m^3/month. Therefore, the operating water storage requirement is assumed to be 160,000 m^3 for five months.

Based on the results from the preliminary water balance, the TMF will operate with surplus water for each year of the mine life, largely due to the large amount of undiverted catchment runoff upstream of the TMF. Apart from the 5% freshwater component that has been assumed in the water balance, the process plant can be supplied from the TMF year round and is not expected to require additional make-up water from other sources.

18.10.2.5 TMF Water Discharge

The water balance model indicates that the TMF will operate in a surplus throughout operations, and water will need to be released from the TMF each year to control water accumulation. This discharge will have to be done to a fresh water environment, and some form of water treatment prior to discharge will be required, most likely for treatment of total suspended solids (TSS). There may also be an opportunity to divert catchment area upslope of the TMF to reduce surplus water accumulation and the need for surplus water discharge.

18.10.2.6 Emergency Spillway

The TMF will be designed to safely manage and store the Inflow Design Flood above the maximum tailings storage level reached at the end of the mine life. Since the dam construction is planned in a single stage, prior to this there will be significantly more stormwater storage capacity in the TMF. An emergency spillway will be incorporated into the embankment abutment to pass storm water from the facility during operations and closure. A dam classification assessment will be conducted in future studies to determine the design storm event for sizing of the spillway.

18.10.2.7 Site Wide Water Management

The site water management plan does not currently include any diversions for the large catchment area upslope of the TMF, however given the water surplus condition, diversions could be used to manage a portion of the upslope catchment area while maintaining the required volume of water in the TMF. Contact water ponds downstream of the processing plant and associated infrastructure, the CH-6 Open Pit and the CH-6 WRSF will be operated as event ponds, and any water contained in them will be pumped to the TMF to provide containment of site contact water.

19 Market Studies and Contracts

19.1 Diamond Pricing and Market Studies

JDS has relied on WWW International Diamond Consultants (WWW) for diamond valuation, as reported in April 2018 (WWW 2018a, and WWW, 2018b). WWW are recognized international leaders in this field and are the valuers to the Federal Government of Canada for the Canadian diamond mines in the north; JDS accepts that it is reasonable to rely on the WWW valuations. JDS has selected diamond price escalation of 1.75% per annum after reviewing other diamond escalation estimates and are of the opinion that the 1.75% rate is a reasonable assumption for current market conditions.

19.1.1 Diamond Valuations

JDS is not able to apply quality control measures to the valuation process performed by WWW International. The reason for this is that diamond valuation is, at best, only partially analytical (in the way that a gold assay process can be termed analytical), as the diamonds are sieved and subjectively classified by colour, clarity, etc. The dollar per carat determinations for various stones, however, is ultimately governed by the valuator's price book. This part of the process is proprietary, governed by a given valuator's view of the marketplace and can vary from valuator to valuator, particularly for larger stones. Even in larger parcels, valuators must then 'model' or extrapolate values in the larger stone size classes, where there may be limited representative samples sizes. The methodology for modelling is also proprietary. These diamond valuation procedures do not lend themselves to quality control measures, which a QP could apply, as with a commercial assay laboratory. At every step, JDS is relying on the valuator's opinions of the diamond market and their subjective view of diamond values. JDS also relies on the valuator's models, which are heavily dependent on their view of the diamond market, their proprietary estimates of the likelihood of finding larger stones in the deposit because of sample-size support, and the perceived value of those larger stones.

The culmination of the process is the average prices for given zones, lobes or pipes. The heavy dependence of the process on economic market assessments, and the proprietary nature of the valuators' assumptions and methods, materially affects the quality of, and confidence in, the Mineral Resource estimate. In this way, the valuations used in the Mineral Resource assessments are markedly different than the concept of analytical mineral assays in, for instance, a precious metal project. The proprietary nature of the processes employed for valuations limit any quantitative assessment of the added risk to the Project. Other than reviewing the WWW reports for transcription errors in the transfer of the valuation figures into the database, no other data verification procedures can be applied. Diamond valuators are experts, but not Qualified Persons, and the Qualified Persons preparing the Mineral Resource estimates and assessing the reasonable prospects for economic extraction have had to completely rely on the WWW diamond values provided.

19.1.2 2018 Diamond Valuations

The diamond valuations for the Chidliak Project are outlined in Table 19-1 and described in the sections below. The base model US$/ct values were used in the economic modelling of the Project.

Table 19-1: Chidliak Project April 2018 Average Diamond Pricing

Pit	Unit	Low Model	Base Model	High Model	Spot (Actual) Valued
CH-6	US$/ct	131	**151**	191	164
CH-7	US$/ct	94	**114**	155	103

Source: WWW, 2018a & WWW, 2018b

19.1.2.1 CH-6

A diamond parcel of 1,117.09 ct from CH-6 was originally valued by WWW International Diamond Consultants Ltd. (WWW) in February 2014. Based on the 2014 valuation, the average value for 1,013.54 carats of diamond in and larger than the DTC 3 size category was US$213/ct and an base case average model diamond value of US$188/ct was reported (WWW, 2014). This valuation exercise was updated in February of 2016 and again in April 2018 based on the March 31, 2018 WWW price book. An updated modelled estimate of average diamond value was generated by WWW (2018a) based on the parcel valuation data by combining a value distribution model (model of average diamond value by size class) with a single SFD model representing the proportion of diamond (by weight) expected in each size class. This model yields a base case average model diamond value of US$151/ct, which was used in the economic model.

19.1.2.2 CH-7

A diamond parcel of 735.75 ct from CH-7 was valued by WWW International Diamond Consultants Ltd (WWW) in February 2016. The parcel was delivered and assessed as six sub-parcels derived from resource domains KIM-1 to KIM-5, inclusively, with diamonds derived from weathered KIM-2 material also assessed separately. The average diamond values for these smaller parcels (ranging from US$44/ct to US$306/ct) varied from US$73/ct to US$154/ct with an overall parcel average value of US$100/ct. A modelled estimate of average diamond value was generated by WWW (2016b) based on the parcel valuation data by combining a value distribution model (model of average diamond value by size class) with a single SFD model representing the proportion of diamond (by weight) expected in each size class. These models yield an estimated average base model diamond value of US$114/ct. This valuation exercise was updated in April 2018 based on the March 31, 2018 WWW price book, yielding the same base case average model diamond value for the parcel of US$114/ct.

19.2 Contracts

No contractual arrangements for transportation, port usage, or shipping exist at this time. Furthermore, no contractual arrangements for the sale of the diamonds into the marketplace exist at this time.

19.3 Royalties

The Project is not subject to any non-governmental royalties. The Nunavut mineral tax royalty has been considered a tax for the purposes of this study and is discussed in Section 23.

20 Environmental Studies, Permitting and Social or Community Impacts

20.1 Environmental Assessment for Mining Projects

One of Peregrine's highest priorities is to manage and mitigate the potential effects of the Project to the surrounding environment. Peregrine is committed to exploration and mining practices that are environmentally responsible and socially acceptable, and dedicated to creating and maintaining a safe environment for the land, its employees, and nearby communities.

The Nunavut Lands Claim Agreement (NLCA), signed in 1993, was the basis for creating Nunavut Territory in 1999. Through the NLCA, surface and subsurface rights for portions of the territorial land base were entrusted to the Inuit with the balance remaining with the Crown. The Designated Inuit Organization under the NLCA is Nunavut Tunngavik Incorporated (NTI), who retains administration of the subsurface mineral rights for Inuit Owned Land (IOL). Surface rights for IOL are vested from NTI to the Regional Inuit Associations (RIA). All other surface and subsurface rights in Nunavut are managed by the Crown through Indigenous and Northern Affairs Canada (INAC), except for the communities and municipalities which are located on Commissioners' lands and managed by the Government of Nunavut.

Five management boards were created within the NLCA, as listed in Table 20-1. These Institutes of Public Government include representatives of NTI, the Crown, and the Government of Nunavut, and are responsible for resource management in Nunavut.

Table 20-1: Nunavut Boards (Institutes of Public Government) and Associated Responsibilities

Board	Responsibilities under NLCA
Nunavut Wildlife Management Board	Wildlife Management
Nunavut Planning Commission	Land Use Planning and Cumulative Effects
Nunavut Impact Review Board	Environmental Assessment
Nunavut Water Board	Water Resource Management
Nunavut Surface Rights Tribunal	Dispute Resolution for Land and Water Resource Management Issues

Source: Nunavut Land Claim Agreement (1993)

The Nunavut Planning Commission is responsible for the development of a Land Use Plan for the Territory of Nunavut. The current Draft Nunavut Land Use Plan (DNLUP) released in June of 2016 recognizes the Chidliak Development Area as an area of High Mineral Potential and also illustrates a proposed transportation corridor to the Project. Upon approval all projects in the territory will need to conform to the Land Use Plan.

The permitting process for exploration and development of any mineral project is continuous and changes as the Project scope changes. As a project advances from exploration to mining, the activities take on a smaller geographic extent as work becomes more intensive. The permitting framework is set out in both the NLCA and the Nunavut Planning and Project Assessment Act (NUPPA).

The Chidliak Phase 1 Development Project area is entirely on Crown Land. At present, the Chidliak mineral claims fall on two categories of lands. Approximately 83% of the Project area is on Crown Land, governed by INAC with the remaining 17% located on Inuit Owned Lands (private lands), where the primary regulatory authority is the QIA.

A class "A" land use permit from INAC is required for exploration on Crown Lands. Peregrine holds land use permit N2018C0002, which authorizes four field camps and mechanical exploration (drilling, trenching etc.). The land use permit is paralleled by a class "B" water use and waste water disposal license, issued by the NWB.

A land use license from the QIA is required to access mineral claims located on IOLs. Peregrine holds two such land use licenses: one for the Chidliak claims (in areas outside the proposed Phase 1 development area) and another for contiguous portions of the Qilaq claims.

- INAC – Class "A" Land Use Permit – N2018C0002 – Surface Exploration and Campsites;
- QIA - Land Use License - Q13L1C005 – Surface Exploration – Chidliak IOL Claims;
- QIA – Land Use License – Q13L1C006 – Surface Exploration – Qilaq IOL Claims; and
- NWB – Water Use License – 2BE-CHI1823 – Water Use and Waste Water Disposal.

New and modified mining projects in Nunavut are subject to an EA and review prior to certification and issuance of permits to authorize construction and operations. The EA is an activity designed to ensure that potential adverse environmental, social, economic, health, and heritage effects, or potentially adverse effects on the local communities and Aboriginal peoples, are addressed before project approval. The two main stages in the EA process are the pre-application stage, when studies and consultations are undertaken, and the application review stage, when project details and effects on environment and communities are submitted as an Environmental Impact Statement (EIS) and reviewed. The scope, procedures, and methods of each assessment are tailored specifically to project circumstances.

The primary environmental review and approval process that applies to the Project is the territorial EA administered by the NIRB. Some federal regulatory requirements and processes that were applicable prior to the NLCA continue to apply in Nunavut. NUPPA came into force on June 9, 2015 and provides clarity to the review and approval process.

In general, an EA contains four main elements:

- Opportunities for all interested parties to identify issues and provide input;
- Technical studies of the relevant environmental, social, economic, heritage, and health effects of the proposed project;
- Identification of ways to prevent or minimize undesirable effects and enhance desirable effects; and
- Compiling the assessment findings and making recommendations about project acceptability.

All project proposals submitted to NIRB are screened. Based upon this screening, NIRB can make three recommendations:

- A review of the Project is not required;

- A review of the Project is required; or
- The Project should be modified or abandoned.

Small exploration level projects are generally only subject to a screening. Large projects such as mining project proposals are subject to a review. The first stage of the review process is to issue scoping and EIS guideline development. This stage identifies valued ecosystem components (VECs) and valued socio-economic components (VSECs) considered to have scientific, ecological, economic, social, cultural, archaeological, historical or other importance. These VECs and VSECs guide the collection of baseline data for the EIS. This is a consultative stage.

Stage two commences when sufficient baseline data has been collected to complete a draft EIS and is submitted to NIRB. A technical and administrative review of the data is conducted as well as a pre-hearing conference. NIRB reviews the work, notes deficiencies and issues a pre-hearing conference report to the proponent.

The third and final stage is the submission of the final EIS and the final hearing. At the conclusion of the hearing, NIRB recommends to the federal Minister of INAC that the Project either proceed or not proceed. The Minister must either accept or reject NIRBs recommendation. If the Project is to proceed and the Minister accepts the recommendation, then a project certificate is granted by NIRB.

A project certificate represents government approval in principle and allows the proponent to pursue the necessary regulatory authorizations required to construct and operate the Project.

20.2 Environmental Studies Background and History

Baseline environmental studies within the Chidliak Project began in 2009 and continued annually to 2016. In addition, several other environmental programs (e.g., stream flow, waterfowl, breeding birds, and fish and fish habitat) were undertaken over a one to three year period (Table 20-2).

The purpose of the environmental baseline work is to collect enough data to satisfy knowledge gaps and effectively design a baseline program for mine development.

Table 20-2: Summary of Baseline Environmental Programs Undertaken at Chidliak since 2009

Survey Type	2009	2010	2011	2012	2013	2014	2015	2016	2017
Water Quality	X	X	X	X	X	X	X		
Stream Flow	X	X	X						
Habitat Analysis	X			X					
Breeding Birds	X								
Waterfowl	X	X	X						
Raptor / Raptor Nest	X	X	X	X	X	X	X	X	
Aerial Caribou	X	X	X	X	X	X			
Aerial Carnivore	X	X	X	X	X	X			
Meteorological		X	X	X	X	X			
Camp Potable Water Quality		X	X	X	X	X	X		
Fish & Fish Habitat		X				X	X	X	
Ecological Land Classification						X		X	
Archaeology	X	X	X	X		X			
Wildlife Observation Logs		X	X	X	X	X	X	X	X

Source: Peregrine (2018)

20.3 Land Capability and Use

The area is located inland, 25 km west of Popham Bay, the nearest marine environment. The area has evolved slightly over the years to reflect Peregrine's potential exploration activities. The current study area for the baseline studies is approximately 3,083 km^2 and 550 to 920 m in elevation with higher land to the east, which is covered by glaciers. Approximately half of the study area drains north and east into Cumberland Sound, while the remainder drains west via the McKeand River and its tributaries. Peak run-off typically occurs in June as a result of snowmelt and subsequently declines from July to October. The climate of the area is typical of the eastern Arctic, being cold in the winter (-25°C to -45°C) and cool to mild in the summer (5°C to 10°C). Precipitation is generally low but snow is possible during all months. Lakes typically have ice until mid-June and freeze up begins in late September.

20.3.1 Archaeology

Peregrine commissioned archaeology surveys in 2009, 2010, 2011, 2012 and 2014. These surveys resulted in the identification and cataloguing of 81 archaeology sites within the Project area, which have been filed with the territorial government. Prior to the Peregrine commissioned surveys, only one archaeological site had been documented in the Project area.

20.3.2 Vegetation

Ecological Land Classification (ELC) is required for project planning and infrastructure layout, identification of sensitive or traditional use plants, and the mapping of special landscape features, such as eskers and cliff faces. For projects within Nunavut, and of similar scope to the Chidliak Project, such as Back River, Hackett River, and Mary River, ELC has been required by the individual project EIS guidelines, as issued by the NIRB. ELC provides a foundation for facilitating discussions of geomorphology, soils, permafrost, ecosystems, rare plants, country foods, traditional use plants, special landscape features, and wildlife habitat. Since ELC mapping supports many downstream activities and assessments, it is often completed early in the Project cycle, in order to reduce project risks. Risk depends on the type of interactions between the Project and the natural environment. The anticipation of such interactions and associated outcomes is critical to effective project planning and risk management. ELC allows for the identification of areas of high ecological function, which can then be avoided (where feasible) during the siting of project infrastructure.

20.3.3 Wildlife

Wildlife baseline studies and wildlife sightings logs were initiated in 2009 and have been in effect for each operational season to the present day. The Chidliak Project environmental baseline study area is centered over the Chidliak camps and main exploration areas, and currently covers an area of 3,083 km^2. The wildlife surveys and wildlife logs kept by camp personnel have indicated a consistently low density of caribou, carnivores, raptors and waterfowl across the study area and project. No large concentration of wildlife species or wildlife high-use areas have been documented to date on the Chidliak Project. Sensitive wildlife areas may imply both a conservative temporal and spatial restrictive-activity zone.

No calving areas have been recorded during wildlife surveys in the study area, and only one carnivore denning area has been documented. However, Peregrine is aware of the general importance of various area landforms (e.g. eskers and other glaciofluvial material, and coastal islands) and watercourses (e.g. McKeand River) to a variety of animals, ranging from caribou and wolves to raptors and seabirds, and limits disturbance to these habitats wherever possible.

20.3.4 Fisheries and Aquatic Resources

Aquatics surveys have been ongoing at four lakes (lentic sampling sites) within the study area and seven stream locations since 2009. A preliminary assessment of fish and fish habitat was also completed for select watercourses in the Project area and for Sunrise Camp Lake in 2010. In 2014, the geographic extent of aquatics studies was expanded to include three additional lakes and four sites along the McKeand River. The scope of the aquatics studies was also expanded to include analyses of primary and secondary producers in lakes and the McKeand River, and analyses of metals concentrations in fish tissues. A preliminary desktop study seeking to identify barriers to fish migration in the McKeand River was also performed.

The 2015 aquatics program repeated sampling in a matter consistent with the 2014 program. In addition, it also included the addition of gill netting in the McKeand River, and aquatic biota sampling (fish, periphyton and benthic invertebrates) at three additional stream sites, due to their proximity to potential future mine development (these sites were also initially assessed for fish and fish habitat as part of the 2010 program). If an updated proposed footprint is provided, any additional areas not previously identified near proposed infrastructure will be sampled.

20.3.4.1 Surface Freshwater Quality (including Sediment Quality)

The objective of the surface water quality sampling program is to determine baseline water quality conditions within the surface waters of the study area, by analyzing for routine parameters, nutrients, total metals, total organic carbon, and oil and grease. The laboratory analytical results indicated that all sampled parameters were within the Canadian Council of Ministers of the Environment for the Protection of Freshwater Aquatic Life (CCME FAL) guidelines at all sampling stations, except for total Aluminum. Twenty-eight surface water quality stations have been identified and sampled in previous years (Tetra Tech EBA 2014 & 2015b).

20.3.4.2 Hydrology and Bathymetric Surveys

In 2009, preliminary stream flow was measured at two locations downstream of kimberlite pipes CH-1 and CH-2, and at a single location downstream at the confluence of these streams. All sampling events are summarized with regional hydrology and climate data, based upon regional climate stations and regional run-off data.

In 2011, bathymetric data was collected from lakes that could be affected by the development of the Project.

20.4 Economic Impacts

The Chidliak Project is expected to provide economic benefits to local communities as a result of training, direct employment and indirect employment. The Project will also generate annual revenues associated with Federal Royalties and income tax for territorial and federal governments.

20.5 Regulatory Approvals

20.5.1 Anticipated Authorizations, Approvals and Permits

Significant EA and regulatory requirements that will be specific for a potential mining project at Chidliak are listed in Table 20-3. Additional ongoing permits will also be required for the life of the Project, such as land use permits and water licenses. Currently, Peregrine has active permits for each of these. Several federal acts apply to the Project, including the Fisheries Act and the Navigation Protection Act.

The organization that represents Inuit under the NLCA is the NTI, which has established the Mining Policy for Nunavut. This policy states that the development of Mineral Resources in Nunavut will be supported and promoted by NTI if there are significant long-term social and economic benefits for the Inuit of Nunavut, and if the development is consistent with protecting the eco-systemic integrity of the Nunavut Settlement Area (NSA).

Various pieces of federal legislation apply to mining projects in Nunavut. The agencies that will be involved in metal mining projects include INAC, Environment and Climate Change Canada, Fisheries and Oceans Canada (FOC), Health Canada, Natural Resources Canada, and Transport Canada. INAC regulates and manages the surface and subsurface Crown Lands in Nunavut. The Fisheries Act, administered by the FOC and Environment and Climate Change Canada, can play a substantial role in permitting mining projects in Nunavut. The requirements of FOC and Environment and Climate Change Canada (under the Metal Mining Effluent Regulations (MMER)) prohibit the harmful alteration or destruction of fish or fish habitat in order to obtain their respective authorizations.

The Navigation Protection Act, administered by Transport Canada, can play a substantial role in permitting all-weather roads in Nunavut where stream crossings are required.

Table 20-3: Major EA and Regulatory Anticipated Requirements Pertinent to a Potential Chidliak Mining Projects

Permit Process	Issuing / Lead Organization	Comments
Pre-environmental Assessment	Nunavut Research Institute	Permits for allowing research activities (baseline studies) in Nunavut
		Wildlife Research Permits
	Department of Environment, Government of Nunavut	
	Department of Culture and Heritage, Government of Nunavut	Archaeological Research Permits
	Fisheries and Oceans Canada	Fisheries Research Permits
	Nunavut Planning Commission	Conformity with Approved Regional Land Use Plans if required
	Community and Government Services, Government of Nunavut	Land Use License for Use of Commissioners Lands
EA	Nunavut Impact Review Board	NIRB Screening, Part 5 Review, Part 6 Panel Review
Post-environmental Assessment	Environment and Climate Change Canada	MMER Schedule 2 Listing
Permits and Licenses	Nunavut Water Board	Water License
	Indigenous and Northern Affairs Canada	Crown Land Surface Leases
		Crown Land Mineral Leases
		Crown Land Use Permits
		Crown Quarry Permits
		Certificate of Occupation for Crown Land
		License of Occupation for Road Alignments on Crown Land
	Fisheries and Oceans Canada	Fisheries Authorizations
Permits and Licenses (continued)	Transport Canada	Navigable Protection Act authorizations; authorization for filling or dewatering a navigable waterway
		Canada Shipping Act
		Arctic Waters Pollution Prevention Act
	Natural Resources Canada	Blasting Permits
		Explosives Magazine Permits
		Radio Licensing
	Environment and Climate Change Canada	Species at Risk Act
		Migratory Birds Act
		Section 36 of the Fisheries Act
		Metal Mining Effluent Regulations

Source: JDS / Peregrine Diamonds (2016)

21 Capital Cost Estimate

21.1 Capital Cost Summary

Estimated LOM capital costs total $521 M, consisting of the following distinct stages:

- Initial Capital Costs – includes all pre-production costs to develop the Phase 1 diamond resource to a 2,000 t/d operation. Initial capital costs total $464 M and are expended over a 36-month pre-production construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $35 M and are expended in operating years 1 through 13; and
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $21 M, and are primarily incurred in Year 14, with costs extending into Year 25 for ongoing monitoring activities.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors.

Table 21-1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q1 2018 Canadian dollars with no escalation.

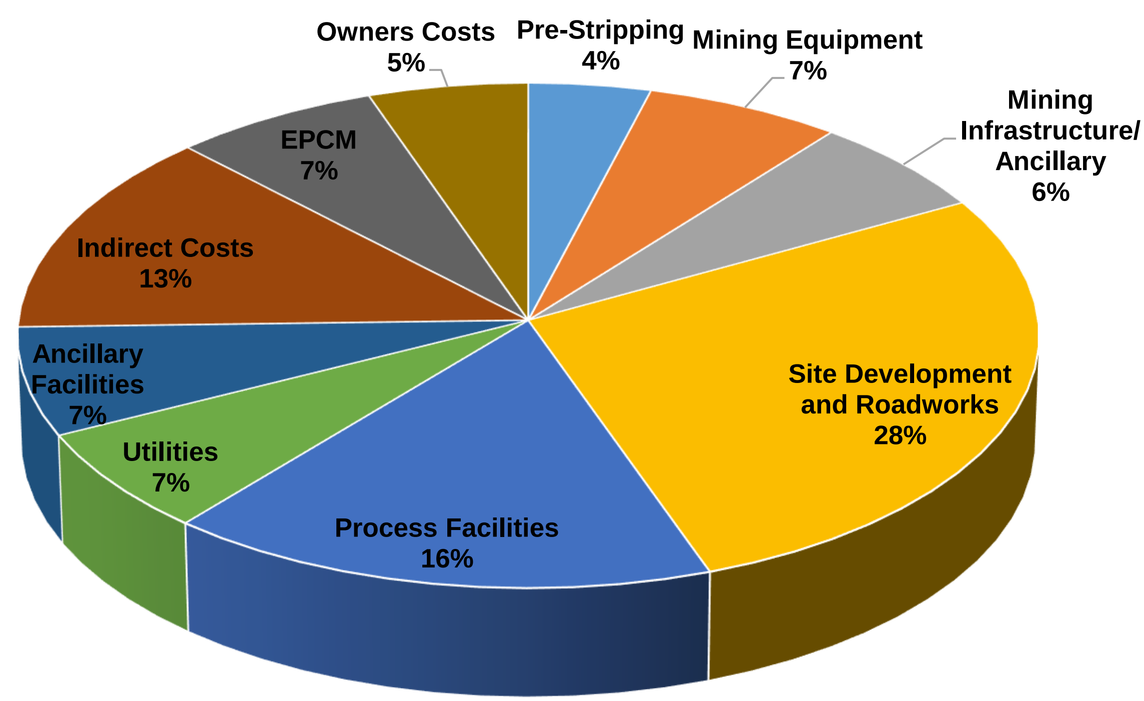
Table 21-1: Capital Cost Summary

Capital Costs	Pre-Production ($M)	Sustaining / Closure ($M)	Total ($M)
Pre-Stripping	16.8	-	16.8
Mining Equipment	27.0	6.9	33.9
Mining Infrastructure/Ancillary	25.8	2.5	28.3
Site Development and Roadworks	112.8	-	112.8
Process Facilities	67.6	22.3	89.9
Utilities	27.0	-	27.0
Ancillary Facilities	28.3	-	28.3
Indirect Costs	53.8	-	53.8
Engineering and Project Management	28.4	-	28.4
Owners Costs	22.0	-	22.0
Salvage Value	-	-	-
Closure Costs	-	16.1	16.1
Subtotal	**409.5**	**47.8**	**457.3**
Contingency	54.9	8.5	63.4
Total Capital Cost	**464.4**	**56.3**	**520.7**

Source: JDS (2018)

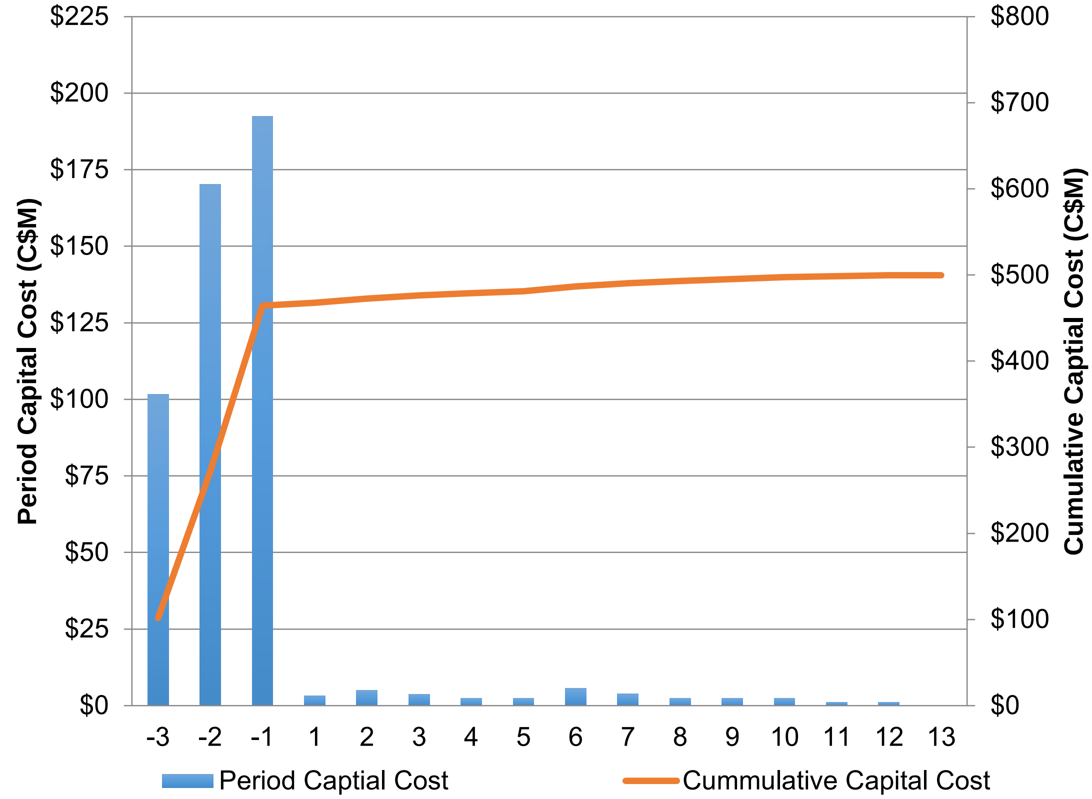
Figure 21-1 presents the capital cost distribution for the pre-production phase. The largest single scope element during development is the construction of the all-weather access road.

Figure 21-1: Distribution of Initial Capital Costs



All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21-2 presents an annual LOM capital cost profile (excluding closure years).

Figure 21-2: Capital Cost Profile (Closure Years not Shown)



21.2 Basis of Estimate

21.2.1 Key Estimate Assumptions

The following key assumptions were made during development of the capital cost estimates:

- The mine fleet will be purchased at the onset of the Project to support the all-weather access road construction and initial site development earthworks. It is assumed that these activities will be performed by a contractor using the Owners equipment;
- Materials for site development earthworks will be sourced from the CH-6 pit limits;
- Following site development earthworks, the mine fleet will be returned to the Owner and the Owners work forces will perform the balance of pre-stripping at CH-6; and
- All surface construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by contractors.

21.2.2 Key Estimate Parameters

The following key parameters apply to the capital cost estimates:

- **Estimate Class**: The capital cost estimates are considered Class 4 estimates (-20%/+30%). The overall project engineering definition is estimated to be 10%;
- **Estimate Base Date**: The base date of the estimate is Q1, 2018. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- **Units of Measure**: The International System of Units (SI) is used throughout the capital estimate; and
- **Currency**: All capital costs are expressed in Canadian Dollars (C$). Portions of the estimate were estimated in US Dollars (US$) and converted to Canadian Dollars at a rate of US$0.78:C$1.00.

21.2.3 Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model),
- Financing costs,
- Currency fluctuations,
- Lost time due to severe weather conditions beyond those expected in the region,
- Lost time due to force majeure,
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule,
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares,
- Any project sunk costs (studies, exploration programs, etc.),
- Costs arising from the establishment of Impact Benefit Agreements,
- Provincial sales tax,
- Closure bonding, and
- Escalation.

21.3 Mine Capital Cost Estimate

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers and in-house cost databases from similar mines in Northern Canada. Table 21-2 summarizes the mining capital cost estimate.

Table 21-2: Mining Capital Costs

Capital Costs	Pre-Production ($M)	Sustaining ($M)	Total ($M)
Pre-Stripping	16.8	-	16.8
Mining Equipment	27.0	6.9	33.9
Mining Infrastructure/Ancillary	25.8	2.5	28.3
Total Mining Capital Cost	**69.7**	**9.4**	**79.1**

Source: JDS (2018)

21.3.1 Open Pit Pre-Stripping

OP pre-stripping costs include all labour and consumables related to pre-production waste stripping from the CH-6 pit. Costs were assembled from first principles using the OP mining schedule as the basis. Database unit costs were applied to labour, equipment, and material requirements.

Stripping of CH-7 (during operations) is included as an operating cost within the mine cost model.

21.3.2 Open Pit Mining Equipment

OP mining equipment quantities were determined through buildup of mine plan quantities and associated equipment utilization requirements. Database unit prices were applied to the required quantities.

21.3.3 Open Pit Infrastructure / Ancillary

21.3.3.1 Open Pit Infrastructure

OP infrastructure includes fixed equipment and materials related to the OP mining process, such as dewatering pumps/piping, survey equipment, shop tools, lunchrooms, and ablution facilities. Cost allowances have been made for the various facilities based on JDS experience at operations in similar climates.

21.3.3.2 Site Services Operations

Site services operations costs include all labour and consumables related to the support of the site during construction. Database unit costs were applied to the determined labour, equipment, and material requirements. These costs are included as an operating expense during production (see Section 22).

21.3.3.3 Site Services Equipment

Support equipment includes all auxiliary equipment purchased by the Owner (light vehicles, cranes, maintenance vehicles, etc.). Support equipment quantities were defined through JDS experience at similar operations in Canada's Arctic region. Database unit prices were applied to the required quantities.

21.3.4 Open Pit Mobile Equipment

OP mining equipment quantities were determined through buildup of mine plan quantities and associated equipment utilization requirements. Database unit prices were applied to the required quantities. A limited fleet of open pit mobile equipment will be used for pre-development and site access construction.

21.4 Processing Cost Estimate

The processing plant cost estimate is based on a proven low-risk Canadian kimberlite flowsheet and includes all processing equipment and the structure. Costs from prior projects for similar northern kimberlite processing plants were used as the basis for the capital estimate. When metallurgical testing and bulk sampling have been undertaken on the CH-6 and CH-7 kimberlites, a project-specific cost estimate can be generated.

21.5 Infrastructure Cost Estimate

Infrastructure costs include site development, access roads, TMF, and site infrastructure including utilities and ancillary facilities. These cost estimates are primarily based on database or recently quoted costs, with factors applied for minor cost elements. Table 21-3 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21-3: Surface Construction Basis of Estimate

Commodity	Basis
Contractor Labour Rates	Database values based on similar Arctic projects
Access Roads	First principles estimate for direct costs, based on an average cross-section for quantity determination Database costs for bridges Factored contractor indirect costs
Bulk Earthworks, including the PFK Containment Facility	Model volumes from preliminary 3D grading model First principle unit rates for drilling, blasting, crushing, loading, hauling, and placement based on the utilization of the mining fleet for earthworks activities
Concrete	Factored quantities and database unit rates, based on similar Arctic projects
Structural Steel	Database factors applied against mechanical equipment costs
Pre-Engineered Buildings	Database unit rates ($/m^2) applied against the building sizes outlined in the general arrangements Database allowances for lighting, small power, electrical/control rooms, and fire detection
Modular Buildings & Warehouses	Database costs from similar Arctic projects for the mine dry, administration offices, mine maintenance building, and mine warehouse
Mechanical Equipment	A combination of quoted and database costs from similar projects Database factor applied against mechanical equipment costs for installation
Piping	Database factors applied against mechanical equipment costs
Electrical and Instrumentation	Database factors applied against mechanical equipment costs

21.5.1 Surface Construction Sustaining Capital

Sustaining capital costs of 3% of the total direct cost of the diamond processing plant are included annually (C$2.0 M/a) for major equipment overhauls and minor capital projects during the LOM. This was reduced to C$1.0 M/a for each of Years 11 and 12 and eliminated for Year 13.

Capital replacements and overhauls are not expected for any of the ancillary buildings or utilities (such as the power generation plant) due to the relatively short mine life. Operating costs are included for the maintenance of these facilities and equipment.

21.6 Indirect Cost Estimate

Indirect costs are those that are not directly accountable to a specific cost object. Table 21-4 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-4: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule
Site Wide Crane Support	Time based cost allowance for the provision of shared crane services to support construction activities
Construction Phase Power Generation	Allowances for power generation to support operation of ancillary facilities during construction, plus an allowance for auxiliary construction power
Temporary Facilities & Utilities	Allowance for the rental of construction offices and ablution facilities to support construction
Contractor Indirect Costs	Factored allowance (1.5%) of direct costs for contractor mobilization/demobilization (exclusive of freight costs) Factored allowance (1.0%) of direct costs for contractor facilities and auxiliary expenses
Logistics & Freight	Allowance per year per kilometre for main access road maintenance Preliminary freight quantity estimate, based on available engineering information Interpolated rates for freight delivery (sea and ground), based on similar projects Interpolated rates for passenger transportation from Ottawa to site (combination flights & bussing) applied against a preliminary man-power schedule Allowance per month for air freight to Iqaluit for periods outside of the sealift season
Start-up and Commissioning	Bulk allowance (6,000 hrs) for pre-operational testing & start-up support Factored allowance (2.5%) of equipment costs for capital & commissioning spare parts Bulk allowance (C$500 k) for first fills, including glycol and ferrosilicon Factored allowance (2%) for the provision of vendor services for commissioning support
Detailed Engineering	Factored allowance (7.5%) of direct costs for engineering and procurement support activities
Project & Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration Database unit (hourly) rates

21.7 Owners Cost Estimate

Owner’s costs are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- **Pre-production processing**: Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production; and
- **Pre-production general & administration**: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production.

21.8 Closure Cost Estimate

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for a surface mine in Canada's Arctic. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF;
- Access road closure;
- Re-vegetation and seeding allowances; and
- Ongoing site monitoring.

The following assumptions were incorporated into the closure cost estimate:

- No salvage value was assigned to any fixed or mobile equipment;
- Contractor labour and equipment was assumed at industry-standard rental rates; and
- The permanent camp was assumed to be used to house the reclamation and demolition crews as long as possible. A self-contained temporary camp was assumed to be used for final decommissioning, which includes removal of the main camp.

These assumptions are typical to the industry for the estimation of closure costs.

The majority (77%) of closure costs are incurred immediately following completion of operations (Year 11), as shown in Table 21-5. Monitoring activities are anticipated to extend to Year 20. For the purpose of the economic model, all costs occurring beyond Year 20 are expressed as a discounted (5% per annum) lump sum in Year 20.

Table 21-5: Closure Estimate Summary

Item	Estimated Cost ($M)
Surface Demolition & Reshaping	5.6
Waste Removal	1.6
Tailing Storage Facility Closure	2.0
Re-vegetation	0.3
Post Closure Site Monitoring & Maintenance	3.2
Road Closure	0.5
Indirect Costs	2.0
Owner's Costs	0.9
Closure Cost	**16.1**
Contingency (at 30%)	4.8
Total Closure Cost	**20.9**

Source: JDS (2018)

21.9 Contingency

A contingency of 15% was applied to the LOM capital costs of the Project excluding Mining Equipment, Pre-Stripping and Closure. Contingency for Closure costs was increased to 30% to reflect a lower level of detail and confidence in the estimate. LOM project contingency totals $63.4 M.

22 Operating Cost Estimate

22.1 Operating Cost Summary

The operating cost estimate (OPEX) for the Chidliak Project has been prepared incorporating both off-site and on-site infrastructure as related to the mine plan and processing schedule. The operating cost estimate is broken into five major sections:

- Mining;
- Processing;
- Freight / Logistics;
- Site Services; and
- General & Administrative.

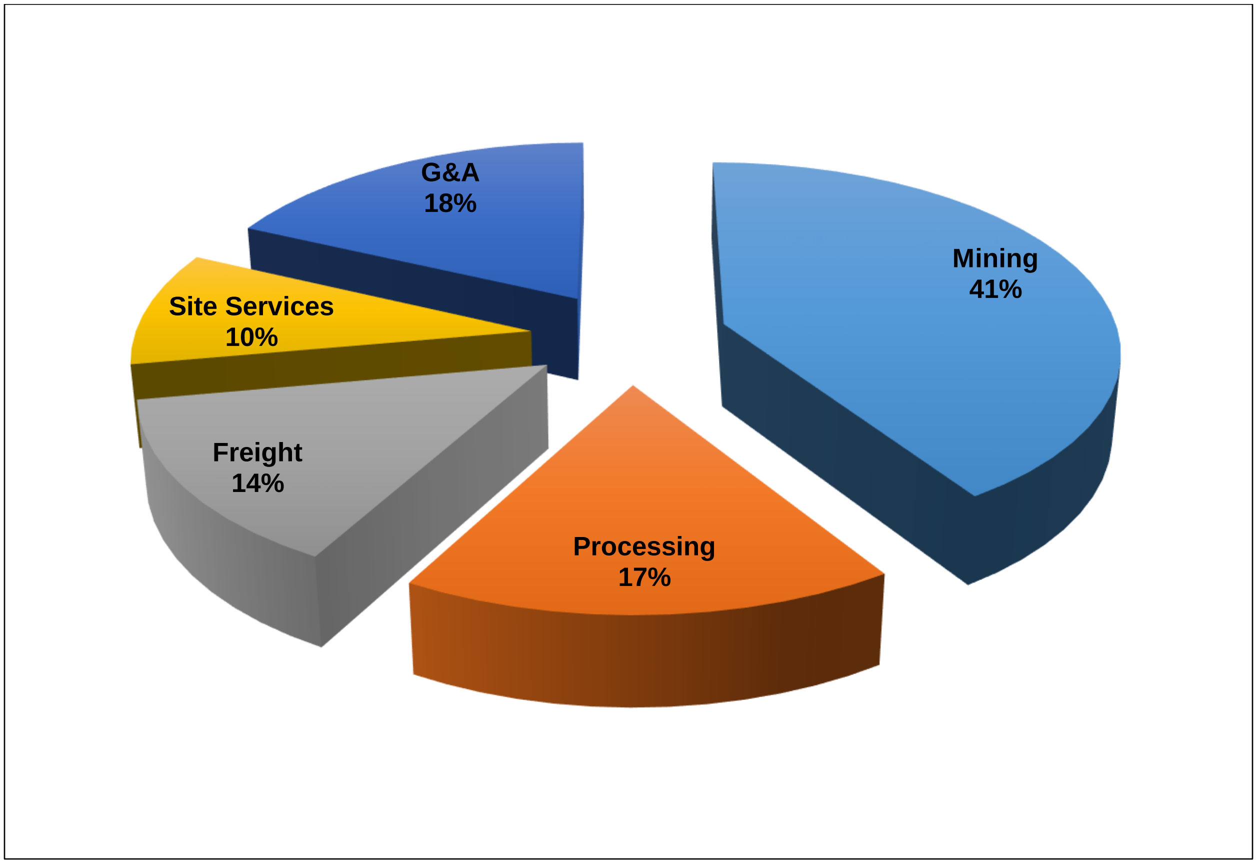
The total operating unit cost is estimated to be $104.8/t processed. Average annual operating costs over the LOM will be $76.4 M and are summarized in Table 22-1. Figure 22-1 show the breakdown and distribution of the LOM operating costs by category. Table 22-2 shows the summary of personnel considered in the operating costs buildup.

Table 22-1: Estimated Operating Costs

Operating Cost	$/t processed	$/ct	LOM $M
Mining	42.8	24.4	406
Processing	17.9	10.2	170
Freight / Logistics	14.9	8.5	141
Site Services	10.2	5.8	97
G&A	18.9	10.8	179
Total	**104.8**	**59.6**	**994**

Source: JDS (2018)

Figure 22-1: Operating Cost Breakdown



Source: JDS (2018)

Table 22-2: Summary of Personnel

Area	Quantity
Mining	
Mining Operations	82
Mine Maintenance	34
Technical Services	18
Subtotal Mining Personnel	**134**
Process Plant	
Process Operations (including technical services)	45
Process Maintenance	17
Subtotal Process Plant Personnel	**62**
G&A	
General Management	2
Human Resources	3
IT & Communications	1
Administration	11
Health & Safety	5
Environmental	5
Security	9
Camp Support	41
Subtotal General & Administration	**77**
Site Services	
Subtotal Site Services	**26**
Total	
Total Personnel - All Areas	**299**

Source: JDS (2018)

22.2 Basis of Estimate

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined proven project execution strategies. The estimate was developed using first principles and applying direct applicable project experience, and avoiding the use of general industry factors. The operating cost is based on Owner owned and operated mining/services fleets, and minimal use of permanent contractors except where value is provided through expertise and/or packages efficiencies / skills.

Virtually all of the estimate inputs were derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

Operating costs in this section of the report include mining, processing, freight / logistics, site services, and general and administrative (G&A) costs. Mine operating costs incurred during the construction phase (pre-production Year -1) are capitalized and form part of the capital cost estimate as Owners' costs. Treatment and refining charges, and non-governmental royalties are discussed in Section 19.

Operating costs are presented in Q1 2018 Canadian dollars on a calendar year basis. No escalation or inflation is included. The fuel and power assumptions are shown in Table 22-3.

Table 22-3: Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.232
Overall power consumption (all facilities)	kWh/t processed	20
Diesel cost (delivered)	$/litre	0.913

Some of the costs incurred during the pre-production period relate to the costs to purchase items such as consumables required for the following year of production. The timing of these costs has been accounted for in the economic analysis as working capital.

22.3 Mine Operating Cost Estimate

Costs for OP mining activities for the Chidliak Project, assumed to be undertaken by an owner-operated fleet, were built up from first principles, as well as JDS experience of similar-sized OP operations and local conditions. OP mining costs for both mineralized and waste material take into account variations in haulage profiles and equipment selection. Equipment efficiency was estimated based on Chidliak conditions (e.g. haul routes for each phase). Local labour rates and diesel fuel pricing estimates were utilized for estimation purposes. The OP mining costs encompass pit and dump operations, road maintenance, and mine supervision and technical services cost.

As shown in Table 22-4, the average mine OPEX for the LOM plan was estimated to be $5.0/t material mined or $42.8/t plant feed, for pit and dump operations, road maintenance, mine supervision, and technical services.

Table 22-4: Mine Average OPEX Estimate by Function*

Mining Operating Cost Category	Unit Cost Estimate ($/t mined)	Unit Cost Estimate ($/t processed)
Drill and Blast	0.91	7.73
Load & Haul	2.91	24.77
Mine General	0.36	3.03
Mine Maintenance	0.59	5.06
Technical Services	0.26	2.24
Total	**5.03**	**42.83**

*Does not include pre-production tonnes mined
Source: JDS (2018)

22.4 Processing Operating Cost Estimate

The Chidliak Project process plant OPEX for the LOM plan totals $17.95 /t processed, as shown in Table 22-5.

Table 22-5: Processing Average OPEX Estimate by Area

Process Plant Operating Cost	Unit Cost Estimate ($/t processed)
Process Operations, including Security, Labour	7.22
Maintenance Labour	2.99
Power	3.25
Process Consumables	1.09
Maintenance Parts	2.23
Sorting Costs	1.17
Total	**17.95**

22.5 Freight and Logistics Operating Cost Estimate

Operating costs for freight and logistics were estimated based on annual consumable quantities for mining and processing and cost estimates provided by local or experienced contractors. Freight and logistics costs encompass the following:

- Passenger movements,
- Charter flights from Ottawa to Iqaluit,
- Charter bussing from Iqaluit to the Chidliak site,
- Airfreight,
- Cargo movement,
- Freight to Montreal port,
- Sea freight,
- Offloading & storage in Iqaluit,
- Transport to site,
- Sea container rental and backhauls,
- Contract services, and
- Main access road maintenance.

The total freight and logistics OPEX for the LOM plan was estimated to be $141 M or $14.86/t processed, as shown in Table 22-6.

Table 22-6: Freight and Logistics Average OPEX Estimate by Area

Freight / Logistics Operating Cost Category	Unit Cost Estimate ($/t processed)
Passenger movements	5.25
Airfreight	1.07
Cargo Movement	7.47
Contract Services	1.08
Total	**14.86**

22.6 Site Services Operating Cost Estimate

Site services operating costs include on-site activities, site services equipment and labour, and infrastructure operations and maintenance costs. The summary of site services costs are shown in Table 22-7.

Table 22-7: Site Services Average OPEX Estimate by Area

Site Services Operating Cost Category	Unit Cost Estimate ($/t processed)
Maintenance	0.72
Fuel	1.05
Labour	6.36
Power	1.39
Material	0.71
Total	**10.23**

22.7 General and Administration Operating Cost Estimate

General and administrative costs comprise the following categories:

- Labour;
- On-site items as such camp catering, health and safety, environmental, human resources, legal, external consulting, communications and office supplies; and
- a satellite office.

The total G&A unit operating cost is estimated at $18.92/t of plant feed processed. Table 22-8 summarizes the annual G&A operating costs.

Table 22-8: G&A Average OPEX Estimate by Area

G&A Operating Cost Category	Unit Cost Estimate ($/t processed)
Labour	7.79
Items – on-site	11.05
Satellite office	0.09
Total	**18.92**

23 Economic Analysis

23.1 Summary of Results

The economic results for the Project are shown in Table 23-1. The Project meets the criteria usually associated with financeable projects: a quick payback period of 2.0 years, a high internal rate of return (IRR) of 31% and a net present value ($669 M) greater than the pre-production CAPEX of $464 M.

Table 23-1: Summary of Economic Analysis Results

	Unit	Value
Mine Life	Years	13
Resource Mined	Mt	9.5
Average Head Grade	ct/t	1.79
Carats Recovered	LOM Mct	16.7
Net Revenue (Net of Royalties)	LOM C$M	3,515.8
Operating Costs	LOM C$M	993.5
Total Pre-Production Capital	C$M	464.4
Total Sustaining & Closure Capital	C$M	56.3
Total Capital	C$M	520.7
Pre-Tax Cash Flow	LOM C$M	2,001.5
Taxes	C$M	669.2
After-Tax Cash Flow	C$M	1,332.3
Pre-Tax NPV$_{7.5\%}$ Discount	**C$M**	**1,052.2**
Pre-Tax IRR	**%**	**38.8**
Pre-Tax Payback	**Years**	**1.9**
After-Tax NPV$_{7.5\%}$ Discount	**C$M**	**668.7**
After-Tax IRR	**%**	**31.2**
After-Tax Payback	**Years**	**2.0**

23.2 Basis of Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in diamond grade/prices, foreign exchange (FX) rate, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and 22 of this report (presented in 2018 dollars). The economic analysis has been run with no inflation (constant dollar basis).

The plant head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

It must be noted that this PEA is preliminary in nature and includes the use of Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized.

23.3 Assumptions

Table 23-2 summarizes parameters and assumptions pertinent to the 13-year LOM plan that were used in the economic analysis.

Table 23-2: LOM Plan Summary

Parameter	Unit	Value
Mine Life	Years	13
Plant Feed Material	Mt	9.5
Total Waste	Mt	75.1
Total Plant Feed Material plus Waste Mined	Mt	84.6
Strip Ratio (waste tonnes : processed tonnes)	t:t	7.9
Throughput Rate	t/d	2,000
Average Head Grade	ct/t	1.79
Carats Recovered	LOM Mct	16.7
Average Annual Diamond Production	Mct/a	1.3

Other economic factors include the following:

- Discount rate of 7.5%,
- Closure cost of $20.9 M was included,
- Nominal Q1 2018 dollars,
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment,
- Working capital calculated as approximately five months of operating costs (mining, processing, freight, site services and G&A) in Year 1,
- All costs and time prior to construction decision considered sunk,

- Results are presented on 100% ownership basis, and
- No management fees or financing costs (equity fund-raising was assumed).

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Table 23-3 outlines the diamond prices and US$:C$ exchange rate assumptions used in the economic analysis.

Diamond prices for both CH-6 and CH-7 deposits used in the study were based on 31 March 2018 price-book, received from WWW International Diamond Consultants and have been escalated annually from 2018 at a rate of 1.75%.

The reader is cautioned that the exchange rate in this study is only an estimate based on recent historical performance.

Table 23-3: Diamond Valuation & Foreign Exchange Rates used in Economic Analysis

Parameter	Unit	Value	Source
Base CH-6 Diamond Valuation	US$/ct	151	WWW (2018a)
Base CH-7 Diamond Valuation	US$/ct	114	WWW (2018b)
Diamond Price Escalation	%	1.75	JDS
Exchange Rate	US$:C$	0.78	JDS

Mine revenue is derived from the sale of diamonds into the international marketplace. No contractual arrangements for rough diamond sales exist at this time. Details regarding the terms used for the economic analysis can be found in the Market Studies (Section 19) of this report.

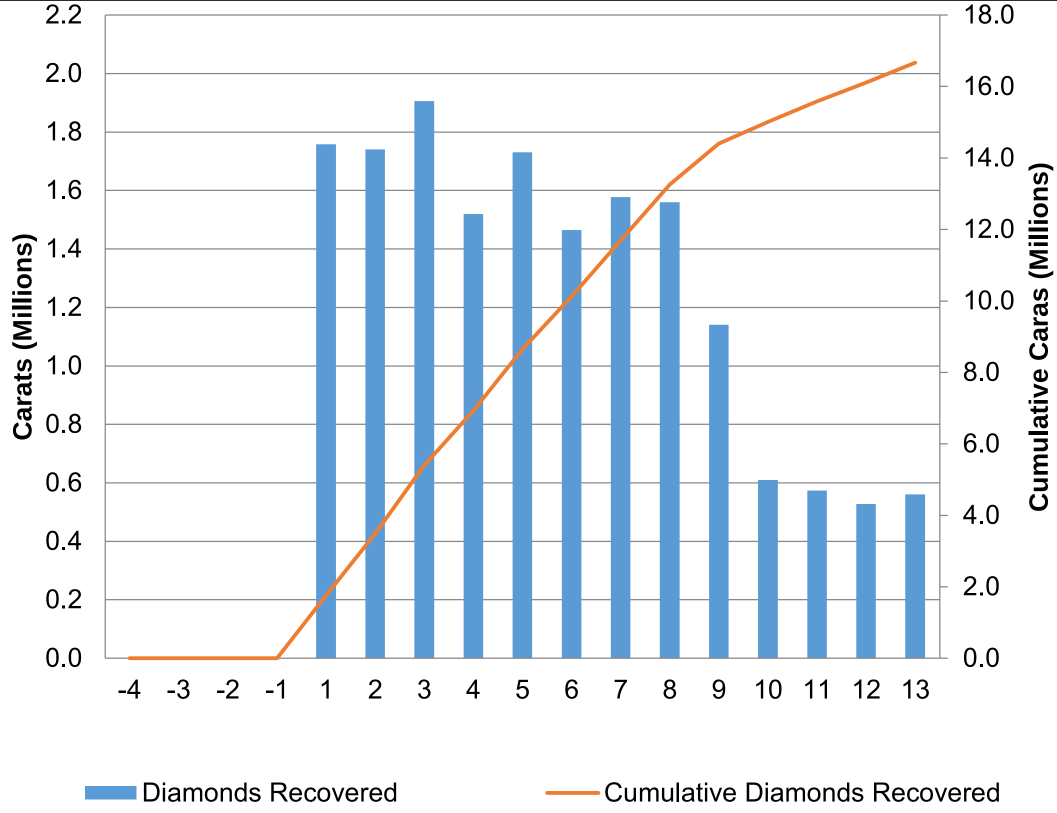
Table 23-4 indicates the Net Revenue (NR) parameters that were used in the economic analysis.

Table 23-4: Net Revenue Parameters

Parameter	Unit	Value
Mine Operating Days	d/a	365
Diamond Recovery from Process Plant	%	98
Selling Cost	%	4
	US$/ct	6.85

Figure 23-1 shows the distribution of diamonds recovered during the mine life. A total of 16.7 Mct will be recovered over the LOM, approximately 1.3 Mct/a. The LOM net revenue, including a 4% selling cost is C$3,516 M before OPEX, CAPEX and taxes.

Figure 23-1: Diamonds Recovered over LOM



23.4 Taxes

The Project has been evaluated on an after-tax basis in order to provide a more indicative, but still approximate, value of the potential project economics. A tax model was prepared by a specialized mining tax accountant, Wentworth Taylor, with applicable Nunavut mineral tax regime. The tax model contains the following assumptions:

- 15% federal income tax rate;
- 12% Nunavut territorial income tax rate;
- Variable 5% to 14% Nunavut mineral tax; and
- Total taxes for the LOM of $669.2 M.

The economic analysis does not incorporate any non-governmental royalties.

23.5 Results

At this preliminary stage, the Project is economically viable with an after-tax IRR of 31.2% and a net present value using a 7.5% discount rate of $668.7 M, using the diamond prices and exchange rates described in Section 19.

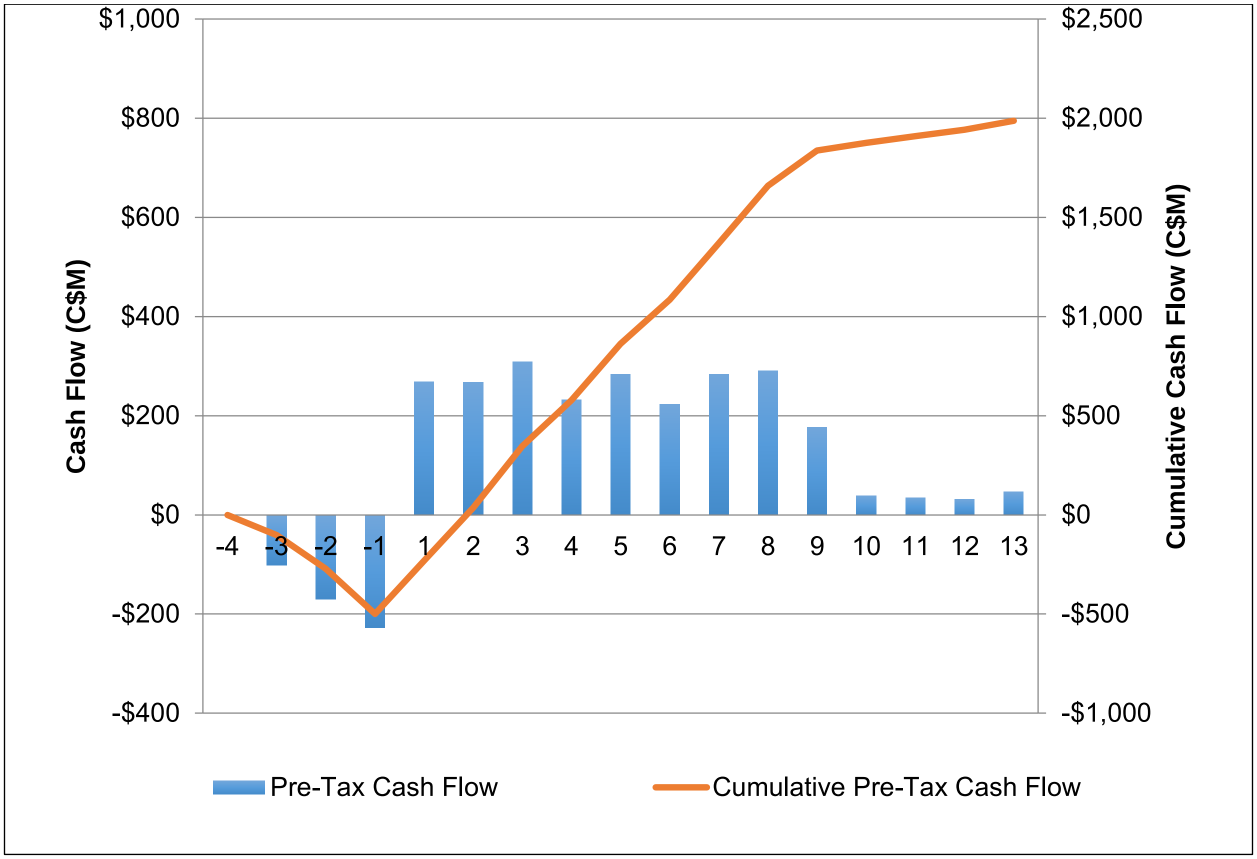
Table 23-5 summarizes the economic results. Figure 23-2 shows the pre-tax projected cash flows for the Project. The cash flow model is shown as Table 23-6.

Table 23-5: Economic Analysis Results

	Unit	Value
CH-6 Base Model Diamond Value	US$/ct	151
CH-7 Base Model Diamond Value	US$/ct	114
Annual Diamond Price Escalation (from 2016)	%/a	1.75
F/X Rate	US$:C$	0.78
Mine Life	Years	13
Resource Mined	Mt	9.5
Throughput Rate	t/d	2,000
Average Head Grade	ct/t	1.79
Carats Recovered	LOM Mct	16.7
	Mct/a	1.3
Net Revenue (Net of Royalties)	LOM C$M	3,515.8
Operating Costs	LOM C$M	993.5
	$/recovered ct	59.6
	$/t processed	104.8
Pre-Production Capital	C$M	409.5
Pre-Production Contingency	C$M	54.9
Total Pre-Production Capital	C$M	464.4
Sustaining & Closure Capital	C$M	47.8
Sustaining & Closure Contingency	C$M	8.5
Total Sustaining & Closure Capital	C$M	56.3
Total Capital	C$M	520.7
Working Capital	C$M	35.0
Pre-Tax Cash Flow	LOM C$M	2,001.5
	C$M/a	154.0
Taxes	C$M	669.2
After-Tax Cash Flow	C$M	1332.3
	C$M/a	102.5
Economic Results		
Pre-Tax $NPV_{7.5\%}$ Discount	**C$M**	**1,052.3**
Pre-Tax IRR	**%**	**38.8**
Pre-Tax Payback	**Years**	**1.9**
After-Tax $NPV_{7.5\%}$ Discount	**C$M**	**668.7**
After-Tax IRR	**%**	**31.2**
After-Tax Payback	**Years**	**2.0**

Source: JDS (2018)

Figure 23-2: Annual Pre-Tax Cash Flow



The annual cash flow model is shown in Table 23-6.

Table 23-6: Cash Flow Model

Item	Unit	Life-of-Mine Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-20
METAL PRICE AND EXCHANGE RATE																
CH-6 Carat Price	US$/ct	175.6	156.3	159.1	161.9	164.7	167.6	170.5	173.5	176.5	179.6	182.7	185.9	189.2	192.5	199.3
CH-7 Carat Price	US$/ct	148.3	118.0	120.1	122.2	124.3	126.5	128.7	131.0	133.3	135.6	138.0	140.4	142.8	145.3	150.5
Escalation	%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	1.8%	2.5%
F/X Rate	US$:C$	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
PRODUCTION SCHEDULE																
CH-6 Resource Mined	Mt	6.2	-	-	0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.3	-	-
CH-7 Resource Mined	Mt	3.3	-	-	-	-	-	-	-	-	-	-	-	0.4	0.7	2.2
Total Resource Mined	**Mt**	9.5	-	-	0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	2.2
Waste Mined	Mt	75.1	-	-	3.7	9.7	9.5	9.6	9.6	8.9	10.1	1.9	0.7	3.4	3.0	5.1
Total Mined	**Mt**	84.6	-	-	3.8	10.3	10.3	10.3	10.3	9.6	10.8	2.6	1.4	4.1	3.8	7.2
Strip Ratio	t:t	7.9	-	-	33.3	15.6	13.1	13.1	13.1	12.2	13.9	2.6	1.0	4.6	4.2	2.3
CH-6 Average Grade	ct/t	2.33	-	-	2.57	2.44	2.43	2.67	2.12	2.42	2.05	2.21	2.18	2.50	-	-
CH-7 Average Grade	ct/t	0.80	-	-	-	-	-	-	-	-	-	-	-	0.81	0.85	0.78
Total Average Grade	ct/t	1.79	-	-	2.57	2.44	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.78
Rehandle	Mt	0.1	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-
MILL SCHEDULE																
Resource Milled	Mt	9.5				0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	2.2
Grade	ct/t	1.79				2.46	2.43	2.67	2.12	2.42	2.05	2.21	2.18	1.59	0.85	0.78
Contained Carats	Mct	17.0				1.8	1.8	1.9	1.6	1.8	1.5	1.6	1.6	1.2	0.6	1.7
CH-6 Contained Carats	*Mct*	14.4				1.8	1.8	1.9	1.6	1.8	1.5	1.6	1.6	0.8	-	-
CH-7 Contained Carats	*Mct*	2.6				-	-	-	-	-	-	-	-	0.3	0.6	1.7
Recovery	%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%
	Mct	16.7	-	-	-	1.8	1.7	1.9	1.5	1.7	1.5	1.6	1.6	1.1	0.6	1.7
SALES AND NSR																
Gross Value	C$M	3,662.2	-	-	-	371.2	373.8	416.6	338.0	391.7	337.4	369.6	371.8	258.1	113.6	320.5
Selling Cost	% of Gross Value	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%	4%
	C$M	146.5	-	-	-	14.8	15.0	16.7	13.5	15.7	13.5	14.8	14.9	10.3	4.5	12.8
Royalties	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Revenues (After Royalties)	**C$M**	3,515.8	-	-	-	356.4	358.8	400.0	324.5	376.0	323.9	354.8	356.9	247.8	109.0	307.7
OPEX																
OP Mining	C$/tonne processed	42.83				51.86	55.31	56.64	60.31	60.87	68.50	29.75	25.75	32.52	32.34	27.55
	C$M	406.1				37.9	40.4	41.3	44.1	44.4	50.0	21.7	18.8	23.8	23.6	60.1
Processing	C$/tonne processed	17.95				18.38	18.36	18.52	18.16	18.36	18.11	18.21	18.20	17.81	17.33	17.28
	C$M	170.2				13.4	13.4	13.5	13.3	13.4	13.2	13.3	13.3	13.0	12.7	37.7

Table 23-6: Cash Flow Model (cont.)

Item	Unit	Life-of-Mine Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-20
Freight/Logistics	C$/tonne processed	14.86				16.41	14.71	14.76	14.69	14.73	14.71	14.71	14.71	14.70	14.71	14.77
	C$M	140.9				12.0	10.8	10.8	10.8	10.7	10.7	10.7	10.7	10.7	10.7	32.2
Site Services	C$/tonne processed	10.23				10.22	10.21	10.24	10.19	10.24	10.22	10.22	10.22	10.22	10.22	10.27
	C$M	97.0				7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	22.4
G&A	C$/tonne processed	18.91				19.14	19.24	19.29	19.21	18.78	18.75	18.75	18.75	18.74	18.75	18.83
	C$M	179.3				14.0	14.1	14.1	14.1	13.7	13.7	13.7	13.7	13.7	13.7	41.1
Total Opex	**C$M**	993.5				84.7	86.2	87.1	89.7	89.6	95.1	66.9	64.0	68.7	68.2	193.4
	C$/payable ct	59.60				48.17	49.52	45.68	59.04	51.79	64.92	42.41	41.02	60.19	111.81	116.42
Net Operating Cashflow	**C$M**	2,522.2				271.7	272.7	312.9	234.8	286.4	228.7	287.9	292.9	179.1	40.9	114.3
CAPEX																
Pre-Stripping	C$M	16.8	-	-	16.8	-	-	-	-	-	-	-	-	-	-	-
Mining Equipment	C$M	33.9	27.0	-	-	-	2.7	1.4	-	-	1.8	1.0	-	-	-	-
Mining Infrastructure/Ancillary	C$M	28.3	12.1	6.7	7.0	0.8	-	-	-	-	1.3	0.4	-	-	-	-
Site Development and Roadworks	C$M	112.8	27.4	74.7	10.6	-	-	-	-	-	-	-	-	-	-	-
Process Facilities	C$M	89.9	4.7	17.8	45.2	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Utilities	C$M	27.0	4.6	11.9	10.4	-	-	-	-	-	-	-	-	-	-	-
Ancillary Facilities	C$M	28.3	8.6	3.6	16.2	-	-	-	-	-	-	-	-	-	-	-
Indirect Costs	C$M	53.8	1.5	21.6	30.7	-	-	-	-	-	-	-	-	-	-	-
Engineering and Project Management	C$M	28.4	5.7	8.9	13.8	-	-	-	-	-	-	-	-	-	-	-
Owners Costs	C$M	22.0	0.4	2.7	18.9	-	-	-	-	-	-	-	-	-	-	-
Closure Costs	C$M	16.1														16.1
Subtotal	**C$M**	457.3	92.0	148.0	169.6	2.9	4.7	3.4	2.0	2.0	5.1	3.5	2.0	2.0	2.0	18.1
Contingency	C$M	63.4	9.7	22.2	22.9	0.4	0.3	0.3	0.3	0.3	0.5	0.4	0.3	0.3	0.3	5.1
CAPEX incl. Contingency	**C$M**	520.7	101.7	170.2	192.5	3.3	5.0	3.7	2.3	2.3	5.6	3.8	2.3	2.3	2.3	23.2
Pre-Production	*C$M*	464.4	101.7	170.2	192.5	-	-	-	-	-	-	-	-	-	-	-
Sustaining & Closure	*C$M*	56.3	-	-	-	3.3	5.0	3.7	2.3	2.3	5.6	3.8	2.3	2.3	2.3	23.2
WORKING CAPITAL																
Working Capital	C$M	0.0	-	-	35.0	-	-	-	-	-	-	-	-	-	-	(35.0)
TAXES																
Income Taxes	C$M	433.7	-	-	-	-	46.2	56.6	41.8	57.2	46.2	61.7	64.2	38.7	2.2	19.0
Nunavut Mineral Tax	C$M	235.4	-	-	-	-	0.3	39.4	29.4	36.1	28.1	36.1	36.9	21.5	0.6	6.9
Total Taxes	**C$M**	669.2	-	-	-	-	46.5	96.0	71.2	93.3	74.3	97.8	101.1	60.2	2.8	25.9
CASH FLOWS																
Pre-Tax																
Net Cashflow	**C$M**	2,001.5	(101.7)	(170.2)	(227.5)	268.4	267.7	309.2	232.4	284.0	223.2	284.1	290.6	176.8	38.6	126.1
Cumulative Net Cashflow	C$M		(101.7)	(271.9)	(499.4)	(231.0)	36.7	345.8	578.2	862.3	1,085.4	1,369.5	1,660.1	1,836.9	1,875.5	2,303.9

Table 23-6: Cash Flow Model (cont.)

Item	Unit	Life-of-Mine Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11-20
Net Cashflow	**US$M**	1,561.2	(79.3)	(132.7)	(177.5)	209.3	208.8	241.1	181.3	221.5	174.1	221.6	226.7	137.9	30.1	98.3
Cumulative Net Cashflow	US$M		(79.3)	(212.1)	(389.5)	(180.2)	28.6	269.7	451.0	672.6	846.6	1,068.2	1,294.9	1,432.8	1,462.9	1,797.0
After-Tax																
Net Cashflow	**C$M**	1,332.3	(101.7)	(170.2)	(227.5)	268.4	221.2	213.2	161.2	190.7	148.9	186.3	189.5	116.6	35.8	100.1
Cumulative Net Cash Flow	C$M		(101.7)	(271.9)	(499.4)	(231.0)	(9.8)	203.3	364.5	555.2	704.1	890.4	1,079.9	1,196.4	1,232.2	1,525.5
Net Cashflow	**US$M**	1,039.2	(79.3)	(132.7)	(177.5)	209.3	172.5	166.3	125.7	148.7	116.1	145.3	147.8	90.9	27.9	78.1
Cumulative Net Cash Flow	US$M		(79.3)	(212.1)	(389.5)	(180.2)	(7.7)	158.6	284.3	433.1	549.2	694.5	842.3	933.2	961.1	1,189.9

23.6 Sensitivities

A sensitivity analysis was performed to determine which factors most affected the Project economics. The analysis revealed that the Project is most sensitive to exchange rate (FX rate), followed by diamond grades/prices and capital/operating costs. The Project is marginally less sensitive to operating costs than to capital costs.

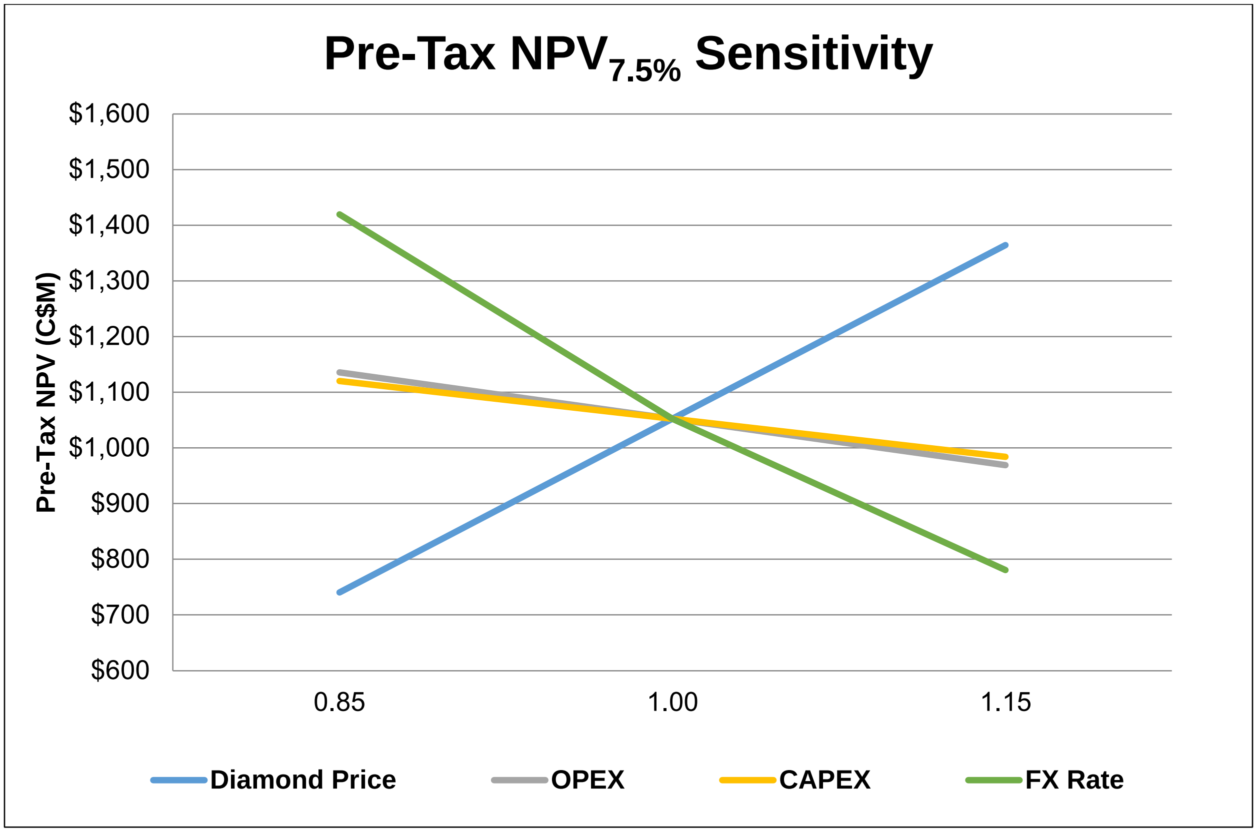
Table 23-7, along with Figure 23-3, outline the results of the sensitivity tests performed on pre-tax and after-tax NPV@ 7.5%.

The Project was also tested under various discount rates. The results of these tests are demonstrated in Table 23-8 and Figure 23-4.

Table 23-7: Sensitivity Results

Variable	Pre-tax NPV @ 7.5% ($M)			Post-tax NPV @ 7.5% ($M)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Diamond Price/Grade	740.2	1,052.3	1,364.4	467.4	668.7	867.9
CAPEX	1,120.4	1,052.3	984.2	736.8	668.7	600.7
OPEX	1,135.6	1,052.3	969.1	722.7	668.7	614.3
FX Rate	1,419.4	1,052.3	780.9	902.8	668.7	493.7

Figure 23-3: Pre-Tax NPV @ 7.5% Sensitivity

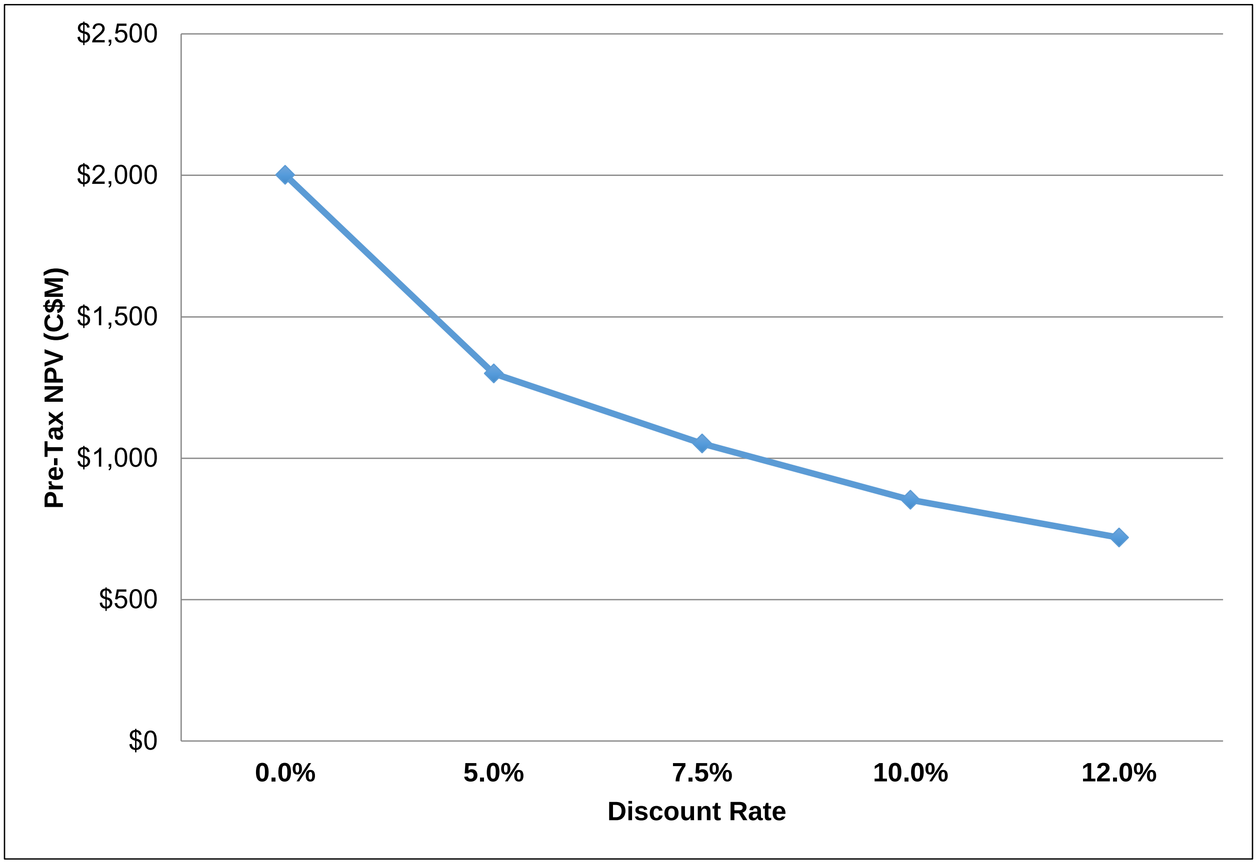


Note: OPEX and CAPEX sensitivity are almost identical

Table 23-8: Discount Rate Sensitivities

Discount Rate	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0%	$2,001.5	$1,332.3
5%	$1,300.0	$841.9
7.5%	$1,052.3	$668.7
10%	$852.5	$529.1
12%	$720.0	$436.5

Figure 23-4: Discount Rate Sensitivity



24 Adjacent Properties

There are no adjacent properties relevant to the scope of this report.

25 Other Relevant Data and Information

The QPs are satisfied that there is no other pertinent or relevant data or information to disclose beyond that which is included in this Report.

26 Interpretations and Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

Using the assumptions highlighted in this report, the Chidliak Project offers sufficient economic potential to be advanced to the next stage of study (Preliminary Feasibility Study).

26.1 Risks

As with any proposed mining project, there are risks. The most significant potential risks associated with Phase 1 development of the Chidliak Project are the level of Mineral Resource estimate, upgrading of existing off-site infrastructure (primarily port construction in Iqaluit), operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, retention of mining personnel due to the remote location, the ability to raise financing, and diamond price variability.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active project management.

Table 26-1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches.

Table 26-1: Main Project Risks

Risk	Explanation / Potential Impact	Possible Risk Mitigation
Mineral Resource Estimate	The resource is currently at an Inferred level, which is considered too speculative geologically to have economic considerations applied in order to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.	Upgrade confidence in Mineral Resource Estimates as proposed in Section 27, which includes additional drilling, bulk sampling, processing and diamond valuation.
Upgrading of Existing Infrastructure Does not Occur.	The existing port infrastructure in Iqaluit is not optimized to support the increased freight requirements for the Project and will require upgrading. This PEA study assumes the Federal Government of Canada will be undertaking these upgrades. The potential impact is increased capital and operating costs and an increased construction period both due to the inefficiencies in the current Iqaluit freight handling facilities.	Work closely with the Federal and Territorial Governments to ensure they understand the benefits of a development at Chidliak. Engage the Canadian Northern Economic Development Agency.
Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced diamond recovery, increased processing OPEX costs, and/or changes to the processing circuit design. If LOM diamond recovery is lower than assumed, the Project economics would be negatively impacted.	Additional sampling and ore dressing study test to be conducted as part of more advanced studies to ensure the ore processing flowsheet is optimized.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the mining cut off grade would increase and, all else being equal, the open pit recovered tonnage would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits could stop or delay the Project.	Continued fostering and further development of close relationship with the local communities. Development of relationships with government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required.
Development Schedule	The Project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the Project economics.	If an aggressive schedule is to be followed, PFS field work should begin in 2019 as detailed in the Section 27. Early engagement with all stakeholders is important.
Ability to Attract and Retain Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the Project, particularly due to the remote nature of the Project. High turnover or the lack of appropriate technical and management staff at the Project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries, flexible work schedules and benefits all help to identify, attract and retain critical people,
Geochemical Characterization	Comprehensive geochemical characterization of the tailings material has not been completed. Should testing confirm either acid generation or metal leaching issues, the proposed TMF closure concept would have to be reconsidered, and may require construction of a low infiltration cover. If the tailings are determined to be geochemically problematic, it may be necessary to minimize contact water, which could require a separate water storage reservoir to provide the necessary processing plant make-up water demand.	Conduct additional sampling and test work.
TMF Design Considerations	Engineering field work and laboratory testing and analysis has not been conducted on the TMF site, impoundment material nor the tailings material. Variations of assumed vs. actual conditions may lead to re-design of the TMF	Conduct additional sampling and test work on the TMF site, tailings material and impoundment material.

Source: JDS / KP (2018)

26.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the Project. The major opportunities that have been identified at this time are summarized in Table 26-2, excluding those typical to all diamond mining projects, such as changes in diamond prices and exchange rates, for example. Further information and assessments are required before these opportunities could be included in the project economics.

Table 26-2: Identified Project Opportunities for Chidliak Project Development

Opportunity	Explanation / Potential Impact	Potential Benefit
Expansion of the Mine – CH-6 Underground	The Mineral Resource at CH-6 has not been fully delineated and there is an opportunity to expand the mineable resource at depth. This added resource would likely be extracted by means of an underground bulk caving operation beneath the pit at the economical cross-over point.	Increase mine life, reduced mining costs at depth.
Diamond Grade & Value at CH-7	The 2015 bulk sample at CH-7 was completed with LDD and as a result, significant diamond breakage was incurred. The reported grade for that sample is considered 15-30% conservative (McCandless, 2016a & 2016c) given the loss of carats due to breakage.	Additional bulk sampling via surface trenching could increase the grade and possibly diamond value, result in additional mine life at CH-7 and should be completed as part of more advanced studies.
Expansion of the Mine – CH-1, CH-44 and other pipes	Additional exploration could bring other kimberlites into the mine plan.	Increase mine life.
Quantify revenue contribution of large diamonds	Determine by additional bulk sampling if the contribution to project value of +10.8 carat diamonds at CH-6 and CH-7 is under-represented by current size frequency distribution and diamond pricing models.	Positive modification to diamond price forecasts and value realization.
Pit Slope Steepening	Pit slope angles could potentially be improved (steepened).	An increase in overall pit slopes for all domains in all pits would reduce the strip ratio.
Project Strategy and Optimization	Detailed planning and strategic trade-off option reviews could result in enhancing project economics.	Planning and executing the Project with the optimum mine design/schedule and processing systems would result in the maximum possible value to shareholders and other economic stakeholders.
Metallurgical Testing	Metallurgical (ore dressing studies), process and equipment specific vendor test work remains to be undertaken at a later engineering stage to optimize the Chidliak Project process flowsheet when the data becomes available.	Test work and trade-off studies are required to determine potential CAPEX and OPEX reduction opportunities e.g. X-ray mineral feed sorting versus conventional DMS, DMS concentrate X-ray sorting versus diamond flotation, fines thickening requirements, fines filtration for stackable waste dump disposal etc.
Tailings Characterization	The tailings split used for the TMF design assumes 20% of the processed material will be PFK tailings, requiring storage in the TMF Physical tailings testing may indicate a higher settled dry density.	If the process flow sheet confirms a smaller percentage PFK tailings, it would result in a commensurate reduction in the TMF size.
Water Balance	A detailed water balance and assessment of the catchment and run-off characteristics may indicate that less water needs to be stored.	Reduction of the TMF size.
Project Schedule Optimization	Build on-site DMS batch plant for processing future bulk samples	Reduce development timeline.
Government Funding for Infrastructure Projects	Engage with the Federal and Territorial Governments to ensure they understand the benefits of a development at Chidliak. Government funding may be made available for infrastructure that benefit the territory as a whole.	Potentially reduce capital costs for major infrastructure.

Source: JDS / Knight Piesold

27 Recommendations

In the opinion of JDS, the Chidliak Phase 1 Diamond Development is of sufficient merit to proceed to the Preliminary Feasibility Study (PFS) stage. This more advanced study will further detail:

- Upgraded Mineral Resource Estimates at CH-6 and CH-7;
- Mine, processing and infrastructure engineering;
- Project scheduling;
- Capital and operating cost estimation; and
- Economic results.

The study will improve the confidence in the Project design and execution and will result in an improved accuracy of project economics.

A PEA should be completed to evaluate the economic viability of underground extraction of CH-6 kimberlite via underground bulk mining methods. This study will determine whether or not to include an underground mine in the PFS.

It is also recommended that environmental monitoring, program planning, and permitting continue as needed to support Peregrine's project development plans.

It is estimated that a PFS and its supporting work programs will cost approximately $27 M over two years. A breakdown of the key components of the next study phase is summarized in Table 27-1. Major components of future work recommended are:

- Deep core drilling at CH-6 to better quantify the TFFE and possibly upgrade it to a Mineral Resource;
- Bulk sample CH-6 via surface trench and LDD in order to potentially upgrade the Inferred Mineral Resource to Indicated level;
- Bulk sample at CH-7 via surface trench to better define the impact of diamond breakage and potentially upgrade the Inferred Mineral Resource to Indicated level;
- Maintaining the collection of geotechnical data to refine mine design.
- Additional sampling and ore dressing study test to be conducted for processing flow-sheet optimization; and
- Continued environmental monitoring, and initiating work in order to develop environmental, water and permitting management plans.

In addition, it is recommended that Peregrine continue and complete a study assessing the feasibility of permitting and constructing a suitably scaled DMS batch plant at Chidliak. This plant would be used to support additional bulk sample processing that may be required as the Project advances to Pre-Feasibility and Feasibility. The study should review the following:

- Cost benefit over other bulk sample processing methods (i.e. off-site third party mineral processing facilities);
- Permitting requirements;
- Engineering requirements;
- Execution plan; and
- Fit with future Phase 1 Diamond development requirements.

Table 27-1: Preliminary Feasibility Study Cost Estimate

Description	Budget Estimate ($)	Comments
Resource Advancement Activity		
Core Drilling Program		
Core Drilling	2,000,000	Pierce point and geotechnical drilling at CH-6 (2,000 m HQ) and CH-7 (1,200 m HQ); drill for caustic samples from CH-6 (1,500 m NQ) and CH-7 (1,500 m NQ).
Operations	1,500,000	All costs associated with the operation of camps and support of field crews (food, meals, salaries, fuel, motels, camp equipment and support flights), capital costs, etc.
Project Management & Supervision	500,000	Supervisory and geologists time unallocated to specific tasks; Land administration including permit fees, annual fees, assessment report preparation; Community relations, etc.
Laboratory Charges	600,000	Caustic fusion analysis of CH-6 and CH-7 core.
Subtotal Core Drilling:	**4,600,000**	
Bulk Sampling Program		
Large-Diameter RC Drilling	4,000,000	Bulk sampling by LDD at CH-6 with 1 x 22" diameter holes to 350 m depth in the high-grade KIM-L domain.
Bulk Sample by Trench – CH-6	3,500,000	Approximately 600 tonnes of kimberlite extracted from the wKIM-L by surface trenching at CH-6.
Bulk Sample by Trench – CH-7	4,000,000	Approximately 1000 tonnes of kimberlite extracted from the wKIM-L by surface trenching at CH-6.
Laboratory Charges	1,000,000	DMS processing of the CH-6 LDD and CH-6 and CH-7 trench bulk samples.
Resource Update	1,500,000	Diamond valuations and resource estimate
Operations	2,500,000	All costs associated with the operation of camps and support of field crews (food, meals, salaries, fuel, motels, camp equipment and support flights), capital costs, etc.
Subtotal Bulk Sampling	**16,500,000**	
Engineering and Environment Advancement Activity		
Engineering Studies		
Geotechnical / Engineering	600,000	Engineering field work and testing to support a PFS.
Infrastructure Engineering Studies	140,000	Engineering studies and permitting for resource development, future camps, infrastructure, access etc., including the beginning the necessary permitting.
Project Management & Supervision	345,000	Supervisory and geologists time unallocated to specific tasks; Land administration including permit fees, annual fees, assessment report preparation; Community relations, etc.
Mineral Processing Studies	100,000	Investigate feasibility of permitting and constructing an on-site DMS facility.
Subtotal Engineering Studies	**1,185,000**	
Environmental Studies		
Monitoring	50,000	Continued monitoring sump and drill sites.
Baseline Work	3,000,000	Collection of baseline data within local study area and All-Weather Road study area.
Community Consultation, Land Use and Heritage Studies	1,150,000	Continued collection of archaeological information and state of knowledge analysis of previously published studies. Commencement of Community Benefit Agreement negotiations.
Developmental Studies and Draft EIS Preparation.	250,000	Commence plan for NIRB project proposal and preparation for Draft Environmental Impact Statement.
Subtotal Engineering and Environment	**4,450,000**	
Total Supporting Work for PFS	**26,735,000**	

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29 Units of Measure, Abbreviations and Acronyms

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3-D	three-dimensions
a	annum (year)
ALT	active layer thickness
AN	Qilaq mineral tenure
ARD	acid rock drainage
B	billion
BHPB	BHP Billiton Limited
C$	dollar (Canadian)
CHI	Chidliak mineral tenure
CHP	combined heat and power plant
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^3	cubic centimetre
ct/s	carats per stone
ct/t	carat per tonne
ct	carat
d	day
d/a	days per annum
DEIS	draft environmental impact statement
DGPS	differential global positioning system
DMS	dense media separation
DNLUP	draft Nunavut land use plan
DTC	diamond trading company
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
FEL	front-end loader
FOC	Fisheries and Oceans Canada
ft	foot

Symbol / Abbreviation	Description
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
Ga	billion years
h	hour
ha	hectare (10,000 m^2)
HG	high grade
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
IDF	Inflow design flood
in	inch
INAC	Indigenous and Northern Affairs Canada
IOL	Inuit-Owned land
IRR	Internal Rate of Return
ISO	International Organization for Standardization
ISRM	International Society for Rock Mechanics
Ja	joint alteration number
Jn	joint set number
Jr	joint roughness number
JDS	JDS Energy & Mining Inc.
k	kilo (thousand)
kg	kilogram
KIM	kimberlitic indicator mineral
km	kilometre
km^2	square kilometre
kt	kilotonne
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per annum
kWh/t	kilowatt hours per tonne
L	litre
L/s	litres per second
LDD	large-diameter drill
LG	low grade
LGM	last glacial maximum
LOM	life of mine

Symbol / Abbreviation	Description
m	metre
M	million
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
Ma	million years
MAE	mean annual evaporation
MAP	mean annual precipitation
masl	metres above sea level
mbs	metres below surface
Mct	million carats
min	minute (time)
ML	million litres
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
MPa	megapascal
Mt	million metric tonnes
MW	megawatt
NG	normal grade
NI 43-101	National Instrument 43-101
NIRB	Nunavut Impact Review Board
NLCA	Nunavut Lands Claim Agreement
NMR	Nunavut Mining Regulations
NQ	drill core diameter of 47.6 mm
NRC	Natural Resources Canada
NSA	Nunavut settlement area
NTI	Nunavut Tunngavik Incorporated
NU	Nunavut
NUPPA	Nunavut Planning and Project Assessment Act
NWB	Nunavut Water Board
OP	open pit
ODS	ore dressing study
OSA	overall slope angles
oz	ounce
P.Geo.	Professional Geoscientist
Pa	Pascal

Symbol / Abbreviation	Description
PAG	potentially acid generating
PCR	pre-hearing conference report
P.E.	Professional Engineer (US)
P.Eng	Professional Engineer (Canada)
PEA	preliminary economic assessment
PFK	processed fine kimberlite
PFS	preliminary feasibility study
Q’	Barton system Q value
QA/QC	quality assurance/quality control
QIA	Qikiqtani Inuit Association
QP	qualified person
RC	reverse circulation
RIA	regional Inuit associations
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SRC	Saskatchewan Research Council
st/kg	stones per kilogram
st/t	stones per metric tonne
t	metric tonne (1,000 kg)
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
t:t	Strip ratio (tonnes waste : tonnes processed)
TFFE	target for further exploration
TMF	tailings management facility
UCS	universal compressive strength
US	United States
US$	United States dollars
V	volt
VEC	valued ecosystem components
VMS	volcanic massive sulphide
w/w	weight/weight

Symbol / Abbreviation	Description
WRSF	waste rock storage facility
µm	micrometre

Rock Type	Description
ACK	Apparent Coherent Kimberlite
CK	Coherent Kimberlite
CRX	Country rock xenolith
HK	Hypabyssal Kimberlite
LSTX	Paleozoic carbonate xenolith
PK	Pyroclastic Kimberlite
RVK	Resedimented Volcaniclastic Kimberlite
VK	Volcaniclastic Kimberlite

Appendix 1: Chidliak Project List of Claims

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
AN001	K15241	26A13	26B16		Active	16-Aug-11	16-Aug-18	189.96	76.87	$0.00
AN002	K15242	26A13	26B16		Active	16-Aug-11	16-Aug-18	619.44	250.68	$0.00
AN003	K15243	26A13	26B16		Active	16-Aug-11	16-Aug-18	549.49	222.37	$0.00
AN013	K15253	26A13			Active	16-Aug-11	16-Aug-18	862.24	348.94	$0.00
AN014	K15254	26A13			Active	16-Aug-11	16-Aug-18	488.73	197.78	$0.00
AN015	K15255	26A13			Active	16-Aug-11	16-Aug-18	958.96	388.08	$0.00
AN016	K15256	26A13			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$928.27
AN017	K15257	26A13			Active	16-Aug-11	16-Aug-19	1,728.60	699.54	$0.00
AN018	K15258	26A13			Active	16-Aug-11	16-Aug-19	988.80	400.15	$0.00
AN024	K15264	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN027	K15267	26A13			Active	16-Aug-11	16-Aug-19	2,299.83	930.71	$0.00
AN028	K15268	26A13			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN029	K15269	26A13			Active	16-Aug-11	16-Aug-19	2,064.30	835.39	$0.00
AN030	K15270	26A13			Active	16-Aug-11	16-Aug-19	2,490.18	1,007.74	$0.00
AN031	K15271	26A13			Active	16-Aug-11	16-Aug-18	40.09	16.22	$0.00
AN032	K15272	26A13			Active	16-Aug-11	16-Aug-18	129.00	52.20	$0.00
AN033	K15273	26A13			Active	16-Aug-11	16-Aug-18	206.60	83.61	$0.00
AN034	K15274	26A13			Active	16-Aug-11	16-Aug-19	1,962.70	794.28	$0.00
AN035	K15275	26A13			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN037	K15277	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$1,427.20
AN039	K15279	26A12			Active	16-Aug-11	16-Aug-21	238.47	96.51	$16,367.64
AN040	K15280	26A12			Active	16-Aug-11	16-Aug-21	2,582.50	1,045.10	$10,330.00
AN041	K15281	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN042	K15282	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN043	K15283	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN044	K15284	26A13			Active	16-Aug-11	16-Aug-21	2,582.50	1,045.10	$10,330.00
AN045	K15315	26A13			Active	16-Aug-11	16-Aug-21	1,497.85	606.16	$5,991.40
AN046	K15286	26A13			Active	16-Aug-11	16-Aug-21	620.00	250.91	$3,720.00
AN047	K15287	26A13			Active	16-Aug-11	16-Aug-21	2,565.50	1,038.22	$10,262.00
AN048	K15288	26A13			Active	16-Aug-11	16-Aug-21	826.40	334.43	$4,958.40
AN049	K15289	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN050	K15290	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN051	K15291	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN052	K15292	26A12			Active	16-Aug-11	16-Aug-21	2,582.50	1,045.10	$10,330.00
AN053	K15293	26A12			Active	16-Aug-11	16-Aug-21	1,111.00	449.61	$6,079.07
AN054	K15294	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN055	K15295	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
AN056	K15296	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN057	K15297	26A13			Active	16-Aug-11	16-Aug-21	1,652.80	668.86	$6,611.20
AN058	K15298	26A13			Active	16-Aug-11	16-Aug-21	1,394.55	564.35	$7,060.71
AN059	K15299	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN060	K15300	26A12			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN061	K15301	26A12			Active	16-Aug-11	16-Aug-19	2,359.00	954.65	$0.00
AN062	K15302	26A12			Active	16-Aug-11	16-Aug-19	516.50	209.02	$21.00
AN063	K15303	26A12			Active	16-Aug-11	16-Aug-19	671.00	271.54	$0.00
AN064	K15304	26A12	26A13		Active	16-Aug-11	16-Aug-19	1,622.00	656.40	$0.00
AN065	K15305	26A13			Active	16-Aug-11	16-Aug-21	671.00	271.54	$4,026.00
AN066	K15306	26A13			Active	16-Aug-11	16-Aug-21	760.00	307.56	$4,560.00
AN067	K15307	26A06			Active	16-Aug-11	16-Aug-19	1,549.50	627.06	$0.00
AN068	K15308	26A06			Active	16-Aug-11	16-Aug-20	2,582.50	1,045.10	$2,568.31
AN069	K15309	26A06			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
AN070	K15310	26A06			Active	16-Aug-11	16-Aug-19	1,291.25	522.55	$0.00
AN071	K15311	26A06			Active	16-Aug-11	16-Aug-19	2,582.50	1,045.10	$0.00
CH015	K12507	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$18,843.83
CH024	K12516	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$7,151.76
CH025	K12517	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,811.16
CH026	K12518	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,996.10
CH027	K12519	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$269.45
CH034	K12526	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,996.10
CH035	K12527	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$244.45
CH036	K12528	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$702.70
CH037	K12529	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$7,573.00
CH038	K12530	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$5,729.59
CH039	K12531	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$491.62
CH046	K12538	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$5,424.17
CH047	K12539	26B2			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$26,283.64
CH048	K12540	26B2	26B7		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,811.16
CH049	K12541	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,249.57
CH050	K12542	26B7			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$587,852.95
CH051	K12543	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,488.55
CH052	K12544	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$701.18
CH053	K12545	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,192.67
CH054	K12546	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$109,403.29
CH055	K12547	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$239,403.29

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH056	K12548	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$10,151.59
CH057	K12549	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,401.18
CH058	K12550	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,031.23
CH059	K12551	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,074.37
CH060	K12552	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$587,852.95
CH061	K12553	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,074.37
CH062	K12554	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,425.62
CH063	K12555	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$5,421.47
CH064	K12556	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$5,732.94
CH065	K12557	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$11,047.44
CH066	K12558	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$14,282.72
CH067	K12559	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$877.89
CH068	K12560	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$9,148.48
CH069	K12561	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,650.82
CH070	K12562	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,347.20
CH071	K12563	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,217.94
CH072	K12564	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,092.27
CH073	K12565	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$10,626.43
CH074	K12566	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,071.71
CH075	K12567	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,298.12
CH076	K12568	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$7,872.55
CH077	K12569	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,140.92
CH078	K12570	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$897.79
CH079	K12571	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$181.06
CH080	K12572	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,885.91
CH081	K12573	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,076.75
CH082	K12574	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$405.62
CH083	K12575	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,270.55
CH084	K12576	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,286.16
CH085	K12577	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$1,274.49
CH086	K12578	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,286.16
CH087	K12579	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,392.53
CH088	K12580	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,703.83
CH089	K12581	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,154.89
CH090	K12582	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$240.56
CH091	K12583	26B7	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$240.56
CH092	K12584	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,154.89

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH093	K12585	26B7			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,703.83
CH095	K12587	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$574.49
CH096	K12588	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,842.49
CH100	K12592	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,883.39
CH101	K12593	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,883.39
CH102	K12594	26B7	26B10		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$647.20
CH103	K12595	26B7	26B10		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$647.20
CH104	K12596	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,883.39
CH105	K12597	26B7			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$741.75
CH135	K12627	26B10			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$5,033.15
CH136	K12628	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$558.35
CH139	K12631	26B10			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$4,903.38
CH140	K12632	26B10			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$5,002.16
CH142	K12634	26B10	26B15		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$166.80
CH143	K12635	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,534.00
CH144	K12636	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,780.17
CH146	K12638	26B10			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$386.87
CH147	K12639	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$492.62
CH148	K12640	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$498.56
CH149	K12641	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,519.85
CH150	K12642	26B10			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$1,733.13
CH151	K12643	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$682.57
CH152	K12644	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$682.57
CH153	K12645	26B10	26B15		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$289.64
CH154	K12646	26B10	26B15		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,303.82
CH155	K12647	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$88.34
CH156	K12648	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$188.34
CH157	K12649	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$611.59
CH158	K12650	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$9,066.05
CH159	K12651	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,519.85
CH160	K12652	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$230,586.94
CH162	K12654	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,273.53
CH163	K12655	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$428.75
CH164	K12656	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$428.75
CH165	K12657	26B10	26B15		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,303.82
CH166	K12658	26B10	26B15		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$89,885.67
CH167	K12659	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$21,198.34

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH168	K12660	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$132,198.34
CH169	K12661	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$121,715.18
CH171	K12663	26B10			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,921.98
CH196	K12688	26B15			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$352.02
CH197	K12689	26B15			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$654.16
CH200	K12692	26B15			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$652.02
CH201	K12693	26B15			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$654.16
CH202	K12694	26B15	26B16		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$85,563.90
CH203	K12695	26B15	26B16		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$4,419.83
CH204	K12696	26B15	26B16		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$637.03
CH206	K12698	26B16			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,004.07
CH207	K12699	26B16			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$4,722.25
CH208	K12700	26B16			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,649.88
CH235	K12727	26B9	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$109,385.95
CH239	K12731	26B9	26B10		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$21,402.92
CH240	K12732	26B9	26B10	26B15/16	Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$63,970.02
CH241	K12733	26B9	26B16		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$430.47
CH242	K12734	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$62,680.05
CH245	K12737	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,408.52
CH246	K12738	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,275.24
CH247	K12739	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,378.45
CH258	K12750	26B9			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$1,131.75
CH259	K12751	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,681.07
CH260	K12752	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,192.43
CH269	K12761	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$16,429.09
CH270	K12762	26B9			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$16,429.09
CH301	K12793	26B7	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$17,345.27
CH302	K12794	26B7	26B8		Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$588.51
CH303	K12795	26B7	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$114,852.19
CH304	K12796	26B7	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,869.28
CH305	K12797	26B7	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$6,351.69
CH306	K12798	26B7	26B8	26B9/10	Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,528.07
CH307	K12799	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$306.48
CH308	K12800	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$204.16
CH309	K12801	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$116,326.67
CH310	K12802	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,146.08
CH311	K12803	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$28.07

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH312	K12804	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$16,724.07
CH313	K12805	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$28,355.50
CH314	K12806	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$158,956.19
CH315	K12807	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$846.08
CH316	K12808	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,574.86
CH317	K12809	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$14,378.93
CH318	K12810	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,306.48
CH319	K12811	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$14,024.83
CH320	K12812	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$13,504.16
CH321	K12813	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,265.30
CH322	K12814	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,050.95
CH323	K12815	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$158,956.19
CH324	K12816	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,965.08
CH325	K12817	26B8			Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$172,651.63
CH326	K12818	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,394.93
CH327	K12819	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,323.89
CH328	K12820	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,204.16
CH329	K12821	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$661.45
CH330	K12822	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$206.48
CH331	K12823	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$335.27
CH332	K12824	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$558.86
CH333	K12825	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$11,673.61
CH334	K12826	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,490.45
CH335	K12827	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,542.40
CH336	K12828	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$48,927.23
CH337	K12829	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$56,281.69
CH338	K12830	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$230,961.34
CH339	K12831	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$337,452.24
CH340	K12832	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,900.11
CH341	K12833	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$956.02
CH342	K12834	26B8	26B9		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$4,364.85
CH343	K12835	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$29,411.21
CH344	K12836	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$29,052.57
CH345	K12837	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$335,643.08
CH346	K12838	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$126,452.24
CH347	K12839	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$9,240.83
CH348	K12840	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$170,495.58

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH349	K12841	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,233.17
CH350	K12842	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$127,497.47
CH351	K12843	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,402.24
CH352	K12844	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$100,683.59
CH353	K12845	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$332.65
CH354	K12846	26B8	26B9		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,539.65
CH355	K12847	26B8	26B9		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$708.45
CH356	K12848	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$100,683.59
CH357	K12849	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,798.23
CH358	K12850	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,594.43
CH359	K12851	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$41,088.03
CH360	K12852	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,233.17
CH361	K12853	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$24,688.93
CH362	K12854	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$41,088.03
CH363	K12855	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,594.43
CH364	K12856	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$617.09
CH365	K12857	26B8			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$617.09
CH366	K12858	26B8	26B9	26A5/12	Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,574.79
CH367	K12859	26B1	26B2	26B7/8	Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$26,211.58
CH368	K12860	26B1	26B2		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$12,647.91
CH369	K12861	26B1	26B2		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,055.03
CH377	K12869	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$15,033.25
CH378	K12870	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,752.03
CH379	K12871	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$13,512.22
CH380	K12872	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$55,194.87
CH381	K12873	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$10,495.84
CH382	K12874	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$20,805.03
CH383	K12875	26B1			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$724.75
CH388	K12880	26B1			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$662.91
CH389	K12881	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$159.75
CH390	K12882	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$495.84
CH391	K12883	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$287,794.25
CH392	K12884	26B1	26B8		Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$299,512.22
CH393	K12885	26B1	26B8		Active, Lease Pending	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$224,512.22
CH394	K12886	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$224,966.86
CH395	K12887	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$62,507.55
CH396	K12888	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$259.75

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH397	K12889	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,193.12
CH400	K12892	26B1			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$719.41
CH401	K12893	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,441.43
CH402	K12894	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$64,659.75
CH403	K12895	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$80,377.54
CH404	K12896	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,036.58
CH405	K12897	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$24,543.49
CH406	K12898	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$25,415.61
CH407	K12899	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$127,493.70
CH408	K12900	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,722.28
CH409	K12901	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$79,817.46
CH410	K12902	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$12,403.52
CH411	K12903	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,435.51
CH413	K12905	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,874.27
CH414	K12906	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,230.62
CH415	K12907	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$11,628.95
CH416	K12908	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$127,493.70
CH417	K12909	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$23,570.73
CH418	K12910	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$57,915.61
CH419	K12911	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$170,915.61
CH420	K12912	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,644.57
CH421	K12913	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$23,570.73
CH422	K12914	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,210.48
CH423	K12915	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,903.71
CH427	K12919	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$2,903.71
CH428	K12920	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$228.95
CH429	K12921	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$8,304.38
CH430	K12922	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,644.57
CH431	K12923	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$39,330.76
CH432	K12924	26B1	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$3,742.20
CH433	K12925	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$102,222.28
CH434	K12926	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$222.28
CH435	K12927	26B1			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$228.95
CH436	K12928	26B1			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$890.00
CH439	K12931	26B1	26B8	26A4/5	Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$100,940.46
CH440	K12932	26A4	26B1		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,648.39
CH441	K12933	26A4	26B1		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$414.38

Claim Name	Claim Number	NTS Map Sheet 1	NTS Map Sheet 2	NTS Map Sheet 3/4	Claim Status	Recording Date	Aniversary Date	Area Acres	Area Hectares	Excess / Deficit Credits
CH442	K12934	26A4	26B1		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,379.91
CH449	K12941	26A4			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$554.57
CH450	K12942	26A4			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$577.80
CH451	K12943	26A4	26A5		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$68,290.92
CH452	K12944	26A4	26A5		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$190.77
CH453	K12945	26A4			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$4,463.34
CH499	K12991	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,266.62
CH509	K13001	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,622.33
CH540	K13032	26A5	26A12		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,695.19
CH541	K13033	26A5	26A12		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,695.27
CH547	K13039	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,637.01
CH548	K13040	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$4,519.49
CH551	K13043	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,746.43
CH552	K13044	26A5	26A12		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$3,485.75
CH553	K13045	26A5	26A12		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$11,130.49
CH554	K13046	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,746.43
CH555	K13047	26A5			Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$2,152.67
CH557	K13049	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,031.06
CH558	K13050	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$68,213.45
CH559	K13051	26A5	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$23,976.58
CH560	K13052	26A5	26B8		Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$1,865.75
CH563	K13055	26A5			Active	17-Aug-09	17-Aug-19	2,582.50	1,045.10	$11,245.65
CH564	K13056	26A5	26B8		Active	17-Aug-09	17-Aug-18	2,582.50	1,045.10	$1,506.77
								778,685.740	315,122.939	

Appendix 2: Diamond Processing Recovery Flow Diagrams

Figure 1: SRC caustic method for diamonds > 106 µm.

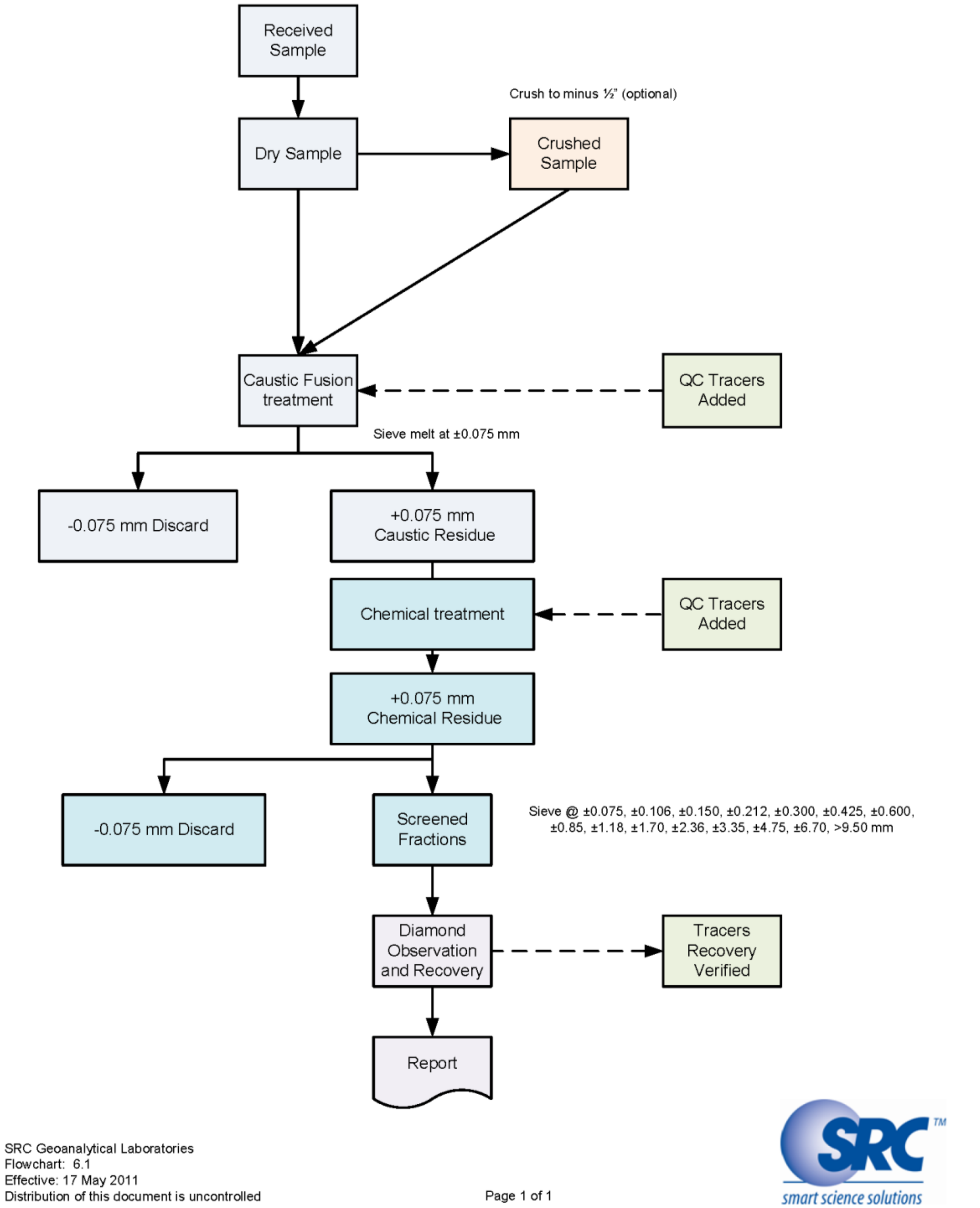


Figure 2: SRC dense media separation process flow diagram.

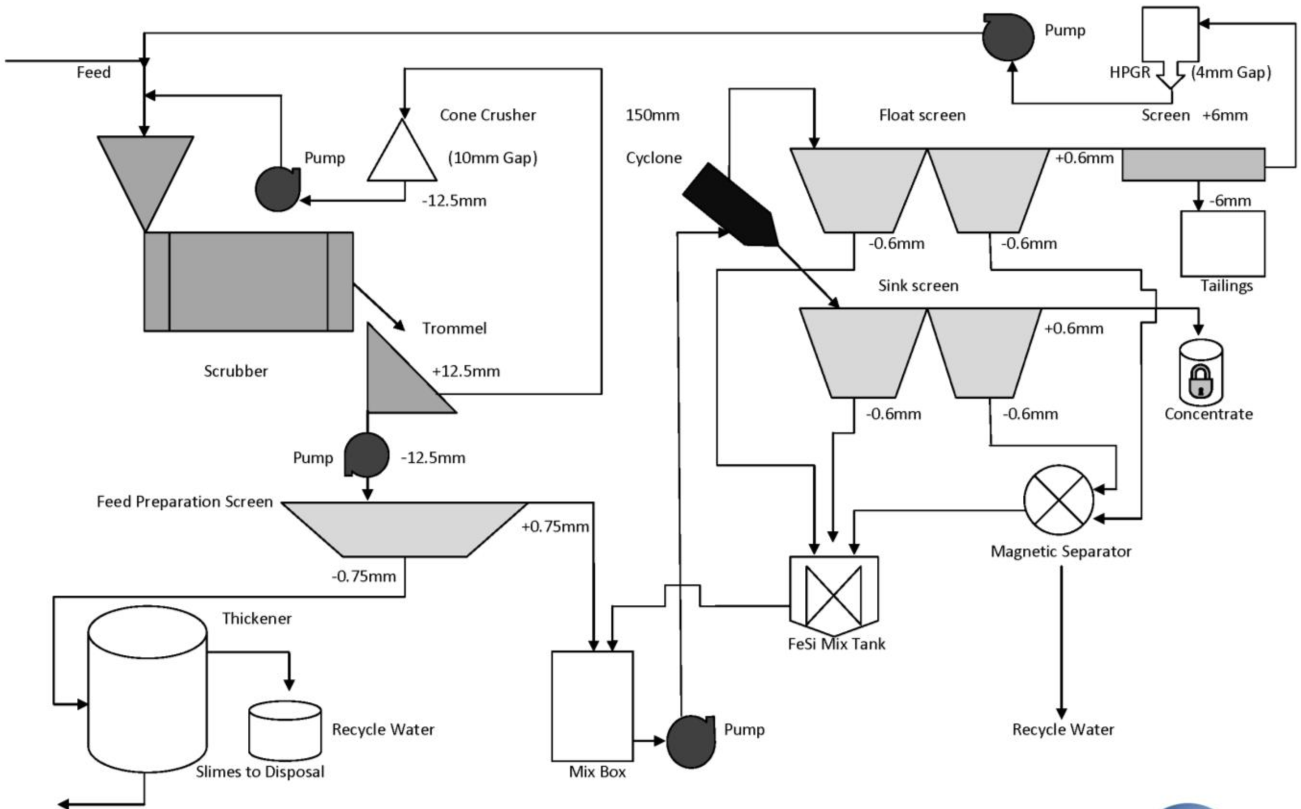


Figure 3: De Beers Sudbury dense media separation process flow diagram.

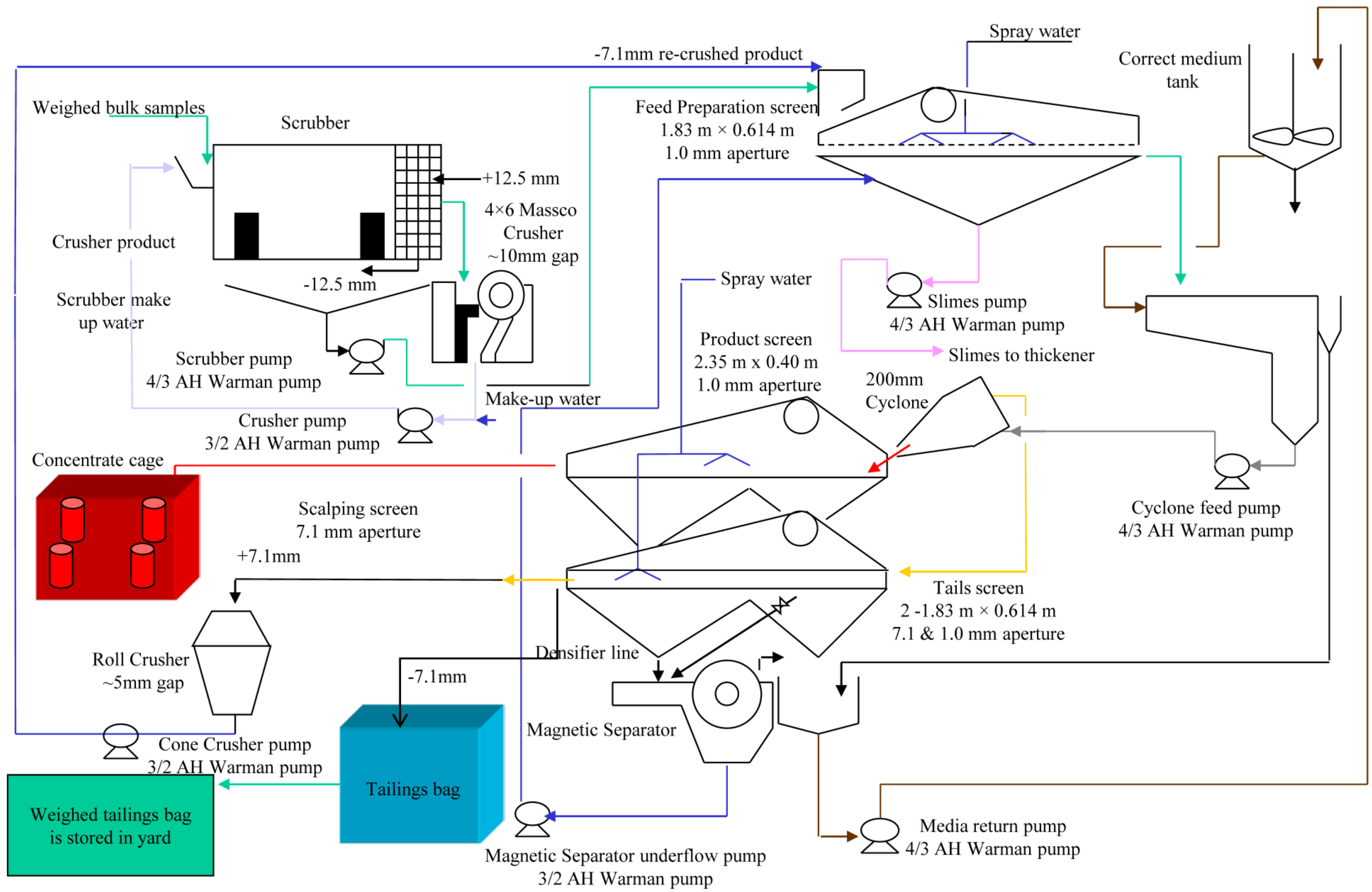
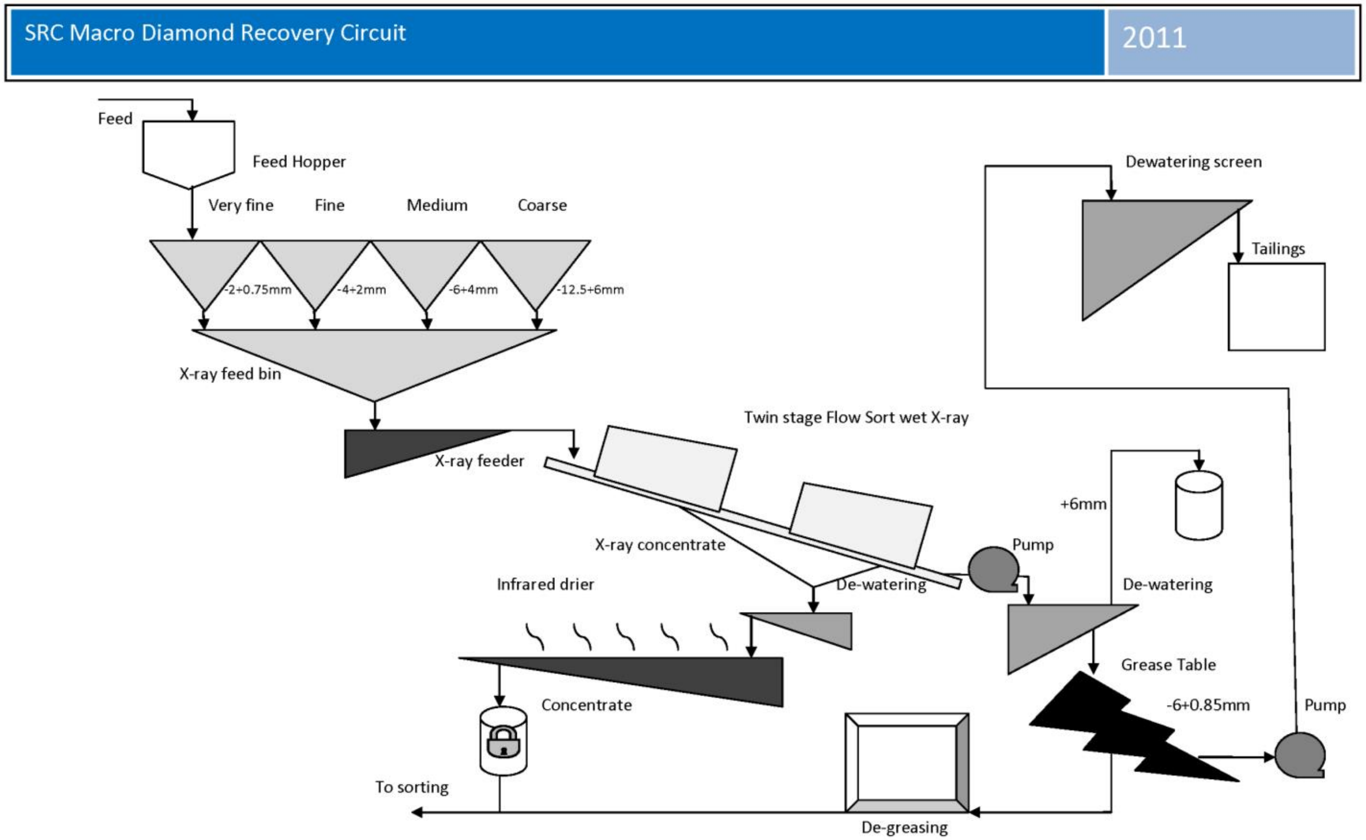


Figure 4: SRC macrodiamond X-ray recovery circuit flow diagram.



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