JERICHO PROJECT

PROJECT DESCRIPTION

Tahera Corporation
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121 Richmond St. West
Toronto, Ontario
M5H 2K1

Month, Day, Year

EXECUTIVE SUMMARY

Benachee Resources Inc., a wholly owned subsidiary of Tahera Corporation (Tahera) plans to construct and operate a diamond mine ("the Jericho Diamond Project") near the north end of Contwoyto Lake in Nunavut Territory (NT). Operations will commence with an open pit mine at the north end of Contwoyto Lake in West Kitikmeot. Ore will be mined eight months of the year (April through November) and processed year round. Fine processed kimberlite (roughly 15% of the plant feed) will be pumped to a lake near the plant site (local name Long Lake). Coarse rejects (the remainder of the plant feed except diamonds) will be stockpiled for use as processed kimberlite impoundment cover material or for other reclamation purposes where a final cover of overburden will be placed. During winter months an ice road will be used to transport materials and supplies to the Jericho site. With current resources the mine and processing plant will have an 8-year life and employ a total of approximately 105 to 175 people (including employees and contractors) with approximately half that number on site at any rotation. Tahera Corporation has actively explored for diamonds in Nunavut for the past seven years (prior to 1999 as its predecessor company, Lytton Minerals).

Location:
350 kilometres SW of Ikaluktutiak (Cambridge Bay) (NT)
420 kilometres NNE of Yellowknife, NWT and 170 km north of the EKATI™ Diamond Mine and Diavik Diamond Project at Lac de Gras, NWT
Jericho Site: 25 kilometres NNW of Lupin Mine
65°59'50" N Latitude, 111°28'30" W Longitude

Proposed Development:
Eight-year operating life on currently proven reserves.
Open pit mine Years 1 to 4 at 330,000 tonnes per year.
Defined resource: 2.53 million tonnes averaging 1.25 carats per tonne with an average value of US $76 per carat (Central Lobe) and US $60 per carat (other lobes).
Processing plant on site.
Implementation of underground mining at Years 5 to 6.
Processing only of North Lobe ore Years 7 to 8.
Ore treatment by Dense Medium Separation (DMS) with cut-off size of 1.15 mm particle size (at present).
Diamond Sorting at on-site facility.
Processed kimberlite disposal on site: stockpile for coarse rejects (-8 to +1 mm) and impoundment for fines (-1mm).

Access and Transportation
1200 m long airstrip at Jericho
Float and ski plane access on Carat Lake (extra-ordinary)
Ice strip for Hercules aircraft in winter on Carat Lake if required
Winter Road Access from Yellowknife to Jericho (mid January to mid March)
All weather road from Contwoyto Lake to the mine site.

Employment

Mine Construction:	25 to 60 including pre-stripping work force
Plant Construction:	20 to 60 people
Operations:	60 to 116 people over 3 years for open pit mining
	48 people over 2 years for underground mining
	40 people over 8 years for processing

Proposed Work Schedule: Mining: 2x12-hour shifts/day, 7 days /week (April through November)
Process Plant: 2x12-hour shifts/day, 7 days per week (year round)
Two week on, two week off roster

Project Schedule
The proposed schedule has the following key milestones:

• Water Licence and Land Lease applications	January 2001
• Draft EIS Submission	January, 2001
• Prehearing Meetings	May & June 2001
• Final EIS Submission	January 2003
• Final Public Hearings	April 2003
• Project Approval (NIRB)	Early June 2003
• Approval from Minister	Late July 2003
• Mobilization and Winter Road Haul	January to March 2004
• Site and Facilities Construction	March – December 2004
• Open Pit Development	March - May 2004
• Full Production	First Quarter 2005

Tahera will select an engineering firm as the engineers for the design, procurement, and construction management of the diamond plant and infrastructure. Dowding Reynard & Associates (Pty) Limited (DRA) of South Africa were responsible for the design and construction of the 10 ton per hour pilot plant formerly located at the Lupin Mine and the engineering design and estimate for the feasibility study.

The selected firm will provide engineering supervision for the project construction and commissioning. A project construction manager will have overall site responsibility. The engineering firm will appoint Canadian sub-contractors to erect the buildings, plant, fuel storage facility, power generators, camp, water services, and processed kimberlite disposal system. The construction program is scheduled to commence in March 2004 and be completed in December 2004.

Plant commissioning is scheduled forDecember 2004. The firm will provide experienced mechanical, instrumentation, and process engineers and, where appropriate vendors will provide commissioning engineers. Tahera Corporation will recruit plant employees, who will participate in the pre-commissioning period and the start up.

Nuna Logistics is an experienced mining contractor with extensive experience in the Northwest Territory and Nunavut. Tahera Corporation intends to appoint Nuna Logistics as the mining contractor for the project. Nuna Logistics has completed the pre-stripping for a major diamond mine in the NWT and is currently implementing site preparation for two other diamond projects in the NWT. Nuna Logistics will mobilize to the project site in February

2004. During February Nuna will complete the ice road from the Lupin Mine to Jericho and establish a camp, diesel storage facility, workshop, explosive storage facilities, and ice access roads on the site.

Tahera Corporation will appoint a project director and project manager to overview the design, procurement, and construction of the project. An accountant will be designated in 2004 to provide financial control services for the project development and the ongoing operation. Tahera will recruit employees for the plant operation during 2004 and implement a training program prior to plant commissioning.

A pre-strip pit will be excavated to expose sufficient kimberlite ore to permit a speedy start up of the processing plant. During the pre-stripping period the open pit will not generate any revenue. Due to the ultimate depth of the planned pit, successive pushbacks will be necessary following the completion of each pre-stripping pit. The width of the pushback will be based on minimum practical mining widths, equipment selection, the required ore schedule, safety, cost, efficiency, and other technical issues.

Mining will be by conventional open pit methods followed by underground mining using open benching or sublevel caving methods. High and medium grade ore from the F6 and F4N zones (central and northern lobes) will be stockpiled immediately north and east of the process plant. The remaining low grade kimberlite will be stockpiled close to the pit, separate from the high grade and waste material, in the event future diamond values increase and make the material economically viable. Also a sample from the south lobe will be taken and processed in the plant to evaluate more accurately the recoverable grade and value of the diamonds. Waste rock, consisting largely of granite, will be placed in dump sites 1 and 2 located immediately to the northeast and south, respectively, of the pit (see Map A, Appendix E).

Underground access will be achieved via a portal in the pit. Ore and waste material from the underground operation will be placed on the high-grade ore stockpiles and the waste dumps. Waste from the underground will come mainly from the decline and level development.

Technical expertise for the construction of diamond processing plants (DMS) has historically been derived from South Africa. Depending on which mining engineer is contracted to supply the processing plant, there is the possibility that the DMS plant will be engineered and constructed in South Africa. Where possible, major equipment has been selected on the basis that it must be supported in Canada. The ore will be processed using conventional diamond processing techniques with the process flowsheet design based on the metallurgical characteristics of the kimberlite ore that was treated in the underground bulk sample program at Tahera’s 10 tonne per hour process plant.

The mining contractor will arrange housing for mine crews. A housing complex constructed of industrial trailers will be erected at the site. Accommodation and kitchen construction and operation will meet or exceed all Workers' Compensation Board, Nunavut Environmental Health and National Fire Code regulations. The accommodation block will have a built-in fire detection system. The kitchen canopy and exhaust ducts will be fitted with a fusible link activated wet chemical fire suppression system. Dry chemical fire extinguishers will be located at all exits with additional units positioned strategically within the building. All doors leading outside will be clearly marked with exit signs, swing outward and be fitted with panic bars (pressure released horizontal latch mechanisms across the centre of the door). Emergency battery operated lighting will be provided in all accommodation and kitchen areas.

Process and potable water will be drawn from Carat Lake from a causeway and pipe constructed out into the lake approximately 90 m to put the intake below where ice damage can occur. Water will be pumped to the plant and accommodation block from a pump house at the shoreward end of the causeway. Approximately 30 m^3/hour will be required.

One fuel storage area will be constructed for all fuel requirements and will be located near the processing plant. The fuel farm will contain twenty vertical tanks that will be shipped in pieces and erected on site. As well, the existing nine tanks used for underground exploration will be moved into the fuel farm area. The fuel farm will be bermed to hold a minimum of 110% of the capacity of the largest tank. The berm will be lined with an impermeable, petroleum-resistant geomembrane.

A lean-to extension to the processing plant will house three 1300 KW, 60 Hz 600 V generator sets (2 + 1 standby) and a 300 kW emergency camp generator all under a 5t maintenance crane. Two additional 800 KW generators will be used, one at the mine shop, one on standby. The 800 KW generator will supply the mechanical shop and the crusher (when required). Generators will be diesel-powered and burn low-sulphur fuel to minimize emissions. Typical fuel consumption for the large generator units is 217 L/hr/unit at full load. Power distribution will be by weatherproof Teck™ cable buried along the access road sides.

Granular fill material will be required for top dressing pads and roads, especially in Year 1 when pre-construction commences. Four borrow areas have been identified from previous geotechnical investigations. One is not near existing roads and will not be exploited unless other nearby sources are exhausted. All borrow areas are north of the exploration camp and two are adjacent the airstrip.

Damming a lake basin south of the proposed location for the processing plant will create a containment area for processed kimberlite. This basin with dams added will be capable of holding all processed kimberlite solids generated by the process plant. Waste process water will be discharged to the containment area. Runoff collected in the catchment and water from the process plant will require discharging annually. Discharge will be to Lake C3 immediately south of Carat Lake and in the same drainage basin. Discharge water will meet Project licence requirements, including being non-acutely toxic to fish.

A series of berms, ditches, and sediment containment ponds will be constructed to collect water for treatment prior to discharge from the site. For eight months of the year water will be frozen; a large percentage of the water on the land will be discharged in the freshet period (typically June) and lesser amounts throughout the summer and early fall. Design of water handling structure will account for this characteristic of small Arctic drainage basins.

A small stream crosses the area that will become the open pit. In order to keep water out of the pit and to protect downstream fisheries resources, a diversion will be constructed in Year 1. The diversion will ensure natural flow volumes are maintained and water quality will not be affected as the engineering design requires protection of the diversion bed from erosion.

A rotating biological contactor waste water treatment plant will be installed to treat and disinfect grey and sewage water from the accommodation, kitchen, and plant. The treatment plant will handle only domestic waste water and will be capable of producing an effluent that is better than NWT guidelines for municipal waste water plant discharges.

Tahera will practice progressive reclamation to the greatest extent possible. Once the open pit is completed, waste rock dumps will be reclaimed; infrastructure no longer required for underground mining will be removed and the site reclaimed. Reclamation trials undertaken by Tahera in the initial years of mining and the reclmation efforts of other Arctic diamond mines will guide revegetation efforts.

Alternatives examined in arriving at the preferred mine configuration included:

- Satellite operation whereby a processing plant would be located at the Lupin Mine and ore stockpiled for winter haul at the Jericho site. This option presupposes an agreement between the two mining companies involved, fully satisfying their needs. As well, a number of regulatory requirements, such as liability and security related matters, make it difficult to contemplate the use of an existing operating site by another tenant.
- A full underground operation and no open pit, which is not economically viable.
- A full open pit operation which would have required removal of excessive amounts of waste rock and would result in leaving more ore in place than the preferred combination open pit-underground operation.
- Alternate locations for the processing plant and processed kimberlite containment area, all of which would entail greater economic and environmental costs.

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n=4noE}=sJ6, ns5]kWvu5 xgw8Nstbsix6S6, w]M4f5 nN/s?9oxli trtbs?9oxli]S6yms3li. xJ3N8qt9lA xqJ5 nN3Dt5 xg6bsix6S5 wvJ6g6bsQxc6g5 vNbu. n=4 nN/six6S6 xg3lt4 s/ci4 yt]Mi4 WoExcDy3i4 mo[lt4 ttCs/6ymiE/q8i4 n=4bcs6g5 WoExa0Jyq8i s/ci yt]MaJi Wbc6gi4]x6rQx6bsJi4 miCs2 wlxi5 W/sifi5 WoExaJi5 bBwsC2 !) b8i4 wv3Cb]n6 WoExc3=zi5]l W8 s/C1ix3=zi5.

w[l3JxEix6bq5 s/C1ix3=1u nNJ5]x6r4bsix6S5 s/C1ix3=1u]y8g]C4tsJj5. w[l3Jx6 nN/sJ6 vobs]hi5 kN4]Dt3Jxi5 N2X6bsix6S6. gJ3u=5 wZl xsMbsiz mo4bc3ix6S5 sz]bd5t8q9lt4 wcNw/6t5 N9os4f]noEp4f5 vtmpq8i4, kNK5 x?lusboEi3j5]k8ix6goEi3j9l vNbomul wfx9M4goEi3j5 xgxZ3i4. gJ3u=sJ6 nN/sym}=sK6 wfx9M4gco6X5 csp0Jt4nu4. wZzb]c z x7myMj5 xixt5t?9ox0JysJ5 SJ3u4 wo/six6S5 w1ui4 s4fxvstQN/6g5 wfx9M4gc6X5 WEz]J/ xs4X5 c2tE0Jyzl w1ui4 xs9M3li WoE0Jy4n6. Xi6g5 c2tDt5 bmw8i s4fx3i xgw8Nsix6S5 xyq5 w[l3Jx2 wlxi xgw8Nscbslt4 s4fx5 xyxi. bmw8i4 s4fx5 yM]bk5]XaJ5 NlNw6ymix6S5 s4fxaiq8i4, yM7j5 s4fwc5b3lt4 x7m gx=3N6gc6X5 tA7u=cs3lt4 GS2M1j5 nebsc5b6gi4 etxi Ntzb x5btcs3ix6S5H. gx=3N6goEi3j5]Xg]baJ6 wr5b6gi4 xsM5t0Jbsix6S6 bmw8i xq3CE/sJi gJ3u=1i x7m wZ}=1i.

wm6 wu6bsJ8N6g3l rsE5]b4u5 wm3u5 x7m h2lo1u5 nN/symJu5 by3j5 !&) ubj5 wo/symix6S6 yfj5 hD4bsJ8N3iE/zi. wm6 X2bsix6S6 WoExc3=1j5 x7m gJ3u=1j5 X2Bxsyu5 y[/zi whxi wmc3isJ2. ciQ/zi #) m3 csb]n5F xg6bsQxc3ix6S5.

xbsy6 s6hx]ly}=4 nN/six6S6 bmw8k5 s6hxl1k5 xg6bsQxc6gk5 x7m WoExc3=sJ2 ciQ/z]i9li. s6hxlcsy}=sJ6 @)_i4 s6hxlcsy}=c3ix6S6 trtbsi4 w]M4f5 x7m N2X6bsJi4. x7ml, (_aJ5 s6hxlcsy}=sJ5 miCs2 wlxi eiDysJ5 k4t6bsix6S5 s6hxlcsy}=1j5. s6hxlcsy}=4 wl]XDbc3ix6S6 XX5tJ8N3li !!)-_u4 xq]MaJu4 s6hxlcsy}=1u4. wl]XDbc6g5 yx7mA8N8q5gu4, s6hxl1u4 y5tDtJ8N8qgu4.

eJ1u4 wMQx6yymix6S5 WoExc3=sJu Wzhi4 !#)) rlKx5i4, ^) Hz 600 V /kEgc3ix6Lt4 G@ + ! yb8Xw]u9liH x7m #) rlKx5 gx=3N6gc6X5 /kEg % T nN3Dt]u9li. m3D4 *)) rMKx5c6g5 /kEbsJ4 xg6bsix6S5, xbsy6 s/C1ix3=s2

WoExc3=z i xbsy6 x4bfc3=1u !_u4 yc9otE0Jtj5, WQxc6t9lA. /kEbw5 s6hxl1j5 xs9]M6tbsc5b3ix6S5 x7m d[?y8q5gu4 wfxMic3lt4 bms8z6X9oxJ6 SJ6 ur8i6nsix3m5. s6hxl2 xg6bsiz @!& L/hr/unit xqic3ix6S5 b]6t9lQ5.]s7mdt5 wosc6bsiq5 y5tDw5gj5]Sj5 ns/symJ4]3ix6S5 x2dt4f5.

gxXw5 xg6bsQxc3ix6S5 nsy0Jbslt4 x2dtosDbslt[l, Wlx6gu4 @))3_u nN/sQx6X5 nN/six6g6. ybm5 SMC6bsJ5 NlNw6bsymK5 yK9osJu kNoEi3j5 cspnMs6gi5. xbsy6 x2dt5 ciQ/z]i8q2S5 x7m WoExaix8qM6 xyq5 ciQ/sJ5 WoExaMs6t8NQ5. bmw8i4 Sx3E/3=symJ5 sx1Nz]i2S5 b1]n3=sJ2 m3D4 u5b3=s2 ciQ/z]i2S5.

Wzh5 SxE/6bsifw5, m3D4 x4gfc3=sJ4, woi3fw5 SxE/6bsifw5 x7m sz]bdbsif5 nN/six6S5 XX5t0Jbslt4 s/C1ix3=1u xsMbsJi4. m3D4 n=4nc6g5 SxE/6bsifw5 ciQ/z]i8ix6S5 s/C1ix3=s2; d[?y8q5gu4 n=4no4 Sx3E/6ymJ2 xf8iz]i8ix6S6 x7m WoEc3=sJ2. x4gfw5 s/c5 i[Qxi sx1Nzil SxE/6bsymJ2 b]n]i8ix6S5. SxE/6bsymJ5 vtms6g5 i[Qxi s4fwzJ2 wic3lt4 x7m xg6bsix8q5g5 vtbslt4 vN1Nzi WoExc3=s2.

s/ci5 yt]Mi WoExaJ5 XXbsiq5 XXbsix6S5 by3u i[Qxi WoExc3=sJ2. wmsisJ6, h2]lbc6g6 XX5t0Jbsix6S6 bmw8i WoExaJi s/ci yt]Mi nebsJi5 WoExc3=1u5. x4bfw5 wmw5 xg6bsJ5 XX5t}=1j5 f=/six6S5. n3?siz vtbs?9oxli]S6yym}=1j5 wm3l WoEc3=1u5 f=/sc5bExc3ix6S6 x3]Ab]n6. f=/sJ6]b4 C3_j5 i[Qxi rsE5 o4 x7m b[Kz5bw8N6 f=/six6S6. wm6 f=/sJ6 mo4bcExc3ix6S6 WoExaJ2 Mwnzi4 mo4bsQxc6gi4, wMslQ5 slExN6]gQxc8q tbsiq5 wcl1k5.

wl]KDbcs6g5, SxE/6ymJ5 wl5g6ymJ5 wm6hw5 XX5tJbsJ5 nN/six6S5 vt5t0Jbsix3mb nl7mEx6bsymi4 wm3i4 f=/s8q8iq8i WoExc3=1i5. bei4 *_i4 srsu wm6 dxaix6S6; Sn8z5tA5 xqJ6 wm6 kNu5 f=/six6S5 xs1N6yt9lA G]J8 usZJ4g6H x7m ur8i6nsix6Lt4 xs/4f5 srx4]n4f9l f=/sJ5. nN/symiq5 wms2 XXbs0Jyq5 rs0Jbsix6S5]b2hjz ur5gj5 srs6b6gu wmc3=sJk5. ur5g6 n3?sJ6 bwmsK6 s4fwzli Sx3E/6ymo3ix6g6. wm6 SxE/6ymJu5 w7u6yd8q9LA wclw9l nS8ixDmlQ5, xyxk5 wq3C0JysJ6 nN/six6S6 n3?stbs0Jbsix6g6 @))@_u. xyxk5 wq3CtbsJ6 n3?sisJi4

]k6rsmt5t i x6S6 wms2 l]k6rsm i z i4 wx8 / i x5 WoExE / q5 xyx8a6tbs i x8qM5 wx8 / i xk5]k6r4bsymJ5 nS8 i x6bsym1mb hD6bwom / s2 l i w[o z wms2.

vw=5g5 x4bfk5 wm3j5 WoExcDbs i x6g6 wo / s i x6S6 n l 7mEx6t5t0JbsJ4ns l i wm3 l x4bf5 s l ExND8]i 6bsQxDtQ l i s4 x7m d3=1u5 f=J5 wmw5 gJ3u}=sJ i 5, wZ}=1u5 WoExc3=1u9 l. n l 7mEx6bs0JysJ6 WoExc3=4 ryx i xq3CE / sJ i 5 x4bf i 5 wm3 i 4 WoExc3 i x6S6 x7m n3?s?9oxJ i 4 ne5tJ8N6S6 kN5tx2 mo4b4nq8 i 4 kNo1 i 5 n3?s?9oxJ i 5 n l m i 6n i 5 kNu wm6 xg6bs i z k5 n3?s?9oxJ i.

vJysmw8N3 l t4 bBwsC4f5 k]b8a6t]E8N3 i x6S5 xJ8qb l 4]bq8 i 4. m i C3u s4fw z J6 Wx i 4bs4X5 x4bfw5 s / c5 k]b8a6tDbs i x6S5; WoExaJ6 xg6bsQxcD8]i 3 i x3m5 m i Cs2 w l x i s / C1 i x3=4]W6bs i x6S6 w i sJ3 l k]b8a6t6bs l i. k]b8a6tE i 3j5]s4]gbsJ5 bBwsC4f8 i yK9osJ i s / C1 i x3=1 i wm8]NDtc3 i x6S5 kNs2 WD6g2 WoExa4v8 i 3 i q8 i 4. xg3 i x6S5 whoymJ i 4 xyq8k5 WoExaym i f i 4 k]b8a6tE i 3u4,]h3 l wvt s / C1 i x3= z 5 x7m bw=4 xsMb z 5.

wodyE / sN / 6g5 cspn6bsJ5 s / C1 i x3=sJmJu wMc6S5:

- c z 5b6tbsymJ4f5 xsMbsJ5 WoExc3=4]l W8 s / C1 i 3= z i x7m n=5 vtbsymJ5 srs4f5 x[/6g6bsJ4n5 psE f1u.]b8N g4yCstc6S6 xqDt4nu4 ne5tdp l i m3D1 i 4 s / C1 i x3=1 i 4 v7X i sJ1 i 4 wMs i c6g i 4 bmw8 i 4]N7m4nod2 l Q5 yK i x i W / symMs8q5g5 wodysd / sJ5.
- m i Cs2 w l x i xsMbsJ6 s4fw z 8q5g5]Ns / tA5]N7m8q8 i s / l x3m5.
- s4fw z J6 m i Cs2 w l x i]W6bs}=sQxc6g6 wo3 i f i 4 x4bf i 4 s / c i 4 xq i 6nu4 n=4nu4 emw0JbsN / Ms6g6 m i Cs2 w l x i x1mJ6 xsMbsJ2 s z]bk5 emw i 6nsc5bC / 3m5 xg6ym8q5g i 4 s / c i 4 xf2o]E4g i WoExaJ i.
- w i Q / s]z D8N6g6 WoExc3=s l i s / c i 4 yt]M i 4 w l oc3=s l i bmw8 i 4]Ns / tA5 wv]Dtc3 i 6nsJ8N6g5 x? l usbk9 l.

TITIRAQHIMANILLUANGIT HIVUNNIURNIKKUT

Benachee Aturuminarnikkut Katimajutigut, tamatkiumanikkut nanminirijaat ukuat Tahera-kut Kuaparisat (Tahera) hanajumaplutik aulapkaijumaplutiklu ujarakhiurnikkut ("uuminga Jericho-kutkunnin Ujarakhiurnikkut Piliriarutainnik") haniani tunuani uumap Tahirjuami (Contwoyto Lake) uvani Nunavut Nunanganni (NT). Havalirniaqtut uvani angmaumanighami ujarakhiurnikkut tunuaani Tahirjuap uvani Uataani Qitirmiut. Havingnik qinirhianiaqtut iingujunik (8) tatqiqhiutinik havakpaklutik uvani ukiumi (April-min November-mun) talvalu havaarijauvakluni ukiuq tamaat. Tungungajuq maniraq (ahu 15%-ngujuq ukuat havaarijaujughat) naavitauvangniaqtut hanianut havarviup initurlianni (taijauvaktuq imaatut Tahiq Takijuq (Long Lake); pijumajaungittuttauq (ilikungurniit uvani havagvingni ukuat kihinngurlugit ujakkat qipliqtut) katitiqtaulutik atuqtaujughat tungungajut maniqqat katitiqtauhimajut hunavaluit talvaluunniit aallallu pivvaangnirijumajait tahapkuninga maturarijaghainnik tahapkununga ilijaujughani. Ukiumi apqutiqaqpangniaqtut hikumi akjarvighaannik hunavalungnik ingilrutighainiklu uvunga Jericho-kut initurliannut. Tajja aturutighait ujarakhiurnikkut ukuallu havagvighaat iingujunik (8) ukiunik hivituviqarniaqtuq talvalu havaktiqarlutik ahu 105-ngunikluunniit 175-ngujunikluunniit inungnik (ilaqarlutik havaktiniklu kaantraaliqijuniklu) avvaulutik havaqattaqpaklutik talvani havagvingni. Tahera Kuaparisat-kut qinirhiaqattaqpaktut ujaqqanik qipliqtunik uvani Nunavunmi ukiunik saivanik (7) talvani (atuliqtinnagu 1999 uumaptauq havagvigijaraluangit, Lytton-kunni Ujarakhiuqtit). Una havagvik nalvaarhihimajut qanurliqaak kitittiqarniriinik tungungajunik maniqqanik turhuangannik talvalu qinirhiavalliavangniaqtut turhuaqarnighanik uumap Jericho-kut Piliriarutainni. Tikittumajaat una nalvaarhinikkut kiinaujaqtaarutigijaujughanik ujaqqanik qipliqtunik haniani Jericho-kut talva taimaa uumap havagvighaat hivitunighaannik piniarumik. Kiinaujaqtaarnikkut piqattarniarumik piliriarutinin tutquqtuqtauqattarlutik nalvaarhiurnikkullu havaagharhiurnikkullu pivangniaqtut.

Iniqarniat: 350 kilometres-nik ungahingniat SW of Iqaluktuuttiaq (Cambridge Bay) (NT)
420 kilometres-nik ungahingniat NNE of Yalunaimi, NWT talvalu 170 km-nik ungahingniat tunuanni Ekati Ujarakhiuqtit talvalu Diavik-kunni Ujarakhiuqtit uvani Lac de Gras, NWT
Jericho-kut Initurlianni: 25 kilometres-nik NNW ukunani Lupin-kut Ujarakhiurvianni
65°59'50" N Takitingani, 111°28'30" W Hiliktinganik

Pijumajaat Havauhighaannik:
Iinik-ukiunik havauhiqarniaqtut tajja talvani havauhighaanni.
Angmaumanikkut ujarakhiurvighaat Ukiunganni 2002-min 2005-mun ukuninga 330,000 tonnes-ngujunik uqumaitkutaqarlutik ukiuq atauhirmit tamaat.
Naunaiqtauhimajut aturnirighat: 2.52 million tonnes-ngunik uqumaitkutaqarlutik piqattarlutik 1.25-ngujunik uqumaitkutaqaqtut ukuninga akiqarniqarhutik Amialikat akiqarniriinik $76 taalaujumik atauhiq uqumaitkutaannik (Akunnganni Ittunik) talvalu Amialikani $60 taalamik uqumaitkutaannik (aallaniklu ittunik).
Havagvighat uvani initurlighaanni.
Atuqtaghaat ukunani ataani ujarakhiurnikkut Ukiuni uvani 2006-min 2008-mun.
Havaarijaghaat kihiani uvani Tununganni Ittup haviliqiniini Ukiunganni uvani 2009-min 2010-mun.
Haviliqiniup havauhighaat ukuningat Qirhuumajumik Akunnganni Aviktirijauhimaniit (DMS-kunnik taijauvaktut) aktiniqarlutik kipingniinik-ahivarniriinik 1.15 mm-ngujunik aktingniqarniriinnik (tajja illutik).

Qipliqtunik Kititiriinik ukunani initurliqarniriinni havagviup.
Havaktauhimajut tungungaujut maniqqat iqqarvighaat initurlini: katitirhimaniini igitauhimajut (-8 min +1 mm-mun) talvalu katitirhimaniinik ukuninga pijjariktunik (-1mm).

Tikittuminarniriillu Aullaarutiqattarniillu

1200 m takitiqarnilik milvingmik uvani Jericho-kutkunni
Puktatiqarniriillu sikiiqarniriillu tingmijjat tikittuminaqtaat uvunga Carat Lakemut (piqattarniqarniini)
Hikumi milvighaanni ukuat tingmijjat Hercules-kut ukiumi uvani Carat Lake-kunni pijaaqaqqat
Ukiumi Apqutiqarnighaanni Yalunai-min Jericho-mun (qitiqquanni January-min qitiqquannut March-mun)
Hilami tamaat apqutit atuqtauvaktughat Contwoyto Lake-min ukununga ujarakhiuqtit initurliinut.

Havaaqarnighat

Ujarakhiuqtit Hanauhighaat:	25-min 60-mun ilagilugit piijaijughanik havaktighat
Havagvighap Hanauhighaat:	20-min 60-mun inungnik
Havauhighaat:	60-min 116-mun inungnit havaktughanik pingahunik ukiunik avatquumajumik ujarakhiurnikkut
	48-ngujunik inungnik malrungnik ukiunik avatquumajumik ujarakhiuqtighat nunap ataani
	40-ngujunik inungnik iinik ukiunik avatquumajumik havaaqaqtughat ukuninga hananikkut
Pijumajaujut Havauhighakkut Aturumajaat:	Ujarakhiurniq: 2x12-nik ikaarninik himmautigiiqattarlutik ubluq atauhiq, saivanik (7) ublunik tamaat (April-min November-mut)
	Havagvighat Igluqpak: 2x12-nik ikaarninik himmautigiiqattarlutik ubluq atauhiq, saivanik (7) ublunik tamaat (ukiuq tamaat)

Malrungnik santinik havakpaklutik, malurningnik santinik havangiqattarlutik pivangniaqtut

Piliriarutip Aturutighaa

Una pijumajaat aturutighaat hapkuninga piqutigilluaqtaghainik:

- Imaqarniriinut Naunaipkutaqarniriillu Nunakullu Aturutighainik ingiqtuutit
- Titirakaffungniit EIS-kutigut Tunihiniinik
- Iniqtiriiqhimajut EIS-kutigut Tunihiniinik
- Havauhighap Angirniriit
- Havauhighap Angirutighait
- Aulapkainighallu Ukiumilu Apqutikkut Akjarutighat
- Igluqpait Hanauhighaat

- Angmaumajumik Ujarakhiurnikkut Havauhighat
- Havakpialirlutik

Tahera-kut tikkuarlugit Dowding Reynard-kullu Havaqataillu (Pty) Ilangannik (DRA) ukunani Kanaganni Africap ingniqiulluarlutik pijauniaqtut, pigilugillu hananikkullu aulapkaijiulluarlutik ukunani ujarakhiuqtit havagvianni igluqpainilu. DRA-kut hanalluarhimajaat ihuarharhimaplunigillu ukuninga 10 ton-ngujunik uqumailrutiqarhutik ikaarnini atauhirmi havagvighami uvani ittumik Lupin-kut Ujarakhiurvianni ukuningalu ingniqinikkut hanauhiannik talvalu qanurliqaak akiqarnighainnik ihivriuqpaghutiklu.

DRA-kut ingniqinikkut munarhilutik pivangniaqtut ukunani piliriarutini hanaliqqata talvalu atuqtittinikkut. Una piliriarutinik hananikkut aulapkaiji tahapkuninga initurliqarniriinnik munarhiuluangniaqtuq. DRA-kut tikkuaqtuiniaqtut Kanatamiuttamik kaantraaliqijughamik igluqpiuqtughamik, havagvighaniklu, uqhurjuallu tutqurvighaannik, qulliliqijughaniklu, tupiqarvighaniklu, imiqtaqattarnighakkullu talvalu tungungajumik maniqqap kuviraqturvighaanniklu. Hananighaat havaarilirumajaat uvani May 2002-mi talvalu iniqtirumaplunigit September 2002-mi.

Havagvighap atuqtittiniqalirumajut uvani October 2002-mi. DRA-kut havaktittivangniaqtut ajuittunik makaninik, hanalrutivalungniklu ingniqijughallu talvalu ajungnaitpat kitulliqaak havaktighat atuqtittinikkut ingniqijitkunnik. Tahera-kunni Kuaparisat havaktighanik qinirhianiaqtut tahapkuninga ilauqataujughanik ukunani atuqtittilihaarnikkut talvalu pilihaarnikkut pinialirumik.

Nuna Logistics-kut ujarakhiurnikkut kaantraaliqijaamingnik ajunngittut tahapkuninga ilihimattiarhutik uvani Nunattiami Nunaanni ukunanilu Nunavunmi. Tahera-kut Kuaparisat tikkuarhilluaqpaktut Nuna Logistics-kunnin ujarakhiurnikkut kaantraaliqijiuplutik ukuninga piliriarutinik. Nuna Logistics-kut iniqtirijut ungavainikkut angijualungmi qipliqtunik ujarakhiurnikkut uvani Nunattiami talvalu tajja atuqtaujughanik initurlinik hannaijaijut ukuninga malruujungnik aallaniktauq ujarakhiurnikkut piliriarutighainnik uvani Nunattiami. Nuna Logistics-kut nuutiqpallianiaqtut ukununga piliriarutighat initurlighaannut uvani February-mi 2002. February-mi atuqtillugu Nuna iniqtiriniaqtut hikumi apqutighaannik ukuningat Lupin-kut Ujarakhiurviannin ukunungat Jericho-kunnut talvalu tupiqturlutik talvunga piniaqtut, uqhurjuallu tutquqtuivighaannik igluqpiurlutik, ajuirhanikkullu, qagaqtautillu tutquqtuivighaannik talvalu hikukkut apqutiliurnikkut pivaklutik uvani initurlighaanni.

Tahera-kut Kuaparisat tikkuarhiniaqtut piliriarutini aulapkaijighaanni ukuningalu piliriarutini munarhijighaanni takuuriqattaqtughamik hanauhighanik, pigilugillu hananikkullu piliriarutip. Kitittijighamin tikkuaqtuiniaqtut uvani 2002-mi kiinaujaqtaarnikkut aulapkaijighaannik ikajurnikkut piliriarutillu havauhighakkullu talvuuna. Tahera-kut havaktighamingnik uvani hanavingni 2002-mi talvalu atuqtittijumaplutik ajuirhanikkut pijumajut havagvighaat atuliqtinnagu. Tahera-kut Hilarjualiqinikkut Atanailluangat ihivriurhiqattaqpangniaqtuq hilarjuakkutigut qanurittaaghaat talvalu aulapkaivaklutik hilarjuakkutigit uqautigiqattarniriinnik hanavalliajjutaannik pilirumik. Munarhilluarniaqtuq iniqtirutighaannik uumap EIS-kut talvalu akulliuqqattarluni havaaqarniaqtuq tahapkuallu ataniulluaqtut piliriarutit atuqpallialirumik naunaipkutainnik.

Una iliktiqtauhimajuq ujarakhiurvik pualrighuqtauniaqtuq takunnarhitilutik tungungajunik maniqqap haviinik pipkarumaplutik qilaminnuaq havauhighaannik uumap havagvighap. Iliktiritillugit hamna angmaumajuq ujarakhiurvik talvuuna kiinaujaqtaarutiqalaittut havanngitkumik. Haffumap itingniqarniannik ujarakhiurviup, qanurliqaak uavaqtuivangniaqtut kitunikliqaak iniqtiritaarumik hapkuninga ilikkuuqtunik iliktiritaarhimajunik ujarakhiurvingnin. Haffumap hiliktingit uavaqtaarhimajut hapkuninga imaatut ilijauvangniaqtut ujarakhiurnikkut hiliktigivaktainnik, hanalrutigijainiklu pijjutighaannik, pijumajaujullu haviliqinikkut aturutighaannik, aanniqtailinighakkullu, akiqarnighakkullu, ajurnaitqijaghakkullu, talvalu aallaniktauq kitunikliqaak pijumajaujunik.

Ujarakhiurutighaat ujarakhiuqpangniaqtut angmaumaluqarnikkut pilutik talvalu imaatullu ataaniittunik ujarakhiuqpaklutik angmaumaniqarlutik talvaluunniit aktikkutaannik pivaklutik ulivighaqarniittumik. Angijualuillu mikitqijaillu naunaipkutait haviqarnikkut ukuninga F6-kunnin talvalu F4N-kunnin ilanganni (akunngannilu tunuanilu ittunik) katitirijauniaqtut qilaminnuaq tununannilu kivallianilu uumap havagviup. Ilakuittauq hakugikpallaangittut tungungajut maniqqat katitiqtauniaqtut hanianut ujarakhiurviup, ilikkuurlutik hapkuningat hakugiktaulungnit talvalu iqqakuurviuvaktunit hunavalungnit, qakugunnguqqalliqaak qipliqtut ujaqqat akittuqpallianiarumik talvuuna atuqtaujungnaqtuttauq kiinaujaqtaarutighakkut. Talvattauq ilangannik uvangat kananarmit ittumin piniaqtut talvalu havaarijauluni uvani havagvianni ihivriuttiarlugit tahapkuninga pijuminaqtunik akitujuniklu qipliqtunik ujaqqanik. Ujaqqat iqqaqtauvaktut, hapkuninga imaqaqtunik ujaqqanik naptujunik, iqqaqtauvangniaqtut ukununga iqqakuurviannut 1-mulluunniit 2-mulluunniit uvani ittumik tunuanni uataani talvalu kananaanni ujarakhiurviup (takulugu Nunaujaq A).

Ataanin maniqqap pijuminarhiniaqtut imaatut ukkuaqarviqarniannin ujarakhiurviup. Haviillu iqqakuurviillu hunavalungnit ukuninga ataaniittunik maniqqap ukununga ilijauvangniaqtut hakugingniringnut haviqarniringnut katitirviannut ukunungalu iqqakuurvingnut hunavalungnit.. Iqqaqtauhimajullu ataaniittunik hapkuninga qailluaqpangniaqtut iqqaqtauhimajunillu piliriarutauvaktuninlu.

Una haviliqinikkut havagviit, tamajaqaqtinniaraat South Africa-min, hanajauhimagiingniaqtut talvalu aullaqtitauvaklutik initurlighanut puuriqhimalutik. Ajurnaitpat angijualuit hanalrutit tigujaujariiqhimaplutik pijauhimajut talva kihimi ikajuqtiqarlutik Kanatami pijaujughat. Hamna havik havaarijauluni pijauniaqtuq aturlutik hapkuninga qipliqtunik ujaraliqinikkut aturniriinik ukuningalu titirauhikkut atuqpaklutik hapkuninga haviliqinikkut havaarivagainnik ukuninga tungungajunik maniqqanik haviannik havaktautaarhimajunin uvani ataaniittunik pihimajut ukunani Tahera-kut 10 tonne-ngujunik uqumaitkutiqaqtunik atauhirmit ikaarnimit uvani havarvianni ittuq Lupin-kut Ujarakhiurvianni initurliqaqtut.

Igluqpakhainik ujarakhiuqtut ihuarhaqtauniaqtut ujarakhiuqtitkut kaantraaliqijit. Igluqpakhat nappaqtiqtauniaqtut talvani initurlianni ujarakhiuqtit. Najugaghallu iggavighallu hanauhiillu havauhighaillu naammagijaujunik Havaktiit Akiliuhiarutighakkut Katutjiqatigiingnit, Nunavunmi Hilarjualirijitkut Pihimattiarniriillu ukuallu Nunarjuami Qaptirinikkut Naunaipkutaita maligainnut. Una najugaghap igluqpaa qaptiritiqaqtut talvani igluqpaup ilanganni. Iggaviup qulaangani talvalu qingaghaini hananiaqtut tahapkuninga atataqaqtumik qaptirutiqaqtunik ilaqarniaqtut.

Paniumajut qaptirutighait ilijauniaqtut tamaita ukkujjat haniani. Tamaita ukkujjat naunaipkutaqarniaqtut anivighat naunaipkutainni, hilamuuqqattaqtughallu talvalu naqittautiqarlutik hanajauniaqtut (nalaumajumik qitqanut natiup ilijauniaqtuq). Paajjituqtunik qulliqarniaqtut tamainnut najugaghat iggavighaplu ilanganni.

Imiqtaqpaklutiklu uvangat Carat Lake-min turhujjakkut hanajauhimajumit ahu ungahiktigijumin 170 m-ngujumin atpanut ilijauluni hikumit ahirulaitkaangani. Immat turhuakkut havagvingnullu hinigvighanullu pijauvangniaqtut hamangat tahirmit. Ahu 30 m³/ubluq tamaat pijumaniaqtut.

Uqhurjuaqarvighaat hanajauniaqtuq tamaita uqhurjuat atuqtaghaat talvalu hanianiinniaqtuq haffumap havagvighap. Hamna uqhurjuaqarvighat tuantinik (20) qattarjungnik ilijauniaqtut uvunga najugaghainut. Talvaluttauq, hapkuat tajja ittut naingujut (9) qattarjuit atuqtauvaktut qinirhiajjutigivaghugit ataaniittunik maniqqap hapkuat nuutauniaqtut hamunga uqhurjuaqarvingnut. Hamna uqhurjuaqarvighaq imaqaqpangniaqtut 110%-ngujunik angitqijaujunin qattarjungnin. Hamna hilataani tunmiraghaat puuqarniaqtut ahivalaaqtumik, uqhurjuatkunnin-aturuminaqtunin maniqqamiilaaqtumin kalikumik.

Atataghaanik ukununga havagvingnut najuqtauniaqtut pingahunik 1300 kW-ngujunik, 60 Hz 600 V-ngujunik huanngaghautinik (2 + 1 himmautighat) uumingalu 300 kW upalungnaqqat huanngaghautighaat tamaita ataaniittut 5 t-ngujut pualritiligjuap. Himmautighaattauq malruujuk 800 KW-ngujunik huanngaghautingnik aturniaqtut, atauhiq uvani illuni ujarakhiuqtit hannavianni atauhirlu uvani illuni Iqqakuurvianni 1 uumap atuqtaghaa hiqupluijup, aturumaniarumikku. Huanngaghautit uqhurjuaqturniaqtut talvalu kajumiittumin-uqhurjuaq ikulaniaqtuq amigaitpallaanik pujuuqunggittugu. Uqhurjuaqtuqpangniaqtut hapkuat 217 L/ikaarniq/ilikkut tatahimaluni. Qulliliqinikkuttauq hanajauniaqtut hilamiutaghami Teck-kunnin alrujaqarlutik hauhimaniaqtunik apqutit hilataanni.

Hiquplurhimajunik qulaaniitughani apqutini aturniaqtut, uvani atuqpallaarniaqtaa 2002-mi hanalingmigumik. Hitamaujunik hitinik ilitturijut maniqqami qinirhiavakkamik qangaliqaak. Atauhiq hanianiingittuq apqutit talvalu aktuqtaulimaittut kihimi ajurhalirumik aturniaqtaat tahamna. Tamaita hitiit tunuani ittut qinirhiajut initurliannin talvalu malruujuk ilanganni ittut milviup.

Pingahuujut haviup katitirvigijaat, malruujuk iqqakuurviik, aturuminaittullu katitirvighaannik talvalu amigaittunik katitirvighaallu hanajauniaqtut katitirvigijaghaat ujarakhiuqtit pijumanngitamingnik. Malruk haviup katitirvigijaghaat ilijauniaqtut haniani havagviup; mikijuqarniittauq katitirvigijaghaat ilijauniaqtuq akunnganni pualrikhuqtauniillu havagviuplu. Iqqakuuqtaujuttauq ungahiaqtumi inniaqtut tahaffumap pualrikhurviup haniani. Amigaittunin katitirvighattauq ungahiaqtumin uvangat pualrikhuqtauvingnit inniaqtut talvalu maniittut tungungajut maniqqat katitirvighaat inniaqtut kivataani haffumap havagviup.

Tungungajut maniqqat tutqurvighaannik hanajauniaqtut tahiqqamik hapurviliurlutik akiani haffumap havagviup. Hamna ataaniittuq, hapurvinik ilaurijauhimmaaqtut hapkuninga ujaqqanik tigummijungnaqtut haffumap havagviup ilanganni. Iqqaqtauhimajuttauq imaqarvighaat tahamunga kuvijauvangniaqtut. Imaqarniriit katitiqtauhimajut talvani immallu uvangat havagvingnit kuvijauqattaqtughat ukiuq tamaat. Kuvijauvangniaqtut uvunga Tahirmut C3 qilaminnuaq ungahiaqtumun uvangat Carat Lake-min talvalu aajjikkiiktunin kuvijaaqaqpaklutik talvani ataanin. Kuvijauhimajut immat Piliriarutikkut laisiqaqtitaunikkut pivaktughajut, ilagilugillu hapkuat tuqunaqangittuugaluit iqalungnut.

Tunmiraghaniklu, attingnirighaniklu halumailrullu tahighannuangannik hanajauniaqtut katitirinighaannik immat havautituqtitaulraarlutik kuvijauvangniaqtut initurlinit. Iingujunik (8) tatqiqhiutinik tahamna imaq qiqumaniaqtuq; angitqijamin imaqarninik nunami kuvijauvangniaqtut talvani kuvinikkut nakuutqijaanni (ahu June-mi) talvalu ikitqijattauq imaqarniriit kuvijauvangniaqtut aujami ukiaghalihaamilu. Hanauhighaallu imaqarniriit atuqtaghainik talva taimaatut aturuminaqtunik tahapkuninga mikijunik Ukiuqtaqtumi turhujjuhigharnikkut pivangniaqtut. Una mikijuq qurluangninga akianungauvangniaqtuq uvunga angmaumajumut pualrikhuqtauhimanianut. Imaqarniriit talvunngaungittaililugit talvalu aanniqtaililugillu iqalukhiurniriit, ahinungarvighaannik hananiaqtut uvani 2002-mi. Hamna ahinungauvighaat qurluaqtitauttiarlutik pijauvangniaqtut talvalu imaqarniqattiarlutik hanauhiqarniaqtut qurluangniriit tammaqtaililugit.

Kaiviqattaqtumiklu imarluktunin havauhighaannik illiriniaqtut havautituqattarvighaat imarluktut anarviillu kuvviannin halummarhivaklutik igluqpaqarninillu, iggavingnillu havagvingnillu. Hamna havautiqarnighaat kuvviviup imarluktunin talvalu nakuutqijauplutik hamani Nunattiap maligarivagainnin atuqtauvaktut kuvinikkut talvuuna.

Hivumuungnikkut nanminiqarumalutik pivangniaqtut Tahera-kut qanurliqaak angijualuugaluarlutik. Tahamna angmaumajuq pualrikhuqtauviit iniqqat iqqaqtauhimajut ujaqqanik nanminiiqtiginiaqtaat; tahapkuallu igluqpait atuqtaulimaiqtut ataani ujarakhiurnikkut ahivaqtauvangniaqtut talvalu tahamna initurliit nanminiiqtiginiaqtaat. Nanminiiqtittinikkut pivangniaqtut Tahera-kut tahapkunani ukiuni ujarakhiurnikkut nauttianik nauttiffaaqpangniaqtut. Aturutighaallu maliglugit aallat ilihimajut pivangniaqtut, ukunanut Ekati-kunni Ujarakhiuqtit Diavik-kullu havauhiannik.

Kitulliqaak ihivriurutigiplugit piju majamingnit ujarakhiurnikkut hapkuninga ilaqaqtut:

- Hilakkuurutikkut havakpangniaqtut qanurliqaak havagvighamik illiriniaqtut uvani Lupin-kut Ujarakhiurvianni talvalu haviit katitirivighaanni uvani ukiumi akjaqattaqtauvaktughat ukunani Jericho-kut initurlianni. Hamna atuqtaujumajauniit angirutaulutik akunnganni ukuak malruujuk ujarakhiuqtiik talva naammaguhuklutik inmikkut pijughak talva taimaatut pihimanngitkaluaramik.

- Tamatkiumajumik ataaniittuniktauq havaaqanngitkaluaramik talvalu angmaumajumik pualriqhuqtauviqangitkaluarmata talvuuna kiinaujaqtungitput taimaatut.

- Tamatkiumajumik angmaumajumin pualriqhuqtauhimaniinik ahivainiarungnautigijagharaluanginnin iqqaqtauhimajut ujaqqanik talvalu haviinnarnik ahu imaqarniarungnarhijunik talvani ilagilunigit angmaumaniit pualriqhuqtauhimaniit-ataani havaarijauniarungnarhijut.

- Kitumulliqaak najugaqarniarungnarhijut havagviillu tahapkuallu tungungajut maniqqap tutquqtuivighaat talva tamaita akittuutivallaarutiginiarungnarhijaat hilarjualiqinikkullu akittuutiginiarungnarhijaraluangillu.

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1.0 INTRODUCTION

1.1 CORPORATE IDENTIFICATION AND PROJECT SUMMARY

Benachee Resources Inc., a wholly owned subsidiary of Tahera Corporation (Tahera) plans to construct and operate a diamond mine ("the Jericho Diamond Project") near the north end of Contwoyto Lake in Nunavut Territory. Operations will commence with an open pit mine at Carat Lake, northwest of Contwoyto Lake in West Kitikmeot (Figure 1.1). Ore will be mined nine months of the year (April through December) and processed year round. Fine processed kimberlite (roughly 15% of the plant feed) will be pumped to a lake near the plant site (local name Long Lake). Coarse rejects (the remainder of the plant feed, except diamonds) will be stockpiled for use as processed kimberlite impoundment cover material or for other reclamation purposes where a final cover of overburden is needed. During winter months an ice road will be used to transport materials and supplies to the Jericho site. With current resources the mine and processing plant will have an 8-year life and employ a total of approximately 110 to 175 people (including employees and contractors); approximately half that number will be on site at any rotation. Tahera Corporation has actively explored for diamonds in Nunavut for the past seven years (prior to 1999, as its predecessor company, Lytton Minerals). The company has identified a number of prospective kimberlite pipes and will continue to explore and develop prospective pipes throughout the life of the Jericho Project. The goal is to find additional economic diamond deposits in the general area of Jericho that would extend the life of the processing plant. Cash flow from the project will be re-invested into exploration and development programs.

CORPORATE DATA

Project Name: Jericho Diamond Project, Nunavut Territory

Owner:

Toronto Office
Benachee Resources Inc.
803 121 Richmond Street West
Toronto. Ontario
M5H 2K1

Benachee Resources Inc. is a wholly owned subsidiary of Tahera Corporation whose corporate office is located as above.

Contact:

Mr. Greg Missal, VP Nunavut Affairs
803 121 Richmond Street West
Toronto. Ontario
M5H 2K1
Phone: (416) 777-1998

Mr. Grant Ewing, VP Investor Relations and Corporate Development
803 121 Richmond Street West
Toronto. Ontario

M5H 2K1
Phone: (416) 777-1998

Tahera Corporation, and its predecessor company Lytton Minerals, conducted diamond exploration in Nunavut and Northwest Territories beginning in 1992. The company carried out mini-bulk sampling at a number of its kimberlite pipes in the late 1990s and conducted an underground mining pilot project at the Jericho site in 1997/98 to extract 15,000 tonnes of kimberlite from the JD-01 kimberlite orebody. Throughout that time no violations of permits and licenses were incurred and required reports were provided to government regulators.

Tahera has experienced no lost-time accidents during exploration programs. A contractor, responsible for job safety for its employees, undertook the bulk sampling. No major accidents occurred during the bulk sampling. A processing plant located at the Lupin Mine site was operated for two years accident-free.

PROJECT DETAILS

Location: 350 kilometres SW of Ikaluktutiak (Cambridge Bay) (Nunavut)
420 kilometres NNE of Yellowknife, NWT and 170 km north of the EKATI™ Diamond Mine and Diavik Diamond Project at Lac de Gras, NWT
Jericho Site: 25 kilometres NNW of Lupin Mine
65°59'50" N Latitude, 111°28'30" W Longitude

Economic Minerals: Gem, near-gem and industrial grade diamonds

Proposed Development:

Eight-year operating life on currently proven reserves.
Open pit mine Years 1 to 4 at 330,000 tonnes per year.
Processing plant on site.
Implementation of underground mining at Years 5 to 6.
Processing only of North Lobe ore Years 7 to 8.
Haul supplies via extension of Lupin winter road to site during mid January to mid April period.
Ore treatment by Dense Medium Separation (DMS) with cut-off size of 1.15 millimeters particle size (at present).
Diamond Sorting at on-site facility.
Processed kimberlite disposal on site: stockpile for coarse rejects (-8 to +1 millimeters) and impoundment for fines (-1millemeter).

ACCESS AND TRANSPORTATION

1200 meter long airstrip at Jericho
Float and ski plane access on Carat Lake (extra-ordinary)
Ice strip for Hercules aircraft in winter on Carat Lake if required
Winter Road Access from Yellowknife to Jericho (mid January to mid March)
All weather road from Contwoyto Lake to the mine site.

UTILITIES

Power Supply: On-site installed Diesel generators at the Jericho site
Water Supply: Carat Lake
Accommodation for Approximately 100 people and offices on site at mine camp; 50 people at exploration camp during construction.

EMPLOYMENT

Mine Construction:	25 to 60 including pre-stripping work force
Plant Construction:	20 to 60 people
Operations:	57 to 116 people over 3 years for open pit mining 48 people over 2 years for underground mining 40 people over 8 years for processing
Proposed Work Schedule:	Mining: 2x12-hour shifts/day, 7 days /week (April through December) Process Plant: 2x12-hour shifts/day, 7 days per week (year round)

Two week on, two week off roster (construction may vary from this rotation)

PROJECT SCHEDULE

The proposed schedule has the following key milestones:

• Permit and Authorization Applications	January 2001
• Draft EIS Submission	January, 2001
• Prehearing Meetings	May & June 2001
• Final EIS Submission	January 2003
• Final Public Hearings	April 2003
• Project Approval (NIRB)	Early June 2003
• Approval fromMinister	Late July 2003
• Mobilization and Winter Road Haul	January to March 2004
• Site and Facilities Construction	March – December 2004
• Open Pit Development	March - May 2004
• Full Production	First Quarter 2005

1.2 CORPORATE ENVIRONMENTAL STATEMENT

It is Tahera’s policy to achieve a high standard of environmental care in conducting its business as a resource company contributing to society’s material needs. Tahera’s approach to environmental management seeks continuous improvement in performance by taking account of evolving knowledge and community expectations.

Specifically, it is Tahera’s policy to:

- Comply with all applicable laws, regulations and standards; uphold the spirit of the law; and where laws do not adequately protect the environment, apply standards that minimize any adverse environmental impacts resulting from its operations;
- Communicate openly with government and the community on environmental issues, and contribute to the development of policies, legislation and regulations that may affect Tahera;
- Ensure that its employees and suppliers of goods and services are informed about this policy and are aware of their environmental responsibilities in relation to Tahera's operations;
- Ensure that it has management systems to identify, control, and monitor environmental risks arising from its operations and to prevent environmental impacts prior to their occurrence;
- Conduct research and establish programs to conserve resources, minimize wastes, improve processes and protect the environment;
- Take appropriate corrective actions should unexpected environmental impacts occur. Appropriate actions will be taken to prevent reoccurrence of such unexpected impacts.

For the Jericho Diamond Project, Tahera's environmental policy will apply, where appropriate, to all its contractors. Environmental clauses outlining contractor responsibilities will be included in contracts for the Jericho Mine.

1.3 REGULATORY REQUIREMENTS

1.3.1 Permits and Licences

The operating mine will require a Water Licence issued under the authority of the Nunavut Water Board (NWB).

A Mining Lease and Land Lease will also be required. Mining leases are issued by Nunavut Tunngavik Incorporated (NTI) or DIAND, depending on land ownership.

The Jericho site is on both Inuit owned and Crown lands. Land leases will be required: for Inuit owned lands from Kitikmeot Inuit Association (KIA), and for Crown land, from Department of Indian and Northern Affairs (DIAND).

Required are the following:

- Section 35 authorization for alteration of fish habitat for construction of the C1 diversion, construction of a water intake causeway in Carat Lake, and use of Long Lake and Stream C3;
- Section 32 authorization for dewatering of Long Lake;
- Section 30 approval for construction of the water intake; and

- Section 20 and 22 approvals to alter stream flows, principally for Stream C3, since natural flows in Stream C1 will be maintained by the diversion.

In addition to the licences and permits listed above, the project will require a quarry permit, and explosives transport, manufacture, and storage permits. As well, permits may be required for camp incinerators and sewage disposal (likely incorporated in the Project Water Licence). Depending on the nature and extent of the operational monitoring program, permits may be required to conduct some of the monitoring activities, e.g. a Fisheries Act Section 52 approval for collection of fish. Table 1.1 provides a summary of key governing legislation that will apply to the Jericho Project. The table is not inclusive but contains the principal acts and regulations pertinent to the Project. Licences, permits and leases will have site-specific conditions attached that will specifically regulate effluents and emissions from the Jericho Diamond Mine.

1.3.2 Authorizations For Exploration Operations At Jericho

The following is a summary of currently held permits and licences to authorize exploration activities at Jericho.

Water License NWBJER9801
Issued By: Nunavut Water Board, Gjoa Haven, Nunavut

Includes: Water use and waste disposal in Nunavut associated with; prospecting, surface drilling, on ice drilling, operation of the bulk sample plant, domestic use and associated activities, environmental baseline data collection, demobilization of equipment, and underground bulk sampling.

Expiry Date: December 31, 2002 **(Applied for extension)**

Crown Land Use Permit N1999C0062
Issued By: Department of Indian and Northern Development, Yellowknife, Northwest Territories

Includes: Activities at the Jericho site such as; bulk sampling, quarrying, airstrip, winter roads, access roads, portal site, fuel tank farm, and laydown areas.

Expiry Date: August 31, 2001 **(Reapplied for)**

Crown Land Use Permit N1998C0874
Issued By: Department of Indian and Northern Development, Yellowknife, Northwest Territories

Includes: Activities associated with the Jericho (Carat) camp (the area inside of electrified fence), and all associated exploration activities on NTS 76E and 76L. Exploration activities include drilling, mapping, prospecting, sampling, and geophysics.

Expiry Date: June 10, 2001 **(Reapplied for)**

Inuit Land Use Permit KTP98C027 (Level II)
Issued By: Kitikmeot Inuit Association, Kugluktuk, Nunavut

Includes: Activities located on Inuit land parcels CO-05, CO-07, and CO-08 including; prospecting, sampling, ground and airborne geophysics, diamond drilling, environmental baseline studies, and all support work.

Expiry Date: September 30, 2002 (Reapplied for)

Inuit Land Use Permit KTP98C036 (Level III)
Issued By: Kitikmeot Inuit Association, Kugluktuk, Nunavut

Includes: Activities located on Inuit Owned Land Parcels CO-07, and CO-08 including; staking, prospecting, exploration work, diamond drilling, bulk fuel storage, trenching, and winter road construction.

Expiry Date: September 30, 2003

2.0 GEOLOGY AND RESOURCES

Upon completion of the geologic model, an analysis of diamond distribution was completed on the drill core samples and underground bulk sample. This resulted in delineation of various geological domains (areas of the deposit with distinct grade distribution and physical characteristics). Within each geological domain, an evaluation of spatial distribution of diamonds was completed in order to select the most appropriate method of estimating the resource.

SRK used the following approach to estimate resources for the Jericho kimberlite:

- Evaluation of Data Quality
- Creation of Geological Model
- Diamond Distribution Analysis
- Grade Estimation
- Diamond Value Assessment
- Resource Classification

2.1 REGIONAL GEOLOGY

The Jericho Diamond Project is located in the northern portion of Slave Structural Province of the Canadian Shield, north of Contwoyto Lake within the Contwoyto-Itchen Lake Region (Figures 2.1 and 2.2). The Slave Structural Province is an Archean granite–greenstone terrain containing belts of 2.67 to 2.70 billion year old metasedimentary and metavolcanic rocks of the Yellowknife Super Group (Bowie, 1994). These rocks were intruded extensively by synvolcanic to post-volcanic granitic plutons dated between 2.58 and 2.63 billion years. The Contwoyto Lake area is transected by a series of NE-NNE and NW trending linear features, indicated by the distribution and shape of a number of small lakes.

Five known kimberlite occurrences have intruded Archean granodiorites near the Jericho Project area. Jericho (JD-1) and JD–2 are located on the south shore of Carat Lake; JD-3 is located under a lake 7km to the southwest. The Tahera-1 kimberlite is located almost 6km due west of the Jericho pipe and the OD Dyke kimberlite is located 8km south of Jericho. An age date of approximately 170 million years has been determined for the Jericho, JD-2 and JD–3 kimberlites (Heaman et al., 1997); the remaining two kimberlites have not been dated. The only other Tahera owned kimberlite in the area is the Contwoyto-1 kimberlite, 40km southeast of Jericho. Although only Jericho kimberlite is part of the current mine plan, after further exploration Tahera may determine that another kimberlite (current or new discovery) is economic and may wish to ammend it to the mine plan. Tahera acknowledges that a new EIS, relevant permits and authorization would have to be submitted for environmental review prior to any significant changes in the mine plan.

The closest competitor kimberlites to the Jericho Project are DeBeers Canada Exploration Inc.'s Muskox, Rush, and Peregrine kimberlites, 15-25km west of Jericho. Within the northern Slave (north of 65.5 degrees), more than 20 kimberlites have been discovered and more than 10 companies are now conducting exploration. There are no diamond mines operating in the northern Slave Craton. In fact, within this area, only the Jericho kimberlite has completed a feasibility study and is being considered for regulatory approval. Other kimberlites in this area are at an early exploration stage; economic viability of the deposits are not yet known.

The only operating mine within 100km of Jericho is the Lupin Gold mine, operated by Kinross Gold through their merger with Echo Bay. The mine is located within the Archean metasediments and Iron Formations of the Yellowknife Supergroup. Further gold exploration potential is possible within this area, although currently very little gold exploration is underway.

Kimberlite is an ultrabasic igneous rock that typically occurs as carrot-shaped vertical pipes; less commonly occurs as dikes; and rarely occurs as sills. It has a distinctly inequigranular texture resulting from the presence of macrocrysts and phenocrysts set in a finer grained matrix. Typically, the matrix consists of olivine and several other minerals with carbonates (commonly calcite), serpentine, and ilmenite being the most common. The macrocrysts and phenocrysts are ferro-magnesian minerals such as olivine, various garnets, clinopyroxene and orthopyroxene. These macrocrysts and relatively early formed matrix minerals are commonly altered by serpentinization and carbonatization. Kimberlite usually contains inclusions of upper mantle-derived ultramafic rocks (i.e. fragments of material sampled from depths of 150 to 200 kilometers (km) below the earth's surface) and variable quantities of crustal rock fragments.

Typically three textural varieties of kimberlite, each associated with a particular regime and style of emplacement are the most common:

- crater facies (near the original land surface)
- diatreme facies (main body of kimberlite that links the surface crater to the root zone), and
- the hypabyssal facies (root zones, dikes, or sills).

2.2 DEPOSIT GEOLOGY

2.2.1 Jericho (JD-1) Kimberlite

The land-based Jericho kimberlite is located within a shallow depression south of Carat Lake and is masked completely by some 10 to 15 meters (m) of glacial till.

The Jericho kimberlite has a roughly NNW trending elliptical body with a length of approximately 300-m and a width of up to 100-m. It has been defined to a depth of 360 m by drilling (Figure 2.3).

The Jericho kimberlite was formed from multiple emplacement phases or events with each event involving a kimberlite of different composition. Three main kimberlite types (facies) corresponding to three major intrusive phases have been identified by differing color, mineralogy, serpentinization, texture and macrocryst count.

The three phases (facies) comprise a precursor dike (JDF2), and three diatreme intrusive stages (JDF4North, JDF4South, and JDF6). The relative order of emplacement of these phases has been interpreted from pipe morphology, contact relationships, and the occurrence of fragments of one phase within another.

- The first intrusive event, the precursor JDF2 forms a dike-like body that extends from the south end of the kimberlite body more than 450 m north to the satellite JD-2 kimberlite body. It has been described as a hypabyssal kimberlite or kimberlite breccia, and is characterized by a gray to dark gray color with a finely crystalline, calcite and oxide-rich groundmass, supporting brown-green olivine macrocrysts of 1 mm to 2 centimeters (cm) in size. Garnet, chrome diopside, and ilmenite, as well as mantle derived xenoliths, make up less than 5% of the rock volume. The kimberlite dike occurs on the east side of the other Jericho kimberlite phases with widths between 5 and 20 m.
- The second intrusive event, the JDF4, forms a rounded lobe in the north (4North (4N)), and an irregular shaped lobe in the south (4South (4S)). The kimberlite comprising this phase is heavily serpentinized, dark olive green, and has a variably fragmental texture. It lacks the finely crystalline calcite and oxide-rich groundmass of the precursor JDF2 dike. This unit has a more fragmental appearance than JDF2, with 15 to 20% olivine macrocrysts; 5 to 10% mantle derived xenoliths, including eclogites, xenocrysts and macrocrysts of garnet, chrome diopside, phlogopite, and ilmenite; and 5 to 10% crustal xenoliths, including granodiorites, diabase, and limestone.
- The third intrusive event, the JDF6 phase forms a central steep-walled pipe filled with light to medium blue-green kimberlite. The macrocrysts are unserpentinized, although the ground mass is mostly serpentine. This phase has the most fragmental texture. It is a clast supported tuffistic kimberlite breccia with 40 to 60% olivine macrocrysts, 5 to 10% crustal derived xenoliths of limestone and granodiorite, and up to 5% mantle derived materials. Based on contact relationships and fragments of older phases carried within it, this phase has apparently cut through all the older phases. West of this central lobe lies a sill like body of JDF6 kimberlite varying between 5 and 20 m in thickness and occurring between 100 and 150 m below surface.

2.2.2 Discovery History And Exploration

The first systematic geological investigation of the Contwoyto Lake area was reconnaissance geological mapping conducted by the Geological Survey of Canada in 1949. Historically, mineral exploration in the north central Slave Structural Province has concentrated on gold and base metal exploration within the Yellowknife Supergroup metavolcanic and metasedimentary rocks. Diamond exploration in the region commenced in the early 1970's when a kimberlite cluster was discovered on Somerset Island.

During 1992 and 1993 Tahera's predecessor companies (Lytton Minerals Ltd. and New Indigo Resources Inc., through wholly owned subsidiary companies Benachee Resources Inc. ("Benachee") and Snowpipe Resources Ltd. ("Snowpipe"), staked a large area of the Slave Province in the area north of the Point Lake kimberlite discovery. In 1993 and 1994 Tahera commenced an aggressive exploration program, consisting of a regional airborne geophysical survey, covering the entire property position at a nominal 250 m line spacing. This survey covered some 5 million acres of ground. Several geophysical magnetic anomalies, displaying characteristics usually associated with kimberlitic intrusions, were identified and prioritized for detailed geophysical follow-up and drilling.

The discovery of the Jericho kimberlite pipes was a process that started with the recovery of kimberlite indicator minerals from till samples collected down-ice of the pipes in the spring of 1994. Intensive follow-up till sampling defined an indicator mineral train that led to the edge of a small lake, referred to as Carat Lake. Airborne and ground geophysical surveys identified a number of geophysical targets within Carat Lake, and on land south of the lake. The geophysical targets within the lake were drilled first, but yielded no kimberlite. Follow-up till samples near a geophysical target, about 600 m south of the lake, yielded indicator minerals, leading to the drilling of the discovery hole 95-JD014, which intersected kimberlite (JD-1 or the Jericho land-based pipe) in February 1995. A small satellite pipe (JD-2) was then discovered in March 1995, by testing another geophysical target near the lakeshore, 350 m north-northwest of 95-JD014.

The geometries of both JD-1(Jericho Pipe) and JD-2 were defined by drilling 79 NQ (47 mm core diameter) delineation holes, combined with 47 PQ (85 mm core diameter) holes, a 100 tonne (t) mini bulk sample holes (1996), and an underground bulk sample extraction in 1996/1997. The drilling shows that the Jericho kimberlite forms an elongate body with a straight, nearly vertical eastern wall, and three lobate pipe-lobe protrusions on the west side. These pipe-like protrusions have been identified as two distinct kimberlite facies (facies 4N and 4S came from the same event). The entire deposit consists of three facies: 2, 4, and 6. Drilling also established that the JD-1 kimberlite is connected to JD-2 by a 1 m wide kimberlite dike.

2.3 RESOURCE MODELING DATABASE

The database for the JD-1 (Jericho kimberlite) consists of:

- NQ diamond drilling results
- PQ (large diameter) diamond drilling results
- Underground bulk sampling results

The drill database consists of 86 NQ and 47 PQ drill holes, totaling approximately 18,000 m and 10,000 m, respectively. The majority of the drilling was completed in 1995 and 1996, although several additional holes were drilled in 1999 and 2000. These latest drill holes were recommended by SRK as part of the feasibility study to better define the geometry of the pipe at several different locations.

In 1997, an underground decline 787.3 m in length (500 m in granite, 287.3 m in kimberlite) was excavated. The aim of the underground development was to: confirm the distribution and volumetric significance of the kimberlite facies types; confirm diamond grade of the main facies types; obtain a parcel of macro diamonds for commercial valuation; and determine the metallurgical characteristics of the kimberlite. A total of 14,555 t of kimberlite were mined from the underground decline, and of that, approximately 9,435 t were processed at the diamond pilot plant constructed at Lupin Mine site.

2.3.1 Drilling

The majority of the 86 NQ holes were drilled so as to intersect the kimberlite at regular intervals, approximately 50 m spacing. The majority of the PQ holes were drilled at 10-20 m spacing within two of the most economically significant facies, as defined by Canamera (including the JDF4N and JDF6 facies). All of the 47 PQ drill holes are vertical, while 21 of the 86 NQ holes were vertical with the remainder drilled at varying orientations ranging from 45 to 82 degrees to the horizontal.

The drilling intersected kimberlite from the topographic surface to a vertical depth of approximately 350 m below surface and included lithological, geotechnical, diamond grade, and survey data. The database is currently maintained at Tahera's office in Toronto.

Down hole surveys of the drill holes, necessary to accurately plot the position of the drill hole trace, were not available for a number of holes drilled prior to 1999. Of the 86 NQ holes, a total of 20 holes have been adequately surveyed for down-hole deflections, while none of the PQ holes have been surveyed. This indicates that a potential problem may exist in accurately locating the down hole trace. Magnitude of the deviation is difficult to quantify. There is increasing uncertainty with increasing hole length (both vertical and inclined) as to the exact position of the trace of each drill hole. In the absence of such invaluable data, the holes are assumed to be straight, given the azimuth and dip of the hole from the surface. Recent drilling completed in 1999 and 2000 showed that the maximum deviation of NQ size inclined holes (average length of 300 m) is less than 5 degrees. It is expected that the drill hole deviation of the large diameter core drilling of vertical holes (PQ campaign) is less than that for the inclined smaller diameter NQ holes. In addition, and most importantly, SRK considers the drill density to be more than adequate for the majority of the pipe to minimize any potential errors in defining geometry of the pipe and the internal facies.

SRK reviewed the drill logs and completed several spot checks to confirm the accuracy of the core logging. The drill logs provide sufficient description and recognition of the lithology, alteration, geological structures, and mineralization to correlate geology between drill holes.

The drill core recovery for the kimberlite is typically greater than 90 percent. A split of the NQ drill core is stored in Tahera's Langley, BC storage unit. All of the kimberlite recovered from the PQ drilling has been processed and is not available for inspection; however, a number of photographs of the PQ holes are available for review.

2.3.2 Sampling and Assaying Procedures

Drill core from the NQ holes was processed by caustic fusion to determine the diamond content, while core from the PQ drilling was processed using the dense media separation (DMS) method.

A total of 1,957 m of split NQ core, approximately 3,465 kilograms (kg) of kimberlite, were processed using caustic fusion. The caustic fusion, carried out at the Tahera North Vancouver Laboratories, uses 10 kg size batches of kimberlite mixed with 100 kg of caustic soda (NaOH) in specially designed reinforced stainless steel alloy buckets. The sample kilns containing the kimberlite and caustic soda mix are heated to more than 600 degrees Celsius in stages to permit maximum reduction of the kimberlite sample without damage to any contained microdiamonds. The hot caustic soda solution is poured off after 24 hours. The kiln residue is washed and cleaned with hydrochloric acid before hand sorting under binocular microscopes. Synthetic diamond spikes are included in caustic soda samples to verify diamond recovery efficiency. A total of 8.31 carats of micro and macro diamonds (+75 micrometers) were recovered. Stones greater than 1 millimeter (mm) were weighed individually; stones between 75 micrometers and 1 rmm were measured and their weights estimated.

Processing of kimberlite from 47 PQ drill holes (a total of 9,666 m of kimberlite weighing approximately 101 t) was carried out at the Canamera Geological Ltd. (Canamera) DMS plant in North Vancouver. The PQ holes were sampled over 18 m intervals, resulting in 413 batches of approximately 250 kg each. The Canamera DMS plant was equipped for multi-stage crushing and sieving to minimize breakage and production of fine material. The DMS unit is a cyclone concentrator, which splits mineral components of crushed ore based on relative density. To control density and aid in the separation ferrosilicon heavy media is added to the water and then the crushed kimberlite slurry is fed to the DMS unit. Specialized equipment for diamond recovery includes an X-ray sorting machine and vibrating grease table. From 47 holes, comprising 413 sample batches weighing 101,913 kg, a total of 2,907 diamonds weighing 138.04 carats were recovered.

2.3.3 Bulk Sample

The aim of the underground development was to:

- Confirm the distribution and volumetric significance of the kimberlite facies types when compared to the drill hole information.
- Confirm diamond grade of the main kimberlite types.
- Obtain a parcel of macro diamonds for commercial valuation.
- Determine the metallurgical characteristics of the kimberlite.

The underground development portal was collared in Archaen granodiorites south of the Jericho kimberlite and driven as a decline into the kimberlite body. Thyssen Mining Construction of Canada developed the decline approximately 500 m to a depth of approximately 65 m below surface and then advanced 287.3 m into the Jericho kimberlite to extract the 14,555 t underground bulk sample. The kimberlite material extracted was separated into the various kimberlite types and 7 piles of kimberlite were created at the portal entrance.

The kimberlite was shipped over the 1996/97 winter road and treated at a 10 t per hour DMS plant built and operated by Tahera at the Lupin Mine site. Each of the kimberlite types was treated separately, although not all material from each pile of kimberlite was processed in its entirety.

A total of 9,435 t of kimberlite from the underground decline were processed at the pilot plant facility at Lupin Mine. A total of 10,535 carats were recovered.

2.3.4 Data Verification

Although independent sampling and assaying to confirm the diamond content was not commissioned by SRK, a detailed review of the sample and assaying/processing protocols developed by Canamera and Tahera was completed to ensure methodologies employed were appropriate.

In addition, diamond distributions determined from NQ and PQ drill samples and the bulk samples were compared and found to correlate very well, defining one population (as discussed later in this report). This good correlation confirms that results of the sampling and assaying/processing methodology are reliable.

2.4 JERICHO KIMBERLITE PIPE - GEOLOGIC MODEL

The Jericho JD-1 kimberlite forms an elongate body with a straight, nearly vertical eastern wall, and three lobate protrusions on the west side. The kimberlite is a NNW trending elliptical body with a length of approximately 300 m and a width of up to 100 m. It has been defined to a depth of at least 350 m below surface.

Canamera Geological Ltd. conducted initial work on the Jericho pipe geology in 1995. Canamera identified 5 main facies within the Jericho pipe, including JDF2, JDF4N, JDF4S, JDF6, and JDF1 (a xenolith-rich kimberlite of JDF4N).

A recent review of the geology and petrography by B.H. Scott-Smith, a well-known kimberlite petrologist, was completed to evaluate the previously defined facies incorporating the most recent drill data. It is interpreted that the Jericho kimberlite was formed from multiple emplacement phases or events with each event involving a kimberlite of different composition, including a precursor dike (JDF2) and three main kimberlite types (JDF4S, JDF4N, and JDF6) corresponding to the three major intrusive phases (Scott-Smith, 2000). The overall size and shape of the Jericho pipe are typical of the transitional area between the diatreme to root zones in the Southern African Kimberlite Model Pipe (Field and Scott Smith, 1998). Since the three pipes are not considered to be tuffisitic kimberlite breccia, the pipes are not considered to be diatremes.

2.4.1 JDF2

Possibly the first intrusive phase, the JDF2S consists of a precursor dike-like body located at the south end of the kimberlite pipe. It has been described as a hypabyssal kimberlite (HK). It is composed of two generations of fresh olivine set in fine-grained seemingly magmatic groundmass. The HK contains only rare xenoliths. Those observed appeared to be granitic rocks, which had undergone reaction with the host kimberlite. Notably, no sedimentary carbonate xenoliths are present. Some of the HK (e.g. drill hole DJ99-02) contained much coarser macrocrysts. It is possible, however, for emplacement processes such as flow differentiation to modify the grain sizes within one hypabyssal intrusion. Overall the nature of the HK suggests that the magmas erupting at Jericho are typical of kimberlites. The main HK is clearly a different phase of kimberlite from the adjacent pipe rocks; the HK contains coarser grained olivine and no ilmenite. The dike occurs on the east side of the Jericho pipe and reaches widths between 5 and 20 m.

2.4.2 Three Main Phases

The other three phases of kimberlite (JDF4S, JDF4N, and JDF6) from each of the three pipes, in contrast, are thought not to be HK. Canamera suggested that they are diatreme-facies kimberlite thus implying that the rocks should be tuffisitic kimberlites (TK), the typical infill of kimberlite diatremes. Many lines of evidence suggest that none of these rocks are TK, including: the overall paucity of xenoliths; dominance of sedimentary carbonate xenoliths from the now-eroded cover rocks; presence of abundant fresh olivine; clast supported textures; apparent lack of microlitic textures in the inter-clast matrix; lack of typical pelletal lapilli; overall lack of fine grained olivines in many rocks; overall lack of the erupting fluidal magma and internal variation in some areas of the core; variable inter-clast matrix; and the variable diamond grades. It is therefore considered unlikely that these rocks are TKs of the diatreme-facies. The features observed during logging are consistent with the rocks in all three pipes being of pyroclastic origin.

2.4.3 Geologic Model

A good understanding of the diamond distribution is essential in estimating the grade and value of any diamond deposit. For this reason, SRK engaged Mr. Tinus Oosterveld, the world leading authority in this field, to develop with SRK the following strategy for the evaluation of the Jericho kimberlite pipe:

- Check sampling results for consistency of recovery efficiency in each individual sampling campaign to assess potential treatment inconsistencies.
- Validate the geological model and compare sampling results and kimberlite types using grade and diamond size frequency to verify the geological model and assess the potential for grouping of sampling results.
- Compare and reconcile different types of sampling to obtain recovery characteristics of the methods of ore attrition and treatment.
- Estimate Resources by applying statistical methods to the sampling results to assess diamond distribution characteristics within each geological zone.
- Assess the large-scale diamond treatment plant recovery to estimate the "recoverable resources" and, with the application of economics, convert that to reserves.

In addition to re-interpretation of the various kimberlite phases completed by B.H. Scott-Smith, SRK completed an analysis of the distribution of diamonds from the PQ drilling to confirm validity of the geological model. The distribution of diamonds was used to further modify geometry of the three main phases, since the distribution of diamonds is considered to be uniform within a given facies. This led to the creation of new geological shapes/lobes, each having a characteristic diamond distribution (Figures 2.4, 2.5, 2.6, 2.7, 2.8). The good correlation that exists between the diamond distribution and the various kimberlite phases has provided a high level of confidence in creating a robust geological model.

As a result of modifications to the geometry, the term facies (defined by Canamera) was abandoned since it implies a known origin. For the remainder of this report the following nomenclature is used.

- JDF2S (former JDF2S and JDF2N)
- North lobe (mostly former JDF4N)
- Center lobe (mostly former JDF6)
- South lobe (mostly former JDF4S)
- JDF1 JDF1

2.5 DATA ANALYSIS

2.5.1 Density

A total of 673 density measurements were completed on the various kimberlite types comprising the Jericho kimberlite using a volume-displacement (non-waxed) methodology under laboratory conditions. These measurements compare well with densities determined from the recent Uniaxial Compression Strength analysis. The density measurements are summarized in Table 2.1.

Since the density does not vary significantly laterally or vertically within the pipe, a density of 2.6 t per cubic meter (m^3) was used to convert volumes to weights for the resource estimate.

2.5.2 Comparison of PQ Holes, NQ Holes and Bulk Samples

2.5.2.1 Center Lobe (JDF6)

Both the PQ holes diamond size frequency distribution by number and the bulk sample diamond size frequency distribution by weight show that the Center lobe contains the largest average stone size within the Jericho kimberlite pipe.

The diamond grade recovered from the PQ drilling is uniformly distributed, laterally across the northern portion of the Center lobe. The southern section shows more variability with a few lower grade holes randomly distributed. Vertically, the highest grades were observed in a zone 100 to 200 m below surface, where the grade is approximately 25% higher than in the zones above and below it. Although the eastern portion of the Center lobe was intersected with only a limited number of PQ holes, NQ drilling has confirmed the continuation of grade in this area.

2.5.2.2 North Lobe (JDF4N)

Both the PQ holes diamond size frequency distribution by number for the Upper zone and the bulk sample diamond size frequency distribution by weight show an average stone size that is smaller than in the Center lobe.

Grade variations were observed across the North lobe, Upper zone, with indications of a higher-grade central core. In addition, higher grades and some vertical zoning were identified near surface. These areas were not separated; it is unlikely they can be differentiated during mining. Although the northern section of North lobe has not been sampled with PQ holes, one NQ hole did confirm a continuation of grade in this area.

2.5.2.3 South lobe (JDF4S)

The South lobe shows a fairly uniform spatial distribution of grades from the PQ drilling and no significant horizontal or vertical trends were observed. Both the PQ holes diamond size frequency distribution by number and the bulk sample diamond size frequency distribution by weight show an average stone size that is smaller than in the North lobe.

2.5.3 JDF1 and JDF2S

The number of sample batches in these areas is small and the sampling is not spatially representative.

2.6 DIAMOND VALUATION

Diamonds produced from the underground bulk have been appraised by a number of diamond valuators:

- Central Selling Organization (CSO), 1997.
- HDM Laboratories, 1998.
- WWW International Diamond Consultants Ltd (WWW).

A summary of the valuations is provided in Table 2.2.

In April 1998, the diamonds were re-valued by HDM Laboratories (including greater than 10.8 carat stones) using the Terrac and Adtec simulated methods of diamond valuation (Table 2.3).

In November 1999 Tahera retained WWW International Diamond Consultants Ltd (WWW) to provide a comprehensive analysis and valuation of the diamonds. A total of 10,530 carats of diamonds were valued. A total of 6 independent valuators representing the Antwerp, Israeli, and Indian diamond markets, were requested to carry out the valuations for WWW.

The combined average price of the valuations was US$61 per carat. For the Center, North, and JDF2S lobes, the combined average prices were US$63, US$51, and US$51 per carat, respectively.

Analysis of the size frequency distributions by WWW shows the Center and JDF2S lobes to be similar, where both lobes show a high incidence of large stones. Two size frequency distribution models were produced, a combined model for the Center and JDF2S lobes and a second model for the North lobe.

Although there are differences in the valuations between the geological zones there was no conclusive evidence to suggest that the quality of diamonds vary within the deposit, therefore, a value model consisting of one value per size class was generated. The model assumes that the better quality large stones “that were absent in the samples” (WWW valuation, 1999), will be recovered in production. Applying the model values per size class to the size frequency distribution model, gives an average dollar per carat value of US$90 per carat for Center and JDF2S lobes, and US$71 per carat for the North lobe.

On the request of SRK the WWW valuation was reviewed by M.H.Oosterveld in March, 2000, and an SRK model was proposed. Mr. Oosterveld believes this SRK model is more realistic and is statistically more justifiable (Table 2.4). The more conservative SRK model was used for final revenue estimates in the feasibility study.

2.7 ESTIMATED RESOURCES

Resource and reserve estimation for diamond deposits is not restricted to the modeling of geology, grade, and density. This is because the value of an individual diamond varies significantly as a function of size, clarity, and color, as well as shape. In addition, the value that can be ascribed to “run-of-mine” diamonds (the figure that must be used in any resource/reserve estimation) should be based on the valuation of a parcel of diamonds that would give a representative valuation. A parcel of 2,000 carats is often considered as sufficiently large to enable variations in the variables of size, clarity, color, and shape to be assessed when performing diamond valuations (AIMM). Resource and reserve estimation therefore requires both estimating the mean quantity of diamonds present, normally expressed as carats per tonne, and the average value of these diamonds, normally expressed in terms of US$ per carat.

In the case at Jericho, the kimberlites have several facies/lobes, each of which tends to have characteristic diamond distributions. It is therefore reasonable to determine a mean grade and value for each lobe and then sum these to give an overall in-situ resource estimate.

The distribution of diamonds recovered from the PQ drilling was compared with the bulk sample data for the Center lobe and North lobe, Upper zone, and a single uniform and continuous distribution was defined for both. This correlation of the diamond distribution between the PQ data and the bulk sample made it possible to calculate a Recovery Factor that could be applied to the PQ grade in order to obtain the recovered grade of the lobe. PQ data is preferred over the bulk sample data as the spatial distribution of the data is much greater (Table 2.5).

For the Center lobe, the PQ drilling recovered 97% of the diamonds (larger stones not recovered from drilling), while the bulk sample recovered 66% of the stones (smaller stones locked up in fine processed kimberlite). Therefore, a “recovery factor” was calculated, based on the grade of ore estimated from PQ drill data, to determine the amount of stones that would be recovered from the processing plant. A recovery factor of 66% (recovery of stones from bulk sample) / 97% (recovery of stones from PQ drilling), or 68%, was calculated for the Center lobe. For the Upper portion of the North lobe, the PQ drilling recovered 77% of the diamonds, while the bulk sample recovered 68% of the stones. This equates to a recovery factor of 68/77, or 88%.

Due to an insufficient amount of bulk sample and PQ data, a recovery factor of 75% was used to estimate the recovered grade of the Lower portion of the North lobe, South lobe, JDF2S, and JDF1 from the PQ data. This percentage is approximately midpoint between the 88% recovery determined for the Center lobe and the 68% recovery determined for the North lobe, Upper zone.

2.8 RESOURCE CLASSIFICATION

In order for a resource to be classified as Indicated (CIMM, 1996), the geometry of the lobe and the diamond distribution must be known with reasonable confidence. Since the geometry is well-defined, there exists a relatively large amount of sampling information, and the diamond distribution determined for the PQ, NQ, and bulk samples are compatible, the upper portion of the North and Center lobes, above the 300 and 200-m levels, respectively, are classified as Indicated Resources. Below these levels, the resource has been classified as Inferred.

The geometry and diamond distribution of JDF2S and JDF1 is known with much less certainty than the other three lobes, and hence, has been classified as an Inferred Resource.

The resources for the Jericho project as of April 26, 2000, and at present, are summarized in Tables 2.6 and 2.7.

3.0 FACILITIES OVERVIEW

3.1 EXISTING INFRASTRUCTURE

Infrastructure related to exploration work to date includes: portal area/kimberlite storage pad, campsite, service roads, three borrow sites, fuel farm, and airstrip and turnaround pad (Figure 3.1). The main camp at the Jericho site, Carat Camp, is situated near the northeastern shore of Carat Lake, and can accommodate approximately 50 people. During construction, Carat Camp will be maintained for initial housing of construction crews. The camp will also be retained for future exploration activity and as an emergency retreat in case of fire or power failure at the mine accommodation complex. A 1,067 m (3,500 ft) airstrip provides all-weather access to the Jericho site for personnel and supplies. Current design of the runway is suitable for Twin Otter, DC-3, DC-4, and C-46 aircraft. The airstrip is also equipped with runway lights and a 10-m tower weather station. An all-weather road, approximately 3.5 km long, connects the airstrip, camp, and Jericho Project site.

Two granular borrow areas (A, C) have been used to build the airstrip, site development area, and access road; a third (D) is undeveloped. A fourth identified site (B) is not connected by road and will not be considered, unless other areas are depleted of suitable material. The borrow sites have also been used for repairs to the road and airstrip, and other purposes as required.

The 3,500 foot (1,067 m) airstrip was constructed in 1996 to support underground (i.e. bulk sampling) exploration activities at Jericho. The strip is located approximately 1 km north of the existing exploration camp on a broad esker that was graded down to provide the 30 m wide running surface. Adjacent the west side of the strip sedge meadows predominate; to the east is esker for a variable distance up to 70 m with lake-stream valley beyond. Elevation of the strip is 500 m above sea level. A turn-around area and laydown area were constructed at the north and south ends of the strip, respectively. Runway lights, powered by a small diesel generator housed at the south end of the strip, allow use of the strip in winter and under poor weather conditions. There is no beacon for the airstrip; the closest beacon is at Lupin airstrip 29 km to the south. A 10-m meteorological tower, installed in 1996 when the strip was constructed, continuously monitors wind speed and direction.

The bulk sampling site (a.k.a. the portal area) is located at the south end of the kimberlite pipe. It consists of the surface facilities pad constructed of granular borrow material, a sample storage pad constructed from granite waste rock, and surface facilities. Total surface area is approximately 12 ha.

A portal for the decline was excavated in 1996 in a granitic bluff adjacent to the kimberlite ore body. This decline extended at a grade of 15% through granite until kimberlite was encountered. Total advance of the decline was 787.3 m; total volume of kimberlite and waste rock excavated was 11,650 m^3.

A mine/groundwater retention pond, shop, office and dry, brine pond, and one 63,000 L fuel tank are located immediately north and adjacent the decline.

A fuel farm is located at Borrow Site A, off the main access road. It contains nine tanks with a total capacity of 587,000 L (8 x 63,000 L; 1 x 83,000 L).

In addition, a laydown area was constructed upland of the north shore of Carat Lake and west of Borrow Site A, and another constructed at the south end of the existing airstrip. The blasting caps magazine and powder magazine (a.k.a. C-can) are located off the access road between the underground sampling site and the main camp.

3.2 MINE

The proposed site layout is shown on Map A (Appendix E). The mining contractor will provide all infrastructure at the Jericho site necessary to produce the required annual feed to the plant.

The infrastructure will include:

- accommodations including office facilities;
- sewage treatment plant attached to, or adjacent to, the processing plant;
- internal mine roads and access road to Contwoyto Lake to link with the Lupin winter road;
- power station (part of the processing plant structure);
- airstrip (existing, with 150 m extension);
- fuel storage facilities;
- ammonium nitrate storage facilities;
- high explosives and caps magazines;
- mine truck shop;
- explosives truck wash shop
- boneyard (i.e. storage area for unused equipment).

Areas associated with the physical mining operations will include:

- open pit;
- two ore stockpiles;
- low-grade ore stockpile;
- two waste rock dumps;
- overburden stockpile;
- sedimentation ponds and berms for water management of pit and dumps;
- 9 km of new mine roads (13 to 22 m width);
- laydown area (part of the plant complex); and
- a diversion of Stream C1 around the open pit.

The required mining fleet (i.e. equipment) for open pit and underground is listed in Table 3.1, together with a list of the processing plant fleet.

3.3 DIAMOND PROCESSING PLANT

The diamond processing plant complex and processed kimberlite containment area (PKCA) are shown on Map A. Proposed infrastructure includes:

- diamond processing plant (40 tonne per hour dense media separation unit);
- powerhouse (attached to plant);
- two ore stockpiles (central and north--referring to deposit lobes);
- laydown area (separate from mine laydown);
- coarse kimberlite storage area;
- recovery plant coarse kimberlite storage area;
- processed kimberlite containment area (PKCA) consisting of two cells and a polishing pond; and
- required connecting road system.

The layout for the process plant complex has been developed to be as compact as possible to minimize surface disturbance and haul distances. Under mine safety guidelines, facilities where people live must be a minimum of one km from blast sites, i.e., the open pit.

3.4 ACCESS AND MINE ROADS

3.4.1 Winter Access Road

Access to the site for initial construction and resupply will be via winter road. An extension of the Echo Bay Lupin road will be required. For Year 1 a winter road will be constructed up Lynne Creek as shown on Map A (Appendix E). The route taken will be identical to that used for the bulk sample mining in 1996-97. Following mobilization, the 3km from Contwoyto Lake to the site will be constructed as an all weather road, allowing for use during years when snowfall may not be sufficient to simply construct an ice road. Annually, this road will be used to resupply the mine with fuel, supplies, and equipment.

The majority of land this road would cross is Inuit held surface rights, which is managed by the Kitikmeot Inuit Association (KIA). The area required for construction of the road will be held under a lease negotiated with the KIA (see Appendix A.2). The road will end as a ramp to facilitate vehicle access to Contwoyto Lake during winter and emergency boat access during summer. The exact dimensions of this ramp are not presently known, but it would be approximately the width of the road (10 m) and would extend no more than 15 m into the lake. As such, the surface area of the ramp would be approximately 150 m^2.

There is no practical alternative to surface transport of bulk materials to the Jericho site (see Section 8). Air transport of fuel (except on an emergency basis) is impractical, as is air transport of bulk explosives. Without surface transport, Jericho would be unable to operate. As well, the only alternate to a winter road, i.e., an all-weather road from Yellowknife (or from Contwoyto Lake should the Bathurst Road become a reality), could not be financed by the Jericho Project.

3.4.2 Mine Access Roads

Map A shows the road layout at the mine site. Three road widths will be used, depending on the intended use. Roads with a 6-m crown will be designed for light trucks, roads with a 9-m crown will be designed for highway trucks or single-lane traffic for ore trucks, and roads with an 18-m crown will be designed for ore trucks to pass each other safely.

4.0 PROJECT SCHEDULE

Key milestones include:

Submission of land lease and water licence applications	January 2001
Submission of a draft EIS	January, 2001
Prehearing meetings	May & June 2001
Submission of a final EIS	January 2003
Final public hearings	April 2003
Project approval by NIRB	early June 2003
Project approval by the DIAND Minister	late August 2003
Mobilization on the winter road	January to March 2004
Mine construction	2nd quarter 2004
Plant construction	2rd to 4th quarter 2004
Full Commercial Production	First Quarter 2005
Production open pit mining	1st quarter Year 2 to 4th quarter Year 4
Reclamation of waste dumps	Year 4
Underground mining	1st quarter Year 5 to 3rd quarter Year 7
Processing of North lobe ore stockpile	Year 8 to Year 9
Final reclamation and closure	Year 9 to Year 10

4.1 PROJECT PERMITTING

As the Jericho Project is located in Nunavut, the Nunavut Land Claims Agreement (NLCA) is applicable to the regulatory approval process of this project. The project will require approval of the Minister of Indian and Northern Affairs (the Minister), and the issue of land and water licences and others to proceed. Tahera commenced the process in November 1999 based on the assumption that the diamond plant would be constructed at the Lupin Mine. However, as the plant will be located at the Jericho kimberlite pipe, Tahera recommenced the approval permitting process January, 2001. In January of 2001 the company submitted a Draft Environmental Impact Statement (along with supporting documents) and permit and authorization applications to the following:

Nunavut Impact Review Board (NIRB);

Kitikmeot Inuit Association (KIA);

Nunavut Water Board (NWB);

Indian and Northern Affairs Canada (DIAND);

Department of Fisheries and Oceans (DFO); and

Natural Resources Canada (NRCAN).

All documents were distributed to an extensive distribution list developed by NIRB. The NIRB determined the project to be significant enough to require a review and recommended to the Minister that a Part 5 Review be conducted as set out in Section 12 of the NLCA. The Minister approved the recommendation of NIRB and requested that an efficient review process be conducted.

Prehearing meetings were scheduled and conducted in the communities of Cambridge Bay, Kugluktuk, and Gjoa Haven (all designated points of hire for the project) during May and June of 2001. In January of 2001, Tahera Corporation underwent a change in Senior Management and a corporate decision was made to pursue a partnership with a major mining company. The intention was to have a major company partner discover and develop a larger project than the one proposed here (i.e. a longer mine life). From September 2001 to September 2002 the partner pursued this objective, but was not successful, resulting in their withdrawal from the arrangement. Immediately following this decision Tahera Corporation decided to resume permitting activities for the Jericho Project.

It is currently anticipated that Final Hearings for the project will be conducted in April 2003. Matters raised at the Final Hearings will determine length of the review period. A recommendation will then be forwarded to the Minister for approval of the Final EIS. Following approval of the EIS, permits and authorizations will be completed with the required stakeholders.

The length of time required for the Minister to approve the NIRB's recommendation will have an impact on the schedule of the project. A timely response from the Minister may allow for some materials and equipment to be transported along the winter road to the site during 2004. Should approvals not be in place by the third quarter of 2003, there is little chance that materials and equipment could be procured to travel on the 2004 winter road, resulting in a lengthy wait until the 2005 winter road season.

4.2 DETAILED ENGINEERING, PROCUREMENT, AND CONSTRUCTION MANAGEMENT (EPCM)

Tahera will select a professional engineering firm (the firm) for the design, procurement, and construction management of the diamond plant and infrastructure. Dowding Reynard & Associates (Pty) Limited (DRA) of South Africa were responsible for the design and construction of the 10 ton per hour pilot plant located at the Lupin Mine and the engineering design and estimate for the feasibility study. The 10 tonne per hour plant was used to process the bulk sample taken from the Jericho kimberlite in 1997.

Should sufficient progress be apparent in the regulatory review, detailed design engineering and procurement will commence in June 2003 to ensure all equipment is available for shipment to the project site on the ice road in quarter 1 2004.

The project engineer will co-ordinate procurement of infrastructure items in Canada. He will also co-ordinate input from consultants with specific expertise on civil, structural, heating, and ventilation required for projects located in arctic conditions.

The selected engineering firm will be responsible for the transport of all equipment and construction materials to the project site during mobilization. The mining contractor for the project (Section 4.3) will complete site preparation and earthworks

A project construction manager will have overall site responsibility. Canadian sub-contractors will be appointed to erect the buildings, plant, fuel storage facility, power generators, camp, water services, and fine processed kimberlite disposal system. The construction phase is scheduled to commence in May 2004 and finish in November 2004.

Plant commissioning is scheduled for December 2004. The engineering contractor will provide experienced mechanical, instrumentation, and process engineers and, where appropriate vendors will provide commissioning engineers. Tahera Corporation will recruit plant employees to participate in the pre-commissioning period and start up.

4.3 EARTHWORKS AND PRE-STRIP

In preparing the Feasibility study and Project Description for the Jericho Diamond Project, Tahera Corporation required a significant amount of information on equipment to be used and mining techniques. An Inuit owned company, Nuna Logistics, were able to provide this information because of their extensive experience in Nunavut and the Northwest Territories. As a result of their involvement in the EKATI™ and Diavik Projects, Nuna Logistics has gained significant experience in the areas of mining and site developments in Canada's north. In addition to mining activities, Nuna Logistics has been solely responsible for the construction of the Lupin Winter Road, which runs from Yellowknife to the Lupin mine site. This combined experience and expertise has led Tahera Corporation to consider Nuna Logistics a preferred contractor for the mining and road construction components at the Jericho Mine site.

Nuna Logistics will mobilize to the project site in February 2004. During February Nuna will complete the ice road from the Lupin Mine to Jericho and establish a camp, diesel storage facility, workshop, explosive storage facilities, and ice access roads on the site. Nuna will utilize the existing diesel storage facility, having a capacity of 600,000 litres.

Pre-stripping of overburden and granite will commence in March 2004. The granite will be used for road construction, ore stockpile pads, plant site, camp site, causeway for water intake, and extension of the existing aircraft runway. The overburden will be stockpiled for use in reclamation. Nuna will also complete the first phase of construction of the PKCA to allow deposition of the first nine months production of fine PK. The PKCA will be completed in Quarter 3, 2005.

Approximately 160,000 tonnes of kimberlite will be mined from the starter pit. Kimberlite will be stockpiled adjacent to the plant and will provide ore feed for the first six months of the operation. Nuna Logistics will complete their work in Quarter 2, 2004 and will return to the site to commence mining in Quarter 2, 2005. Additional mining equipment will be mobilized to the site at that time.

Equipment and construction materials for the plant and infrastructure will be stored on future ore pads close to the plant site. Approximately 4 million litres of diesel will be delivered to the site for mining, construction, and power for the diamond plant. In Quarter 1, 2005 additional fuel storage capacity will be constructed, sufficient for a full year's operation.

4.4 TAHERA CORPORATION

Tahera Corporation will appoint a project director and project manager to overview the design, procurement, and construction of the project. An accountant will be designated in 2004 to provide financial control services for the project development and the ongoing operation.

Tahera will recruit employees for the plant operation late in 2003and in 2004, or as the progress of regulatory approval will allow. Skilled employees will be sought for these positions; some positions will be filled through training programs (prior to plant commissioning), and some through apprentiships, where feasible.

Tahera's Environmental Manager will monitor environmental conditions and manage environmental matters during the construction phase. He/she will liaison with authorities during the project permitting phase, as required.

Tahera will appoint mining consultants to complete final pit design and mine scheduling for the starter pit. The mining contractor will provide input into the final schedule and then negotiate the mining contract with Tahera Corporation. The mining consultant will audit mining practices and review pit slope design on an ongoing basis.

4.5 UNDERGROUND DEVELOPMENT

Underground development is scheduled to commence in Quarter 1, Year 5; however, this date is dependent on the timing of Project approval. Tahera will appoint an experienced mining contractor to develop the access decline from the open pit to the underground production levels. The contractor will also be responsible for diamond drilling to further delineate the ore body, as required, prior to final slope design by mining consultants.

5.0 MINING AND RESERVES

5.1 INTRODUCTION

SRK Consulting provided mine design engineering for the Jericho Diamond Project Feasibility Study. This section discusses the mine plan recommended; alternatives are discussed in Section 22.

Mining will be by conventional open pit mining methods followed by underground mining using open benching or sublevel caving methods. High and medium grade ore from the F6 and F4N zones (central and northern lobes) will be stockpiled immediately north and east of the process plant. The remaining low grade kimberlite will be stockpiled close to the pit, separate from the high grade and waste material, in the event future diamond values increase and make the material economically viable to process. Also a sample from the south lobe will be taken and processed in the plant to evaluate more accurately the recoverable grade and value of the diamonds. Waste rock, consisting largely of granite, will be placed in dumpsite 1 or 2 located immediately to the northeast and south of the pit, respectively (see Map A, Appendix E).

Underground access will be achieved via a portal in the pit. Ore and waste material from the underground operation will be placed on the high-grade ore stockpiles and waste dumps. Waste from the underground will come mainly from the decline and level development.

Surface and underground mining contractors will carry out mining activities. It is unlikely that one contractor will be used for both surface and underground operations, as the expertise required of each method is quite different. The surface contractor will provide all infrastructure requirements related to open pit operations (including accommodations, catering, power, water supply, fuel storage, explosives storage, mine office, and maintenance facilities). The surface contractor will also be responsible for all site earthworks (i.e., associated with the open pit, PKCA, roads, and all buildings, including the DMS processing plant). The contractor will make use of existing infrastructure where it makes sense to do so.

Tahera will provide personnel (staff or consultant) for mine planning, grade control, and surveying. The mine manager will be responsible for contract supervision.

5.2 MINING GEOTECHNICS

Designing pit walls is an important part of open pit mine planning and design. The objective is to design the wall to maximize economic benefits and at the same time provide for a safe environment. The cost of waste removal must be balanced against the cost of wall instability and safety.

The following sources of information were used in pit wall design for the Jericho project:

- Drillhole database

- Underground decline evaluation
- Past record and relevant experience from other diamond projects
- Relevant industry practice and experience from other cold region projects (including diamond open pits in Russia)
- SRK's professional opinion

The following methodology was used for the geotechnical assessment:

1. Drillhole data assessment
2. Underground exposure assessment
3. Permafrost assessment
4. Kinematic analysis
5. Empirical base case design
6. Numerical analysis
7. Slope design

A detailed geotechnical review and assessment of the Jericho Project was undertaken by SRK (Jericho Feasibility Study, 2000). Other geotechnical assessments at the Jericho site have been undertaken since 1995 by various independent geotechnical engineering consultants (Bruce Geotechnical Consultants Inc. [BGC], 1995, 1996a, b, c, 1997; Nilsson Mine Services Ltd. 1996; Rescan Engineering Ltd, 1997).

The following rock mass characterization of the main geological domains is based on Laubscher's rock mass classification data obtained by SRK from the core, laboratory strength measurements, and the site visit:

- Overburden has been characterized as frozen glacial till consisting of gravel, gravel sand, sand, and silt with layers of massive ice. Unless protected from melting, the till has to be excavated at shallow angles of 2:1. The construction of a permeable rip-rap covering will prevent sloughing and unraveling of till faces and could allow steeper angles.
- Granodiorite (granite) country rock has been classified as strong to very strong rock with good rock mass quality (Class 2A-2B).
- Pre-contact granite which is the fractured and altered granodiorite adjacent to the kimberlite has low strength and has fair to very poor rock mass quality (Class 3B-5A). The presence of ice will enhance the strength properties.
- Kimberlite has been described as medium strong to strong with good to fair rock mass quality, (Class 2B-3B). The geotechnical properties of the different kimberlite types, (facies) are somewhat similar.

Individual geotechnical units are also illustrated in schematic section (Figure 5.1)

There are sections of kimberlite that are moderately susceptible to weathering within the Jericho pipe. However the weathering tests carried out on 1999/2000 drillcore samples show low susceptibility to weathering within the mine life.

5.3 MINING OPTIONS

Two open pit mining scenarios were studied to compare the cashflows of relatively large, low unit cost equipment and shallow overall pit wall slopes versus using small, higher unit cost equipment with steep overall pit wall slopes and lower overall strip ratio.

The following two mining scenarios were studied:

1. 777 Option: Open pit mining using a Cat 5130 shovel (Cat 992C loader backup) with Cat 777A/B/C trucks to the 310 elevation (overall wall slope of 44° to 53°) followed by underground mining to the 230 elevation; and
2. D300 Option: Open pit mining using a Cat 5130 shovel (Cat 992C loader backup) with Cat 777A/B/C trucks to the 400 elevation for the mid-stage and ultimate pits and a Cat 980 loader with Cat D300 trucks to pit bottom at the 310 elevation (overall wall slope of 46° to 58°) followed by underground mining to the 230 elevation.

[**Note**: The option of only recovering the overburden by open pit techniques and then mining underground was also considered. The analysis showed that only the contact ore zone would be economic, due to the higher mining costs and lower (diluted) mining grade. In addition, there is a higher confidence level in operating cost and productivity parameters for open pit mining, hence the increased risk of only using the underground mining methods].

Cashflow analysis indicated that both the 777 and D300 scenarios are relatively equivalent. However, the D300 option, which makes use of steeper overall pit wall slopes (i.e., up to 58°), is considered a higher risk method and was therefore rejected. If pit wall stability indicates that the steep slopes are achievable then the D300 option could be revisited as it will result in lower overall stripping and potentially lower reclamation burden.

5.4 OPEN PIT DESIGN

5.4.1 General

The kimberlites in the Jericho Project are characteristically composed of softer, more erodable materials than the surrounding granodiorite. Kimberlite bodies are generally associated with surface depressions usually containing sediments and till, or shallow lakes. In the case of an open pit mine plan overlying sediments have to be removed, or the water drained.

Pit operations will begin with a starter pit in the pre-stripping period to feed the plant during commissioning and the first six months of the production period. Waste from the starter pit will be used for site earthworks (e.g., roads, building foundations, stockpile pads, etc.).

Due to the vertical pipe nature of the orebody two successive pushbacks will be required. Pushback widths, nominally 40m, are based on minimum practical mining widths for the selected equipment.

It is assumed that the NWT Mine Health and Safety Act (or Nunavut replacement legislation) will be the applicable governing legislation for the safe development and operations of the Jericho mine.

5.4.2 Pit Design Process

The pit design process started with a review of previous designs and costs (Prefeasibility Study November 1999). The overhead, mining, and process costs were reviewed and updated based on more detailed estimates from Nuna Logistics, Dowding Reynard & Associates, and Tahera. The updated costs, a new resource model, and a new diamond valuation were used to produce new optimized pits. A starter, intermediate stage, and ultimate pit were manually designed based on the optimized pit for the Cat 777 option.

The following procedure was used to develop the final pit design:

- Review previous work (iterations) to develop overall pit angles and unit costs
- Update and verify cost and pit slope angles
- Run Whittle 4X pit optimizer using updated data and the latest diamond valuation
- Select base case pit shell including a 20% profit margin
- Develop a detailed mine design (with bench configuration and ramps) based on the selected pit shell
- Calculate reserves within the designed pit
- Develop a schedule from calculated pit reserves

5.4.3 Pit Optimization and Design Parameters

Pit optimization parameters consist primarily of slope angles, the geological model, diamond valuation, and costs. The geological model and diamond valuation are discussed in Section 2.

5.4.3.1 Pit Slope Angles

The geotechnical assessment discussed in Section 5.2 formed the basis of the pit wall slope calculation. The recommended pit wall design slopes listed in Table 5.1 are based on rock mass ratings, available structural data, and experience with similar projects in the NWT and Russia. Values in Table 5.1 will be reviewed as pit production progresses and more geotechnical data becomes available. Inter-ramp angles are derived from the graph shown in Figure 5.2.

5.4.3.2 Pit Optimization Input Slope Angles

Pit slope angles entered into the Whittle 4X program are overall pit angles for each rock type. To estimate the overall pit wall angle the number of ramp intersections and the approximate location of the pit wall are necessary. This knowledge is combined with the recommended inter-ramp pit angles, Figure 5.3, to arrive at an overall pit angle. Refining the estimate is an iterative process using previous pit optimizations and pit designs. To further complicate the input parameters, the inter-ramp angle varies with the height difference between the design ramp spirals. This is necessary to avoid an excessively conservative overall wall angle for small pits with tight spiral ramps. The final recommended overall angles are illustrated in Figure 5.4.

A marketing cost equal to 5% of revenue was included in the pit optimization. An underground mining cost of $45/t and underground dilution of 20% were applied to determine the economic crossover from open pit to underground operations.

5.4.3.3 Pit Optimization Results

The pit optimization process was used for the following:

- to identify the ultimate economic pit;
- to locate pit stages within the ultimate pit that meet practical mining constraints (i.e., equipment, safety); and
- to locate the economic crossover between open pit and underground mining.

Tables, graphs, and figures of the pit optimization results for the 777 and D300 options (see Section 5.3 above) are available upon request. The optimized pit results for the 777 option were used for the detailed manual pit design.

5.4.4 Detailed Pit Design

While pit shells generated from optimization routines provide a relatively good preliminary indication of resources amenable to economic exploitation by open pit methods, a workable design is normally required before reliable material inventories can be produced. Equipment logistics, pit wall configurations, and access are a few of the considerations that impact on the overall design.

The vertical pipe shape of the Jericho orebody and resulting deep conical shape of the optimized pit shell requires a spiral ramp access that can result in a significant discrepancy between the optimized pit reserves and the detailed pit design reserves. This is usually due to the difficulties with accessing the lower ore in the optimized pit shell and errors in estimating the overall angle during the optimization. It must be stressed that pit shells generated by pit optimization routines often contain less waste and are marginally deeper than workable designs. Therefore the detailed (workable) pit design is primarily for the purpose of improving the accuracy of the ore reserve and the waste volume.

The optimized pit shell used for the ultimate pit design was selected as follows:

1. Locate the breakeven marginal cut-off shell
2. Calculate the ratio of 'worst case discounted cashflow' versus 'total tonnes' contained in the shell and add 20% (i.e., to account for a 20% profit margin)
3. Locate the shell with the 'worst case discounted cashflow' to 'total tonnes' ratio that was closest to the value calculated in the previous step (i.e., breakeven marginal cut-off plus 20%)

The selected optimized pit shell served as a general guide for the detailed pit design. The shell indicated the general pit location, shape, and bottom elevation. The goal of the final design was to produce a workable pit that maximized cashflow opportunity. Optimum cashflow was achieved by concentrating the pit design and schedule on the much higher valued central zone. Although the pit optimizer, which optimizes on NPV, indicated a second pit bottom to the north to extract part of the northern lobe, mining this pit extension proved to have a detrimental effect on cashflow and was therefore omitted from the final design. This extension of the open pit may be a viable economic opportunity for the future.

The pit design is in accordance with Nuna Logistics' (the currently preferred contractor) experience and available equipment fleet. Bench height will be 10m with double benching to 20m in granite and single benching in all other materials. Berm widths are typically 8m (10m in overburden). The minimum catch berm width complies with the Mine Health and Safety Act. The bench configuration is illustrated in Figure 5.3. Pit design parameters are based on this configuration and are summarized in Table 5.1.

Due to the shape of the kimberlite pipe and the optimized pit shell, the ramping system (at 10% gradient) becomes a very tight spiral at the pit bottom. In this area, the recommended inter-ramp angles were reviewed and slightly steepened to reflect the short vertical distances between segments of the ramp. The ramp width is 22m, which includes a ditch and berm, and is based on the use of CAT 777 trucks (two lanes). The pit bottom and push back widths assume efficient production for the equipment size utilized. The use of single lane ramps (14m wide including ditch and berm) for the last 5 benches was incorporated into the design to minimize waste stripping for the final benches. The final bench of each pushback will be excavated from above (i.e., 'pig rooted') with an excavator to maximize ore tonnage and eliminate additional ramping and stripping. Pit bottom is at elevation 310m (about 180m below surface)

All three pit designs (starter, pushback, and ultimate) are illustrated in Figures 5.5, 5.6, 5.7, 5.8, 5.9 and 5.10.

5.4.4.1 Mine Plan & Reserves

For the purpose of this report, reserves are defined as that portion of the diluted measured and indicated resources considered economically and legally extractable. This definition is consistent with both the American Institute of Mining Engineers (AIME) and Australasian Joint Ore Reserves Committee (JORC) guidelines for the reporting of resources and reserves. The mineable reserves for the Jericho Diamond Project are based on an open pit design utilizing the results of a number of economic pit optimization studies (Whittle 4X) and cash flow analyses as guidelines for pit boundaries.

The ultimate detailed pit design (Figure 5.7) contains reserves of 1.92 Mt ore at a recovered grade of 1.25 c/t, (see Table 5.2). Waste, including low grade, inferred kimberlite material, is estimated at 16.1 Mt giving a strip ratio of 8.4 to 1.

In order to minimize the payback period and maximize NPV, the central ore will be processed first. All northern lobe ore will be stockpiled until the central ore is exhausted, or to provide plant feed should problems occur with the supply of central lobe ore. Stockpiles for central and northern lobe ore will be constructed immediately north and east of the plant (see Map A).

The ultimate pit will be approximately 350m wide by 400m long and will cover an area of approximately 10 hectares (the pit is roughly oval, not square). Permafrost extends to a depth of approximately 540m, thus it is assumed there will be essentially no groundwater seepage into the pit. The small stream crossing the northern pit area will be diverted into Carat Lake via a surface diversion as indicated in Map A.

Pre-production is scheduled for quarters 1 and 2 of Year 1 (Table 5.3). Sufficient granite will be excavated in Q1 to provide the necessary fill for the site infrastructure. The approximately 161,000 t of ore mined in Q2 of Year 1 will be stockpiled, then used for plant commissioning in Q4 Year 1 to cover plant production until mining re-commences in Q2 Year 2.

Ore mining was scheduled to maintain a plant processing rate of 330,000 t per year. Due to operational and weather constraints unique to the far north (e.g., winter road access, white outs, freshet, freeze up, equipment maintenance in extreme cold, etc.) open pit mining will be carried out in Quarters 2, 3, and 4 of each year. Ceasing mining operations in Q1 will allow time for resupply over the winter road, holidays for contract labour, and buffer in the event mining operations experience difficulty in the operational quarters.

Accelerated production will require the construction of two main ore stockpiles (for central and northern lobe ore) and one low-grade stockpile (see Map A). Ore stockpiles will be largest at the end of Year 4 when the central and northern lobe stockpiles will contain about 400,000 and 500,000 t of ore, respectively. The low-grade stockpile will then contain 1.7 million t.

Pit waste rock consisting largely of granite will be deposited in the two dumpsites indicated on Map A (south and northeast of the pit). Waste dumps and stockpiles will be constructed in 10m lifts as described in Section 14.

5.4.5 Mining Method

The surface mining contractor will carry out conventional open pit mining using a combination of loaders, hydraulic shovels, and off-highway trucks. Current cost estimates are based on a fleet of Cat 777A/B/C trucks with a Cat 992C loader and a Cat 5130 hydraulic shovel/backhoe. Pioneering drills will be used to establish the first level bench, and then DML drills will be used for all production blast holes. Blast design details, parameters, and costs are based on estimates from Nuna Logistics and NWT Rock. Since operations are principally in permafrost, all blast holes are assumed dry. Therefore, explosives will be ANFO mixed on site and delivered to the hole by an explosives supplier. Further study is required to optimize powder factor, fragmentation, and preshear. Kimberlite and waste rock from the pit will be hauled to an ore stockpile, the overburden stockpile, or a waste rock pile (see Map A).

5.4.5.1 Ore Stockpiles

Ore stockpiles at Jericho will require a maximum capacity of 2.6 million t. The largest volume of storage will be at the end of Year 4 when the central lobe, northern lobe, and low grade stockpiles will contain 400,000 t, 500,000 t, and 1.7 million t, respectively. The main ore stockpiles will be located to the north and east of the plant and the low-grade stockpile will be located immediately east of the central lobe site (see Map A). The stockpiles will be constructed in 10m lifts similar to waste rock dumps described in Section 14.3.

5.4.5.2 Waste Rock and Overburden Piles

Overburden and waste rock will be hauled to designated waste piles located immediately to the south or northeast of the pit (Map A). The overburden and waste rock piles are designed to accommodate 14.5 million t of overburden and waste rock. Loose overburden will be used for construction of diversion channels, settlement ponds, and for reclamation purposes. About one million tonnes of granite will be used for site earthworks (road beds, building foundations, stockpile pads, etc.). The majority of the waste rock will be granite, granodiorite, or diabase dikes (up to 5%).

5.4.5.3 Explosives Storage and Use

A maximum of 2300 t of ammonium nitrate will be used during the maximum production year of open pit mining. In addition, up to 26,000 kg of high explosives may be used annually. The hauling contractor will transport the ammonium nitrate over the winter road. Ammonium nitrate will be stored approximately 1.5 km east of the open pit on the all weather road connecting the open pit to Contwoyto Lake (Map A). It will be stored on a pad in the tote bags used for transportation. ANFO use by year is listed in Table 5.4.

High explosives (Magnifrac™) and caps will also be transported during the winter haul. Powder and explosives magazines will be located on separate pads on the same all weather road as the ammonium nitrate. Storage will be in locked, tamper-proof seacams, or similar units approved for high explosives storage by Workers Compensation Board and Natural Resources Canada (NRCan).

Ammonium nitrate and fuel oil will be delivered to the blast site and mixed down hole.

An explosives truck wash will be established between the open pit and the high explosives storage areas on the all weather road to Contwoyto Lake (Map A). The truck wash will comply with the Federal Explosives Act requirements.

A factory licence will be applied for prior to mine construction to permit explosives use on the site. Given that the underground mining contractor will be different from the open pit contractor, a different licence may be required for underground operations. This will be determined in advance of underground mining planned to commence in Year 5.

5.5 UNDERGROUND DESIGN

The economic open pit operations will go down to the 310 level. At this depth, it is more economic to convert to underground extraction as the strip ratio rapidly increases. The underground reserves then continue to the 230 level and constitute some 615,000t in the center lobe over a vertical height of 80m (or approximately two years of production). The underground extraction of kimberlite can only start once the open pit is complete.

5.5.1 Selection of Mining Method

Selection of a mining method nearly always involves a compromise between cost, resource utilization, dilution, and production rate. In this respect it is useful to consider the evolution of most underground kimberlite operations. All of them started with open pits (of one sort or another), which then converted to glory hole methods until control of dilution became critical, when caving methods were used.

Current underground kimberlite operations favor block caving if it is applicable. Considerable sub-level caving has been tried but there is very little done today. Open benching methods have also been widely applied in many operations. There have been some trials with open stoping and vertical crater retreat, but these have shown variable results. There are no backfill operations utilized in underground diamond mining to date.

The most recent conversions to underground in kimberlite mines have been with the open benching technique at De Beers' Finsch Mine. Currently EKATI™ in NWT is developing an open bench mine.

5.5.2 Open Benching: Detailed Description

Open benching, also known as glory hole mining, will start at the base of the open pit. Very little ore would be at risk at any one time as it is a "top-down" method with unbroken ore in the floor. Open benching is similar in concept to open pit, but with access to the "benches" within the waste rock and kimberlite. In effect it is a very steep walled pit (as steep as the walls of the pipe). Safety of men and equipment is assured since they both operate from the security of tunnels.

5.5.2.1 Layout Elements for Open Benching

The layout elements are:

- Level interval;
- Production drift spacing and size;
- Slot development.

Figure 5.11 illustrates the key design elements. The key elements depend on:

- The angle of repose of the broken muck and the angle of the lowest hole in the ring
- The angle of the apex together with the level interval will in turn determine the spacing of the production development
- Whether the cross-sectional dimensions of the production drift will constrain the size of drill and therefore drill hole size as well as other pieces of operating equipment
- Hole size and deviation; hole size will also affect damage as well as chargeability.

5.5.2.2 Level Interval

Currently, the typical industry level intervals for Open Benching and Sub-level Caving (SLC), a viable alternative to open benching, are 25m and above. The concern with high level intervals is the face stability and drill hole accuracy. Considering these issues, a compromise of 20m level interval was reached for the purposes of this feasibility. There is opportunity to increase this dimension (and reduce costs) when there is more information on geometrical variability and rock mass competence.

5.5.2.3 Production Cross-Cut Spacing

The cross-cut spacing affects the overall height of the face: the larger the interval the higher the apex between the cross-cuts. Another factor to consider is how easily the Open Benching layout can be changed to an SLC layout (if dilution is excessive and has to be managed). A conservative cross-cut spacing for SLC might be 10m. The typical dimension used in the industry for open benching has been 20m (which is conveniently a multiple of 10m). A 20m dimension was selected as appropriate.

5.5.2.4 Slot Development

Although the slot would be adjacent to, or part of, the contact zone no problems with ground conditions are anticipated due to the permafrost, thus the slot does not have to be excavated away from the contact. A conventional slot is assumed with a raise at one end of the slot drift. The first slot raise will be "drop-raised" from the pit floor. Subsequent slot raises will either be mined in advance (and covered with a concrete plug, left with a crown, or filled with dry material) or be mined from below when required. There are good examples of mines pulling 20m in one blast when adequate free-face is available (use of large hole slots or groups of large holes). The Ridgeway Mine in Australia is a good example. The Ridegeway Mine is a copper/gold, sub-level caving operation in very strong and competent rock with a geometry which, although much larger, is similar to Jericho. Once the slot raise is available the slot will be excavated with a series of parallel hole rings which will be blasted, then pulled empty, before the next one or two rings are broken.

5.5.2.5 Drill Hole Size

The economics of drilling depend on the hole size: larger holes, in a blasting context, are cheaper. The drill hole size is usually a compromise between fragmentation (a larger number of smaller holes yields better fragmentation), damage to the next ring (larger holes produce more damage), and drill accuracy (larger holes are more accurate), as well as the ability to charge up holes (charging large up-holes is difficult). For the purposes of the feasibility a 76mm hole diameter has been assumed, as this is fairly typical for the industry. During open pit operations the fragmentation and damage factors will be used to refine the hole size. The largest hole size possible will be used, as this will reduce operating costs.

5.5.2.6 Drill Pattern

The layout for the rings is based on the design powder factor, which defines the burden and spacing at the toes of the holes. A typical value for open stoping is 0.5 to 0.8kg/m^3. Kimberlite is a difficult rock to break as it "absorbs" blast energy (lower gas pressures developed due to less confinement in the hole). A reasonable compromise value for the design powder factor would be 0.7kg/m^3. A 76mm diameter hole will have a load of approximately 4.3kg/m for pneumatically loaded ammonium nitrate-fuel oil (ANFO).

The burden should be small relative to the spacing, but the burden is also critical in terms of protecting the brow and having adequate distance from the last ring and the open bench face. A reasonable compromise of 2m burden and 3m spacing results in a powder factor at the toe of 0.70kg/m^3. It is believed that the layout will be conservative and leave potential to widen the spacing dependent on results. The approximate tonnage per metre of long hole will be 8t/m.

It is recommended that maximum blasts of three rings be taken. It is also recommended that a two ring buffer be left to protect the driller from the open draw point. Safety bays should be included to protect the Load-Haul-Dump (LHD) operator during remote control loading and the long hole driller during drilling near the brow.

A ring inclination of 90^{o} has been assumed. In SLC the rings are usually inclined to the cave to assist with draw and to protect the brow. But in non-choke conditions this is probably unnecessary. Inclining the ring back would improve face stability, but would reduce brow stability.

The assumption should be made that remote control secondary breakage will be required. There is insufficient information to estimate the proportion of broken material that will need secondary breakage. A machine such as the "weasel" (used at the Brunswick Mine) would be practical together with the non-explosive "boulder buster".

5.5.2.7 Development Sizes

The following development sizes have been assumed for the feasibility:

- Ramp and access: capital development in granite will be 5m by 5m;
- Production drift: on the level the production drift in granite will be 4.5m by 4.5m;
- Production cross-cut: the production development in kimberlite will be approximately 4m by 4m.

5.5.2.8 Layout and Pipe Shape

Open benching relies on a repeatable layout, but as the pipe geometry is irregular and area reduces with depth, the layout has to adapt to geometrical changes. The production drifts, where they run parallel to the contact zone, have to be kept at a stable distance.

Having sub-production drifts within the pipe could reduce the amount of waste development. But the acute intersections that would be formed could be very unstable and would significantly interfere with the ring layout and retreat of faces. There should be no intersections in the pipe apart from those with the slot.

Layout guidelines are illustrated in Figure 5.12. This figure also shows suggested stand-off distances for the permanent ramp and for waste development on a level.

5.5.2.9 Sequence and Face Geometry

There are a number of factors that need to be considered for sequence and face geometry:

- The lag between one level and another will determine the overall face angle
- The lag will also affect the size of muckpile left in place and the length of the remote mucking
- The lag between faces on a level will determine face shape and stability

The face height and lag between faces will control the overall face angle. The average individual face height would be approximately 25m. The kimberlite is reasonably competent and the faces move relatively fast. A nominal overall face angle of 50^{o} has been assumed. With results from the open pit this angle will be refined.

To achieve an overall face angle of 50^{o} or less with an average face height of 25m the lag must be at least 21m. A typical minimum length for an LHD with the bucket down would be 10m. There has to be some control over how far the LHD can advance beyond the brow or there would be a risk of sending it over the edge of a bench. An easy way of doing this would be to ensure that the back end of the machine is kept close to the brow. This would leave a small 5m to 8m long muckpile at the end of each blast/muck cycle. Proper procedures and operator training will be required to implement these procedures. These considerations are illustrated in Figure 5.13.

The most favorable face shape on a level is concave, as illustrated in Figure 5.14. This figure also illustrates the sequence in terms of individual faces. This will impose scheduling constraints as each production drift must consider its face position relative to other faces on the level and on the levels above and below.

The corners between individual faces on a level will tend to be unstable and the faces should be kept as close together as possible. Some constraint should therefore be imposed to maintain overall shape and to prevent the shape from becoming too extreme.

Ground Support and Road Beds

The rock mass is competent and only precautionary support will be required apart from the contact zone. The kimberlites are not rapidly degrading so no unusual road bed construction methods will be required. The primary support method will be split sets, a form of friction rock bolt, with the addition of mesh in the backs for the production cross-cuts and tighter bolting patterns across the contact zones. The bolt spacing is a nominal 1.5m by 1.5m in granite and 1.2m by 1.2m in the kimberlite.

5.5.3 Overall Underground Mine Layout

The underground mine will be accessed from the 370m bench in the open pit. The decline will be mined at a 15% gradient. Overall underground mine layout is illustrated in Figure 5.15.

Each level will include a muck bay and truck loading bay as well as access to the return air raise. The return air raise may be equipped with a ladder for a second means of egress although the short life of the project should not require this.

The ventilation system is designed to force ventilate the levels, ensuring that warm air is not drawn into the production development. The ramp portal will include a ventilation door and by-pass system with a 283 kW fan to force 100m^3/s underground. This quantity is a conservative estimate, assuming all equipment is operating at the same time. The 100m^3/s is equivalent to 4m^3/s/kilotonne of ore per month, which also indicates a conservative estimate. Every effort will be taken to reduce the amount of air that is circulated (but meeting regulatory requirements) to minimize the amount of thawing during the summer months. Most of this thawing will be restricted to the granites, which are extremely stable regardless of permafrost conditions.

On the level the air will again be routed via regulators to ensure there is adequate air to all work places. Once a draw cross-cut is completed it will be blocked off. At the end of each production level drift there will be a return air raise that will be long-hole raised from level to level as required.

The initial ventilation raise will be driven from the first production level to the 290m bench using an Alimak and a 2.4m by 2.4m raise cross-section.

The individual level layouts are available upon request.

The underground mine is limited and the layout is straightforward and lends itself to operation by a contractor.

5.5.4 Operating Procedures

The ore extracted from the underground mining will be trucked to surface; it is anticipated that a 5cu.yd LHD will be used for this process. The long hole drilling will utilize a drill rig suitable to small cross-section headings such as a Quasar, which has been successfully used in underground kimberlite operations. The kimberlite can be drilled very easily and only one drill rig will be necessary. It is likely that all drilling in both granite and kimberlite will have to be dry due to the permafrost. The underground Nanisivik operation used dry drilling with success. Dry drilling requires additional equipment for dust collection, but faster drilling and greater m/bit are achievable, which should offset the additional costs.

A single LHD could achieve production with a two unit operating truck fleet. A second LHD would be used for development and for production back-up.

The chosen contractor will determine the final equipment fleet. For the purposes of this feasibility Procon Mining and Tunneling Ltd. (Procon) were used to estimate production costs and to ensure that the layouts, development sizes, and mining procedures would be practical.

5.5.5 Dilution and Recovery

Open Benching is not a selective method and has limited ability to control dilution. Dilution will be "planned" (due to layout and waste that has to be mined for the purpose of recovery of the resource) and "unplanned" (which will be mostly from instability of the contact zone). Since the outline is not yet sufficiently accurate to determine either the planned or the unplanned dilution the feasibility has assumed a conservative estimate for dilution and recovery. The in situ resource of approximately 615,000t has been diluted by 30% with waste material (i.e. grade reduced by 30%), with a recovery of 70% of the in situ resource. Overall this gives a mineable reserve of 615,000t at a grade of 0.99 recovered carats per tonne (in situ resource of 615,000t at a recovered grade of 1.42c/t).

5.6 PRODUCTION SCHEDULE

Table 5.3 presents a breakdown of the open pit production tonnes by rock type, time frame, and phase. The open pit schedule is based on three pit phases (starter, pushback, and ultimate) mined in sequence with some waste stripping of a latter phase overlapping the ore production of the earlier phase. This is illustrated in Figures 5.5 to 5.7. The production schedule has the following attributes:

- Mine life is approximately 8 years including 3 years of open pit production followed by 9 months of no production mining (underground mine development) then 2 years of underground production and finally 2 years of processing from stockpiles only;
- Mining of waste rock, low grade ore, and each main ore type (with recovered grade and diamond product in carats) is shown by time frame for both open pit and underground operations;
- Stockpiled tonnes are tracked by ore type and source (i.e., open pit or underground) for each quarter;
- Processing of open pit and underground ore, including recovered grade and diamond product for each main ore type, is shown by quarter;
- Pre-production mining occurs in Q1 and Q2 of Year 1 (i.e., Quarters 5 and 6);
- During the 9 month delay between open pit and underground production, the central lobe ore on the stockpile is processed. The richer central lobe ore is preferentially processed to maintain high cashflow; and
- Underground production of the central lobe ore is fed directly to the mill. North lobe ore remains on the stockpile.

6.0 ORE PROCESSING

6.1 INTRODUCTION

Tahera will select a professional engineering firm (the firm) for the design, procurement, and construction management of the diamond plant and its required infrastructure. Dowding Reynard & Associates (Pty) Limited (DRA) of South Africa were responsible for the design and construction of the 10 tonne per hour (t/hr) pilot plant located at the Lupin Mine and the engineering design and estimate for the feasibility study. The 10 t/hr plant was used to process the bulk sample taken from the Jericho kimberlite in 1997. Tahera Corporation has utilized information from DRA for the purposes of preparing the relevant sections in the EIS.

6.2 PROCESS DESCRIPTION

The 330,000 t per annum processing plant and fines waste disposal facilities will be located in a steel frame, metal clad building at the Jericho site. Stockpiling by the plant will ensure a continuous supply of ore is available for a year-round, twenty-four hours per day operation.

Processing of kimberlite diamond bearing ore is unique in that every effort is made to recover all the commercially valuable diamonds, while minimizing breakage. Specially designed equipment is used to sort the diamonds from the Dense Media Concentrate [Dense Media Separation (DMS)]. Currently, X-ray machines built by Debex (manufacturing subsidiary of De Beers) are being proposed.

The ore will be processed using conventional diamond processing techniques. The process flowsheet design is based on metallurgical characteristics of the kimberlite ore that was treated during the Jericho bulk sample program in 1997 at Tahera's 10 t/hr process plant.

Main steps in the proposed process flowsheet are crushing, scrubbing, dense medium separation (DMS), X-ray and magnetic separation, cleanup and sorting of the diamonds (Figure 6.1).

To accommodate the above main steps the process plant can be divided into six main areas:

- Crushing, Scrubbing and Screening Circuit – where run-of-mine (ROM) ore, that has been transported from the ore stockpile to the ROM feed bin by a loader, is crushed, wet scrubbed, and then screened to provide feed material suitable for the Dense Medium Separation stage.
- Dense Medium Separation (DMS) Plant – where washed and sized feed is split in the DMS cyclone (on the basis of density) into diamond bearing concentrate and a fine PK product. The diamond bearing concentrate is directed to the recovery circuit. The PK product is screened to produce an undersize fraction (-8+1mm), which reports to a PK stockpile, and an oversize fraction (+8mm), which reports to the recrush circuit.

- Recrush Circuit – where oversize DMS PK is crushed (Closed Side Setting or [CSS] 6mm) to further liberate locked-in diamonds
- Recovery plant – where DMS concentrates are dried, sized, and processed through X-ray machines to recover luminescent diamonds from the non-luminescent waste. The diamond-bearing fraction is processed through magnetic separation (permaroll) to produce a magnetic and non-magnetic fraction. The non-magnetic fraction is directed to the sort-house.
- Sorting house – where diamonds are sorted, weighed, and cleaned in an automated acid process using hydrofluoric and hydrochloric acid to dissolve any surface coatings and non-diamond products that would impede diamond sorting.
- Thickener and Slimes Disposal-where the (minus 1mm) slime fraction (+/-10-15% by volume) is thickened for disposal as slimes to PKCA and water is recovered for use in the plant.

Primary and secondary crushers will be located inside the building and ore will be fed to the crushers in a wet state. Processing the kimberlite ore is essentially mechanical, with only minimal use of reagents. Reagents used in the process include ferrosilicon, hydrofluoric acid, hydrochloric acid, lime, and flocculent (Percol E-10) (Material Safety Data Sheet [MSDS] can be found in the Spill Prevention, Countermeasures and Control Plan, Appendix D.2.4). The ferrosilicon is recovered as part of the process and the acids are recycled, or neutralized if no longer useable.

6.2.1 Crushing, Scrubbing and Screening Circuit

A schematic flow sheet of the diamond recovery process is provided in Figure 6.1. Ore is transported into the stockpile area in trucks and dumped onto the active stockpile (refer to Section 5). The stockpiled ore is loaded into the run-of-mine feed hopper using a Cat 966 front-end loader, or equivalent. A static grizzly scalps off (i.e. removes) any +400 mm rocks; oversize rock is broken using an hydraulic rock breaker located inside the plant. The ore is withdrawn from the hopper at a controlled rate by an apron feeder and discharges directly onto the crusher sizing screen feed conveyor, which transports the material to a combined scalping/dewatering screen for washing and fines removal. The screen oversize (+ 50 mm) drops into the primary jaw crusher, which is sized to handle a maximum particle size of 400 mm and is housed inside the building to protect it from the weather.

From the jaw crusher the ore passes onto the scrubber feed conveyor and then to a feed chute where it is mixed with the DMS and recrush circuit effluent, before entering the scrubber. The scrubber is sized to handle both the head feed and the recirculating load from the secondary crusher. Heated water is added to accelerate thawing of the ore in the scrubber. The scrubbed ore is discharged onto a double deck screen where coarse material (+28mm) is sent to the secondary crusher and the minus 1.2 mm fines removed via the screen underpan and sent to the degrit cyclone for water recovery. The screen product (-25 mm + 1.2 mm) is conveyed to the DMS 60 t feed storage bins. The +28 mm material is conveyed to a bin ahead of the crusher. It is drawn out using a vibrating feeder and choke fed into the crusher. The crusher product reports to a transfer conveyor that discharges onto the scrubber feed conveyor where it is fed back to the scrubber.

6.2.2 DMS Plant

The DMS plant is a gravity separation step unit that offers the advantages of being able to make sharp separations, removes products continuously, and treats a wide size range. The heavy medium used will consist of a slurry mixture of water and ground ferrosilicon. Heavy minerals, including diamonds, will report to the spigot of the cyclone ("the sinks") while lighter particles will exit the top of the cyclones via the vortex finder ("the floats").

Material from the DMS storage bins will be fed by a belt feeder onto the DMS feed conveyor, which feeds into a pulping box where it will be mixed with magnetic separator effluent (dirty water) and process water. The material will be screened with the oversize (+1mm) discharging to the primary mixing box, where it is mixed with ferrosilicon of the correct density. From the mixing box, the slurry will be pumped to the DMS cyclones. The separation occurs in the DMS cyclone, where (depending on their density) particles float or sink in the medium. The higher density particles move outwards and down the walls of the cyclone and are discharged through the spigot ("sinks"), while the less dense particles ("floats") move towards the central axis of the cyclone and are discharged through the vortex.

The floats discharge onto a drainage panel where most of the ferrosilicon is recovered[1]. After the drainage panel the floats pass onto a double deck float screen where the +8mm material is screened off to report to the re-crusher feed circuit. The concentrate ("sinks") report to the concentrate drain screen where the ferrosilicon is drained off. The remaining concentrate is then stored in a hopper ahead of the rotary drier in the recovery circuit.

Any residual ferrosilicon is washed off the gravels and recovered as an overdense product on the wet drum magnetic separator. The ferrosilicon is returned to the correct medium sump for re-use.

6.2.3 Tertiary Crusher Circuit

The -25 mm + 8 mm DMS kimberlite from the float screen is discharged onto the tertiary crusher feed shuttle conveyor and transported to a small bin. Material is drawn from the bin by the variable speed crusher feed conveyor and choke fed into the throat of the cone crusher. The crushed material passes onto a double deck screen, and the –8 + 1.2mm fraction from the screen is fed onto the DMS surge bin feed conveyor. The + 8 mm oversize is recycled back to the tertiary crusher feed shuttle conveyor and the screen undersize (- 1.2 mm) gravitates to the scrubber undersize sump and is pumped to the degrit cyclone.

[1] 300 g/t are lost, 80% to coarse PK and 20% to fine PK.

In order to reduce building height, use is made of a variable speed belt feeder to choke feed the tertiary crusher. The circuit is closed through a screen, optimizing diamond liberation and relieving the loading on the DMS plant.

6.2.4 Rejects System

Coarse plant kimberlite rejects (-8 + 1.2 mm) are held in a bin and are transported to the coarse kimberlite rejects stockpile as required with either a Cat 300 articulated dump truck or a 40 t class tandem dump truck.

6.2.5 Recovery Plant

The concentrate is removed from the hopper using a vibrating feeder and fed into a diesel-fired rotary drier. The drier has a drying capacity (to a maximum moisture content of 12%) of 1.5 t per hour (t/h). Dried material is then cooled and discharged onto a concentrate sizing screen. After dedusting (removal of - 1 mm particles) the material is split into a number of size fractions for passing through the X-ray sorting machines. The ore drier and dedusting areas will be fitted with a negative fan and air scrubber to remove particulates. Scrubbed particulates will be fed to the fine kimberlite discharge stream and then pumped to the PKCA.

By using a low-powered stabilized X-ray source, the need for X-ray tube cooling water has been eliminated.

6.2.6 Sorting House

Each concentrate size fraction that passes through the X-ray machines is kept separate and reports to individual glove boxes. The rejects from these machines are conveyed to a recovery kimberlite bin for later removal by truck and storage in a fenced area near the plant.

From the glove boxes the sorted diamonds will be weighed and cleaned in an automated acid process using hydrofluoric and hydrochloric acid to dissolve any surface coatings and non-diamond products that would impede diamond sorting. This will be done in a chemical fume hood equipped with scrubbers to remove the acid fumes. The effluent from the scrubbers will be neutralized and discharged to the PKCA.

Following processing and cleaning, the diamonds will be weighed and then valued on site by a Government appointed independent valuator, prior to marketing to the Canadian and world diamond markets.

An important aspect of this process is that it is essentially mechanical, with only minimal use of reagents. The only reagents used are ferrosilicon, which is recovered from the DMS plant, and acid for a final diamond cleaning. Acid is recycled and fumes are vented through a scrubber; spent acid will be neutralized.

6.2.7 Thickening and Fine Processed Kimberlite Disposal

Fine processed kimberlite (PK) generated in the plant is collected in the scrubber's desliming screen underpan and pumped to the degrit cyclone. The overflow from the cyclone reports to the Hi-rate thickener. The degrit cyclone spigot product reports to a dewatering screen, where the grits are removed and discharged onto the –8mm DMS PK conveyor. The underflow from the degrit screen reports to the screen underpan and is then pumped through a dewatering cyclone. The dewatering cyclone overflow reports to the Hi-rate thickener for further treatment.

To accelerate settling, flocculent is added at the thickener feed well; coagulant may also be required. The clarified water overflows into a launder and back to the plant process water tank for reuse in the plant (make-up water is added to the process water tank). The settled fines are moved to the centre of the thickener by the rake and then pumped to the transfer sump. The fine kimberlite is then pumped to the selected impoundment area.

6.2.8 Services

The following services are required:

- raw water;
- process water;
- compressed air;
- diesel fuel;
- power; and
- fire protection.

Raw water is pumped from Carat Lake and is used for both potable water, and make-up water in the plant. The plant make-up water is fed into the boiler and then used on the crusher sizing screen and in the scrubber to melt ice and thaw the frozen ore. The tank will be located outside adjacent to the plant or accommodation building and sized appropriately. It will be insulated to prevent the water from freezing.

Process water is used as spray water on the screens, dilution water in the DMS module and wash down water on the plant. The process water pump is fed from the clarified water tank, which in turn is fed by the thickener overflow launder. Magnetic separation effluent is reused in the DMS plant to reduce the amount of water required in the plant.

The plant has a stand-alone compressor with two air receivers, one for instrument air and one for plant air. The instrument air is used to open and close valves and operate other instruments; plant air is used to agitate the circulating medium sump in the DMS plant.

C40 Diesel fuel is used to generate power, it also fires the water boiler and the rotary drier in the recovery plant. C50 Diesel is used in the earthmoving vehicles and trucks.

All power is diesel generated at 60 Hz 600 Volts and transformed for lighting, control circuits, and overland lines. Maximum power requirements for the processing plant are 2422 kW with an anticipated running load of 1730 kVa. Power supply and transmission lines for the Project are discussed further in Section 12.

Waste heat from the power generators will be used to indirectly heat the plant building and process water for use in the scrubber. A radiator-based system with a supplementary boiler will be used to heat glycol and circulate it through the plant.

Fire suppression will consist of infrared and smoke detector units together with a zoned alarm indicator panel to be located within the security control room. A separate alarm system will be provided for the plant building; this will cover areas within the recovery and sorthouse and will be linked to the security office with a tie in to the overall site system. Due to the large number of electrical drives in the plant, preference is given to dry chemical and foam type fire extinguishers over water-based systems. Two fire water pumps, one electric and one diesel powered jockey, will be connected to the raw water tank and feed a hose reel based hydrant system. The take-offs are strategically positioned to give maximum coverage and are only to be used on non-electric fires.

6.2.9 Plant Building and Warehouses Facilities

The plant, including plant office, will be housed in a sheet-metal clad building approximately 40 x 90 m (subject to detailed engineering). Sheet-metal panels will be pre-sprayed with insulation. The plant will be sited in an area with minimal overburden, which will be removed prior to construction. Suitability of the site will be confirmed through surface inspection by a geotechnical engineer and drilling as required. Footings will be anchored in bedrock, a layer of crushed rock spread to level the site, and a concrete pad poured to hold the building. The pad will be thicker in areas where equipment requires more support; thickness of the pad will be determined by detailed engineering. Using this method of construction, permafrost will not be an issue.

Shipping containers will be converted to serve as in-plant or external cold storage units. Containers will be founded on crushed rock pads. In areas with significant overburden, 1.5 m of crushed rock will be used to prevent increase in the active layer. Where bedrock is near the surface, all overburden will be stripped, set aside for reclamation if suitable, and sufficient crushed rock laid down to level the site.

6.3 PROCESS DESIGN CRITERIA

Table 6.1 lists a summary of plant design criteria.

6.4 PROCESSING SCHEDULE

Table 6.2 provides a projected processing schedule based on the mining rate discussed in Section 5 (i.e.330,000 t per year). After plant commissioning in Year 1, ore will be fed to the plant at a rate of 330,000 t per year, initially from the Central ore stockpile and, in the last two years of mine life, from the North ore stockpile. Based on the June

2000 Feasibility Study, low grade ore will not be economic to process. However, location of additional reserves, or a material change upward in diamond values may alter this situation.

The coarse and fine kimberlite amounts do not include the water produced and assume 85% and 15%, respectively, for percentages of feed. This is the ratio expected under normal production. The increase in coarse fraction expected will not significantly affect the storage area for coarse kimberlite rejects. Processed kimberlite and water management are discussed in Sections 15 and 16, respectively.

7.0 HUMAN RESOURCES

7.1 INTRODUCTION

It is currently planned that mining and hauling operations will be contractor-operated. Tahera will operate the plant and appoint full time employees. Tahera Corporation and outsourcers will provide administration and technical services.

A maximum of 116 people will be employed for open pit mining (Years 2 to 4), 48 during underground operations (Years 5 to 7), and 40 for the processing plant (Years 1 to 8.5). Tahera is committed to a target of 60% Inuit employment by Year 5 of operations. The Project will operate as a fly-in-fly-out camp with a two-weeks-on-and-two-weeks-off rotation. Employees will be flown between the mine and the point of hire.

7.2 OPEN PIT MINING

As previously discussed, Tahera has determined Nuna Logistics, a 51% Inuit-owned firm, to be the preferred contractor for open pit mining at the Jericho Diamond Project. Table 7.1 provides a breakdown by phase of operation. The actual number of employees may change slightly from those shown. In Year 4 production will only occur on one shift. Nuna Logistics will provide administration services to support the site crew. The cost associated with this service is included in the contractor rates. Nuna will be responsible for monthly planning and progress reporting.

7.3 UNDERGROUND MINING

The projected workforce for underground mining is shown in Table 7.2. Underground mining will take place from Years 5 to7.

7.4 DIAMOND PROCESSING PLANT

Table 7.3 provides a breakdown of positions required for work in the processing plant.

The plant manager will be responsible for site administration of the mining contractors and other third party contractors, and also for safety and environmental monitoring at the plant. The manager will implement a safety management system, with all employees being required to participate in the program.

Plant Operators will be trades people and maintenance personnel who will be trained in diamond plant operations (see Table 7.3 for a list of personnel). The plant is contained in a small building, which will enhance close monitoring of operations. A Project Logic Controller (PLC) system will be installed in a central control room to assist operators.

The diamond sorters will be specially trained prior to the start of operations.

7.5 CATERING

Catering will be required throughout the life of mine. The catering requirements for the project will be place to tender at the appropriate time. Details of employment breakdown are shown in Table 7.4.

7.6 SECURITY

Tahera will employ a security team at the mine site, responsible for all components of the site, including the mine area and processing plant. A security manager will oversee the security team and will report to a corporate official. During construction of the site and prestripping, the security team will be present in reduced numbers; as the construction phase nears completion, the number of security personnel on site will increase.

Security officers will also monitor safety related concerns at the site and be trained to recognize and report basic environmental concerns. The security team will be comprised of at least two women, who will handle security matters related to female employees. In addition, security personnel will be trained in basic medical emergencies, and will have knowledge of the safety plan that is in place during mine operations.

7.7 ADMINISTRATION AND TECHNICAL SERVICES

As a large number of services required for the operation of the mine will be contracted positions, Tahera will be responsible for a limited provision of administration and technical services. Tahera has only included the cost of additional staff required to support the mine operation. The mine manager will be responsible for reporting stores consumption, annual purchase orders, and daily management reporting. The operations accountant will co-ordinate procurement and cost reporting. Tahera senior management will be responsible for the annual budgeting and operating plans. All plans will be submitted to the Board of Directors for approval.

Table 7.5 shows the positions required.

7.8 EXPLORATION

Exploration will continue throughout the mine’s life and facilities at the mine or the current exploration camp will be used by exploration crews, depending on the complement of mine personnel on site (space availability in the accommodation complex) and the size of the exploration crew. Exploration crews to date have ranged between 4 and 20 people and further exploration will likely be within this range. No new facilities will be required to accommodate exploration crews at Jericho.

7.9 RECRUITMENT AND TRAINING

Tahera management will seek to recruit Inuit employees where possible from the communities. Extensive community consultations have indicated that there is a source of skilled labour in the region, which has a long mining history.

Tahera is committed to achieving its goal of having 60% of its workforce be Inuit. This would include Inuit people working for Tahera directly and with its contractors. Although it will not be possible to achieve this goal at startup, the company believes the goal is achievable over a five-year period. To achieve this objective Tahera will liaise closely with the employment and training officers in the territory and will establish as a prerequisite a culture at the operation that supports a multi-cultural group. Tahera Corporation is a member of the Kitikmeot Employment Training Partners (KETP) and will work co-operatively with the Association, providing employment opportunities through this Association and other sources that are able to deliver trained and qualified Inuit employees. Provisions for training will be part of the Inuit Impact and Benefits Agreement.

It is crucial to have a clear indication of when the environmental review and permitting process will be complete prior to engaging training programs. Organizations such as KETP and Nunavut Arctic College conduct training programs on an ongoing basis and will be provided information on Tahera's requirements at the appropriate time. During the last year, KETP has conducted two successful heavy equipment operators courses and an industrial cooking course. In addition, companies such as Nuna Logistics (mining/construction) and Secure Check (security requirements) are aware of Tahera's proposed timetable.

Once the company is in a position to begin hiring employees, job postings will be placed on the company's website, advertisements will be placed in local and national newspapers, and community employment officers and Inuit Organizations will be notified. It is anticipated that some prospective employees may wish to leave current jobs and join Tahera Corporation.

Tahera Corporation will provide the opportunity to incorporate two apprentices into the processing plant team. It would be preferable if these apprentices were Inuit, but preference will also be given to Northern candidates. These individuals will be hired through a government-approved apprenticeship program for training in the plant, one on each of the day shift rotations. The goal of this program is to integrate the apprentices into the full time process plant team, upon completion of their programs, and then continually reintroduce two new apprentices as graduations are completed.

Where unique employment opportunities exist, such as the requirement for diamond sorters, Tahera will work with local employment officers and Inuit organizations to identify individuals who have the skills and aptitude necessary to enter in to a training program. As the diamond industry expands in Canada, a greater number of appropriate training courses are being established for these and other skills.

It will be essential to employ a number of senior personnel at the mine site who have extensive experience in the operation and maintenance of a diamond plant. Many of these people may have to be recruited externally, but it is possible Canadian candidates may be available.

Tahera will expect all contractors to follow the same philosophy as that held by Tahera Corporation with regard to recruitment, training, safety, and environmental responsibility.

8.0 ACCESS, SUPPLY, AND TRANSPORTATION

8.1 ACCESS

8.1.1 Access Methods

Surface access to the site will be on the winter road constructed annually from Yellowknife to the Lupin mine site. A continuation of this road will run north from Lupin on Contwoyto Lake for approximately 27km and terminate near the north end of Contwoyto Lake. From Contwoyto Lake to Jericho site an all season road will be built to facilitate access from the ice road. A number of methods for transport are listed in Table 8.1.

Figure 8.1 shows the winter road route from Yellowknife as it existed in 2000.

8.1.2 Road Construction

Three road widths will be required (Map A). Table 8.2 provides details.

Roads will be constructed, or widened, with 1.4 m of ROM waste rock over soil to prevent permafrost melting. Soils will not be stripped to construct roads but will be left as insulation to assist in preventing melting of permafrost. On bedrock areas enough ROM rock will be used to make a suitable running surface. Road running surfaces will be top dressed with esker material.

8.1.3 Airstrip Extension

A 150 m extension of the airstrip will be constructed in Year 1 in response to the need for additional landing and takeoff room for Hawker Siddley (HS) 748 aircraft. Approximately 18,000 m^3 of esker material will be used for construction, which will take place in the spring when the ground is frozen. The airstrip will be extended in a northerly direction at the present width (30 m running surface). No water bodies will be directly affected by construction and any sediment from runoff will be absorbed by tundra prior to reaching any water body. Drainage control is discussed in Section 16.

8.2 SUPPLY AND TRANSPORTATION

8.2.1 Overview

In Year 1, the mine and plant construction supply will commence in February (weather permitting). All materials and supplies required for construction and mining will be moved to site on the winter road. No equipment larger than that currently carried on the road will be transported.

During operation each winter, fuel and other supplies will be trucked on the Lupin winter road to the Jericho site as required. Trucking will be under Echo Bay's/BHP Billiton's License of Occupation for the winter road, as per the current arrangement with other users, plus a Land Use Permit for the winter road north from Lupin. All mine site roads will be under mining leases (DIAND for Crown land and KIA for Inuit-owned land). Either the mining contractor or other contractors will be used to haul bulk supplies. Aircraft will transport people, food, and summer resupply materials.

8.2.2 Mine and Process Plant Construction

Table 8.3 provides a list of anticipated supplies and trucking requirements. The anticipated schedule for aircraft for the construction period will be up to two Twin Otter or HS 748 class aircraft per week chartered from Yellowknife or Cambridge Bay. From time to time, additional aircraft flights may be required.

8.2.3 Mine Operation

Table 8.4 provides a list of the anticipated resupply materials and equipment for the mine operation period. Resupply will be: for open pit mining and processing during Years 2 to 4, inclusive; mostly for underground development and processing during Year 5; for underground mining and processing in Years 6 and 7; and for processing only in Years 8 and 9.

8.2.4 Mine Closure

Certain mine equipment and facilities will be dismantled and removed from the Jericho site after each phase of mining. Major reclamation will occur in Year 9, but the supply requirements will be less than those for operation. After Year 10 only monitoring activities will continue. All supplies and materials required for monitoring will be airlifted in Twin Otter, or smaller aircraft, to the Jericho strip or Carat Lake. If the airstrip is reclaimed, access will be to Carat Lake or via helicopter. Table 8.5 provides a worst-case anticipated trucking requirement, assuming no building waste materials can be landfilled in the open pit. Materials will be hauled out in the year after closure on the winter road.

Open pit mine mobile equipment, buildings, and fuel tanks not required by the underground mining contractor will be removed as backhauls on the winter road in Year 7 (January to March). A worst-case estimate of additional trucking is 10 loads for fuel trucks, 10 loads for mobile equipment, and 30 loads for miscellaneous materials.

9.0 ACCOMMODATION

9.1 HOUSING

The mining contractor will arrange housing for mine crews, which will also include the housing of all other minesite personnel. A housing complex constructed of industrial trailers will be erected at the site. Trailers will be placed on a crushed rock pad as discussed for process plant warehousing. As well, mats will be placed under the trailers to prevent permafrost melting and differential settling if building footings are on overburden and not bedrock. While conceptual at this time, possible layouts for the camp are shown in Attachment 9.1. Accommodation and kitchen construction and operation will meet or exceed all Workers' Compensation Board, Nunavut Environmental Health, and National Fire Code regulations. The accommodation building will be connected via an Arctic corridor to the plant.

9.2 HEATING

The accommodation block will have its own electrically powered boilers and wall-mounted heaters, independent of the processing plant.

9.3 FIRE PROTECTION

The accommodation block has a built-in fire detection system. The kitchen canopy and exhaust ducts will be fitted with a fusible link activated wet chemical fire suppression system. Dry chemical fire extinguishers will be located at all exits with additional units positioned strategically within the building. All doors leading outside will be clearly marked with exit signs, swing outward, and be fitted with panic bars (pressure released horizontal latch mechanisms across the centre of the door). Emergency battery operated lighting will be provided in all accommodation and kitchen areas.

10.0 WATER SUPPLY

The water supply consists of five components:

- intake and causeway with shore-based pump house;
- distribution pipeline;
- water storage tank for raw water storage;
- distribution system in the processing plant; and
- distribution system in the administration building.

A Section 35 authorization and a Section 30 approval will be required for the water intake, both of which must be obtained from Department of Fisheries and Oceans (DFO).

10.1 INTAKE DESIGN AND CONSTRUCTION

Figure 10.1 shows the proposed structure. A causeway will be built from near the mouth of Stream C1 (Map A, Appendix E) out approximately 90 m into Carat Lake to place the inlet below maximum winter ice thickness. The causeway will be constructed of run-of-mine rock and top dressed with esker or overburden material, if suitable overburden is available. Construction will take place in the spring and summer of Year 1. During construction, a silt fence will be placed around the area to prevent excess sediment drifting away from the construction site in Carat Lake. Up to 3,300 m^3 of fill material will be required. Table 10.1 lists quantities of materials required which correspond to the construction shown in Figure 10.1. Pending final confirmation of depth at 90 m from shore, the causeway length may require some small adjustments.

The intake pipe will be 45 cm outside diameter high density polyethylene (HDPE) pipe with a 5 cm wall thickness. The pipe will be buried in the causeway on one side. The intake end of the pipe will be fitted with a screened cage that meets Fisheries and Oceans guidelines (DFO 1995). The guidelines pertinent to the Jericho water intake are for subcarangiform swimming mode fish (fish that swim like trout or salmon), e.g. Arctic char, Arctic Grayling, lake trout with a minimum fork length size of 25 mm. These criteria limit intake water velocities to 0.11 m/s with a screen mesh size of 2.54 mm. The required effective screen area (ESA) is provided by the following:

$$\text{ESA } (m^2) = \text{Open Screen Area}/\%\text{ Open Area}/100$$

where open screen area and % open area are derived from tables provided in the guide.

The intake velocity for the water supply will be a maximum of 10.3 L/s, thus requiring an open screen area of 0.09 m^2. Use of #45 wedge wire screen provides a % open area of 69, thus providing an ESA requirement of

0.13 m^2. The proposed use of a 60 cm diameter screen (i.e. 0.20 m^2) meets DFO's guidelines, as well as providing a small safety margin.

The intake will be in an area of low concentration of fish year round, based on aquatic studies conducted at the site (Appendix B.1.4, Aquatic Baseline Report). As well, small fish of the criterion size are highly unlikely to survive 90 m offshore, as they would be prey for larger fish found in deeper water of the lake.

At the shore end of the causeway, two electric pumps (one running /one standby) will be mounted side by side and will pump water to the plant through an insulated heat traced pipe. A non-return valve will be placed on the delivery line and a footvalve on the suction line. A small bypass line will feed a heater and will circulate warm water through the pump to prevent freezing. A system of valves will allow switching between the pumps, which will be started remotely. Both pumps, a 4160V transformer, and associated switchgear will be housed in a heated enclosure converted from a shipping container.

10.2 DISTRIBUTION SYSTEM

From the Carat Lake pumphouse a 15 cm (6 inch) outside diameter HDPE pipe will transport water to the plant complex area. The pipe will be routed along access roads (Map A). The pipe will terminate at the raw water supply tank adjacent to the plant or accommodation building, where piping will connect to the plant and accommodation building. The system is sized to provide up to 33 m^3 per hour raw water for process use and 2 m^3 per hour for potable use. Actual anticipated usage will be approximately 30 m^3 per hour for the plant and 1.25 m^3 per hour potable water (300 L/day/person for a 100 person camp).

A chlorinator and filter system will be placed on the potable water feed. Raw process water for the plant will not be chlorinated.

10.3 MINE WATER

Water for the mine drilling operations will be trucked to the pit by the water truck; water will be drawn from the water supply tank. The requirement for mine water will be variable depending on whether kimberlite or granite is being drilled, but will be a small amount compared to other uses. In summer the water truck will also draw water for watering roads. When available, water for dust suppression will be drawn from water recycled from the pit and/or other sediment collection ponds.

10.4 PROCESS WATER

The plant will require process make up water to balance water lost in the processed kimberlite fines slurry, in the ore drier, and on the coarse kimberlite rejects, all of which accounts for the 27 m^3 per hour.

11.0 FUEL STORAGE

One fuel storage area will be constructed for all fuel requirements and will be located near the processing plant (Map A). The fuel farm will contain twenty vertical tanks that will be shipped in pieces and erected on site. As well, the existing nine tanks used for underground exploration will be moved into the fuel farm area. The general arrangement is shown in Figure 11.1. The fuel farm will be bermed to hold a minimum of 110% of the capacity of the largest tank. The berm will be lined with an impermeable, petroleum-resistant geomembrane.

The existing tank storage areas will be reclaimed. Stained soils will be incinerated to remove petroleum residues. If this treatment is not acceptable, the stained soils will be land farmed, or as a last resort removed by a hazardous waste contractor. Potential soil volumes are small (a few cubic meters). Berms will be removed and liners disposed of; method of disposal will depend on the condition of the liners, which if oil-stained will be treated as hazardous waste.

12.0 POWER

12.1 GENERATION

A lean-to extension to the processing plant will house three 1300 kW, 60 Hz 600 V generator sets (2 + 1 standby), and a 300 kW emergency camp generator all under a 5 t maintenance crane. Generators will be diesel-powered and burn low-sulphur fuel to minimize emissions. Typical fuel consumption for the units is 217 L/hr/unit at full load for the 1300 kW generators. The mine mechanical shop will have a separate 800 kW generator and a standby 800 kW generator, which will power the shop and in-pit crusher, as required. The fuel consumption for 800 kW generators will be 134 L/hr/unit at full load.

12.2 DISTRIBUTION

Power will be delivered to the plant and mine substations via armored cable at ground level. A Westinghouse (or similar) IQ analyzer situated at the powerhouse will be used to monitor usage.

12.3 ENERGY BALANCE

As a result of the high cost of diesel power at the mine site, project design has been optimized to require the least amount of power consistent with safe and efficient operation. The generators have been sized to produce only the required amount of power, thus minimizing fuel requirements. The combination of generator sets proposed will allow units to be taken off line. For example, during summer months the electrical demands are less, in part because baseboard heating units in the accommodation complex can be switched off. Mine mobile equipment fuel consumption will be closely monitored and economies of use implemented wherever possible. Figure 12.1 shows facilities requiring power. The figure is conceptual, pending detailed engineering, which may result in slight configuration changes. Power loads are approximate estimates, pending detailed engineering for the project. Table 12.1 provides an energy balance for the Jericho Project for the year of maximum mine production (Year 3). Year 1 will require approximately half the amount of fuel (essentially equals energy consumption) as Year 3. Underground mining, because of the reduction in fuel use, will require less than 2/3 the amount of fuel. Fuel consumption in the last two years of processing will be less than half of Year 3 consumption.

The accommodation block will have its own electrically powered boilers and wall mounted heaters. Waste heat from the generators will be used to indirectly heat the plant building and process water for use in the scrubber. A radiator based system with a supplementary boiler will be used to heat glycol and circulate it through the plant and offices. All external pump houses will be electrically heated.

Pumps, spray irrigation, and the in-pit crusher will be used on an intermittent basis, generally more during the summer when the load for building heat is substantially reduced.

13.0 BORROW PITS

Borrow areas were identified through reconnaissance prior to commencement of underground bulk sampling in 1996. Three of the four sites identified are shown on Map A. Volumes of fill available were quantified by geotechnical drilling in 1996 (BGC 1996b). The GSC investigated the potential for occurrence of ice lenses prior to the BGC study. BGC drilling reduced the potential area for ice lenses. The esker complex and kame delta (glacial outwash deposit) north of Carat Lake are the closest sources of granular material. Extensive geological reconnaissance in connection with mineral exploration activities over an eight-year period has not identified any practical alternate sources with respect to environmental or economic costs of extraction.

For the Jericho Project, granular materials will be drawn from the already exploited esker sites permitted for exploration bulk sampling. Maximum use will be made of run-of-mine (ROM) crushed rock and overburden where practical for roads, pads and berms. For some purposes, especially in early Year 1 when stripped overburden will be frozen or contain high amounts of moisture, use of esker granular material will be necessary, for instance where graded fill is required. Four sites with sufficient esker material to consider extraction were originally identified in the search for suitable fill material in 1995 and 1996. One area about 1 km east of the airstrip, designated Borrow Pit B (SRK 1998) was rejected from consideration, because of its distance from existing roads. The three remaining sites will be used as required. Two sites (A and C) have been exploited for exploration activities to date including construction of the airstrip, roads, and pads at the portal and exploration camp. Borrow Area D will be drawn on for fill only if necessary.

The conceptual schedule for fill material use is shown in Table 13.1. Quantities may change with detailed engineering and on site conditions found during construction.

During operation, additional fill material will be used to raise impoundment dikes and for repair and maintenance of existing structures.

14.0 STOCKPILES AND DUMPS

14.1 ORE STOCKPILES

Three ore stockpiles will be constructed during life of mine. Based on the type of kimberlite to be stockpiled, they are designated Central Lobe, Northern Lobe, and Low Grade (Map D, Appendix E). Drainage control and reclamation of these stockpiles are discussed in Sections 16 and 21, respectively.

14.1.1 Central Lobe Ore Stockpile

The Central Lobe Ore Stockpile will sit on a pad located north of the processing plant. This ore will be mined from Years 1 through 6 and processed first over the same time period. By the end of Year 6 all Central Lobe ore will be processed and the pad can be reclaimed. The pad will occupy 4 ha north of the processing plant and will be capable of holding 500,000 t. During times when mine production exceeds processing, the ore will be stockpiled in lifts up to 10 m high.

14.1.2 Northern Lobe Ore Stockpile

The Northern Lobe Ore Stockpile will be located east of the processing plant. This ore will be mined by open pit in Years 3 and 4 and stockpiled for processing in the final two years of operation (Years 7 and 8). The pad will occupy 4 ha and, with present reserves, will hold 500,000 t. Ore will be stockpiled in 10 m lifts.

14.1.3 Low Grade Ore Stockpiles

One or two low grade ore stockpiles will be constructed east of the processing plant. Low grade ore is currently not scheduled for processing. Should economics improve, e.g. higher than expected grade or external market conditions, the ore will be processed following completion of North Lobe ore processing.

The Low Grade Ore Stockpile pad will occupy 7.4 ha. Ore will be placed in lifts 10 m high with 15 m set back berms and the stockpile will have a capacity of slightly under 2 million t.

14.2 OVERBURDEN STOCKPILE

Overburden will be stripped for open pit development in Years 2 and 3 as shown in Table 14.1.

Overburden will be stockpiled directly northeast of Waste Rock Dump No. 2 and will occupy an area of 12.5 ha. When stripped this overburden will be frozen and may have a high moisture content. Run-of-mine waste rock will be used to construct a toe berm on the low sides of the stockpile area to prevent the overburden from slumping downslope. Overburden with a low organic content will be used for fill in preference to esker material in Year 2 and later, but most overburden will be retained for reclamation purposes.

14.3 WASTE ROCK DUMPS

Waste rock, granite, granodiorite, and diabase dikes (up to 5%), will be hauled to two waste dumps located immediately south and northeast of the pit area at Jericho (Map D). Waste Rock Dump 1 will have an area of 21.7 ha, hold 6.9 million t, and at completion will consist of four 10 m lifts with 15 m set backs on subsequent benches. Waste Rock Dump 2 will be 22 ha, hold 6 million t, and be of similar construction. The angle of repose of waste rock is estimated to be 37 degrees and the swell factor approximately 30%. The average slope angle (taking into account benches) will be 19° (Figure 14.1). The long term stability of the dumps takes into consideration their height and the underlying topography. As evident from Map D, the existing topography at Dump 1 is relatively flat, sloping slightly toward Carat Lake. The topography at Dump 2 is flat on the western two thirds, has a wide gentle swale trending north-south at the two thirds point, and small hills on the eastern side. As discussed in Section 16, drainage control will take the swale into account; the waste rock will be piled against the eastern hills. The factor of safety for the dumps will be more than adequate to prevent instability or failure (the normal requirement for long-term stability of granitic or volcanic waste rock is 27°).

Overburden and waste rock dumps will experience the highest concentration of haul truck traffic, due to volume of material to be moved. The schedule of waste rock placement is discussed in Section 5. Some waste rock may be used as fill to up-grade the Jericho site land access road to Contwoyto Lake.

The locations of overburden and waste rock dumps were selected to minimize impact within the Jericho site area. Site drainage is relatively straightforward to control and the sites are mostly founded on bedrock.

14.4 COARSE KIMBERLITE REJECTS

The coarse kimberlite reject stockpile will be located east of the processing plant, will occupy an area of 9.1 ha, and will be built in three 10 m lifts with 15 m setbacks. A total of 2.2 million t can be placed on a stockpile with the proposed configuration. A total of 1.19 million t of coarse PK will be generated and approximately 135,000 t will be used as cover on the PKCA cells during reclamation. About half the proposed site is currently a shallow pond approximately 1 to 1.5 m deep which will be pumped out, if practical, prior to placement of coarse rejects. Water will be discharged into the Long Lake drainage system in the summer of Year 1 upon authorization from DFO.

Once PKCA Cell 1 fills, coarse kimberlite rejects will be trucked directly from the processing plant to Cell 1 and be spread on the surface as part of reclamation (Section 21).

15.0 PROCESSED KIMBERLITE CONTAINMENT AREA

15.1 PROCESSED KIMBERLITE CHARACTERISTICS

There will be two processed kimberlite (PK) streams from the diamond separation:

- fine PK (-1 mm) from degritting, following primary and secondary scrubbing. and fines from the DMS waste (-1mm); and,
- coarse PK (-8 mm to +1 mm) from the DMS and diamond recovery plants.

Fine PK streams will be combined prior to exiting the processing plant. The fine fraction of the PK (-1mm) will be directed to thickeners for water recovery and volume reduction and then pumped to the processed kimberlite containment area (PKCA) via an insulated and heat-traced pipeline. Flocculants may be required to obtain the desired settlement characteristics. The coarse PK rejects will be handled as a solid, as discussed in Section 14. The fine portion is estimated to comprise at most approximately 15% of the total PK mass (approximately 870,000 m^3 fines at 45% slurry density or 1.8 million m^3 at 25% slurry density). The coarse fraction is estimated to comprise approximately 85% of the PK mass (2 million m^3 at a bulk dry density of 1.5 g/cm^3).

15.2 PKCA LAYOUT AND DEVELOPMENT SCHEDULE

A conceptual design for the PKCA was provided by SRK and is briefly summarized in this subsection. More information is available in Mine Waste and Water Management (Appendix D.2.1). The study will be updated as required at the detailed engineering phase after Project approval. Map A provides an overview of the concept. The PK management facility will consist of two adjacent storage cells, cell 1 to the east and cell 2 to the west. As well, a polishing pond will be established immediately west of the storage cells. Four dams (east, southeast, north, and west) and five small perimeter dikes constructed at low points around the impoundment will provide confinement of PK. A divider dike will be constructed within the impoundment to develop the PK cells. The dams and perimeter dikes will be built to an elevation of 524 m above sea level (the existing Long Lake mean elevation is 515.5 m). Figure 15.1 shows the bathymetry of Long Lake based on a July 1999 survey. Maximum height of the four dams and internal divider dike will be 7 to 9 m. Perimeter dikes will be 1 to 2 m high. The dam at the polishing pond will be built to an elevation of 518 m (above sea level), corresponding to a height of about 4 m.

The total design storage capacity for the Long Lake storage cells is 1,580,000 m^3, provided the confining dams are built to an elevation of 524 m and 1 m is maintained as dry freeboard.

In the mine construction year (spring of Year 1) dams will not be raised to their final design height (actual height to be determined pending final engineering). As well, the divider dike between cells 1 and 2 and the southeast and south dikes will not be constructed in Year 1. Construction of these dikes and raising dams to their final design height will occur in spring of Year 2.

The proposed PKCA drainage basin is small and ideally suited in that regard for fine PK storage. As well, the lake is bedrock controlled along all of the north side and much of the south side. Ground inspection by SRK and borehole data have indicated the dams can be footed on bedrock; there will be no need for a reliance on permafrost to form a seal between dams and their foundations. Waste management is discussed further in Mine Waste and Water Management, Appendix D.2.1, which includes a discussion of geotechnical investigations for dams. Geotechnical investigations are also discussed in the Environmental Management Plan (Appendix B.3.1).

15.3 PKCA OPERATION

15.3.1 Operation

Jericho processed kimberlite (PK) has been confirmed to be non-acid generating from geochemical tests completed by SRK in 1998 (refer to Appendix D.1.5). Given current reserves a total of approximately 870,000 to 1.8 million m^3 (350,000 to 720,000 t) of fine PK will be produced. If additional reserves are identified, the PK containment area can be expanded either to the north or by raising dikes. Site environmental management would not change significantly, i.e., clean water will be routed around the impoundment expansion and effluent will be routed through the west polishing pond prior to discharge.

PK supernatant water will require discharge each summer to maintain storage capacity. PK discharge is discussed below under Water Management.

For simplicity, the site arrangement map shows fine PK discharge at a fixed point in the impoundment. The main PKCA will be divided into two sections as shown on Map A. Initially fine PK will be spigotted into the east section in winter and the west section in summer, against the dams, as dictated by good engineering practice. Winter deposition will lead to ice formation; a change of cells for summer operation will be required to ensure frozen PK thaws and the volume in cell 1 reduces. Supernatant water will flow naturally into the west pond through the permeable divider dike, then into the polishing pond, and from there to the environment. Once the east pond (cell 1) is filled with fine PK, the west half of the main PK pond (cell 2) will be used exclusively, again spigotting fine PK initially against the west dam to keep water away from the upstream dam face. Fine PK will be discharged from the processing plant year round through a four-inch heat-traced line from the plant.

15.3.2 Processed Kimberlite Effluent Water Discharge Quantity

The water balance for the PKCA will determine the amount of discharge required each open water season. Water balance for the facility is treated in more detail under Mine Waste and Water Management (Appendix D.1.6, Section 16). Between 105,500 and 118,250 m^3/month will require discharge, assuming a 4-month period and depending on the mean annual runoff that occurs at the site. If over a 3-month period, monthly discharge would range from 140,700 to 157,700 m^3/month in Stream C3, which flows into Lake C3.

15.3.3 Processed Kimberlite Effluent Water Quality

Expected characteristics of the kimberlite fines flowing into the PKCA (with the exception of ammonia), based on leach tests conducted by SRK in 1998 (Appendix D.1.5) are listed in Table 15.1. Ammonia is approximately an order of magnitude higher than expected, based on nitrogen model predictions and experience at EKATI™ Mine (EKATI™, pers. comm. June 2000). Overblasting during underground mining is a possible cause.

No dilution in Lake C3 will be required to reach receiving environment water quality (assuming ammonia is low or oxidizes in cell 2 or the polishing pond). Two times dilution will be required to meet drinking water guidelines. No drinking water will be drawn from Lake C3; dilution will occur in the lake as supernatant diffuses into lake water. PK water will be tested for chemical parameters, and for acute toxicity with rainbow trout bioassay (or as specified in the Project's Water Licence) prior to discharge and at the end of the discharge season. Water will be monitored at the outflow from the polishing pond to ensure it meets Water Licence criteria.

16.0 WATER MANAGEMENT

16.1 SITE WATER BALANCE

A site water balance, based on a regional analysis, was developed by SRK Consulting and is presented in the Mine Waste and Water Management Report (Appendix D.2.1). A positive water balance exists at Jericho, which will entail discharge of water from water management structures at the mine site.

16.2 WATER MANAGEMENT PLAN

The water management plan was developed conceptually by SRK Consulting (Appendix D.2.1) and will be updated pursuant to additional field investigations required to support detailed engineering. Map E (Appendix E) shows the proposed water management facilities. The processed kimberlite containment area (PKCA) will be required to treat fine processed kimberlite from the diamond processing plant. A diversion will be required to route Stream C1 around the ultimate open pit.

Key elements of the water management plan are:

- *Water diversion ditches* located to the southeast of Dump 1 and around the south and east sides of the pit. The purpose of the ditches will be to route clean water away from the waste rock and open pit. All other facilities are located in the upper part of the catchment, and will not require significant clean water diversion.
- *Water collection ditches* located downslope of the waste dumps, overburden pile, and ore and low grade ore stockpiles. Water intercepted by the collection ditches will flow into one of the collection ponds, into the sediment control pond, or into the Long Lake PKCA.
- *Stream C1 Diversion* located to the northwest of the pit will divert surface water from the C1 catchment area away from its natural channel over the pit area. The diversion is shown on Map E.
- *Collection ponds* located south of Dump 2, south of the coarse kimberlite rejects pile, and north of the low grade ore and central lobe stockpiles. Depending on the water quality, water from the collection ponds will be discharged directly to the environment, pumped to the sediment control pond, or pumped to the Long Lake PKCA impoundment.
- *Sediment control pond* located northwest of Dump 1. The sediment control pond is designed as a settling pond to remove suspended sediments from the water. The pond will be sized to meet this objective.
- The *Long Lake PKCA*. In addition to providing storage for the fine PK, the PKCA will provide additional storage time to allow settling of fine particles and possibly ammonia degradation.
- A polishing pond at the outlet of the PKCA for final polishing of effluent prior to discharge to the environment.

17.0 SEWAGE AND WASTE DISPOSAL

17.1 WASTE WATER TREATMENT PLANT

17.1.1 Location

A waste water treatment package system will be installed at the site, housed in a stand alone building next to the plant. The building will be founded in a manner similar to the accommodation building and will be supplied heat from the plant.

17.1.2 Operation Of The Rotating Biological Contactor Waste Water Treatment Plant

The waste water treatment plant (WWTP) employs aerobic digestion of raw sewage to reduce the biochemical oxygen demand (BOD) and total suspended solids (TSS) in the treated effluent concentration to less than 30 mg/L each. Field operation of such equipment has consistently yielded effluent qualities of less than 20 mg/L for these parameters.

In addition to aerobic digestion, chlorine is injected directly into the clarified plant effluent stream ensuring the complete destruction of any pathogens and bacteria prior to its discharge. Annual discharge to the PKCA will be 11,000 m^3.

Sludge from the plant will be periodically removed as required and placed in a fenced berm a minimum of 30 m from the accommodation building. If odor or other issues become problematic the treatment plant can be equipped to dewater the sludge and the sludge can be incinerated.

Additional information on plant operation and quality of discharge can be found in the Environmental Management Plan (Appendix B.3.1).

17.2 SOLID WASTE DISPOSAL

Kitchen wastes will be burned daily in the camp incinerator, which will be an approved design.

Industrial solid wastes that cannot practically be recycled will be landfilled at a proposed and permitted site north of Dump 1. The landfill will drain to the waste dump and all water from the waste dumps will be treated.

Additional information on waste management is provided in the Environmental Management Plan (Appendix B.3.1).

18.0 HAZARDOUS MATERIALS

Table 18.1 lists the hazardous materials that will be used for mining and diamond processing at the Jericho Project. Hazardous materials management is discussed further in the Environmental Management Plan (Appendix B.3.1 and Hazardous Materials Management (Appendix D.2.3).

As a general practice, all chemical handling will strictly follow requirements as set out in Material Safety Data Sheets (MSDS). All employees will be provided with Work Place Hazardous Materials Information system (WHMIS) training. All hazardous materials will be stored in approved containers. Small liquid petroleum containers (e.g. 205 L barrels) will be kept in silled areas where accidental spills will be contained.

19.0 EMISSIONS

19.1 MINE

Emissions will be principally dust from mobile equipment and blasting. Some additional emissions of NOx, SOx and CO will result from diesel engines in mobile and stationary equipment. Finally, incineration of burnable garbage will result in a small amount of emissions, principally suspended particulates, from the incinerator.

The United States Environmental Protection Agency has developed emission factors for many mining and other industrial operations (the so-called AP-42 emission factors). The AP-42 factors were used to develop emission predictions for operations at the open pit mine for Year 3 (highest production year). These factors were used to estimate dust (total suspended particulate) emissions for the Jericho mine site. Major facilities are shown on Map A. Table 19.1 lists emission factors for waste dump 1, low grade ore stockpile, and central lobe ore stockpile. These areas will be active during the highest production period of mining; they represent the highest combined sources of dust. However, dust from waste dumps and ore stockpiles is not expected to be problematic, based on EKATI™'s operating experience. Dust from roads will be controlled with watering during months when temperatures are above freezing. Air quality is discussed further in the Air Quality Report (Appendix D.1.1).

Mobile and stationary equipment will generate exhaust emissions. Table 19.2 provides a summary.

19.2 PROCESSING PLANT

Emissions will be limited to the ore drier (principally water vapour), air from the acid scrubber, and residual suspended particulates from the crusher cyclone(s), as well as a small amount of exhaust gases from the diesel used to heat the unit. Emission factors are listed in Table 19.2 for the ore drier; emission factors were not available for the other equipment mentioned. Emissions from the drier will be scrubbed, thus removing much of the suspended particulates. The scrubber will be shared with the X-ray unit; the design is pending.

19.3 PKCA BEACHES

The PKCA beaches may generate dust if controls are not put in place. The amount of dust generated will depend on the amount of exposure of the beaches and the moisture content. During the summer beaches will be wet. During winter, as long as snow cover is in place, dust will not be problematic. Dust could be problematic at times when snow cover is blown off beaches and the surface freeze dries. Management of similar matters at the nearby Lupin Mine has consisted of spraying water on fine PK beaches just prior to winter freeze up. Limited use of snow fences has also been employed. These management techniques will be used at Jericho if dust from the PKCA beaches becomes problematic. Additional dust management will be evaluated should it be needed. Tahera will also draw on EKATI™'s experience of dealing with dust.

19.4 POWER GENERATION

The largest stationary emission source will be the power generators. Power generation was previously discussed. Table 19.2 provides an estimate of emissions based on US EPA AP-42 emission factors.

20.0 OFF-SITE FACILITIES

No Tahera-owned off site facilities are planned. Any facilities required, e.g. for hiring, expediting, etc. would be rented or leased from existing owners; most off-site services, such as those suggested above, would be contracted out.

21.0 SITE REHABILITATION AND CLOSURE

21.1 TEMPORARY CLOSURE

Should the Jericho Mine be put on care and maintenance, the stability of infrastructure such as dams, dikes, and berms would be maintained throughout the closure period. All infrastructure would remain in place and be maintained at a level that would allow a smooth transition to operation upon re-commencement of mining. Discharges from waste control facilities, such as the PKCA basin or sediment control pond would continue to be monitored prior to discharge, if required.

21.2 RECLAMATION

Reclamation is outlined in this section; further information is provided in Appendix B.3.2.

21.2.1 Reclamation Activities After Construction

21.2.1.1 Access Roads

Reclamation after construction will not be possible on the access roads shown on Map A as they will remain active throughout the mine life. Should ground disturbances inadvertently occur adjacent to the road, these will be reclaimed the summer following the inadvertent disturbance and revegetated where practical and possible. Should small spurs be required for construction purposes only, these will be reclaimed at the end of their active use and revegetated.

21.2.1.2 Sedimentation Berms and Ponds

No sediment berms, ditches, or ponds will be reclaimed after construction, except for any temporary ponds that may be required to remove sediment from disturbed areas during the summer after construction. No runoff is anticipated during construction as construction is planned for winter and early spring.

21.2.2 On-Going Reclamation

21.2.2.1 Borrow Areas

Once borrow areas are no longer required, they will be reclaimed. Borrow pits are exclusively on eskers or kame deltas and soils are granular, thus not presenting surfaces easily eroded by wind. However, any steep micro-slopes will be subject to water erosion during the summer. Removal of esker surface material will increase the depth affected by freeze-thaw; potential exposure of ice-rich soils could result in further melting and slumping. This in turn, may lead to additional potential for erosion. During active use, esker borrow areas will be managed to minimize any potential for water erosion. Once areas are no longer active, steep slopes will be regraded to the angle of repose or 3:1, as appropriate, and revegetated (if revegetation trials indicate a possibility of success). Exposed ice-rich soils will be covered with ROM rock to insulate them against further thawing.

21.2.2.2 Waste Rock Dumps

Waste rock dumps will remain active until the end of open pit mining. In Year 4 they will be reclaimed, except for a small portion of one of the dumps required for underground waste rock (estimated 57,000 t). Open pit mine equipment on site for that phase of mining will be used for reclaiming the dumps. Final regrading of dump slopes will be to attain a 2:1 slope (26°) by pushing material down onto benches. Top surfaces will be compacted from traffic use and will be ripped or scarified to loosen the surface and provide microhabitat for plants. Dumps are expected to be dry microhabitats and inimical to plant growth. If revegetation trials indicate the potential for successful revegetation of the dump tops, salvaged soil will be placed on the top or flat surface of the dump to a maximum depth of 0.3 m, fertilized, and seeded.

Sections of dumps that remain active throughout mine life will be reclaimed at closure in the manner indicated above. Side slopes will be left in a stable condition (not subject to water or wind erosion), but will not be revegetated. Slopes will be coarse rock to retard water and wind erosion. Moisture content will be minimal; the probability of successful revegetation on these slopes will be very low. The scarcity of organic soils will make its use unwise in areas with a low probability of successful revegetation. Any planting will take place in the spring or fall to avoid the possibility of moisture stress to seedlings during summer months.

21.2.2.3 Open Pit

The open pit will remain for a number of years as a large opening in the ground. To prevent accidental entry or fall into the pit by animals (such as caribou) or by people who may visit the site after closure, a rock berm will be placed around the lip of the pit. Any additional barrier requirements will be addressed as needed. The rock berm will be built of rock mined from the open pit after the perimeter is pushed back to its final position. Waste rock from the mine will be dumped directly in place and dozed into a berm.

21.2.2.4 Processed Kimberlite Containment Area

Cell 1 of the processed kimberlite containment area (PKCA) (Map A) will be filled prior to end of mine life; exact time of this occurrence will depend on the relative use of cells 1 and 2. Use of the cells will be determined during operation of the processing plant. Once cell 1 is filled it will be reclaimed the following winter by placing a geotextile fabric on the fine PK surface, followed by coarse kimberlite. The 0.3 to 0.5 m layer of coarse kimberlite will provide a buffer between PK and overburden, will prevent fine PK from being forced to the surface by frost heaving or groundwater upwelling, and will act as a filter for runoff, thus helping to improve drainage characteristics of the cap. Coarse kimberlite generated during winter operation of the plant will be placed directly on the cell, rather than the normal practice of placing coarse kimberlite on the rejects stockpile. Any additional coarse kimberlite required will be taken from the stockpile. The plant's front end loader and dump truck will be used for this operation. Once the coarse kimberlite buffer is placed, it will be top dressed with up to 0.3 m of overburden from the overburden stockpile. This will either occur in the winter or summer, depending on driving conditions on the cell. Revegetation, as indicated from reclamation trials, will take place either in the spring or fall, depending on timing of completion of the overburden placement. Organic overburden will be retained for reclamation of the PKCA as this facility has the best chance of successful revegetation. The proposed facility will occupy a shallow valley, where water naturally collects (a lake and meadows presently occupy the site); with the proposed impermeable east and west embankments, this site can be expected to provide a moist microhabitat for plant growth after closure and reclamation.

21.2.2.5 Central Ore Stockpile

The central ore stockpile will be completely processed by the end of Year 6. During the summer of Year 7 the pad will be reclaimed by scarifying and grading down the perimeter as required. Top dressing the margins with overburden will be undertaken, if reclamation trials indicate probable success.

21.2.2.6 Access Roads

Access roads are expected to be required throughout mine life. When access roads are no longer required, the road surface will be scarified, or ripped, and the surface revegetated, if reclamation trials indicate probable success.

21.3 FINAL RECLAMATION AND CLOSURE

21.3.1 Waste Rock Dumps and Low Grade Ore Stockpile

The part of the dump used during the underground mining phase and the low grade ore stockpile will be reclaimed in the manner discussed above in Section 21.2.

21.3.2 Open Pit

At the location where Stream C1 presently exits the proposed pit area, a swale, bridge, or flat culvert suitable for fish passage will be inserted in the berm. The opening will be screened or fenced off with coarse screening preventing people or larger animals from entering the pit through this gap. This reclamation activity will occur in Years 8 and 9 during final reclamation.

The final pit will not be revegetated on closure; it will be allowed to flood (see Reclamation Plan, Appendix B.3.2). Runoff, precipitation, and melt water will not exit the pit until it is filled. From these sources alone, and based on long-term average precipitation, it is estimated that approximately 150 to 200 years would be required to fill the pit, excluding snow blown into the pit during winter and evaporative loss in the summer.

To ensure only surplus water is directed back to the pit, a weir will be constructed in the diversion dam for Stream C1. Any water over weir height would flow both to the C1 diversion and over the weir, into the old Stream C1 channel and thus into the pit.

An alternative to allowing the pit to flood would be to backfill with waste rock. This alternative was examined and rejected on economic and practical grounds. Cost to backfill the pit would be approximately $2 per tonne. Backfill could not occur until the end of mining in Year 7. At that time much of the mass of waste rock dumps is expected to be infiltrated by permafrost. This analysis leads to the conclusion that backfilling the pit would make mining the deposit uneconomic.

21.3.3 Processed Kimberlite Containment Area

The remaining cell (2) will be reclaimed at end of mine life in a manner similar to cell 1, except that all coarse kimberlite must be taken from the stockpile as the plant will not be operating. The polishing pond will not be reclaimed to land. The polishing pond dam will have a permanent spillway constructed of concrete or rip rap armor that will pass a 200 year flood for the small drainage basin left at end of mine life. Water quality in the polishing pond will need to be at CCME receiving environment quality prior to final abandonment.

21.3.4 Coarse Kimberlite and Recovery Plant Reject Stockpiles

The remainder of the coarse kimberlite and recovery plant reject stockpiles not used for reclamation will be sloped to 24° and covered with up to 0.3 m of overburden to prevent dust generation from fines incorporated in the rejects. A shallow ditch will be constructed on the downslope side of the stockpile to capture any runoff and prevent export of sediment overland in the initial years after reclamation and prior to surface consolidation. Monitoring will continue at the site through this time.

21.3.5 North Ore Stockpile Pad

The ore stockpile pad will be scarified and revegetated, if success is indicated from reclamation trials. Overburden will be added to the perimeter of the pad if reclamation trials indicate probable plant growth success.

21.3.6 Mine and Access Roads

Mine and access roads no longer required will be scarified or ripped and possibly revegetated. Road edges will be graded off to form a gentle slope, yet minimize additional disturbance of tundra. The road between the camp and the airstrip will be left in a stable condition until final closure, and may be left unreclaimed at final closure, if requested by a government agency or by a third party who agreed to assume responsibility for its maintenance.

21.3.7 Sedimentation Ponds, Berms and Ditches

Once no longer required, as determined by water quality monitoring after mine closure, sediment ponds and berms would be breached and normal drainage patterns restored at the site. Ditches will be stabilized where they pass through overburden; ditch portions in bedrock will remain as constructed. To the greatest extent possible, ditches will be altered to return drainage to pre-disturbance conditions. Prior to leveling, any sediment in ponds would be removed and placed on one of the dumps, then covered with waste rock or overburden to retard wind or water erosion. Conceptually, all dikes and berms would be flattened and revegetated. Any remnant of berms or pond dikes left after resloping would be stabilized and revegetated.

21.3.8 Borrow Areas

Any borrow areas active to the end of mine life will be reclaimed and revegetated as discussed above. Some borrow material may be required for final reclamation; such a borrow area would be one of the last sites to be reclaimed and revegetated.

21.3.9 Airstrip

The airstrip will be kept open until final closure, for use by Tahera Corporation reclamation personnel. When no longer required the airstrip will be scarified or ripped and vegetated, pursuant to probable success as shown in reclamation trials. Alternatively, the strip could be left for others to use, if requested by government agencies or by a third party willing to assume the airstrip land lease.

21.3.10 Infrastructure

Infrastructure at mine closure will include:

- portal to the underground access ramp;
- vent raise (if outside the pit);
- the accommodation and mine office;
- the fuel farm;
- the explosives magazines;
- truck shop;
- explosives truck shop;
- the laydown areas; and
- the ammonium nitrate cold storage area.

The portal and vent raise will be sealed and all buildings will be torn down. If acceptable to permitting authorities, building scrap will be placed in the open pit and covered. If not, buildings will be removed from the site.

Foundations will be covered with soil. The fuel farm tanks will be emptied into tanker trucks, disassembled, and removed as will all warehousing and trailers. At least part of the exploration camp will be maintained for post closure activities accommodation. Weatherhaven™ tents can be air freighted from the site at abandonment. The explosives magazine trailers (likely steel bulk shipping containers) will be removed.

The water intake will be sealed off at the Carat Lake end and the piping removed from the upland sections. During winter, the causeway will be graded down to a 3:1 or less slope. Any in-water infrastructure, such as mine rock cover on pipes, will be left, as it will likely have become useable fish habitat.

All non-salvageable scrap metal left at closure will also be placed in the open pit and covered, or removed from the site as per buildings. All necessary removal of infrastructure will take place the first winter of final closure, thus obviating the necessity of constructing the winter road beyond the end of mine life plus one year. Further details on infrastructure are presented in the Reclamation Plan (Appendix B.3.2)

21.3.11 Soils Testing

Any soils suspected of being contaminated (stained) with petroleum hydrocarbons will be tested and those not meeting criteria for industrial purposes (at the time of mine closure) will be remediated. Remediation will preferably be done on site, e.g., by land farming, but may be done off site, in which case soils would be transported off site by truck during the first year after mine closure, or later by aircraft if required (see Reclamation Plan, Appendix B.3.2, for more information).

21.4 MINE ABANDONMENT

Mine abandonment refers to the stage at which all reclamation activities aimed at rehabilitation and stabilization have been completed, all infrastructure removed (or transferred to third parties to manage, e.g. if the site is taken over be a guide-outfitter), and the only activity is post closure monitoring.

A reclamation plan is provided in Map F. Objectives of abandonment are presented in the Reclamation Plan (Appendix B.3.2).

21.4.1 Infrastructure

Infrastructure remaining at abandonment will depend on the intended use of the site. Assuming complete closure, all buildings will have been removed, and all roads and the airstrip rehabilitated and permanently stabilized against erosion. At the election of Transport Canada, or other government agency, the airstrip may be left intact, likely with removal of the landing lights and associated cabling. This would only occur if the federal land lease for the airstrip can be terminated by Tahera while leaving an unreclaimed airstrip.

21.4.2 Drainage Controls

Natural drainage patterns (to the extent possible) will be re-established at closure. At abandonment, all drainage systems will be confirmed to be stable. The Stream C1 diversion channel will remain intact to ensure the lower end of the stream does not dewater. The diversion dike will be stabilized for long-term maintenance-free operation.

21.4.3 Sedimentation Ponds

Sedimentation ponds will remain in place until post-closure monitoring indicates water quality is acceptable for discharge directly to the environment. At that point berms will be removed and pond surfaces revegetated. Final abandonment of the site will not take place until ponds can be removed.

21.4.4 Land Use at Abandonment

21.4.4.1 Wildlife Habitat

Disturbed areas, other than mesic and moist soil microhabitats will only very slowly revegetate. Wildlife habitat lost to create dumps, pads, and roads will regain pre-disturbance productivity at the same rate as vegetation returns. Every practical effort will be made to accelerate this process.

The waste rock piles may be used by raptors as lookout perches for prey. Ground squirrels are known to nest in natural piles of rock and may use crevices in the waste rock as burrows where adjacent vegetated areas can provide forage. Tops of dumps will be used by birds, small mammals, and carnivores for foraging. The open pit will be flooded as discussed and the area of the pit would be permanently lost as terrestrial wildlife habitat (10 ha).

21.4.4.2 Fish Habitat

A lake will form in the open pit from drainage water as discussed previously (end pit lake). Tahera Corporation understands end pit lakes are not considered fish habitat by DFO. Nonetheless, once the pit fills and water once again flows in the pre-mining Stream C1 channel, fish will have access to the filled open pit, which will form a small lake. The lake will be deep (approximately 100 m), but will have narrow shallow margins around the edge and a somewhat larger shallow area where the pit access road slopes into the pit. The end pit lake will follow a primary succession sequence. The lake will likely remain permanently oligotrophic, similar to Carat and other lakes in the area.

21.4.4.3 Recreation

There is little chance of future recreational opportunities at the Jericho site, unless people chose to fish at Carat Lake (unlikely given the lack of access and proximity to Contwoyto Lake, which supports large game fish). However, should a guide-outfitter decide to make use of the site, some of the site facilities could be used for recreational purposes, such as hunting and fishing at Contwoyto Lake. Since caribou frequent the site occasionally in large numbers, wildlife photography could also be promoted. The setting of the site is quite photogenic, as is tundra vegetation, especially in the fall of the year.

21.5 MONITORING AFTER CLOSURE

Monitoring after mine closure will be in two phases:

- immediately post closure, until Tahera is assured long-term facilities, such as waste rock dumps and the C1 stream diversion are stable; and
- longer-term monitoring of water quality of site runoff to ensure the improved quality predictions bear out and water quality at the site improves and does not degrade.

21.5.1 Post Closure Monitoring

During this period immediately after mine closure, sedimentation ponds, berms, and outfall (if required) will be maintained. Water quality will be monitored on a monthly basis for parameters controlled by the Jericho Project Water Licence in place at the time of closure, until such time as those parameters meet CCME (or applicable at the time) guidelines for protection of freshwater aquatic life. At that point, sediment control structures will be dismantled and reclaimed. As waste dumps will be reclaimed after the end of open pit mining in Year 4 there will be several years monitoring data to ascertain the trend in runoff water quality from these areas.

Rock dumps and infrastructure associated with the Stream C1 diversion will be inspected on closure by a qualified geotechnical engineer for stability and their report recommendations implemented

Monitoring will be in late winter/early spring (April) and monthly during the open water period (July, August, September).

21.5.2 Post Abandonment Monitoring

Post abandonment monitoring will continue until resource agencies, notably Nunavut Water Board and DIAND, agree to cessation. For mines without an acid generation problem, monitoring for a five-year period after closure is typical; however, each mine is judged on its own merits. Monitoring will include annual visual inspection of rock dumps and the Stream C1 diversion as well as the four water quality monitoring sites sampled once annually in mid to late summer.

Biological monitoring will continue only if runoff water quality from the site does not achieve the receiving guidelines or better. Once these guidelines are achieved, biological monitoring will cease.

21.6 AESTHETICS

Tahera will address the issue of aesthetics on closure to the extent practical. No infrastructure scrap will be left; it will be burned, buried, or taken off site. However, evidence of past mining will remain; it will not be practical to level the two waste rock dumps nor to completely refill the open pit. Waste rock dumps will be approximately the same elevation as the lower of the surrounding hills.

All areas that can be practically revegetated will be revegetated. The site will slowly green up, but such processes take decades on the Arctic tundra and therefore for many years after mining, bare rock and soil will be visible. A total of 221.8 ha will be disturbed by mining and, except for the open pit, will slowly return to natural vegetative cover. For comparison, the hamlet of Kugluktuk is over 1000 ha, or four times the size of the Jericho Mine footprint.

The probable aesthetics of the reclaimed and abandoned mine site must be viewed in the context of the setting. The Jericho Project is in a remote location with access limited to snowmobile in the winter and aircraft year round. Prior to exploration activity at the site in the early 90's, people did not use the site to any significant extent.

21.7 COST

The estimated total reclamation cost for the Jericho Project is $7.35 million. This will be reduced by approximately $530,000 after the end of Year 4 with reclamation of waste rock dumps, and removal of unneeded open pit mining equipment. An estimate provided by the probable open pit mining contractor is attached in Attachment 21.7.1.

22.0 PROJECT ALTERNATIVES

22.1 THE NO PROJECT ALTERNATIVE

From a strictly economic view, the no project alternative would result in a significant lost opportunity, since tax and royalty revenues to government, and employment and business contracting opportunities to individuals and companies would be lost. Beyond that, the attraction of Nunavut to other potential diamond mine investors would be lost. An operating diamond mine in Nunavut will act as a lightning rod to attract other investment in the diamond (as well as precious metals) industries. The potential for development of a secondary industry (cutting) exists with a mine in Nunavut. Finally, many of the job skills learned in the mining industry are directly transferable elsewhere, e.g., heavy duty mechanics and heavy equipment operators, environmental and geological technicians.

The government of Nunavut receives transfer payments from the federal government of about $21,000 per resident (Howatt, 2002). In 1996, 44% of the workforce was employed in government or government related services such as education and health care. In 2000-2001 the hamlets of Cambridge Bay and Kugluktuk employed a total of 228 people and had a combined payroll of $4.3 million (NPC, 2002). A large number of the government jobs are taken up by a specialized non-Inuit labour force (NPC, 2002). Mineral exploration activity in the Region has increased in the past few years. In all of Nunavut Natural Resources Canada reported that $37.4 million was spent in 1999 or 7.4% of the mineral exploration dollars in Canada, $62.1 million or 12.5% was spent in 2000 and $61.3 million or 12% was spent in 2001 (NRCan, 2002). In 2002 it is estimated that $67.8 million or 13.5% of the country's total exploration dollars will be spent in Nunavut (NRCan, 2002). Of the $61.3 million spent on 36 different projects in 2001, fifty percent of the projects were located in the Kitikmeot Region (DIAND, 2001). From the above it can be seen investment in mineral exploration in the Kitikmeot Region is substantial. However, to maintain and encourage this exploration it is imperative that there be an expectation that economic mineral discovers can be exploited. This in turn requires a favourable regulatory climate which can only be provided by government. The competition for mining investment dollars is the entire world. Infrastructure in many parts of the world is much better developed than in Canada's north. A major incentive to investment then has to be a stable regulatory regime that is seen to be fair and equitable and not overly onerous. If regulatory burdens are too high, investment dollars will flow elsewhere.

From an environmental view, the no project alternative would mean no impacts from mining. However, viewed in a regional context, negative impacts from the Jericho Mine will be very limited, both spatially and temporally. Environmental management plans will ensure impacts are reduced to the extent practical and limited to the site. Residual impacts will be few and mostly medium term (life of mine). Management plans have been made adaptive and flexible to expected and unexpected changes that may occur.

22.2 MINING

The Jericho kimberlite is the only kimberlite resource belonging to Tahera Corporation that can be economically developed at this time. If other kimberlite pipes are proven to be economic, they may be added to the mine plan or a separate mining facility would be developed. In either circumstance, this would entail a separate environmental impact assessment and permitting process.

22.2.1 Open Pit

Alternatives of the open pit would not appreciably affect environmental impacts of the project, but are important for economic considerations. Both paced and accelerated production were considered as alternatives for open pit mining. Accelerated production will require the construction of two main ore stockpiles (for central and northern lobe ore) and one low grade stockpile (see Map A). Ore stockpiles will be largest at the end of Year 4 when the central and northern lobe stockpiles will contain about 400,000 and 500,000 t of ore, respectively. The low grade stockpile will then contain 1.7 million t.

The alternative to accelerated mining was to mine at a rate in balance with the process plant (i.e., 330,000 t per year). Estimates indicated higher unit mining costs (due to the lower production rate) and a high fixed cost (due to the carrying costs for equipment trapped on-site), resulting in less than optimum cashflows.

The choice of accelerated production will not result in any increase in the footprint size, nor in the final configuration of mining units such as waste dumps or pads.

An alternative to the combined open pit and underground mining is an entirely underground operation. In remote and high cost areas like the Jericho Diamond Project area, project development would not be feasible without initial open pit mining. Only open pit mining provides both high extraction and development rates, relative to capital expenditure, to repay the heavy investment costs required for the infrastructure. Further, as discussed in Section 5.3, dilution of ore is less with open pit mining. Finally, a conventional and cost effective means such as a combination of open pit and underground mining methods, may positively contribute to securing financing for the project, compared with a completely underground mining operation.

At approximately 180 m depth, open pit mining will cease and underground mining will commence, as previously discussed. As the open pit reaches greater depths, waste rock removal becomes excessive, and underground mining will become more economical. The economics of underground mining are based on the geometry and value of the kimberlite, the cost of ramp and sublevel development, rock mechanics considerations, and the direct mine operating costs.

Mine operation (i.e. open pit mining) by Tahera was considered, as opposed to the use of a mining contractor. The conclusion was reached that the additional capital needed to purchase equipment, would make it difficult to for the company to finance the project. Conversely, mining contractors have an inventory of equipment that can be used on other projects and they are also better positioned to deal with the depreciation of the equipment. Finally, it is felt that sufficiently experienced and qualified Nunavut contractors exist to carry out the open pit mining operations at Jericho.

22.2.2 Underground

Selection of a mining method nearly always involves a compromise between cost, resource utilization, dilution, and production rate. In this respect it is useful to consider the evolution of most underground kimberlite operations. All of them started with open pits of one sort or another, which then converted to glory hole methods until control of dilution became critical and then caving methods were used.

Current underground kimberlite operations favor block caving if it is applicable. Considerable sub-level caving has been tried but there is very little done now. Open benching has also been widely applied. There have been some trials with open stoping and vertical crater retreat but with variable results. There are no backfill diamond operations.

There are three main groups of methods that can be considered:

- **Caving***:* natural (block/panel caving); or pre-break, such as open stoping with pillar wrecking or sub-level caving;
- **Open***:* partial extraction (with permanent pillars) or open benching;
- **Supported***:* various cemented or uncemented backfill methods.

Some of these can be eliminated as being inapplicable under any circumstances:

- **"Natural" caving**: not possible as the pipe is too small; the footprint at the lower part of the pipe has a Hydraulic Radius of 10m and the kimberlite needs a HR over 20m to cave;
- **Partial extraction**: not possible as the kimberlite is too weak and the deposit too large; much too much ore would be tied up in pillars and stability could not be guaranteed.

Two methods were then considered further:

- **Open benching**: similar to sub-level caving, but pulled empty after each blast, so not a choke method, and draw points would be pulled open;
- **Sub-level caving**: this method would be used, if problems with dilution and open benching need managing.

Before finalizing on method choice and then design it is useful to consider the rock mass and geometry described in previous sections.

22.2.2.1 Design Background Information

Rock Mass

The rock mass can be discussed under the following headings:

- Wall rock: the granites are extremely competent and strong;
- Kimberlites: the ore rocks are reasonably competent and there is no indication of rapid weathering, which can be a feature of kimberlites;
- Contact zone: there is a well developed contact zone which will affect mining costs and the amount of dilution;
- Permafrost: all the underground mining will be within the permafrost zone and this will greatly enhance stability during underground development and extraction;
- Major weakening structures: there are no indications of major through-going structures that would significantly affect underground mining and in any event the permafrost is an effective “cement”.

Geometry

The amount of drilling is adequate to assess the indicated grade, but the exact outline of the center lobe is not available. The resource tonnage at this stage is conservative in terms of where the center lobe boundaries have been drawn. The critical consideration for the underground mining is the degree of geometrical variability of the outline, as this will affect the amount of planned dilution included in the estimate.

The current outlines are sufficiently accurate for the choice of method, mine design and layout, and for the costing of the underground extraction.

By the time final layouts are drawn for the underground there will be much more detail on geometry from the existing open pit as well as additional drilling.

22.2.2.2 Open Benching

The two prominent examples that have been used for reference are Finsch and Koffiefontein mines. Layouts from Finsch are shown in Figure 22.1. The basic layout is similar to sub-level caving. Open benching is very similar in concept to an open pit; the only difference being that drilling, blasting, and mucking are done from within drifts. At Finsch and Koffiefontein the value of the kimberlite is relatively low and every effort was made to keep open benching competitive with block caving. This resulted in an aggressive layout with long drill holes and wide spaced development.

The problems experienced with the method as operated have been as follows:

- Instability of the production or stope faces;
- Loss of holes and rings;
- Very large slope failures that have "threatened" the mine's ability to produce;
- Mud and water causing significant operating problems;
- Major slope failures through contacts between facies; major dilution problems where these failures were through waste.

These problems resulted in conversion to block caving. It is important to distinguish between Jericho and the South African (SA) operations. SA mines have much larger pipes with large level intervals and the faces are very high. They also have significant differences between facies, which constitute major weakness planes and very long holes with much more potential for deviation and misfires. Many of the problems could have been overcome with a tighter development layout, admittedly at higher costs. But the most important difference is the permafrost, which makes the short term ground behaviour very different. The amount of deterioration at Jericho will be limited, and it is believed that the mine site will maintain the frozen conditions.

It is still important to recognize the problems that have been experienced and ensure that the mine layout and design for Jericho allow for correct management of the risks. The concerns with the open benching method at Jericho might be:

- **Remote control loading**: much of the draw would be in the open and operators would have to stand in a safety bay;
- **Open bench stability**: individual face heights would need to be limited;
- **Coarse fragmentation**: small blasts in potentially damaged and unstable faces; need for secondary blasting;
- **Water and mud management**: surface water during freshet and from ground water combined with kimberlite fines; strict management procedures would be required to prevent any build-up of water and/or mud;
- **Ventilation**: open draw points; would need to regulate air onto levels;
- **Loss of brows**: over-break of brows and need for charging and hole cleaning in an open draw point;
- **Pipe wall stability**: the pipe walls and contact zones may be a source of loose material and/or massive failures; dilution and possible risk at open draw points.

There is little concern with open benching near surface or the base of an open pit as face heights and wall exposure would be moderate.

Open benching is likely to be the least expensive of the methods, as long as it performs adequately. The major concern would be with dilution from the pipe wall. Dilution cannot be controlled and all waste would find its way to the draw point and finally to the process plant.

22.2.2.3 Sub-Level Caving

Sub-level Caving (SLC) is a choke method and it relies on broken material to confine the blast and keep the broken ore near to the draw point. Although it is considered a high dilution method it works well in massive, steep ore bodies as the dilution mostly comes from above, where it is mixed with the ore that is left behind after each individual ring blast.

The method is illustrated in the series of sketches in Figure 22.2.

As each ring is drawn it mixes with sub-economic material from above and around the ring. An economic draw point shut-off value determines the overall recovery and grade factor that is achieved. There should be a reasonable visual difference between granite and kimberlite and visual grade control at the draw point is a standard procedure for SLC. The overall dilution can be limited at the expense of recovery.

SLC is ideally suited to very strong and competent rock masses. The kimberlite at Jericho should be adequate.

SLC would be more expensive than open benching, because the geometry for effective draw requires a tighter development layout, illustrated in Figure 22.2, and more drill holes due to a higher powder factor for choke blasting. An SLC layout is also less adaptable to a variable geometry, due to the much higher draw angle (need to ensure a regular layout whatever the ore geometry). But the use of SLC may overcome some of the concerns with open benching, such as:

- No requirement for remote control loading; no need for remote secondary breakage;
- Draw points always full of muck so less problems with ventilation;
- Some control of dilution; large failures would fall on top of broken ore; but typically dilution achievements are around 20/30% ;
- No open benches.

SLC has some of the same problems as open benching, such as concerns with brow stability and loss of drill holes. But there are also some additional concerns:

- Mud and water may collect in the bottom of the open pipe; this could be risky to operators if muck rushes occur and can also lead to very "sticky" muck which could cause operating problems;
- Wet broken material could freeze, bridge, and interfere with the draw.

The very serious problems experienced at some SLC operations in SA kimberlites were a function of the height of broken material above the draw points. The height of broken material would be much less at Jericho, as there is not a ready source of diluting material. However, it should be noted that a minimum depth of broken material is needed to generate adequate choke conditions for dilution control.

SLC is feasible and would overcome some of the concerns with open benching, but would be more expensive and create some additional concerns. It is also less flexible and at this stage (with only 80m of open height) there seems little advantage to SLC. But it is a very good method for risk management. We know that it will work and it would be easy to adjust the Open Benching method to SLC by tightening the development (a doubling of the production cross-cuts) and still achieve similar production rates but at slightly higher costs.

22.3 ORE PROCESSING

An alternative to mining and processing at Jericho is construction of a processing plant at Lupin Mine, stockpiling of ore at Jericho and campaign hauling of ore to Lupin over a winter road to the Lupin Mine. This alternative was discussed in detail in Tahera's November 26, 1999 Project Proposal (Tahera 1999). Pre-feasibility level studies carried out for Tahera by SRK Consulting showed lower capital costs, but higher operating costs, because of the requirement to haul ore 29 km to the Lupin site. Some environmental impacts, such as those related to the process kimberlite containment area, might be less with a satellite operation, since it has been speculated that PK containment areas would be shared under this scenario. A satellite alternative requires a high level of mutual technical cooperation between owner and tenant. A number of regulatory requirements, such as liability and security related matters, make it difficult to contemplate the use of an existing operating site by another tenant.

There are no economic alternatives to the proposed method of ore processing. The recovery process of diamonds from kimberlite is well tested and proven. No alternate process is available to be considered.

In arriving at the selected site location for the processing plant, a number of alternatives were considered.

i) In the valley next to the winter road access – the ground conditions were considered to be too wet.

ii) On the ridge on the hill above the winter road access; however, a suitable area for coarse PK disposal could not be found. This option also added approximately 1km to the fine PK disposal pipeline.

22.4 MATERIALS HANDLING FACILITIES

An analysis of site alternatives for mine components (waste rock dumps, ore stockpiles, PKCA) has been conducted. The three waste dump sites shown in Map B are viable, however, analysis favours splitting the waste into Dumps 1 and 2. Although Dump site 1 has the capacity to hold all planned waste rock, use of Dump site 2 will reduce the height of a single waste dump located at Dump site 1 and allow for more flexible management of ROM waste rock and lower trucking costs. Evident from Map A is that drainage control at Dump site 3 would be more problematic.

Ore stockpiles, for economic reasons, need to be placed close to the processing plant. While alternate locations are possible, the placement preferred results in the least amount of distance for rehandling. This in turn will reduce fuel consumption, meaning less fuel needs to be hauled to site in winter and less emissions from mobile equipment will result during operations.

22.5 PROCESSED KIMBERLITE DISPOSAL

Several options for PKCAs were examined at various times. All but the Long Lake option were previously discussed by SRK (1998). In 1999, the use of Long Lake was examined. Preliminary geotechnical engineering and environmental studies suggested Long Lake would provide an acceptable location for processed kimberlite fines. Map B shows the location of all basins examined for PK fines disposal, as well as a number of alternate sites for the plant and PKCA. From both economic and environmental considerations, the preferred site has advantages. Use of Long Lake for PK fines will result in a much shorter fine PK line. Further, no water bodies will need to be crossed, thus eliminating one source of accidental spills to the receiving environment. Long Lake has a very limited fish population, i.e. slimy sculpins and burbot. Both Key and Lynne lakes contain trout and char. Pocket Lake and the adjacent unnamed lake (south of Key and Lynne lakes) do not have fish, but ore would have to be hauled a much greater distance from the pit to a processing plant located south of Lynne Lake. As well, there is a small potential for ore to be accidentally spilled into water courses that would require crossing by the ore trucks.

Likewise an alternative for the plant to be located east of the existing exploration camp would require PK fines to be piped a much greater distance and over the stream connecting Key and Lynne lakes. There is a well-used caribou trail corridor that traverses the area east of Lake C4, and passes just west of Lynne and Pocket lakes. Any infrastructure in this corridor has the potential to interfere with caribou movement and thus increase potential Project impacts to caribou.

22.6 PROCESS WATER RECYCLING

Plant make up water requirements will be 23.7 m^3/hour. This water will be drawn from Carat Lake as previously discussed. Free water in the PKCA is unlikely during winter; all water may be locked up in ice. Fine slurry from the plant will likely freeze rapidly in winter and would be the main source of water flowing into the PKCA, effluent from the Waste Water Treatment Plant (WWTP) being the other. Water could be recycled from June through

September and reduce the required draw from Carat Lake by approximately 56,000 m^3. This would represent from 18 to 21% of the required discharge from the PKCA to Lake C3, which flows into Carat Lake.

There are a number of disadvantages to this system:

- Recycling would not result in a totally closed PKCA facility, as water would still need to be discharged each year to maintain a balance in the facility.
- Two separate water supply systems would be required: one for winter and one for summer at added capital and operating expense.
- Water from Carat Lake will be required for potable use year round and for the processing plant in the winter; thus the water supply infrastructure discussed previously would still be required and no reduction in surface disturbance would result from recycling water from Long Lake.
- Approximately 2 km of supply line would be required to run from the west end of Long Lake to the processing plant and a pump and reclaim barge would be required at Long Lake. This would add to capital and operating expense.
- Either a diesel pump, a power transmission line, or a separate generator would be required at the barge site with some increase in the risk of a fuel spill into Long Lake. A fuel spill could require cessation of PKCA discharge until the fuel could be soaked up (if that were possible). Cessation of discharge for any significant period would force an increase in discharge rate when discharge recommenced in order to draw down the PKCA the normal amount by the end of the open water season.
- Discharge water quality from the PKCA would remain unchanged.

In summary, recycling would add significant capital and operating costs but would provide very little, if any environmental benefit.

22.7 TRANSPORTATION ALTERNATIVES

The only economical method of transporting bulk materials and supplies such as fuel and ammonium nitrate (the two largest items required for mining) is presently via winter road (the Echo Bay Lupin Winter Road), which currently connects Yellowknife with EKATI™, Diavik, and Lupin. A 29 km extension on Contwoyto Lake would be required to connect Jericho Mine to the existing winter road. A short, 3.5 km extension up Lynne Creek, across Lynne Lake and through a draw to the Jericho Mine, or all weather road north of Lynne Lake would be required to connect the Mine to Contwoyto Lake. The winter road typically operates from January through March each year.

Capital and operating cost refinements favor the latter alternative of an ice road over Contwoyto Lake.

If an all weather road were built, either from Yellowknife to Kugluktuk or from Bathurst Inlet to Lupin Mine, a shorter winter road from the Jericho site to connect to the all weather road would be required. An all-weather road to complete the connection to Jericho Mine could not be supported by the current economics of the Jericho Project.

22.8 DIAMOND MARKETING

Rough diamond marketing is a complicated process, historically dominated by the Diamond Trading Company (DTC) (formerly the Central Selling Organization (CSO) operated by DeBeers); however it is becoming increasingly common for companies to market their product independently or through alternative marketing routes to the DTC.

A final decision regarding diamond marketing will be made prior to the commencement of commercial production.

23.0 POTENTIAL FUTURE DEVELOPMENT

Kimberlites tend to occur in clusters of 2 to 70 occurrences per group. Measurements in Botswana, Siberia, Australia, and Lac de Gras indicate that the diameter of these groupings can be up to 50 km. In addition to the three Jericho kimberlites, the Contwoyto-1 kimberlite was discovered in Fall 1998. There is further potential for the discovery of additional kimberlites as Tahera continues to explore the Jericho Diamond Project area. The Jericho and Contwoyto kimberlites can be considered to represent a kimberlite group.

Tahera will only advance prospective kimberlite discovery through the evaluation process of caustic fusion, mini-bulk sample, bulk sample, and through to the feasibility study. Additional economic kimberlites may have the effect of rescheduling the production sequence, additional mine sites, and infrastructure. Further improvement to the project economics may be realized with future discoveries resulting in positive socioeconomic effects. Although under the control of regulatory agencies, Tahera expects that any new exploitable kimberlites found outside the footprint of the present proposal would require a new environmental impact assessment and additional permitting.

There are no economic kimberlite deposits that could add to Jericho production at the current time.

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TABLES

TABLE 1.1
SUMMARY OF KEY LEGISLATION GOVERNING THE JERICHO PROJECT

Legislation	Jurisdiction	Governed Activities/Target Environments
Northern Inland Waters Act	Nunavut	Water Licence; use and disposal of water
Territorial Land Act	Federal	Leases on federal lands
Territorial Land Regulations	Federal	Land leases on federal lands
Canada Mining Regulations	Federal	Mine and mineral leases on federal lands
NWT Mine Health and Safety Act and Regulations	NWT/Nunavut	All mining operations; safety orientation
Nunavut Land Claims Agreement (NLCA)	Federal-Nunavut	Agreement providing for establishing regulatory bodies in Nunavut (NWB, NIRB, NWMB, NPC, SRT, DIOs [KIA])
NLCA	Federal-Nunavut	Under NIRB, establishment of a monitoring program pursuant to issue of a Project Certificate by the Minister of DIAND
NLCA	Federal-Nunavut	KIA administers surface rights access
Environmental Protection Act	Nunavut	Discharge of contaminants to air, water, land
Public Health Act	Nunavut	Public Health
Fisheries Act[1]	Federal	Destruction or alteration of fish habitat and fish populations
Canadian Environmental Protection Act	Federal	Air and water quality
Canadian Water Quality Guidelines	Federal	Receiving water quality
Canadian Drinking Water Guidelines	Federal	Drinking water quality
Canadian Air Quality Guidelines	Federal	Ambient air quality
Explosives Act	Federal	Natural Resources Canada permit to manufacture and/or store explosives (e.g. ANFO)
Navigable Waters Protection Act	Federal	No navigable waters will have obstructions to navigation constructed

NWB: Nunavut Water Board
NIRB: Nunavut Impact Review Board
NWMB: Nunavut Wildlife Management Board
NPC: Nunavut Planning Commission
SRT: Surface Rights Tribunal
KIA: Kitikmeot Inuit Association

1 The Metal Mine Liquid Effluent Regulation (MMLER) is used to set effluent objectives for metal mines on federal lands in Canada. Although the MMLER may be used as a guide for diamond mines, it does not strictly apply.

TABLE 2.1 SUMMARY OF DENSITY OF INDIVIDUAL KIMBERLITE TYPES		
Facies/lobe	**Count**	**Density (tonnes/m^3)**
Center lobe	229	2.58
North lobe	261	2.57
South lobe	28	2.67
JDF2S	10	2.68
JDF1	145	2.57
Total	**673**	**Average 2.58**

TABLE 2.2 DIAMOND VALUATION BY CSO FOR BULK SAMPLE				
Valuation Date	**Lots Valued**	**Facies**	**Weight (Carats)**	**Value per Carat (US$)**
July 29, 1997 (i)	1, 2	6	4189.67	**57.97**
	3, 4	2N	331.18	**58.14**
	5, 6	6W	1794.66	**62.43**
November 17, 1997 (ii)	1	6	2227.60	**57.55**
	2	4N	1420.50	**47.74**
	3	2S	175.90	**45.03**
	4	4S	169.50	**44.69**
Total/Average			**10,139.51**	**57.70**

Notes i) July 29, 1997 – Diamonds >10.8 carats not included in value. A total of 234.07 carats later valued at $US30,000,

ii) November 17, 1997 valuation includes a total of 61.58 carats for stones >10.8 carats

CSO (Central Selling Organization) replaced by DTC (Diamond Trading Company)

TABLE 2.3 DIAMOND VALUATION BASED ON TERRAC AND ADTEC METHODOLOGY			
Valuation Date	**Facies**	**Weight (carats)**	**Value per Carat (US$)**
July 14, 1998	2 North	330.48	**76.65**
	2 South	175.327	**33.75**
	4 North	1419.033	**49.92**
	4 South	168.823	**38.73**
	6	6551.72	**77.52**
	6 West	1884.98	**62.01**
Total/Average		**10,529.63**	**69.65**

TABLE 2.4 THE COMPARISON BETWEEN THE WWW MODEL AND THE SRK MODEL		
Lobe	**WWW model (U$/ct)**	**SRK model (U$/ct)**
Size Model for the centre lobe and JDF2S	90.00	76.00
Size Model for JDF4 North lobe	71.00	60.00

TABLE 2.5 SUMMARY OF THE PQ DRILL DATA			
Lobe	**Stones per tonne**	**Carats per 100 tonnes**	**Carats per stone**
North – Upper (JDF4N)	21.48	88.04	0.041
North – Lower (JDF4N)	15.85	94.65	0.060
South (JDF4S)	15.43	48.88	0.032
Center (JDF6)	44.38	208.84	0.047
JDF2S	11.97	83.21	0.070
JDF1N	7.57	57.98	0.077

Note: i) Based on recovered stones >0.0106 carats (approximately 1.0mm CSO sieve).
ii) Upper and lower portion of North lobe separated at 390 meter level.

TABLE 2.6 JERICHO PROJECT – INDICATED RESOURCES (APRIL 26, 2000)						
Lobe	**Tonnage (‘000)**	**PQ Grade Carats/t**	**Recovery (%)**	**Recover. Carats/t**	**Value US$/t**	**Value/t CDN$/t**
North-upper	603	0.88	88	0.77	46.20	66.99
North-lower	908	0.95	75	0.71	42.60	61.77
Centre	2,156	2.09	68	1.42	109.34	158.54
Subtotal	**3,667**	**---**	**---**	**2.90**	**198.14**	**---**

TABLE 2.7 JERICHO PROJECT - INFERRED RESOURCES (APRIL 26, 2000)						
Lobe	**Tonnage (‘000)**	**PQ Grade Carats/t**	**Recovery (%)**	**Recov. Carats/t**	**Value US$/t**	**Value/t CDN$/t**
North-lower	484	0.95	75	0.71	42.60	61.77
Center	86	2.09	68	1.42	109.34	158.54
South	1,145	0.49	75	0.37	22.20	32.19
JDF2S	694	0.83	75	0.62	47.74	69.22
JDF1	992	0.58	75	0.44	26.10	37.85
Total	**3,401**		**---**	**0.52**	**---**	**---**

Note:
i) Based on a density of 2.6 tonnes/cubic meter.
ii) Recovery of stones from PQ drilling correlated with bulk sample.
iii) Recovery of Center lobe used for South and North lobe.
iv) Exchange rate of US$1.00 = CDN$1.45.
v) Indicated for North lobe, above 300m level; Center lobe, >200m level; South lobe, >270m level.
vi) Value of US$77/carat used for Center lobe and JDF2S, US$60 used for North and South lobes.

TABLE 3.1
MINE MOBILE EQUIPMENT

Equipment	No.	Equipment	No.	Equipment	No.	Equipment	No.
Construction		Open Pit Operation		Plant Operation		Underground Operation	
Cat 777 Ore Truck	1	Cat 777 Ore Truck	7			Scooptram	3
Cat D300	2	Cat D300	-			Scissor Lift	1
Cat 992 Loader	-	Cat 992 Loader	1			Service Tractor	3
Cat 980 Loader	1	Cat 980 Loader	-	Cat 966E Loader	1		
Cat 5130 Shovel	-	Cat 5130 Shovel	1	Skid Steer Loader Cat 226	1		
JD992E Excavator	1	JD992E Excavator	-				
D10 Dozer	-	D10 Dozer	2				
D9 Dozer	1	D9 Dozer	1				
D6 Dozer	2	D6 Dozer	-	D6R Dozer	1		
6" Drill Cat 3406 Engine	-	6" Drill Cat 3406 Engine	3				
16G Grader	-	16G Grader	1				
824 RT Dozer	-	824 RT Dozer	1				
Cat C563 Packer	1	Cat C563 Packer	-				
Service Truck	1	Service Truck	1			Service Truck	
Fuel Truck	1	Fuel Truck	1			Fuel Truck	
Water Truck	-	Water Truck	1			Water Truck	1
Tractor and Lowboy	1	Tractor and Lowboy	1			Tractor and Lowboy	
Tandem	1	Tandem	1	Cat D300 articulating dump trk	1	Tandem (40 ton)	3
Pickup Truck	2	Pickup Truck	4	Pickup Truck	1	Pickup Truck	1
Bus	1	Bus	1				
Crane 22T	1	Crane 22T	1				
Compressor (800 cfm)	1	Compressor	-				

No. based on maximum use month for construction (May Year 1)

No. based on highest production month for production (Oct Year 2)

TABLE 5.1 **PIT DESIGN PARAMETERS**			
	Kimberlite	Granodiorite	Overburden
Slopes			
Inter-ramp < 100m	41 – 70 degrees	60 – 65 degrees	26.5 degrees (2:1)
Inter-ramp > 100m	41 degrees	60 degrees	--
Bench Configuration			
Bench height	10 m	10 m	10 m
Bench face angle (°)	70	85	--
Bench width (m)	8	8	--
Berm interval (m)	10	20 (double bench)	1
Ramp Configuration			
Gradient (%)	10	10	10
Width (m)	22 (14*)	22 (14*)	22 (14*)
Operating Limits			
Minimum mining width – pushback (m)	40	40	40
Pit bottom (m)	50	50	50

* Single lane ramp used for the bottom 500m of each pit shell.

TABLE 5.2 **DESIGN PIT MATERIAL SUMMARY BY BENCH**										
Bench	**Ore**				**Waste**			**Inferred**	**Total**	**Strip**
Toe	**Central**	**N.Upper**	**N.Lower**	**Total**	**Overburden**	**Granite**	**Precontact**	**LoGrade**	**Material**	**Ratio**
Elev.(m)	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**(w/o)**
500EL	-	-	-	-	-	-	-	-	-	-
490EL	-	-	-	-	46,534	4,746	-	-	51,280	-
480EL	-	-	-	-	508,799	750,341	-	-	1,259,140	-
470EL	-	-	-	-	708,690	1,302,404	13,000	23,400	2,047,494	-
460EL	41,600	-	-	41,600	318,292	1,366,544	72,653	93,600	1,892,689	44.5
450EL	70,200	59,800	-	130,000	5,400	1,488,608	132,600	166,400	1,923,008	13.8
440EL	83,200	62,400	-	145,600	-	1,229,613	137,800	153,400	1,666,413	10.4
430EL	88,400	75,400	-	163,800	-	1,130,081	135,200	153,400	1,582,481	8.7
420EL	101,400	97,700	-	199,100	-	879,546	104,336	163,800	1,346,781	5.8
410EL	96,200	94,459	-	190,659	-	796,110	116,633	153,400	1,256,802	5.6
400EL	93,600	57,698	-	151,298	-	616,546	109,038	158,600	1,035,481	5.8
390EL	96,200	35,683	-	131,883	-	544,518	101,198	165,842	943,442	6.2
380EL	98,800	-	14,028	112,828	-	371,918	102,439	154,106	741,291	5.6
370EL	98,800	-	6,315	105,115	-	319,321	93,413	129,783	647,632	5.2
360EL	101,400	-	137	101,537	-	225,607	89,279	71,458	487,881	3.8
350EL	88,400	-	-	88,400	-	188,598	88,694	48,115	413,806	3.7
340EL	90,510	-	-	90,510	-	122,327	51,650	20,805	285,292	2.2
330EL	92,595	-	-	92,595	-	81,187	37,446	748	211,976	1.3
320EL	94,741	-	-	94,741	-	21,312	21,532	-	137,585	0.5
310EL	79,886	-	-	79,886	-	646	7,657	233	88,422	0.1
Totals	**1,415,933**	**483,140**	**20,480**	**1,919,552**	**1,587,715**	**11,439,973**	**1,414,568**	**1,657,090**	**18,018,897**	**8.4**
Recovered Grade (ct/t)	1.42	0.77	0.71	1.25						
Recovered Carats	2,010,624.18	372,017.69	14,540.76	2,397,182.63						

TABLE 5.3
MINING SCHEDULE MATERIALS SUMMARY

Period	Q#	Waste Overburden Tonnes	Granite Tonnes	Precontact Tonnes	Total Tonnes	Low Grade SU-D Tonnes	F1N Tonnes	Ore Total Tonnes	CU-D Tonnes	NU-D Tonnes	NL-D Tonnes	Total Tonnes	Total Pit Total Tonnes	Phase*
OPEN PIT														
P2-Q1	5	373,121	50,000	16	423,137	-	-	-	-	-		-	423,137	St
P2-Q2	6	397,702	300,000	66,467	764,170	8,388	36,397	44,785	160,668	-		160,668	969,622	St, Pb
P2-Q3	7													
P2-Q4	8													
P3-Q1	9													
P3-Q2	10	452,329	1,663,373	165,752	2,281,454	107,947	176,950	284,897	67,210	16,440	-	83,649	2,650,000	St, Pb, Ult
P3-Q3	11	363,236	1,874,450	106,985	2,344,671	64,356	84,051	148,406	156,923	-	-	156,923	2,650,000	Pb, Ult
P3-Q4	12	1,326	1,353,511	85,571	1,440,408	44,837	27,985	72,822	95,880	30,896	-	126,777	1,640,006	Pb, Ult
P4-Q1	13													
P4-Q2	14	-	2,038,770	156,328	2,195,098	68,082	78,389	146,471	175,804	132,627	-	308,431	2,650,000	Pb, Ult
P4-Q3	15	-	1,906,082	193,368	2,099,450	132,818	148,072	280,889	42,953	230,451	-	273,404	2,653,743	Pb, Ult
P4-Q4	16	-	956,431	250,410	1,206,841	195,650	139,227	334,877	70,163	72,725	-	142,889	1,684,607	Ult
P5-Q1	17													
P5-Q2	18	-	657,678	93,413	751,092	62,788	139,796	202,584	98,800	-	20,343	119,143	1,072,819	Ult
P5-Q3	19	-	536,532	229,623	766,155	97,894	42,483	140,378	280,310	-	137	280,447	1,186,979	Ult
P5-Q4	20	-	103,145	66,635	169,780	836	146	981	267,222	-	-	267,222	437,984	Ult
Subtotals		**1,587,715**	**11,439,973**	**1,414,568**	**14,442,255**	**783,595**	**873,495**	**1,657,090**	**1,415,933**	**483,140**	**20,480**	**1,919,552**	**18,018,897**	
UNDERGROUND														
P6-Q1	21													
P6-Q2	22													
P6-Q3	23		57,000									57,000		
P6-Q4	24								50,000			50,000		
P7-Q1	25								82,500			82,500		
P7-Q2	26								82,500			82,500		
P7-Q3	27								82,500			82,500		
P7-Q4	28								82,500			82,500		
P8-Q1	29								82,500			82,500		
P8-Q2	30								82,500			82,500		
P8-Q3	31								68,600			68,600		
Subtotals			**57,000**						**613,600**			**670,600**		
Totals		**1,587,715**	**11,496,973**	**1,414,568**	**14,442,255**	**783,595**	**873,495**	**1,657,090**	**2,029,533**	**483,140**	**20,480**	**2,590,152**	**18,018,897**	

* Phases as follows:

- St = Starter pit
- Pb = intermediate pushback
- Ult = ultimate pit

Ore types:

- CU-D = Central lobe ore, recovered grade of 1.42 ct/t
- NU-D = North Lobe upper ore, recovered grade of 0.77 ct/t
- NL-D = North Lobe lower ore, recovered grade of 0.71 ct/tn
- SU-D = South Lobe inferred material, recovered grade of 0.37 ct/t
- F1N = North F1 facies inferred material, recovered grade of 0.44 ct/t

TABLE 5.4 EXPLOSIVES USE AT JERICHO								
	OPEN PIT				**UNDERGROUND**			**TOTAL**
Date	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	
Low Grade Ore (mt)	44,784	506,100	762,300	344,000	-	-	-	**1,657,184**
Low Grade Ore (bcm)	17,225	194,654	293,192	132,308	-	-	-	**637,378**
Northern Ore (mt)	-	47,300	435,800	20,400	-	-	-	**503,500**
Northern Ore (bcm)	-	18,192	167,615	7,846	-	-	-	**193,654**
Central Ore (mt)	160,700	320,000	289,000	646,300	50,000	330,000	233,600	**2,029,600**
Central Ore (bcm)	61,808	123,077	111,154	248,577	19,231	126,923	89,846	**780,615**
Waste (mt)	416,502	5,249,650	6,066,600	1,687,100	57,000	-	-	**13,476,852**
Waste (bcm)	157,171	1,981,000	2,289,283	636,642	21,509	-	-	**5,085,605**
Overburden (mt)	770,824	816,891	-	-	-	-	-	**1,587,715**
Overburden (bcm)	428,236	453,828	-	-	-	-	-	**882,064**
Total Material (mt)	1,392,810	6,939,941	7,553,700	2,697,800	107,000	330,000	233,600	**19,254,851**
Total Material (bcm)	664,438	2,770,751	2,861,245	1,025,372	40,740	126,923	89,846	**7,579,316**
Anfo Use (LG Ore) kg	13,780	155,723	234,554	105,846	-	-	-	**509,903**
Anfo Use (NF Ore) kg	-	14,554	134,092	6,277	-	-	-	**154,923**
Anfo Use (CF Ore) kg	49,446	98,462	88,923	198,862	10,385	68,538	48,517	**563,132**
Anfo Use (Waste) kg	125,736	1,584,800	1,831,426	509,313	11,615	-	-	**4,062,891**
Anfo Use (OB) kg	141,318	149,763	-	-	-	-	-	**291,081**
Total Anfo Use (kg)	**330,280**	**2,003,302**	**2,288,996**	**820,298**	**22,000**	**68,538**	**48,517**	**5,581,930**

[a] All waste rock will be used for construction in Year 1

[b] Amount stored minus amount processed; stored ore leachate to Lake C1; N from PK to PKCA and Lake C3.

[c] All potential N available in ore assumed leached in process; amount to coarse kimberlite proportional to surface moisture volume.

[d] Assumes make up water = water lost; nitrogen lost proportional to water reporting to coarse and fines.

TABLE 6.1 PROCESS DESIGN CRITERIA		
Jericho Site Details		
Location	28 km NW of Lupin Mine Nunavut 66°N (lat) 111° 30'E (long)	
Altitude	400 to 600m above sea level.	
Ambient Temperatures Mean Average Air Temperature Maximum Recorded Minimum Recorded Annual Mean Rainfall Equivalent Average Evaporation	-45 to +24°C -11.5°C +31°C -54°C 300mm. 100mm.	
Maximum Wind Speed (design) Average Prevailing	110 km/hour 18 km/hour (NW)	
		Units
General Design Criteria		
Throughput	330 000	t/y
Annual Operating	365	Days
Plant Availability	90%	%
Running	329	Days
Plant Utilization	90%	%
Expected Running	296	Days
Actual Running	7 096	Hours
Required Feed rate to Make Call	46.5	tph
Design Process Rate	50 (dry)	tph

TABLE 6.2
ORE PROCESSING SCHEDULE

Ore	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Total**
					(Tonnes)					
Central Ore	34,000	330,000	330,000	323,100	330,000	330,000	330,000	22,400		2,029,500
Northern Ore				6,900				307,600	189,100	503,600
Total Ore Processed	34,000	330,000	330,000	330,000	330,000	330,000	330,000	330,000	189,100	2,533,100
Coarse Kimberlite (solids) (85% of feed)	28,900	280,500	280,500	280,500	280,500	280,500	280,500	280,500	160,735	2,153,135
Fine Kimberlite (solids) (15% of feed)	5,100	49,500	49,500	49,500	49,500	49,500	49,500	49,500	28,365	379,965
Total Kimberlite Processed	= total ore processed									

TABLE 7.1
NUNA LOGISTICS OPEN PIT MINE WORKFORCE

Class	**Position**				
		Year 1	**Year 2**	**Year 3**	**Year 4**
Supervisory					
	Superintendent	2	2	2	1
	Foreman	4	4	4	2
	Safety/Training	2	2	2	1
	Tool Crib Attendant	2	2	2	1
	Administrator	2	4	4	2
Operators					
	Truck Driver	10	18	26	15
	5130 Backhoe Operator	0	4	4	2
	Front End Loader Operator	4	2	2	1
	Dozer Operator	8	6	6	3
	Crusher Operator	2	2	2	1
	Crusher Helper	0	0	0	0
	Grader Operator	2	2	2	1
	Other Equipment Operator	2	2	2	1
	Labourers	2	2	2	1
Mechanical					
	Mechanic	8	12	16	8
	Welder	4	8	10	5
	Serviceman	4	4	4	2
	Labourer	2	2	2	1
Drilling and Blasting					
	Superintendent	2	2	2	1
	Driller	8	8	6	3
	Blaster	2	2	2	1
	Helper	10	10	8	4
	Mechanic	4	2	2	1
Surveyors					
	Crew Chief	2	2	2	1
	Instrumentman	2	2	2	1
Total		**90**	**106**	**116**	**60**

TABLE 7.2 UNDERGROUND MINING WORKFORCE YEAR 5 TO YEAR 7			
Position	**No. per Shift**	**No. per Rotation**	**Total**
Project Superintendent	1	1	2
Shift Boss	1	2	4
Surveyor	1	1	2
Clerk/Expeditor	1	1	2
Lead Mechanic	1	1	2
Mechanic	2	4	8
Electrician	1	1	2
Development Miner	2	4	8
Raise Miner	1	1	2
Longhole Driller/Blaster	1	2	4
Scoop/Truck Operator	2	4	8
Rockbolter	1	2	4
Total			**48**

TABLE 7.3 DIAMOND PROCESSING PLANT EMPLOYMENT			
Position	**No. per Shift**	**No. per Rotation**	**Total**
Plant Manager	1	Alternate	1
Plant Engineer	1	Alternate	1
Operators	4	8	16
Sorters	3	3	6
Loader Operator	1	2	4
Truck Driver	1	2	4
Security Officer	1	2	4
Security Foreman	1	Alternate	1
Security Manager	1	Alternate	1
Apprentices	1	1	2
Total			**40**

TABLE 7.4 CATERING EMPLOYMENT		
Position	**No. per Rotation**	**Total**
Supervisor	1	2
Cook	2	4
Cook's Helper	2	4
Janitor	3	6
Total		**16**

TABLE 7.5 ADMINISTRATION AND TECHNICAL SERVICES POSITIONS		
Position	**Number**	**Remarks**
Operations Accountant	1	Toronto based; accounts, inventory management, payroll.
Environmental Manager	1	Part-time (21 weeks/annum). Consultants will be utilized in support.
Human Resources And Community Liaison	1	Part-time (8 weeks/annum)
Expediter	Outsource	Part-time, mainly for winter road transport
Technical Services	Outsource	Consultant for mine planning, geology, geotechnical audit, plant audit.
Purchaser	Outsource	Annual inventory procurement

TABLE 8.1 JERICHO SITE ACCESS	
Surface	Winter road: Yellowknife to Lupin on existing road (528 km) Lupin to Jericho on Lupin road extension (Year 1) (32.5 km) Lupin to northwest end of Contwoyto Lake (Year 2 to Year 9) (29 km) All-weather road: Northwest end of Contwoyto Lake to Jericho (Year 2 to Year 9) (3.5 km)
Fixed Wing Air	Year Round 1220 m gravel strip Winter and Summer Carat Lake (emergency only)
Helicopter Air	Year Round Emergency Exploration - possible

TABLE 8.2
ROAD CONSTRUCTION

Road	Width (m)	Length (km)	Material	Fill Vol (m^3)
Exploration Camp - airstrip	6	1.5	existing	n/a
Pit to dump 1 SE	18	0.3	ROM	7,688
			15 cm Crush	810
			Esker	270
Dump 1 SE to exploration camp	6	1.2	ROM	12,750
			15 cm Crush	1,080
			Esker	360
Pit to Contwoyto Lake	9	3.5	ROM	50,313
			15 cm Crush	4,725
			Esker	1,575
Pit to processing plant	18	1.5	ROM	38,438
			15 cm Crush	4,050
			Esker	1,350
Processing plant to PKCA	9	2.1	ROM	30,188
			15 cm Crush	2,835
			Esker	945
Pit to pump house	6	0.3	ROM	3,188
			15 cm Crush	270
			Esker	90

ROM: run-of-mine PKCA: processed kimberlite containment area

TABLE 8.3
LOADS REQUIRED FOR COMMISSIONING

Buildings, Supplies and Equipment	No Trucks	Year
Accommodation/mine office	25	Year 1
Plant Building	51	Year 1
Plant	31	Year 1
Pumps	4	Year 1
Mine Truck Shop	25	Year 1
Explosives Truck Shop	22	Year 1
Powder Mag	3	Year 1
Caps Mag	1	Year 1
Generators x 4	5	Year 1
Fuel Tanks x 5	5	Year 1
Fuel x 3 M L	71	Year 1
Airstrip Equipment	1	Year 1
Fire Protection	2	Year 1
Tools + Furnishing	3	Year 1
Incinerator	1	Year 1
Miscellaneous	40	Year 1
Ammonium Nitrate	70	Year 1
Powder + Caps x 26,000 kg	1	Year 1
SUBTOTAL	361	Year 1
Fuel Tanks x 5	5	Year 2
Fuel (part of resupply)	0	Year 2
Ammonium Nitrate (part of resupply)	0	Year 2
Powder + Caps (part of resupply)	0	Year 2
SUBTOTAL	5	Year 2
Open Pit Mobile Equipment		
992 FEL x 3	15	Year 1
777 Ore Truck x 4	4	Year 1
D300 Ore Truck	1	Year 1
Excavator	1	Year 1
D10 x 2	4	Year 1
D9 Dozer x 2	4	Year 1
D6 Dozer	1	Year 1
RT Dozer	1	Year 1
Drill x 2	4	Year 1
Packer	1	Year 1
Explosives Truck	1	Year 1
Miscellaneous	10	Year 1
SUBTOTAL	47	Year 1
992 FEL	5	Year 2
777 x 3	3	Year 2
D10	2	Year 2
Miscellaneous	2	Year 2
SUBTOTAL	12	Year 2
Processing Plant Mobile Equipment		
966 FEL x 2	4	Year 1
D300 x 2	2	Year 1
Tandem Dump Truck	1	Year 1
Skid Steer Loader + Miscellaneous	1	Year 1
SUBTOTAL	8	Year 1
Underground Mobile Equipment		
Scooptram x 3	3	Year 3
Scissor Lift	1	Year 3
Service Tractor	1	Year 3
Tandem (40 t)	3	Year 3
SUBTOTAL	8	Year 3
TOTAL	441	

TABLE 8.4 ANNUAL LOADS REQUIRED FOR RESUPPLY					
Supplies and Equipment - Open Pit Mining	**No. Trucks**	**Year**	**Supplies and Equipment - Underground Mining**	**No. Trucks**	**Year**
Fuel x 5 M L	112	2 - 4	Fuel x 0.73 M L	20	5 - 7
Ammonium Nitrate x 2300 t	66	2 - 4	Ammonium Nitrate x 254 t	8	5 - 7
Powder + Caps x 26000 kg	1	2 - 4	Powder + Caps x 10 t (estimate)	1	5 - 7
Miscellaneous	10	2 - 4	Miscellaneous	5	5 - 7
SUBTOTAL	189	2 - 4	SUBTOTAL	34	5 - 7
Supplies and Equipment - Plant			**Supplies and Equipment - Plant**		
Fuel x 5 M L	112	1 - 8	Fuel x 5 M L	112	1 - 8
Gasoline x 10,000 L	0.5	1 - 8	Gasoline x 10,000 L	0.5	1 - 8
Jet Fuel x 5000 L	0.5	1 - 8	Jet Fuel x 5000 L	0.5	1 - 8
Ferrosilicon x 120 t	4	1 - 8	Ferrosilicon x 120 t	4	1 - 8
Miscellaneous	5	1 - 8	Miscellaneous	5	1 - 8
SUBTOTAL	122	1 - 8	SUBTOTAL	122	1 - 8
Supplies and Equipment - Catering			**Supplies and Equipment - Catering**		
Miscellaneous (estimate)	1	1 - 8	Miscellaneous (estimate)	1	1 - 8
TOTAL OPEN PIT MINING	312		TOTAL UNDERGROUND MINING	157	

NOTE: Fuel in excess of annual requirements will be transported to site each year to top up the 10 ML storage capacity for the purpose of having a reserve on site.

TABLE 8.5 TRUCKING REQUIREMENTS FOR MINE CLOSURE[1]	
Description	**Number of Loads**
Plant building	51
Plant	24
Mine buildings	47
Accommodation	23
Generators	4
Fuel Tanks	10
Mobile Equipment	10
Miscellaneous	10
TOTAL	179

1 Assumes all materials and equipment removed from site.

TABLE 10.1 MATERIALS SCHEDULE FOR WATER INTAKE CAUSEWAY			
Component	**Material**	**Dimensions (m)**	**Volume (m^3)**
Cap	Crushed ROM Waste Rock or Esker	3.25 x 1 x 90	293
Top	ROM Waste Rock	7.2 x 1 x 90	648
Middle	ROM Waste Rock	11 x 1 x 47	517
Bottom	ROM Waste Rock	14 x 1 x 91	1,274
Toe	ROM Waste Rock		514
Total			3,246

TABLE 12.1
JERICHO PROJECT ANNUAL ESTIMATED ENERGY BALANCE - OPEN PIT MAXIMUM PRODUCTION YEAR 3

Item	Amount (L)	Power (KW)	Energy Value (Giga-Joules)	Coversion Factor[a]
TOTAL DIESEL CONSUMPTION	**9,893,080**		**405,684**	0.04101
Mobile Equipment Fuel	4,617,200		189,337	0.04101
Process Rotary Drier	156,200		6,405	0.04101
ANFO Production[b]	101,860		4,177	0.04101
Power Generation (from next line)	5,119,680	3,700	209,942	
TOTAL POWER GENERATION	**5,119,680**	**3,700**	**209,942**	
Main power generators[c]	3,801,840	2600	155,902	0.04101
Process plant	3,363,166	2300	137,913	0.04101
Accommodation	438,674	300	17,989	0.04101
Accommodation Standby	144,000	300	5,905	0.04101
Auxillary power generator	1,173,840	800	48,135	0.04101
Mechanical Shop	440,190	300	18,051	0.04101
Pit Crusher	586,920	400	24,068	0.04101
Mine pumps	73,365	50	3,008	0.04101
Explosives truck shop	73,365	50	3,008	0.04101
Gasoline (small motors)	10,000		308	0.03081
Jet Fuel (helicopter use)	4,920		165	0.03360
TOTAL PROPANE CONSUMPTION	**25,000**		**531**	
Cooking	25,000		531	0.02122
WASTE HEAT RECYCLED[d]	**(220,095)**	**150**	**(9,025)**	
Process Plant	(146,730)	100	(6,017)	0.04101
Boilers	(73,365)	50	(3,008)	0.04101
TOTAL ENERGY USE (Giga-Joules)			**406,688**	

NOTES

a 41.868 GJ/tonne of oil equivalent; source UK Petrol. Ind. Assoc.

b Assumes 5% fuel oil in ANFO. Diesel SG 0.89. ANFO use from Table 5.4

c Main power generator fuel consumption 217 L/hr/generator;
other power generator fuel consumption pro-rated.

d Factored into total fuel use; equals equivalent fuel and energy savings

TABLE 13.1 ANTICIPATED CONSTRUCTION FILL REQUIREMENTS				
Fill Material	**Year 1**		**Year 2**	
	Tonnes	**m^3**	**Tonnes**	**m^3**
Run of Mine Rock	290,000	109,000	509,000	192,000
6 inch (15 cm) Crush	25,000	9,500	36,500	14,000
1 inch (2.5 cm) Crush	9,100	3,500	39,000	14,700
Esker[a]	81,000	43,000	73,000	38,000
Total	**405,100**	**165,000**	**657,500**	**258,700**

BCM: bank cubic metres

[a] In Year 2 overburden low in organic content from the stockpile will be substituted for esker where practical.

TABLE 14.1 OVERBURDEN STRIPPING SCHEDULE			
Year 1		**Year 2**	
Tonnes	**m^3**	**Tonnes**	**m^3**
771,000	428,000	817,000	454,000

TABLE 15.1
JERICHO PROCESSED KIMBERLITE EFFLUENT QUALITY

Parameter	**Conc**	**CCME**	**Health Canada**	**MMLER**
	(mg/L)			
Total Dissolved Solids	1155		500	
Hardness (CaCO3)	654			
pH	8.84			>5 (>6)
Total Suspended Solids (supernatant)	8	<10mg/L increase		50 (25)
Turbidity (NTU)	5.2			
Conductivity (uS/cm)	1424			
Alkalinity (CaCO3)	39			
Chloride	510		250	
Fluoride	0.35		1.5	
Sulphate	16		500	
Ammonia N	17.3	2		
Dissolved Metals				
Aluminum	0.048	0.1		
Barium	0.75		1	
Calcium	146			
Copper	0.002	0.002	1	0.6 (0.3)
Magnesium	70			
Manganese	0.0028		0.05	
Molybdenum	0.0066			
Nickel	0.025	0.025		1.0 (0.5)
Potassium	34			
Silicon	2.79			
Sodium	33		200	
Strontium	5.26			
Zinc	0.021	0.03		1(0.5)

CCME — Canadian Council of Ministers of the Environment
Health Canada — Canadian Drinking water guidelines
MMER — Metal Mine Effluent Regulation (of the *Fisheries Act*)

TABLE 18.1			
HAZARDOUS SUBSTANCES INVENTORY - JERICHO MINE SITE			
Substance	**Estimates of On Hand Quantities**	**Risk of Spill**	**Comments**
Diesel	10 million litres	low	In fuel farm with containment berm
Ammonium Nitrate	2300 tonnes	low	Powder, 1 t bags, no proximity to water
Sodium Nitrate	to be determined	low	25 kg bags, palletized in C-can
High Explosives (Magnifrak™)	26 tonnes	low	Stick powder in boxes in a magazine
Blasting caps	To be determined	low	In boxes in a magazine
Hydraulic Oil	6 - 205 L barrels	low	Stored in covered warehouse in silled area
Motor Oil	5 - 205 L barrels	low	In mine shop in a silled area or outside the mine shop
Jet Fuel	24 - 205 L barrels	low	Stored at airstrip, no proximity to water
Gasoline	up to 10,000 litres	low	In fuel farm with containment berm
Varsol	205 L	low	In mine shop in silled area
Petroleum grease	50 - 20 L pails	nil	In mine shop or cold storage containers
Transmission Oil	6 - 205 litre barrels	low	In mine shop in silled area
Sulphuric acid (battery acid)	small quantities	low	In mine shop in silled area
Ethylene glycol (vehicle antifreeze)	6 – 205 litre barrels	low	In mine shop in silled area
Ethylene glycol (heating system)	not applicable	very low	In pipes in heating system
Ferrosilicon	120 t, non-hazardous		
Hydrofluoric acid	small quantities	low	In fume cupboard in plant
Hydrochloric acid	small quantities	low	In fume cupboard in plant
Sodium hydroxide	small quantities	low	In lab in plant; in controlled drainage area
Acetone	small quantities	low	In fume cupboard in plant
Flocculent - Percol E-10, or equiv.	2 t, non-hazardous	low	In plant controlled drainage area
Slaked lime	to 10 t	low	Powder in bags on pallets and in container for use in controlled drainage area of plant
Floor Dry	small quantities	nil	In the accommodation complex and mine shop

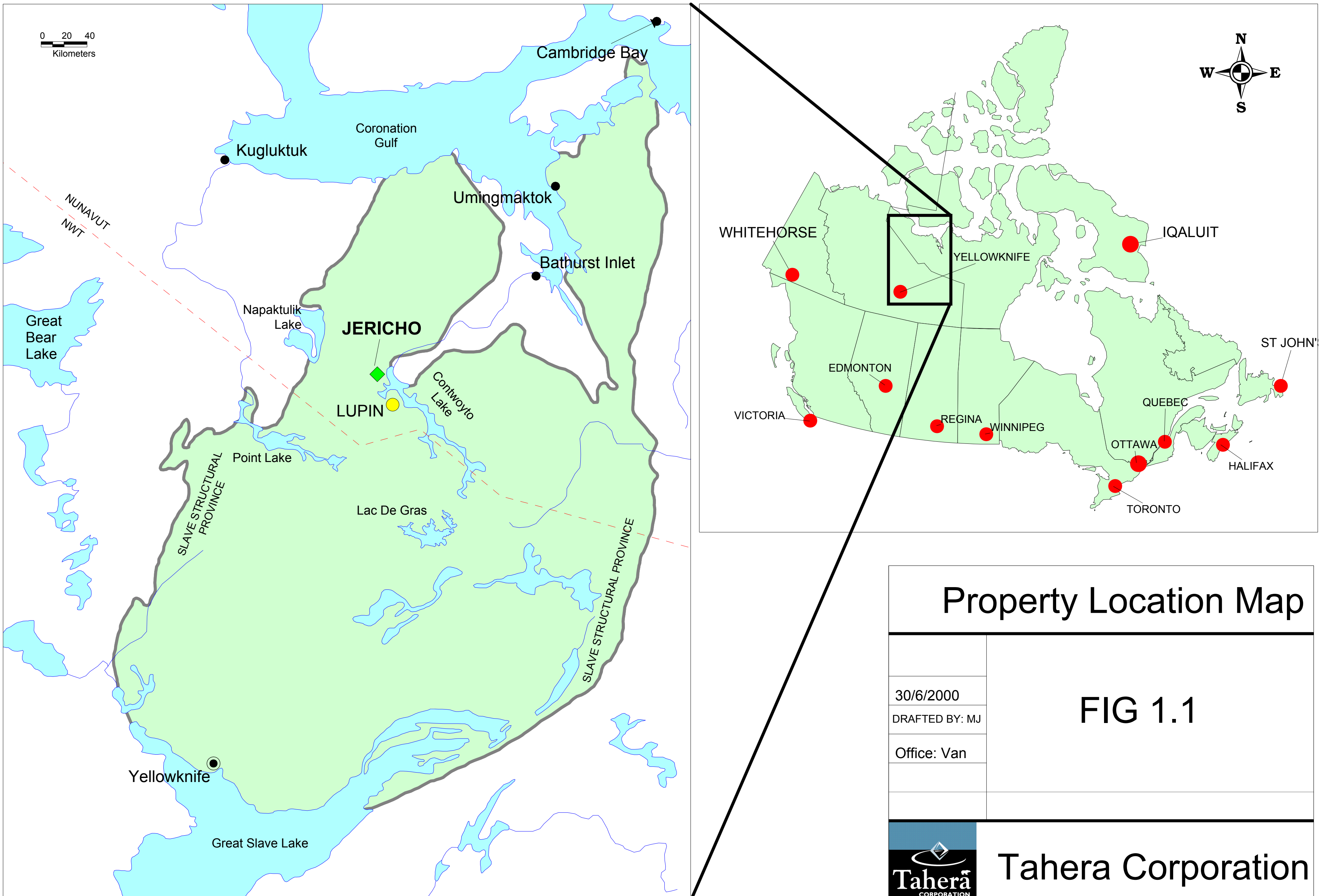
TABLE 19.1 FUGITIVE DUST EMISSION RATES FROM WASTE DUMP AND ORE STOCKPILES			
Facility	**Assumed Area (ha)**	**TSP Emission Rate (g/s)**	**PM-10 Emission Rate (g/s)**
Waste Dump 1	21.7	5.7	2.85
Low Grad Ore Stockpile	5.4	1.42	0.71
Central Lobe	4	1.05	0.53

Source: Levelton 2002 (Appendix D.1.1)

TABLE 19.2 EXHAUST GAS EMISSION RATES FROM STATIONARY AND MOBILE SOURCES						
Source	**NOx (g/s)**	**SOx (g/s)**	**CO (g/s)**	**TSP (g/s)**	**PM-10 (g/s)**	**CO_2 (g/s)**
Plant Generator	2.85	1.95	1.21	0.15	0.123	254.7
Camp Generator	0.66	0.45	0.28	0.035	0.028	58.8
Shop Generator	1.756	1.20	0.74	0.095	0.076	156.7
Ore Drier	0.167	0.115	n/a	0.199	n/a	146.5
Cat 777 Ore Truck 6 x 938 hp	1.79	0.242	2.215	0.104		137
D10 Dozer 1 x 570 hp	1.09	0.147	1.345	0.063		83.3
D9 Dozer 1 x 405 hp	0.776	0.105	0.956	0.045		59.2
Cat 992 Loader 1 x 800 hp	1.53	0.207	1.89	0.089		117
Cat 5130 Shovel 1 x 800 hp	1.53	0.207	1.89	0.089		117
Service Truck 1 x 300 hp	0.575	0.078	0.708	0.033		43.8
Fuel Truck 1 x 300 hp	0.575	0.078	0.708	0.033		43.8
Pickup 5 x 200 hp	0.383	0.052	0.472	0.022		29.2
Cat 966E Loader 1 x 285 hp	0.546	0.074	0.673	0.032		41.7
Cat D300 Tandem 1 x 285 hp	0.546	0.074	0.673	0.032		41.7
D6R Dozer 1 x 175 hp	0.335	0.045	0.413	0.019		25.6

Source: Levelton 2002 (Appendix D.1.1)
n/a: not available
Emission factors for mobile equipment based on AP-42 factors for diesel industrial engines (g/hp-hr):
NOx: 6.9; SOx: 0.93; CO: 8.5; PM/TSP: 0.4; CO_2: 526

FIGURES



Property Location Map

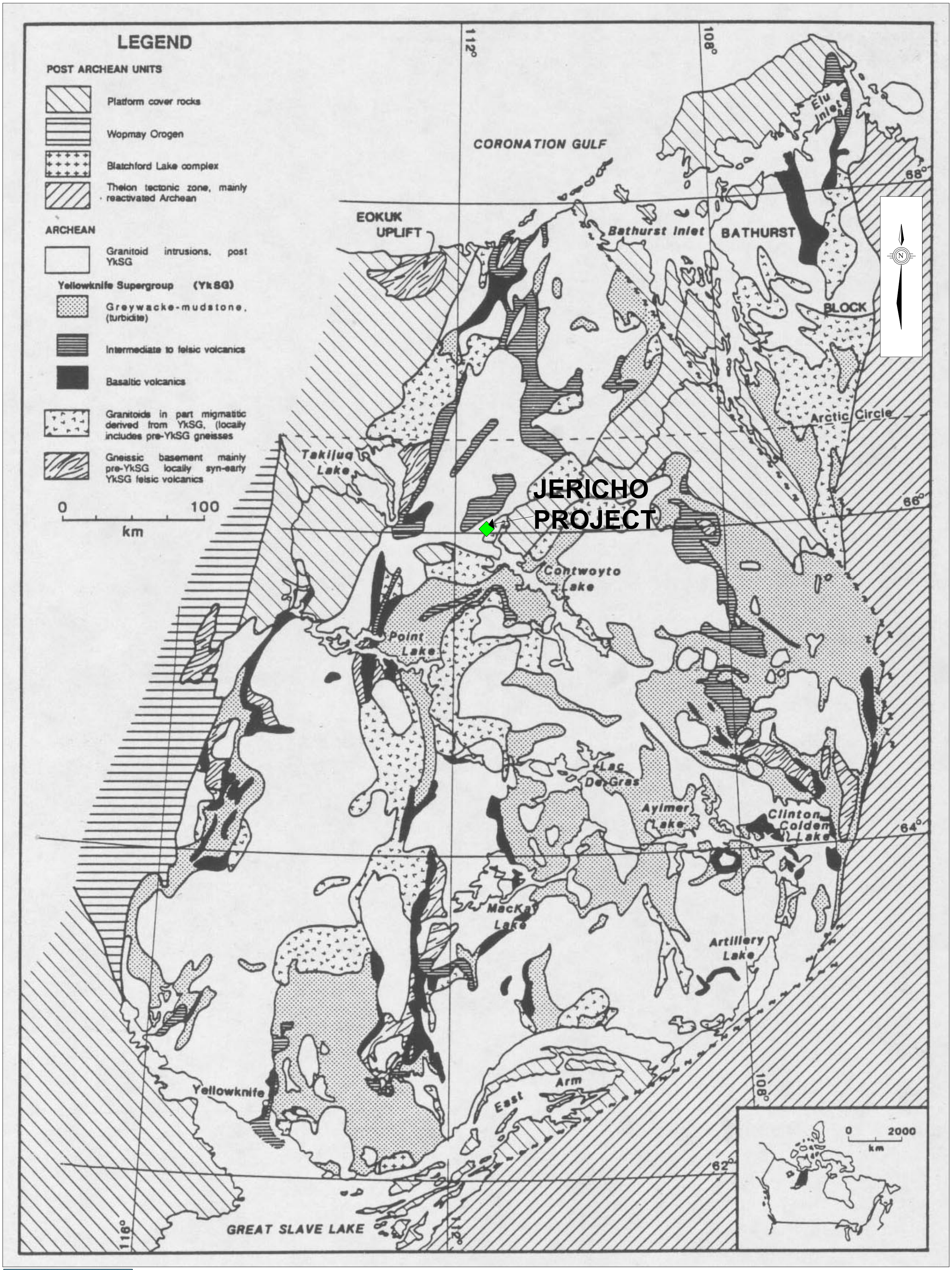
30/6/2000

DRAFTED BY: MJ

Office: Van

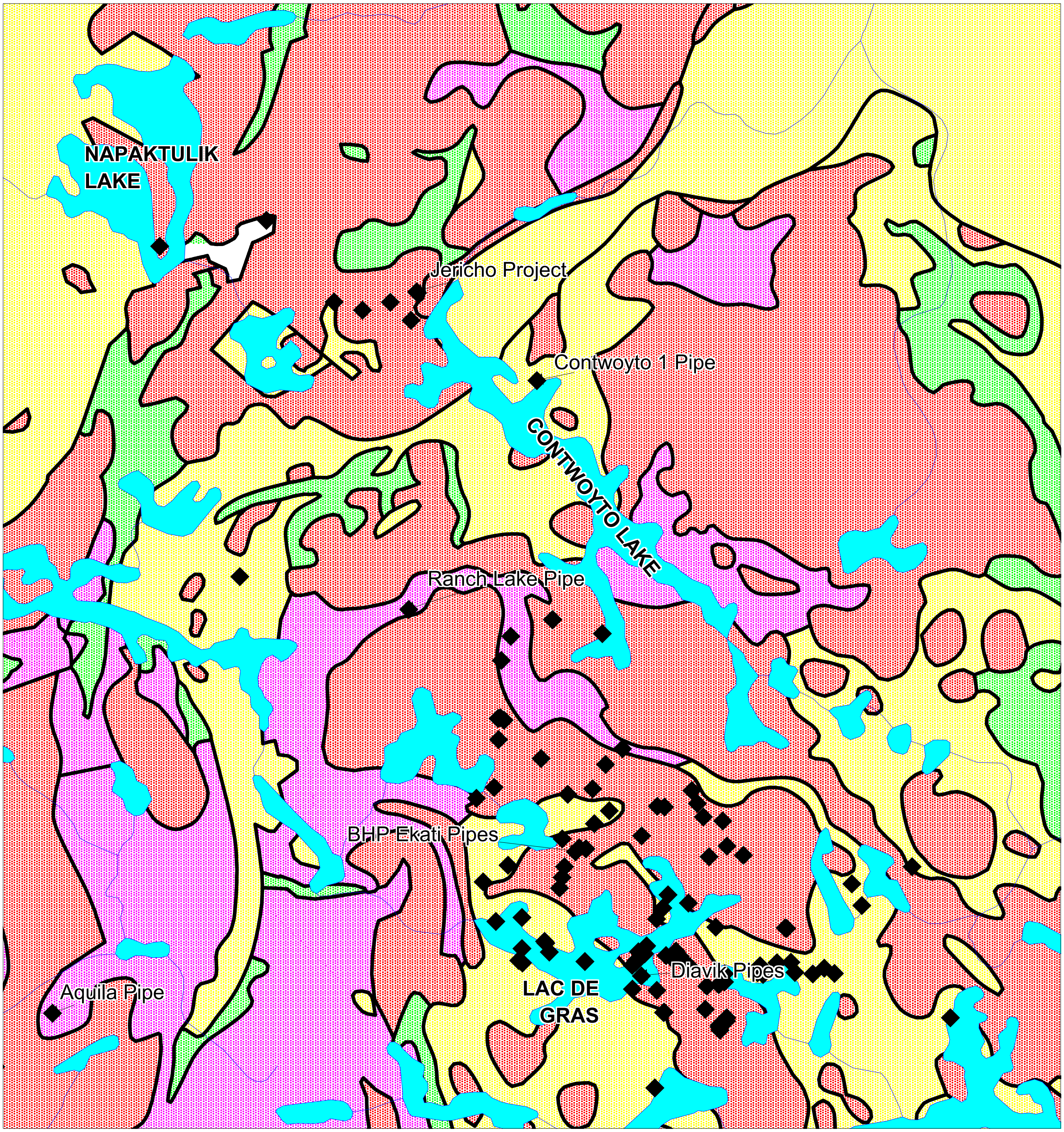
FIG 1.1

Tahera Corporation



Tahera CORPORATION

FIGURE 2.1
JERICHO DEPOSIT REGIONAL GEOLOGY
(AFTER McGLYNN, 1977)



Source: P. Hoffman and L. Hall

Legend

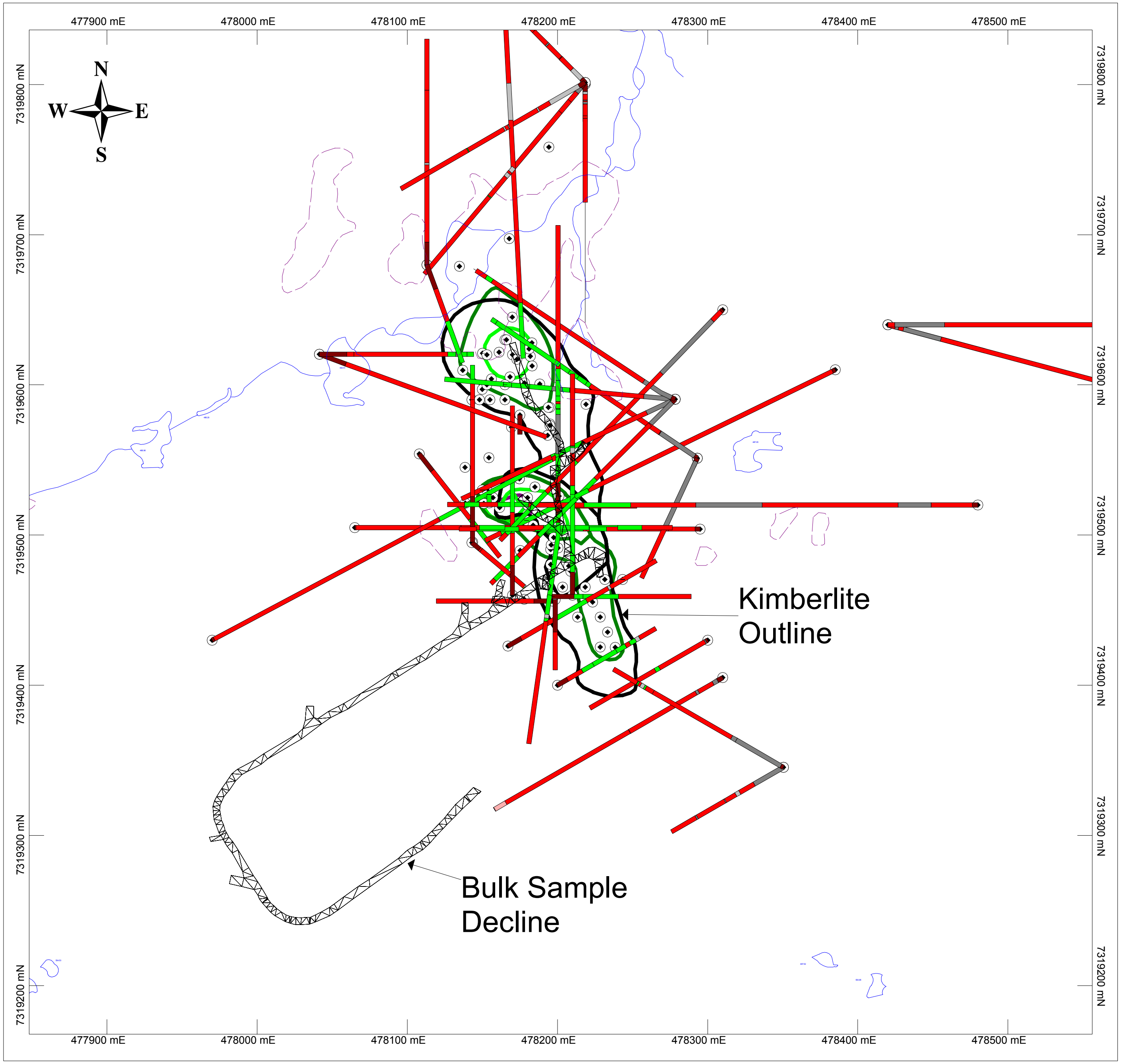
- INTRUSIVE ROCKS
- METAMORPHIC ROCKS
- SEDIMENTARY AND VOLCANIC ROCKS
- SEDIMENTARY ROCKS
- VOLCANIC ROCKS
- ◆ Kimberlite

Tahera Corporation

Fig. 2.2	CENTRAL SLAVE DETAILED REGIONAL GEOLOGY
Date:5/7/2000	
Author: MJ	
Office: VAN	
Drawing:	
Scale: 1:1500000	Projection: UTM Zone 12 (NAD 27)

20 0 20 40

kilometres



Tahera Corporation

Fig. 2.3	Jericho Kimberlite Pipe
Date:23/6/2000	
Author: MJ	
Office: VAN	
Drawing:	
Scale: 1:4000	Projection: UTM Zone 12 (NAD 27)

0 75 150

metres

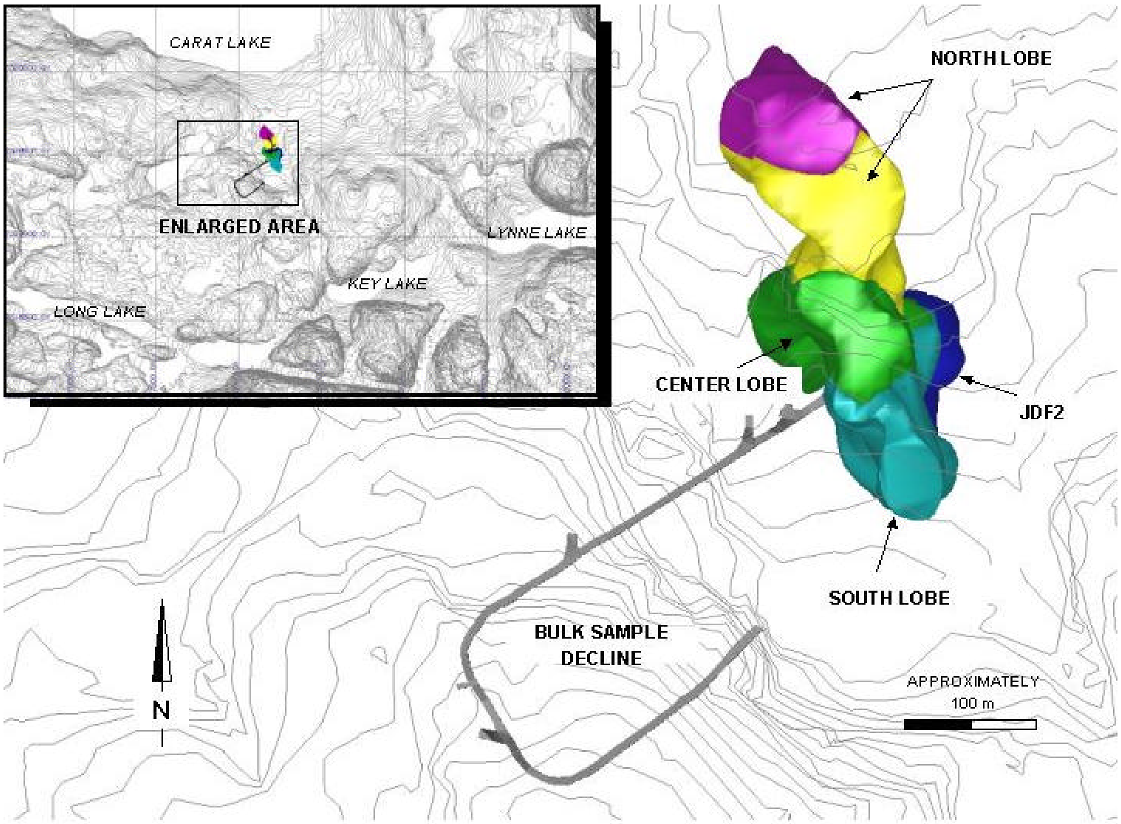
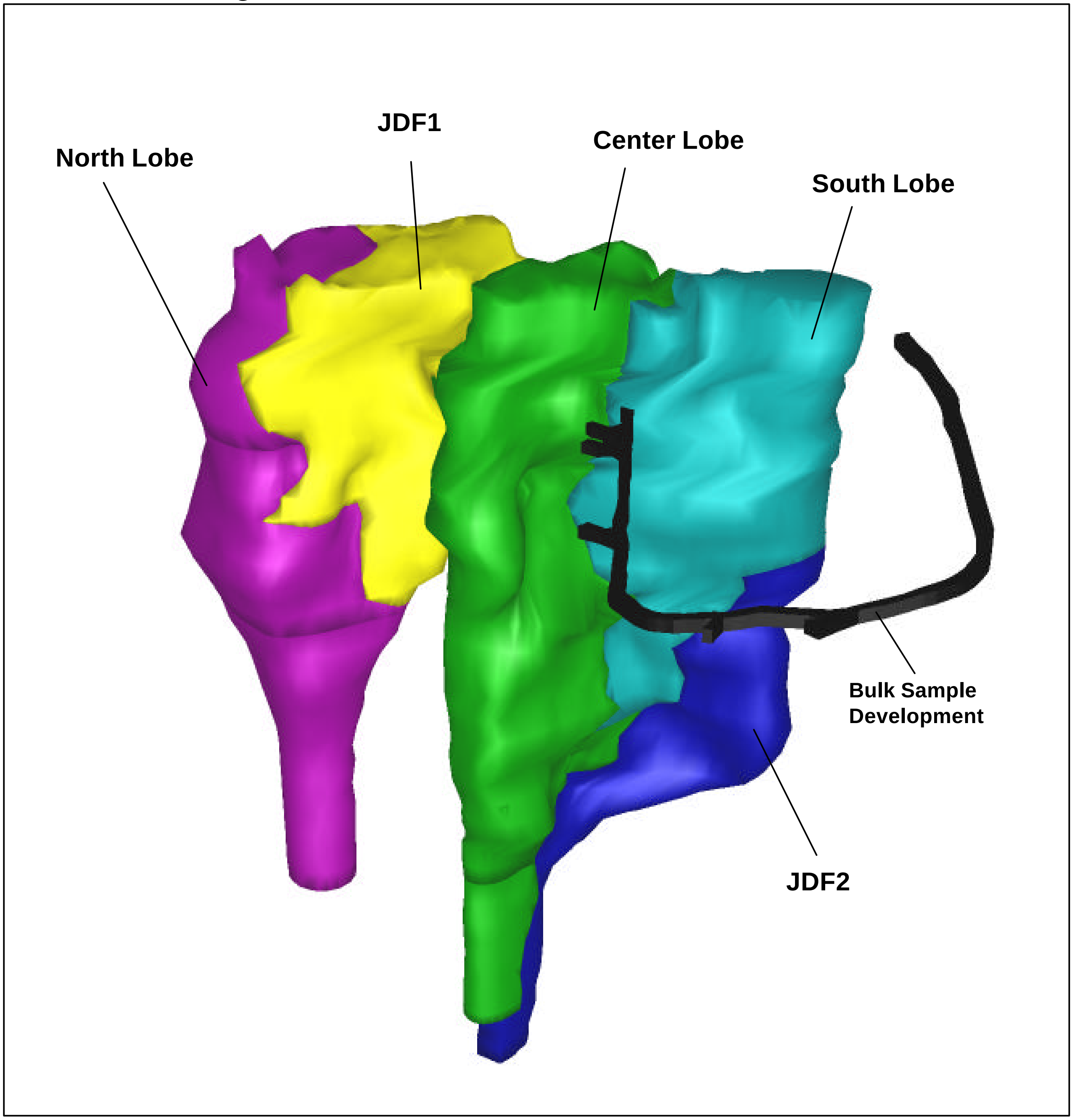
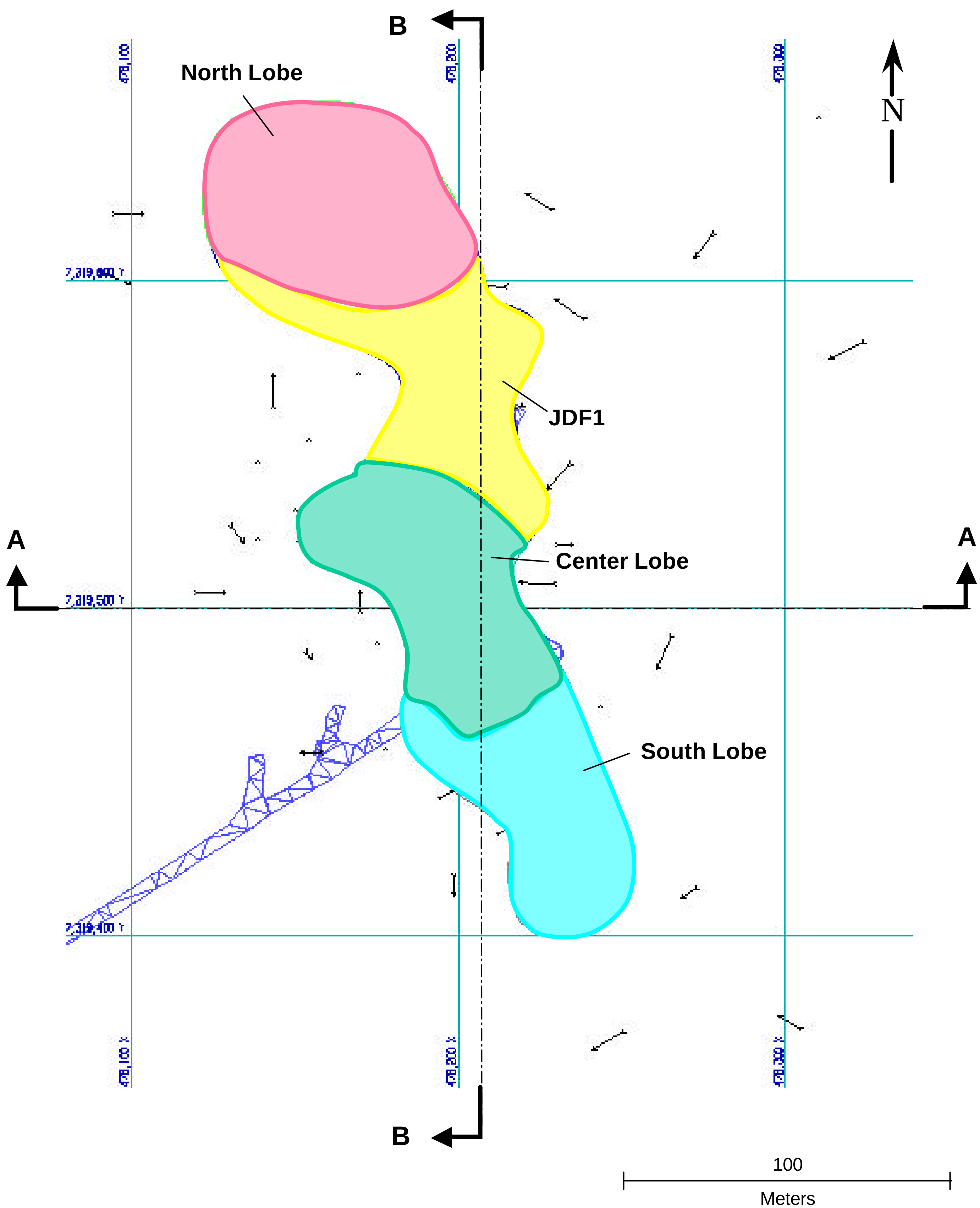


Figure 2.4 Jericho kimberlite pipe location and major kimberlite units.

3D View looking Northeast



SRK Consulting Engineers and Scientists	Jericho Diamond Project Feasibility Study			
TAHERA CORPORATION	**3D View showing Geometry of various Lobes**			
	PROJECT: 2CT005.06	DATE: June 2000	APPROVED:	FIGURE:: 2.5



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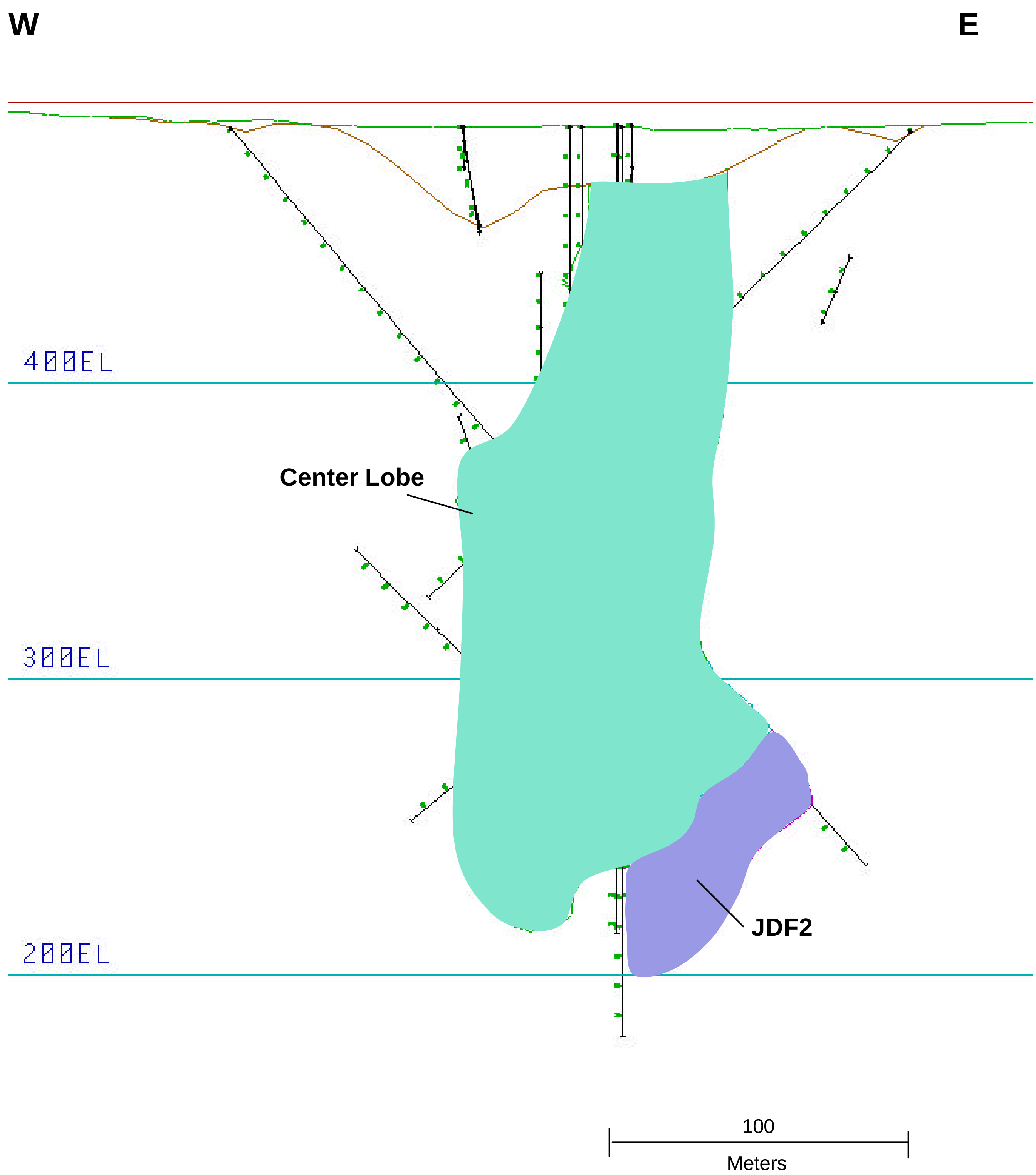
Jericho Diamond Project
Feasibility Study

Kimberlite Lobes at 410m Elev. with Development

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		2.6



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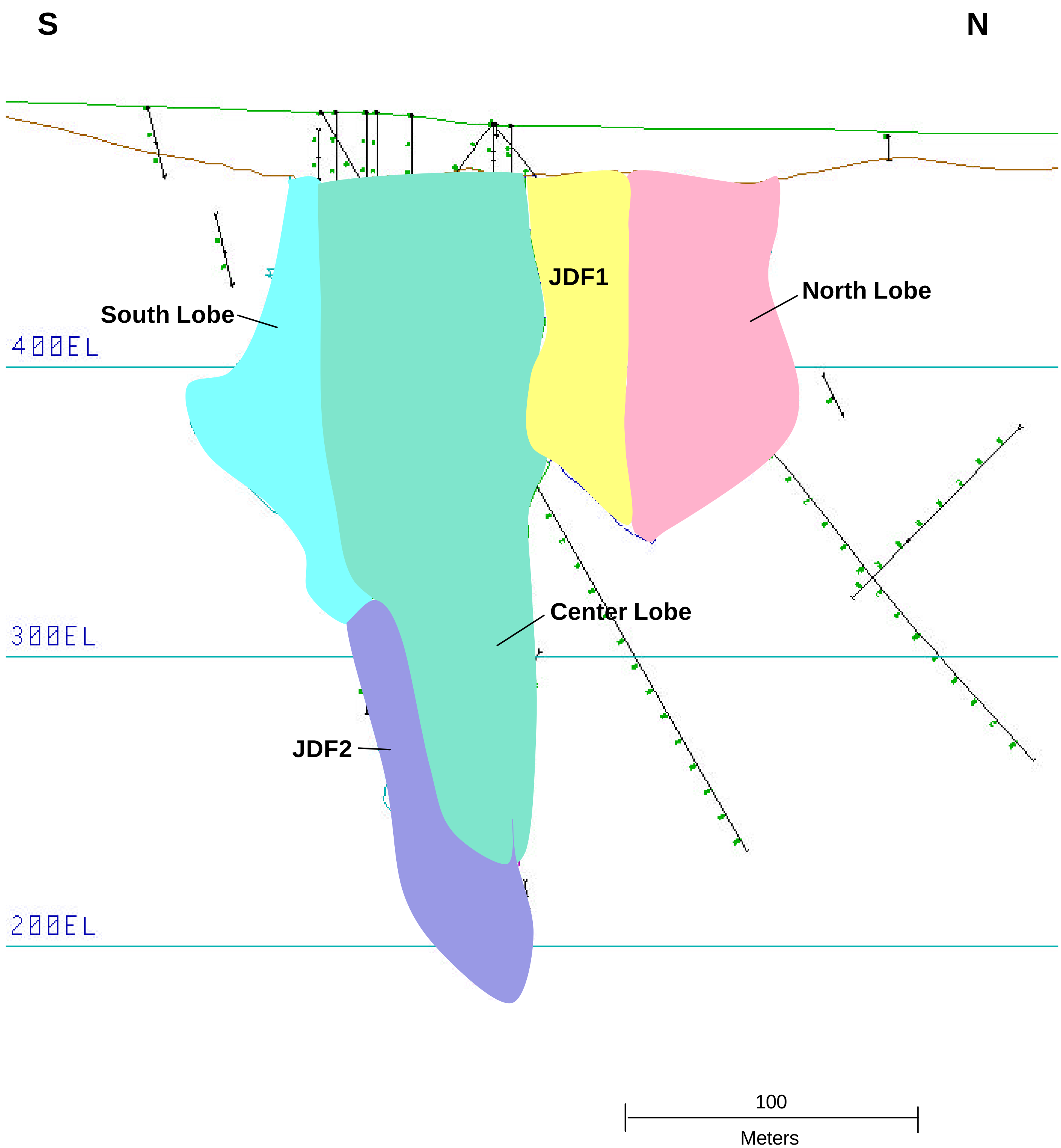
Jericho Diamond Project Feasibility Study			
Kimberlite Lobes - Section A-A (7319500N)			
PROJECT: 2CT005.06	DATE: June 2000	APPROVED:	FIGURE:: **2.7**



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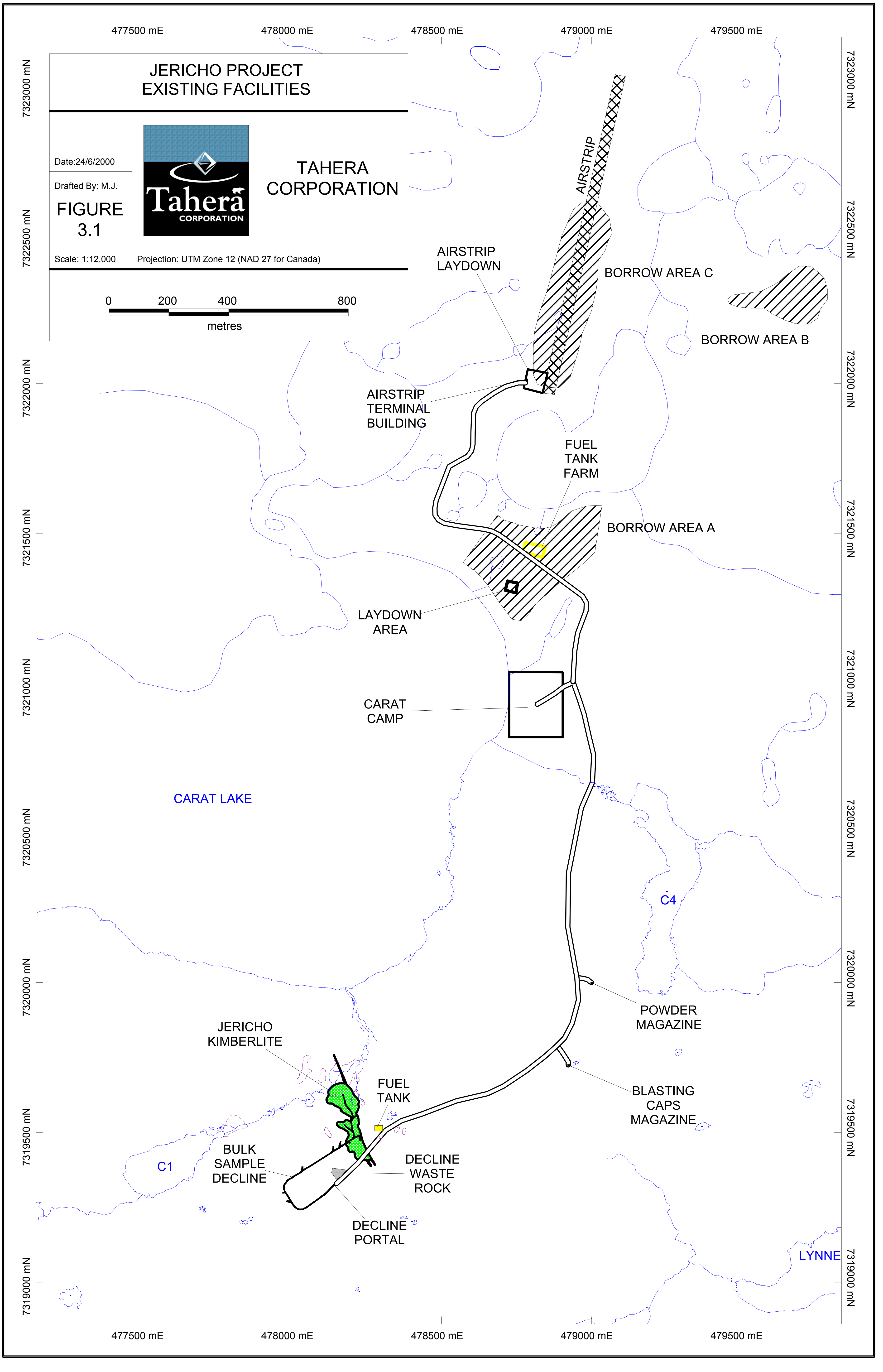
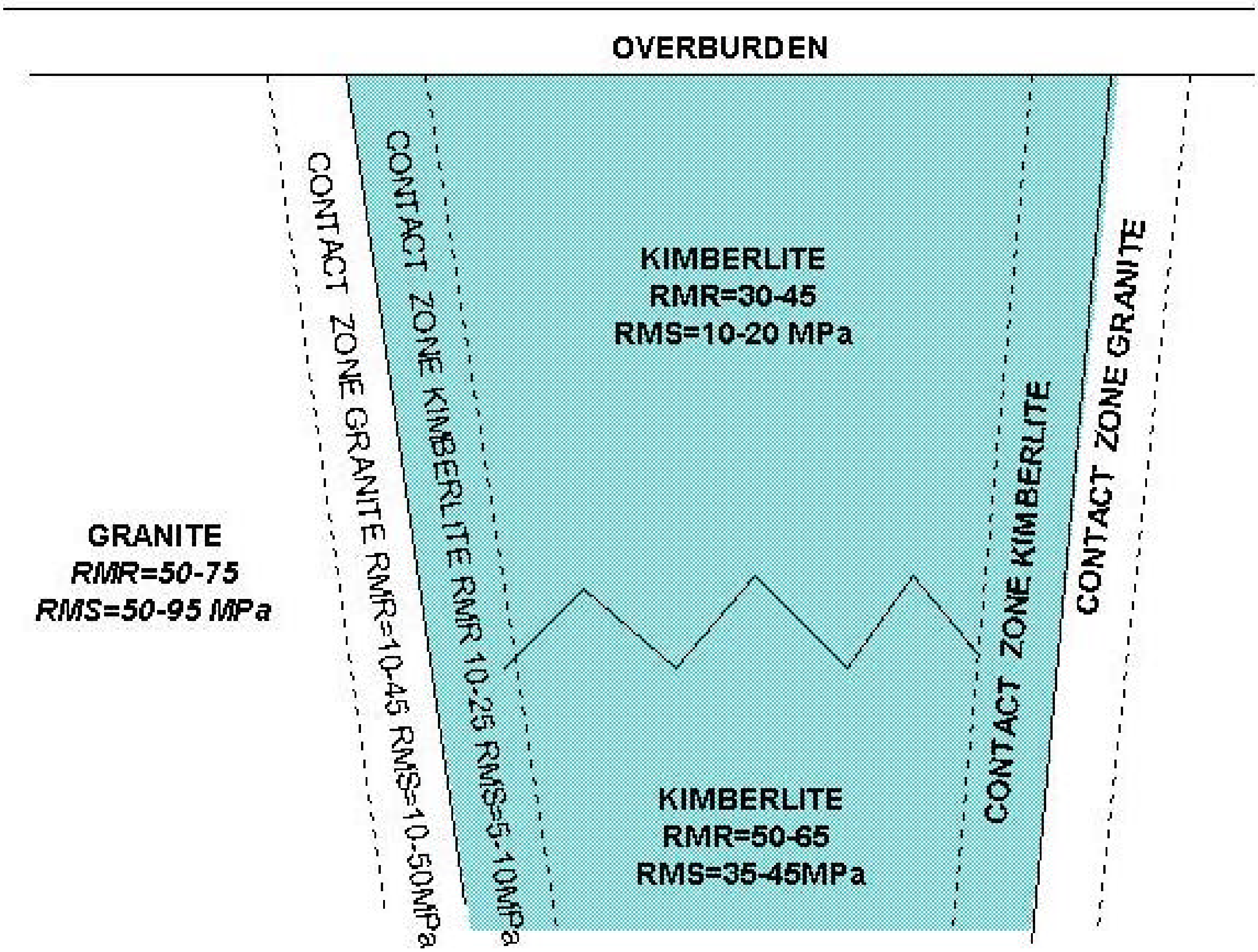
Jericho Diamond Project
Feasibility Study

Kimberlite Lobes - Section B-B (478205 E)

PROJECT:	DATE:	APPROVED:	FIGURE:
2CT005.06	June 2000		2.8


JERICHO PROJECT
EXISTING FACILITIES
Date:24/6/2000
Drafted By: M.J.
FIGURE
3.1
Taherā
CORPORATION
TAHERA
CORPORATION
Scale: 1:12,000
Projection: UTM Zone 12 (NAD 27 for Canada)
0
200
400
800
metres
477500 mE
478000 mE
478500 mE
479000 mE
479500 mE
7323000 mN
7322500 mN
7322000 mN
7321500 mN
7321000 mN
7320500 mN
7320000 mN
7319500 mN
7319000 mN
AIRSTRIP
AIRSTRIP LAYDOWN
BORROW AREA C
BORROW AREA B
AIRSTRIP TERMINAL BUILDING
FUEL TANK FARM
BORROW AREA A
LAYDOWN AREA
CARAT CAMP
CARAT LAKE
C4
POWDER MAGAZINE
BLASTING CAPS MAGAZINE
JERICHO KIMBERLITE
FUEL TANK
BULK SAMPLE DECLINE
C1
DECLINE WASTE ROCK
DECLINE PORTAL
LYNNE



NOTES:

- *ILLUSTRATED ARE TYPICAL VALUES ESTIMATED BY SRK, BASED ON DRILL CORE REVIEW*
- *SCHEMATIC VERTICAL SECTION, NOT TO SCALE*
- *LAUBSCHER'S RMR AND RMS*

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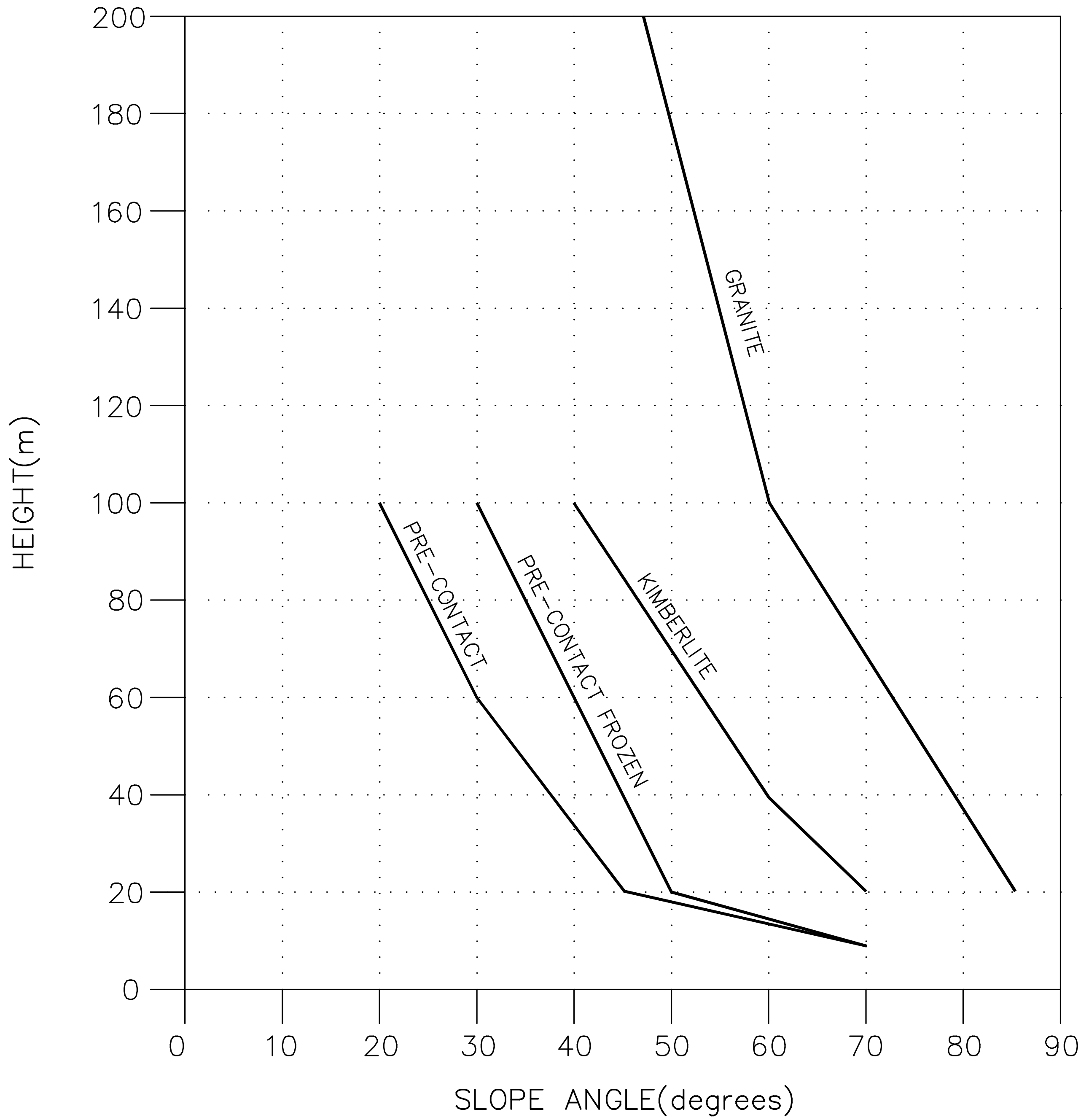
Jericho Diamond Project
Feasibility Study

Schematic Section Through the Pipe and Major Geotechnical Units

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		**5.1**



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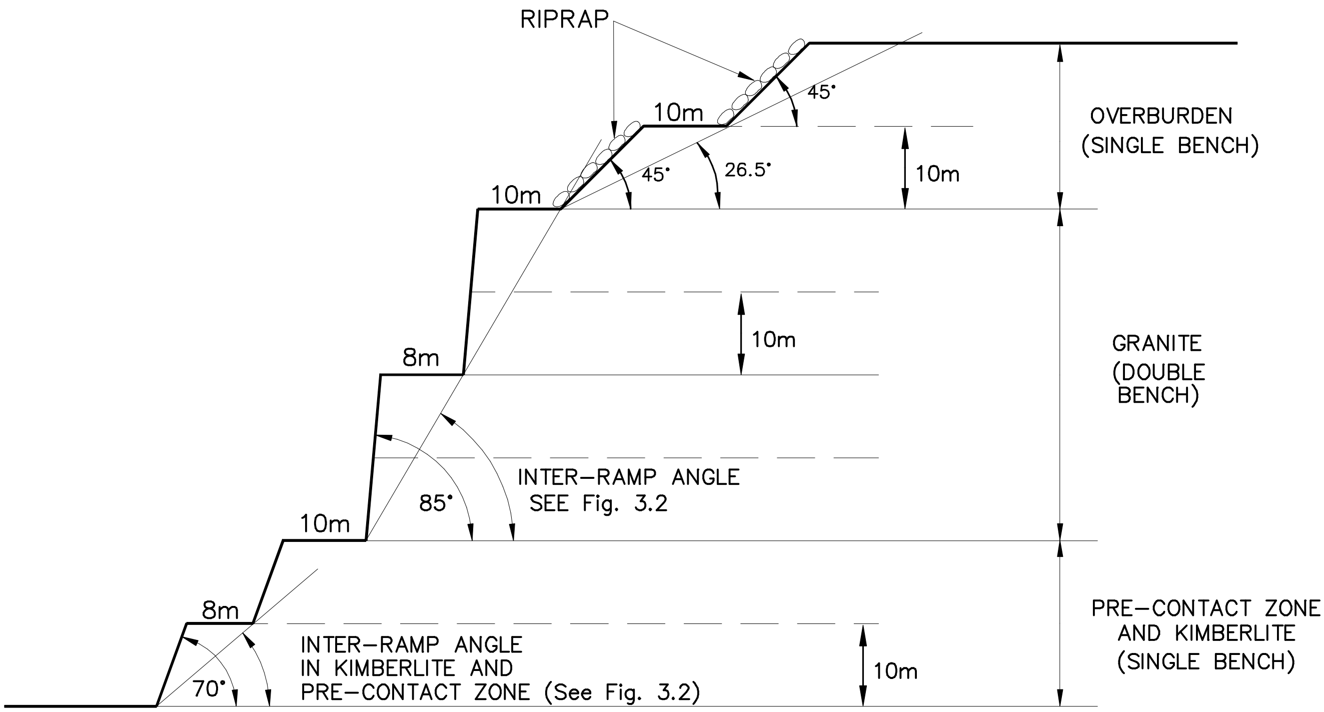
JERICHO PROJECT

PRELIMINARY INTER-RAMP SLOPES

PROJECT NO.	DATE	APPROVED	FIGURE
2CT005.06	JUNE 2000		5.2



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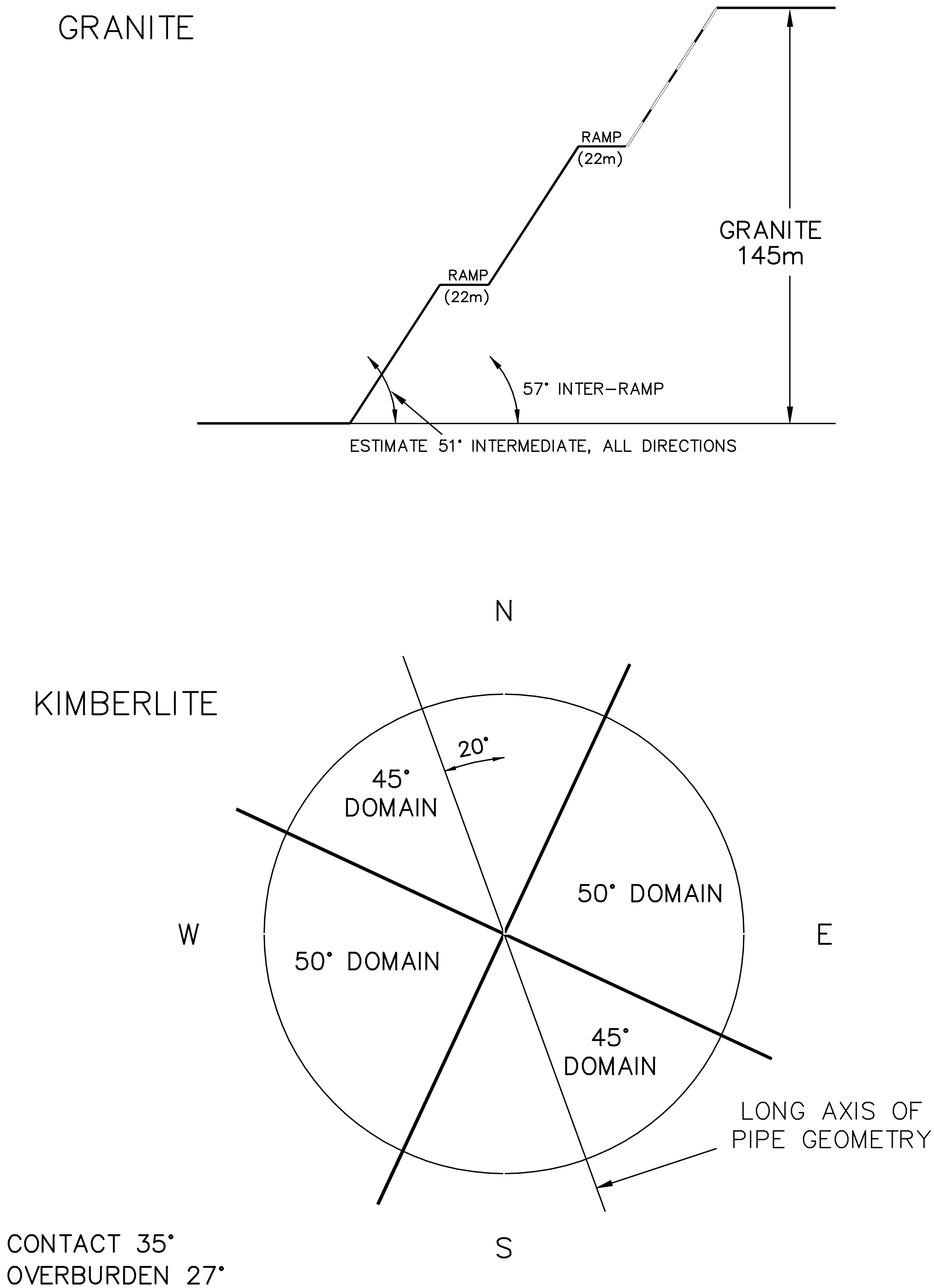
JERICHO PROJECT

BENCH CONFIGURATION AND
INTER–RAMP PIT ANGLES

PROJECT NO.	DATE	APPROVED	FIGURE
2CT005.06	JUNE 2000		5.3



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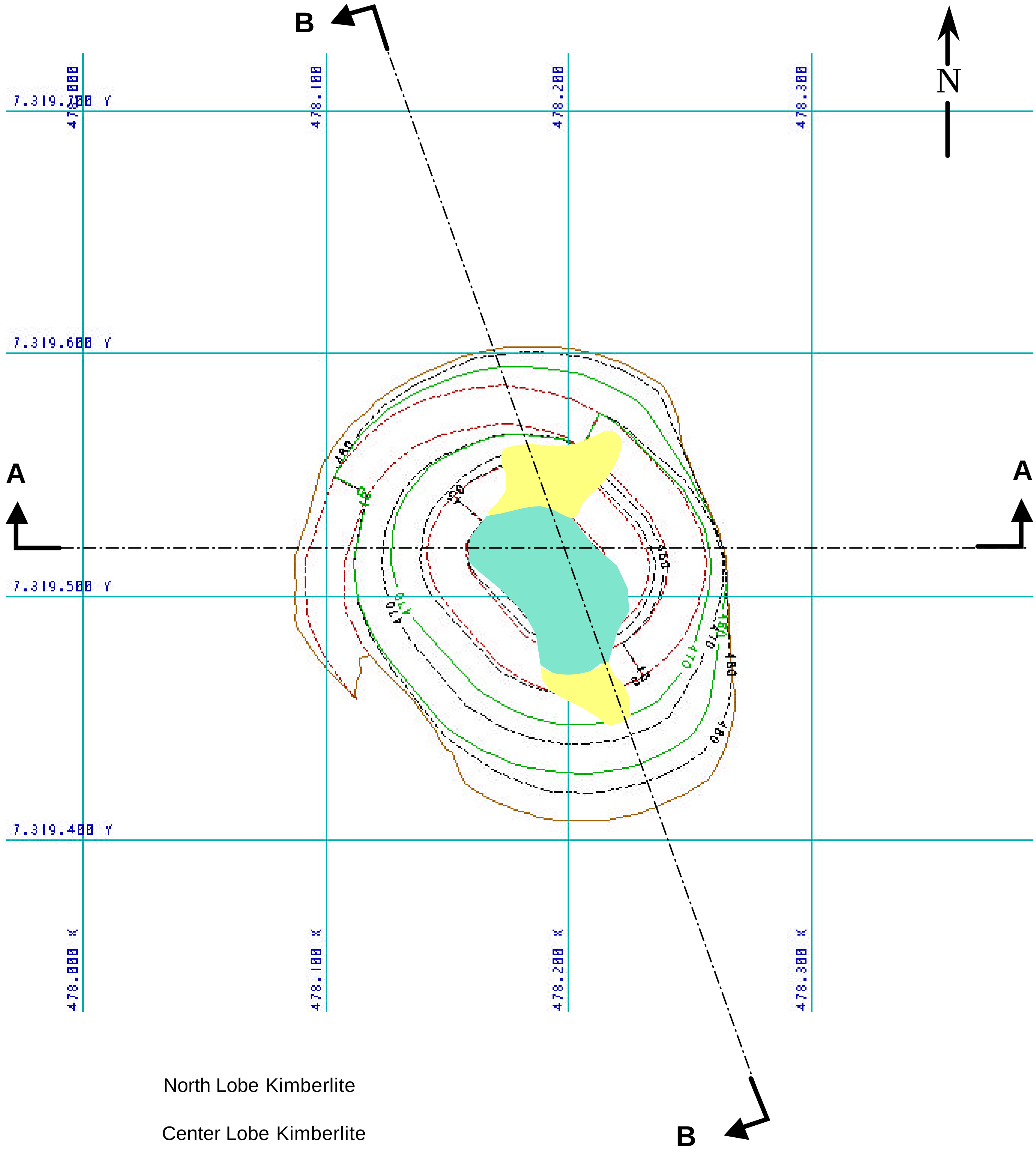
JERICHO PROJECT

OPTIMIZATION PARAMETERS–
OVERALL PIT ANGLES

PROJECT NO.	DATE	APPROVED	FIGURE
2CT005.06	JUNE 2000		5.4



North Lobe Kimberlite

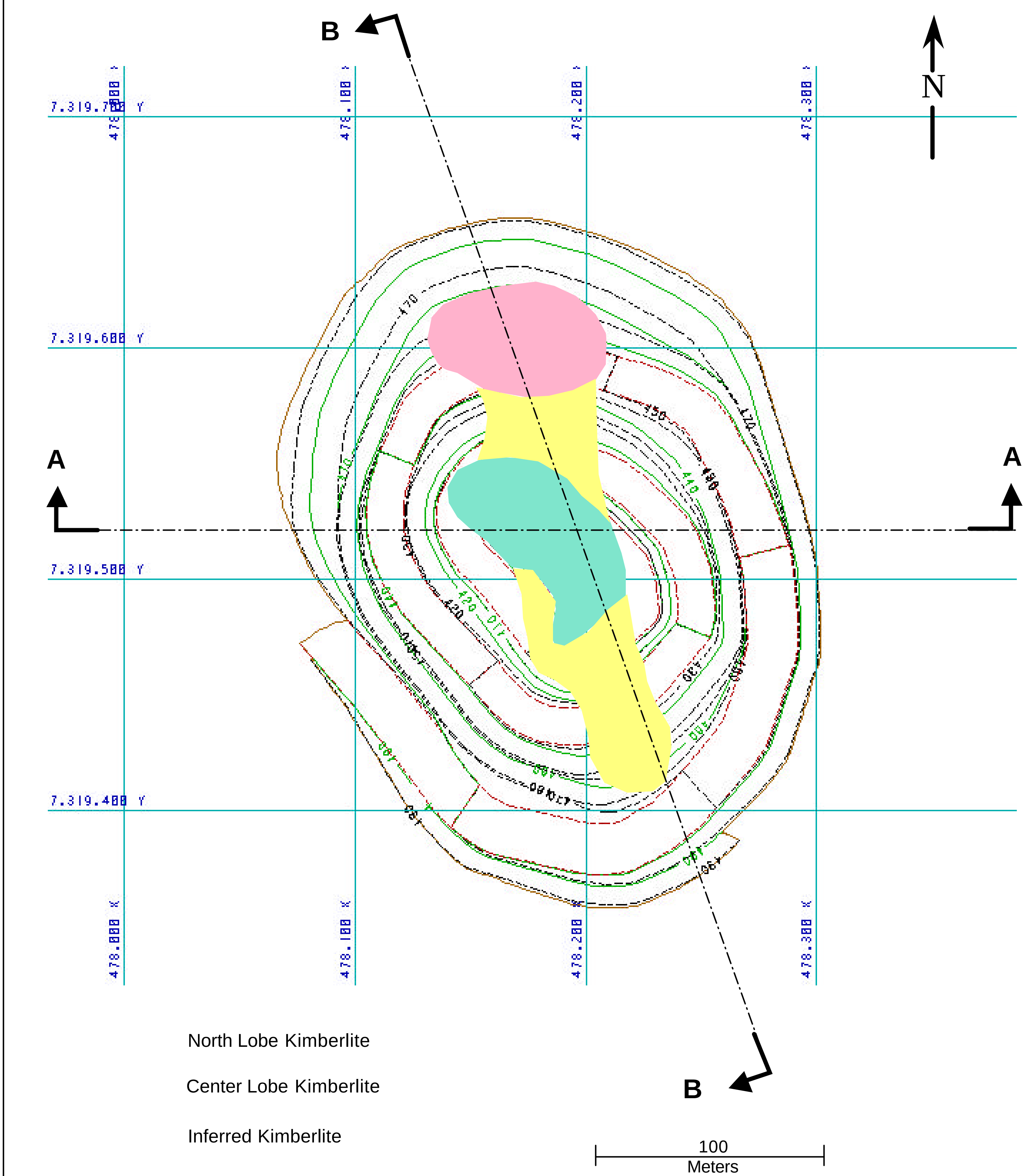
Center Lobe Kimberlite

Inferred Kimberlite

100
Meters

SRK Consulting Engineers and Scientists	Jericho Diamond Project Feasibility Study
TAHERA CORPORATION	**Design Starter Pit**

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		**5.5**



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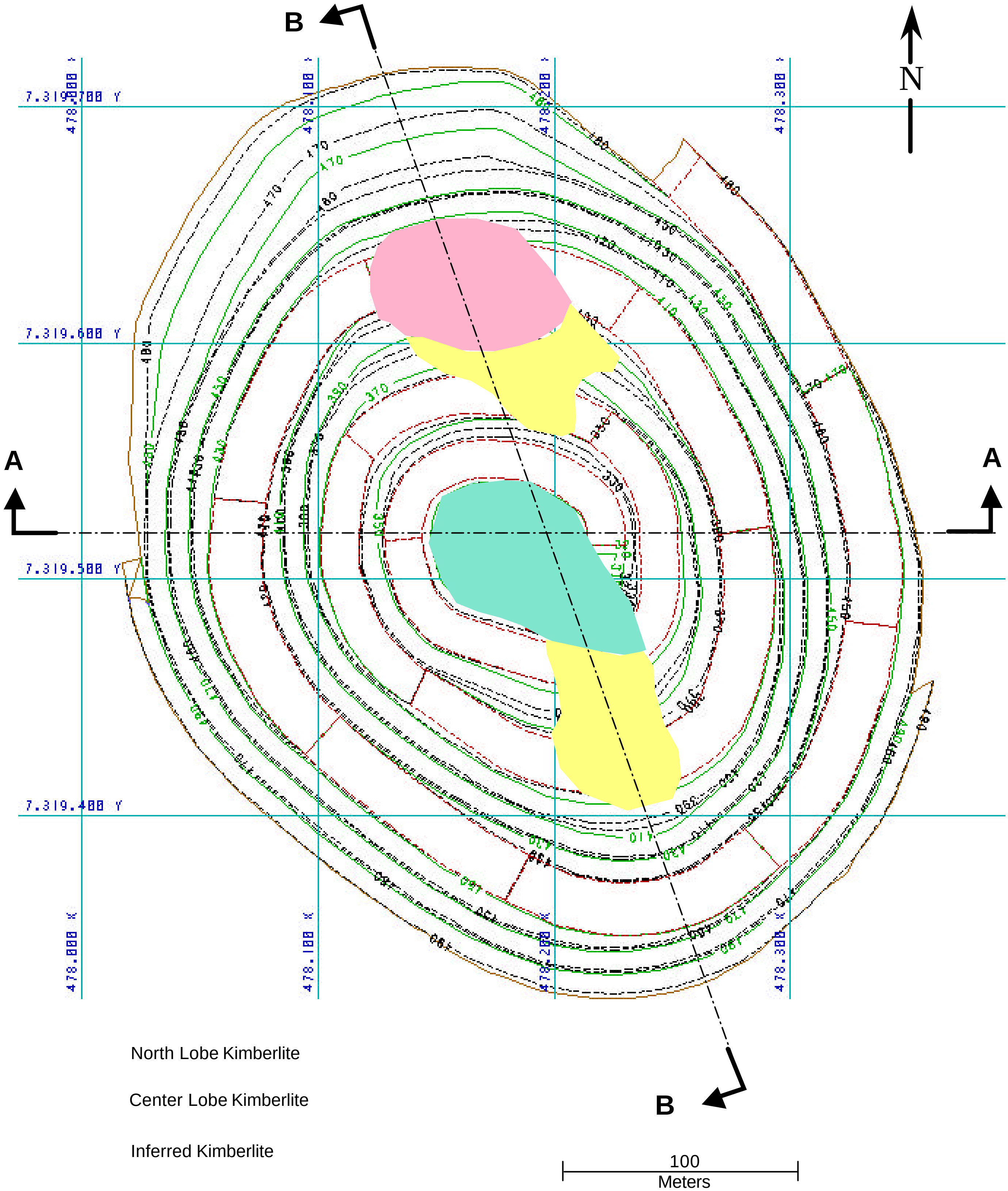
Jericho Diamond Project
Feasibility Study

Pushback Pit

PROJECT:	DATE:	APPROVED:	FIGURE:
2CT005.06	June 2000		5.6



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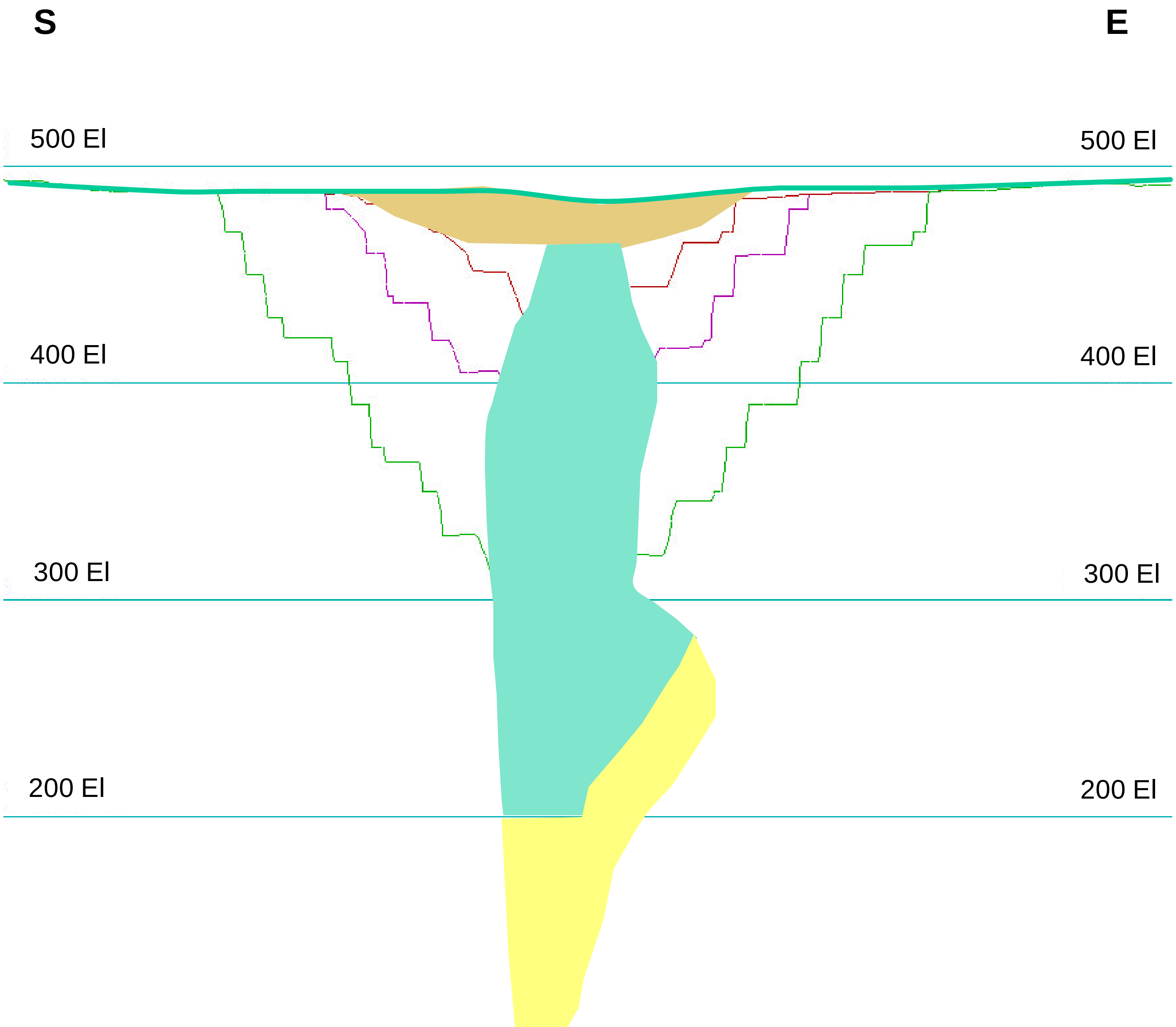
Jericho Diamond Project
Feasibility Study

Ultimate Pit

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.7



North Lobe Kimberlite

Inferred Kimberlite

Center Lobe Kimberlite

Overburden

100
Meters

* For Section location see Fig. 3.5, 3.6 & 3.7

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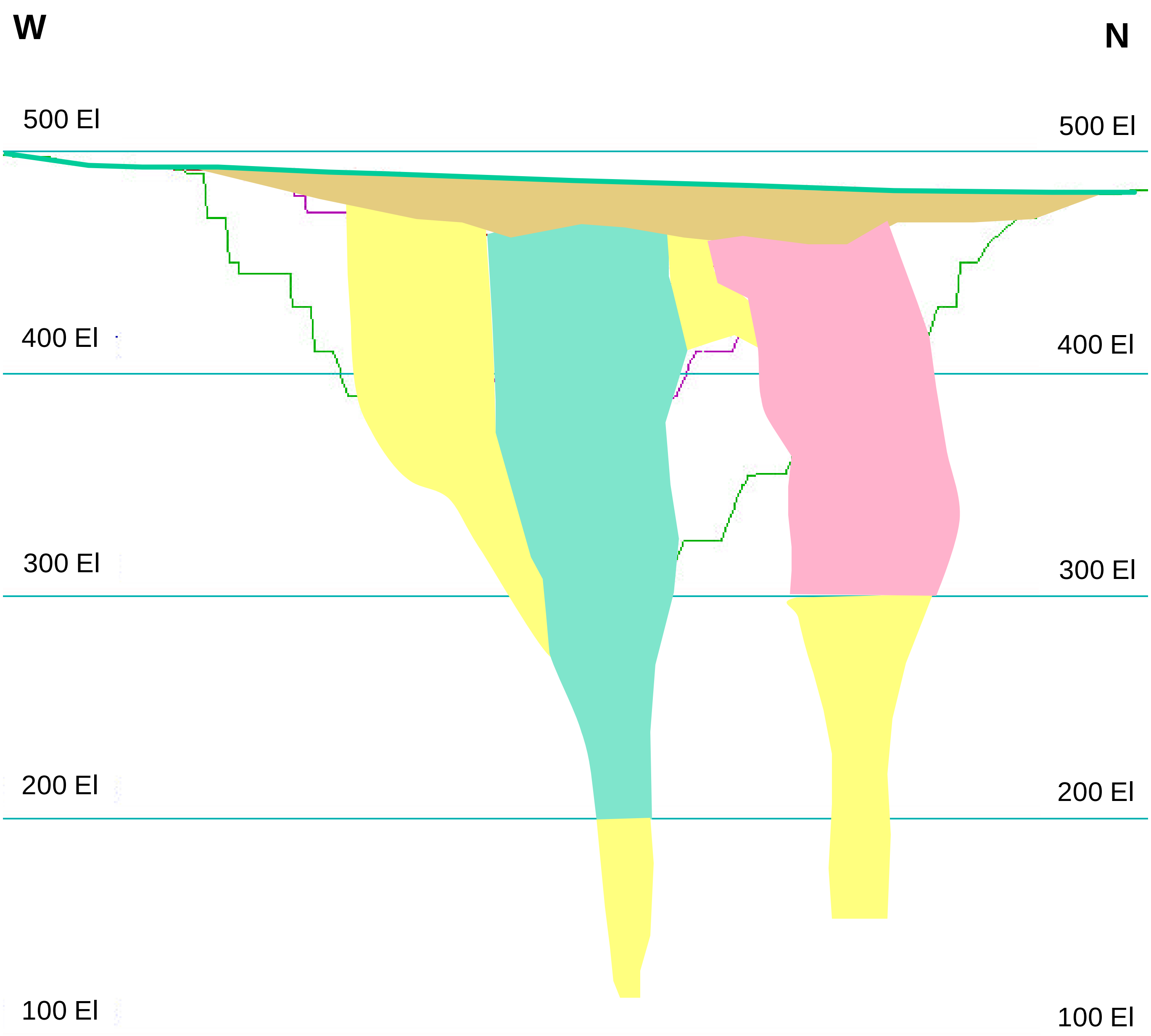
Jericho Diamond Project
Feasibility Study

Section A-A*
(7319520N)

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		**5.8**



North Lobe Kimberlite

Inferred Kimberlite

Center Lobe Kimberlite

Overburden

100
Meters

* For Section location see Fig. 3.5, 3.6 & 3.7

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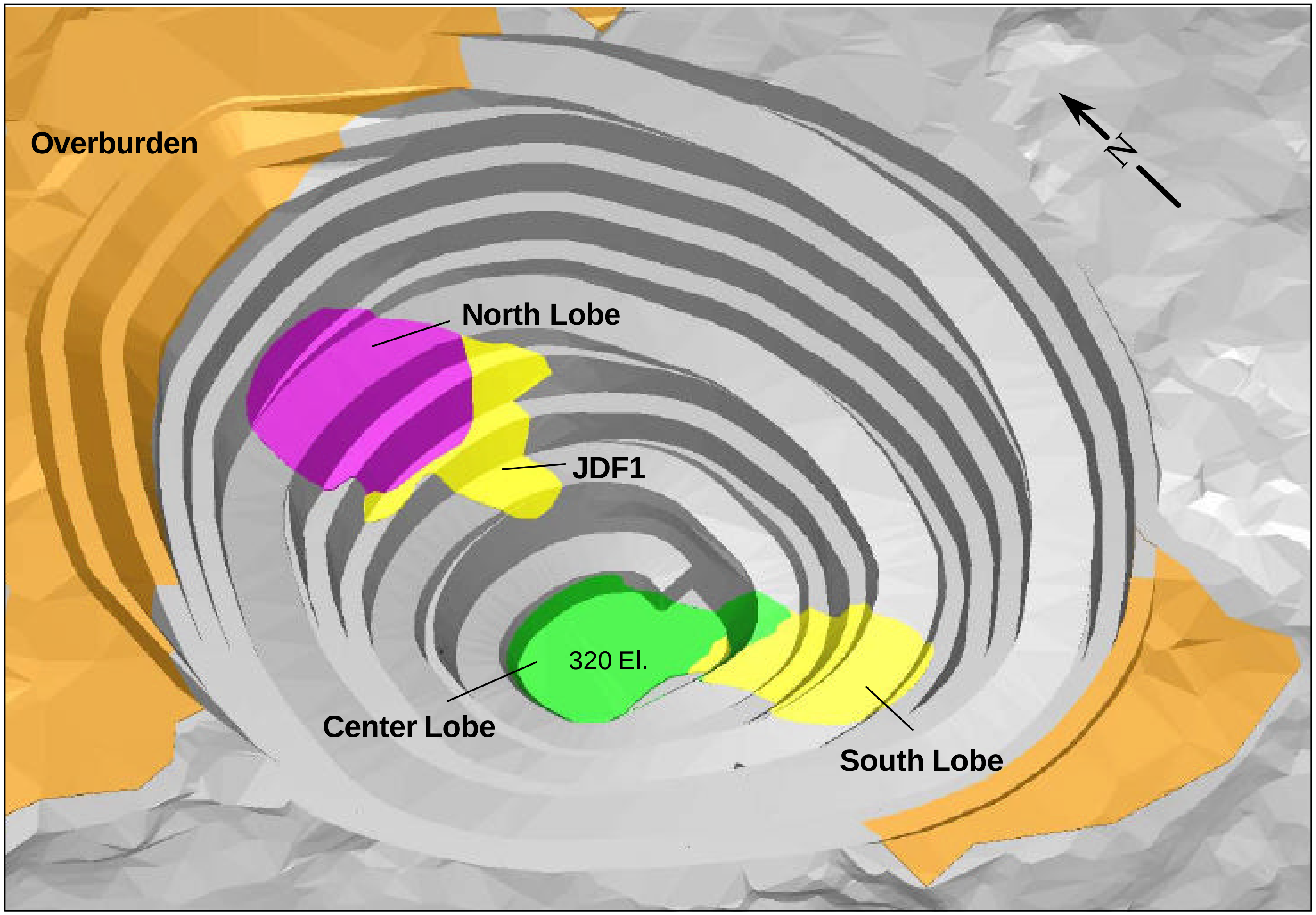
Jericho Diamond Project
Feasibility Study

Section B-B*

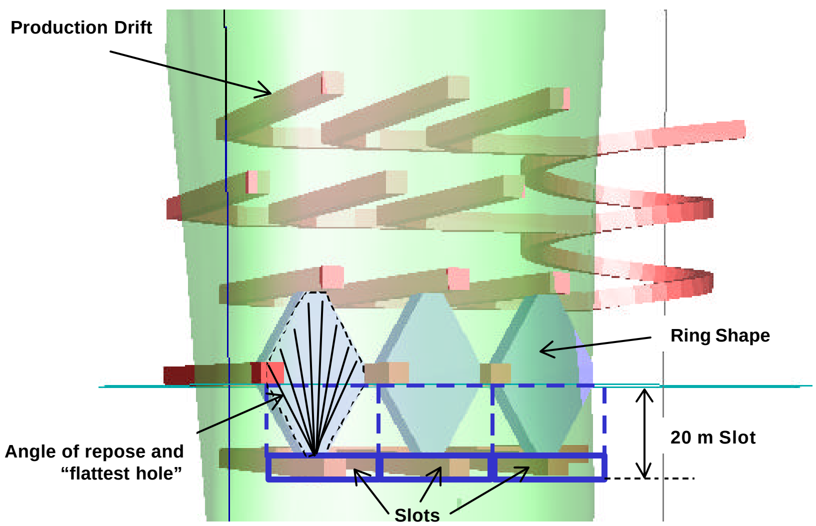
PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.9



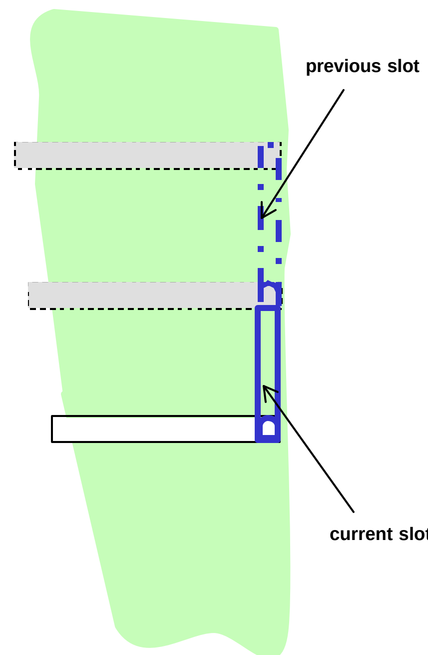
SRK Consulting Engineers and Scientists	Jericho Diamond Project Feasibility Study
TAHERA CORPORATION	**Overall Open Pit Layout**

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.10

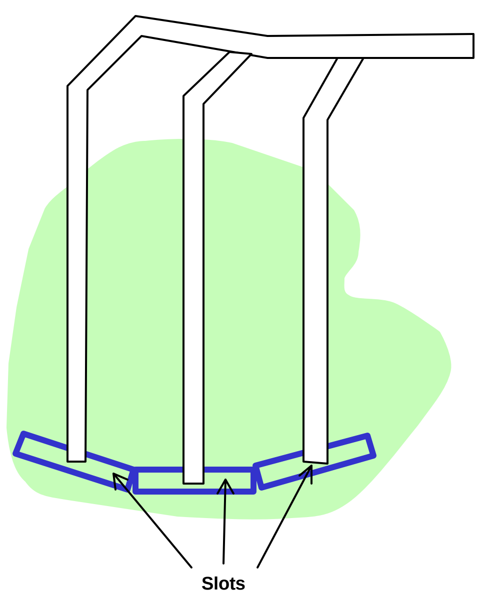
3-D View of Development



Section View



Plan View



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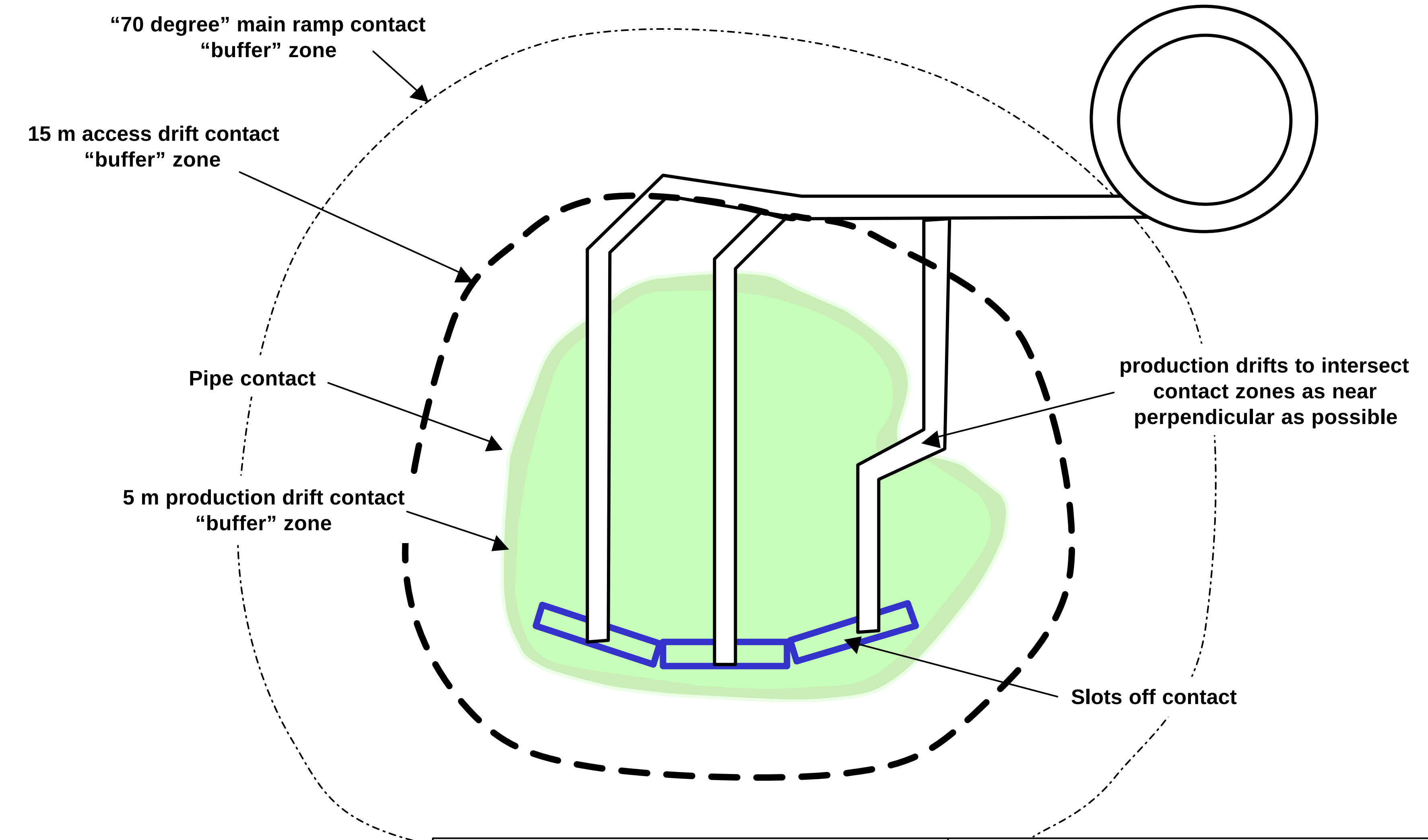
Jericho Diamond Project
Feasibility Study

Key Design Elements for Open Benching

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.11



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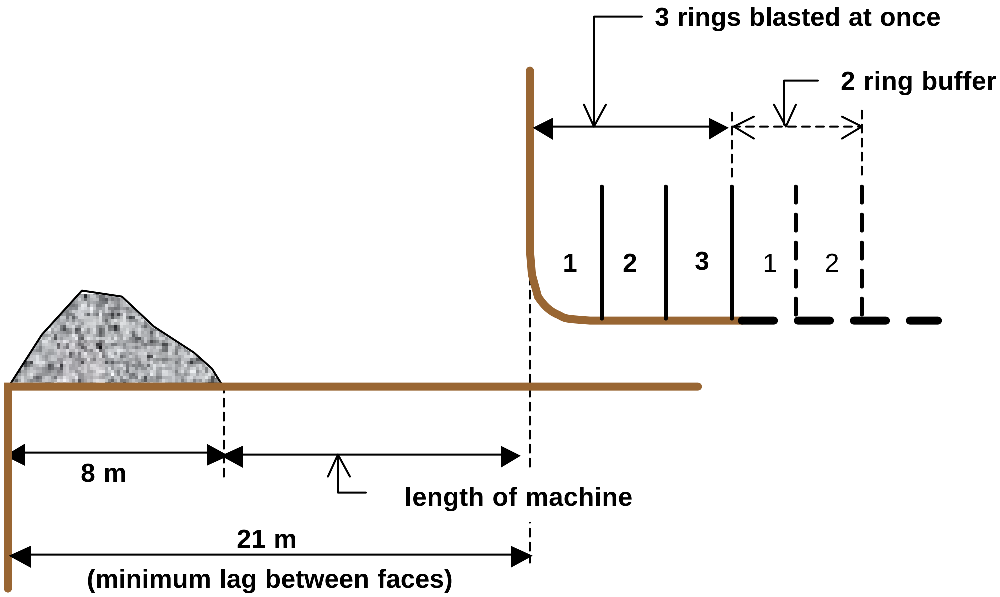
Jericho Diamond Project
Feasibility Study

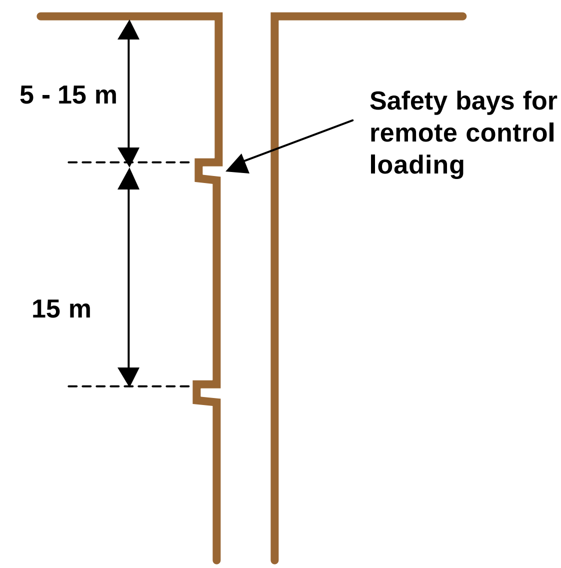
Guidelines for Development Layout in Plan View

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.12

Face Development



Safety Bays



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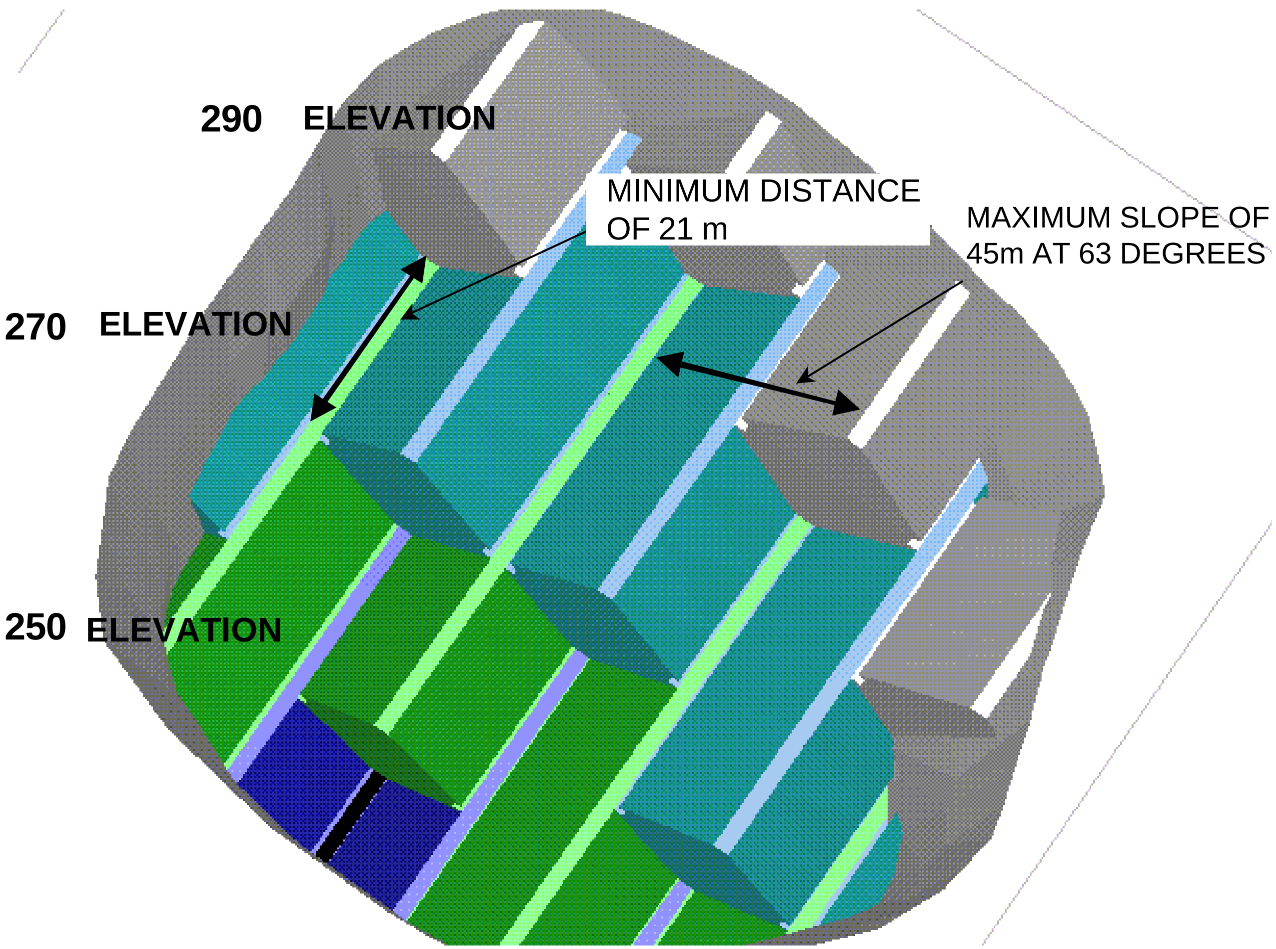
Jericho Diamond Project
Feasibility Study

Schematic of Face Geometry and Safety Bays

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.13



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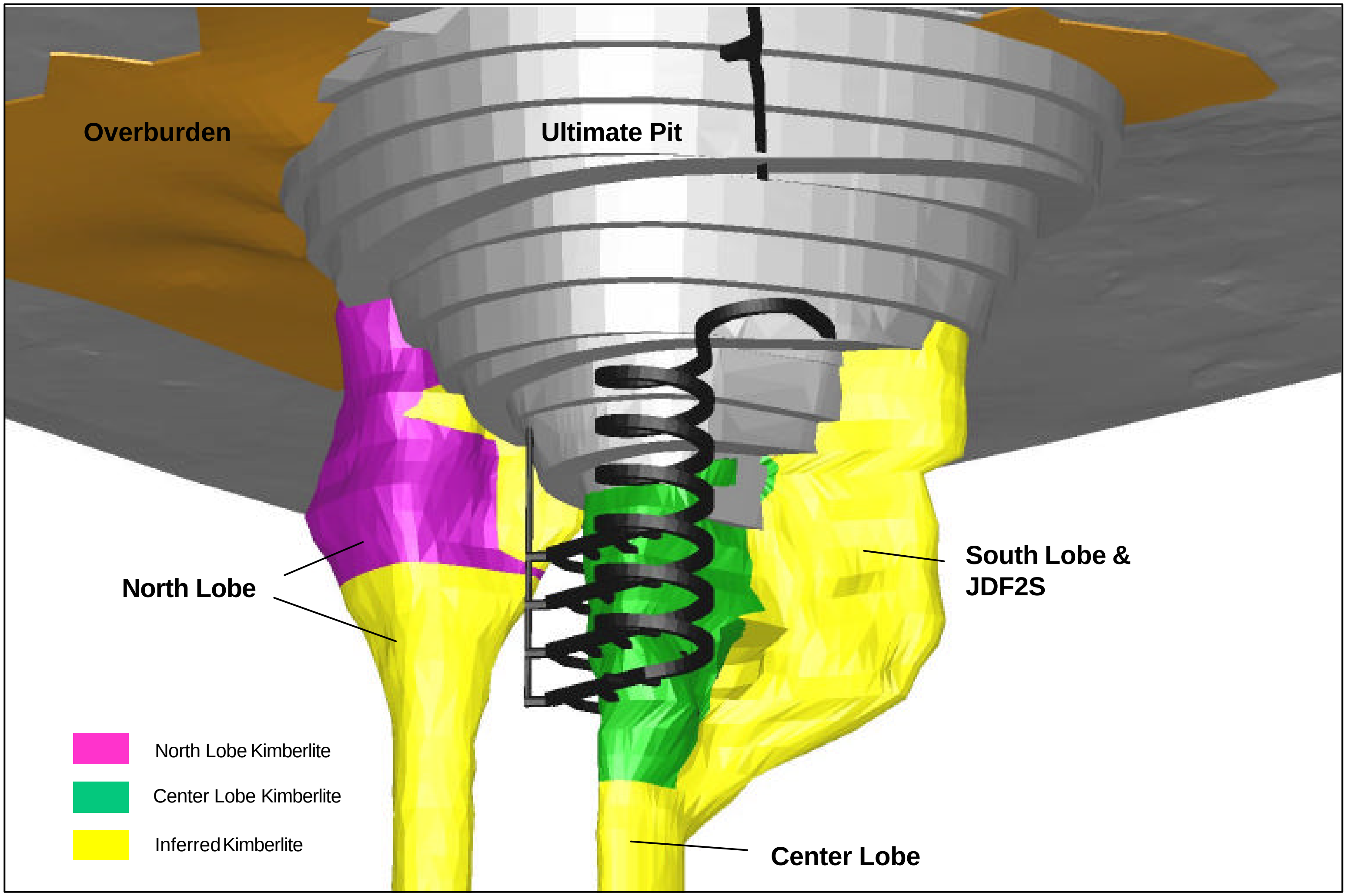
Jericho Diamond Project
Feasibility Study

Isometric View of Open Benching Face Position

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		5.14



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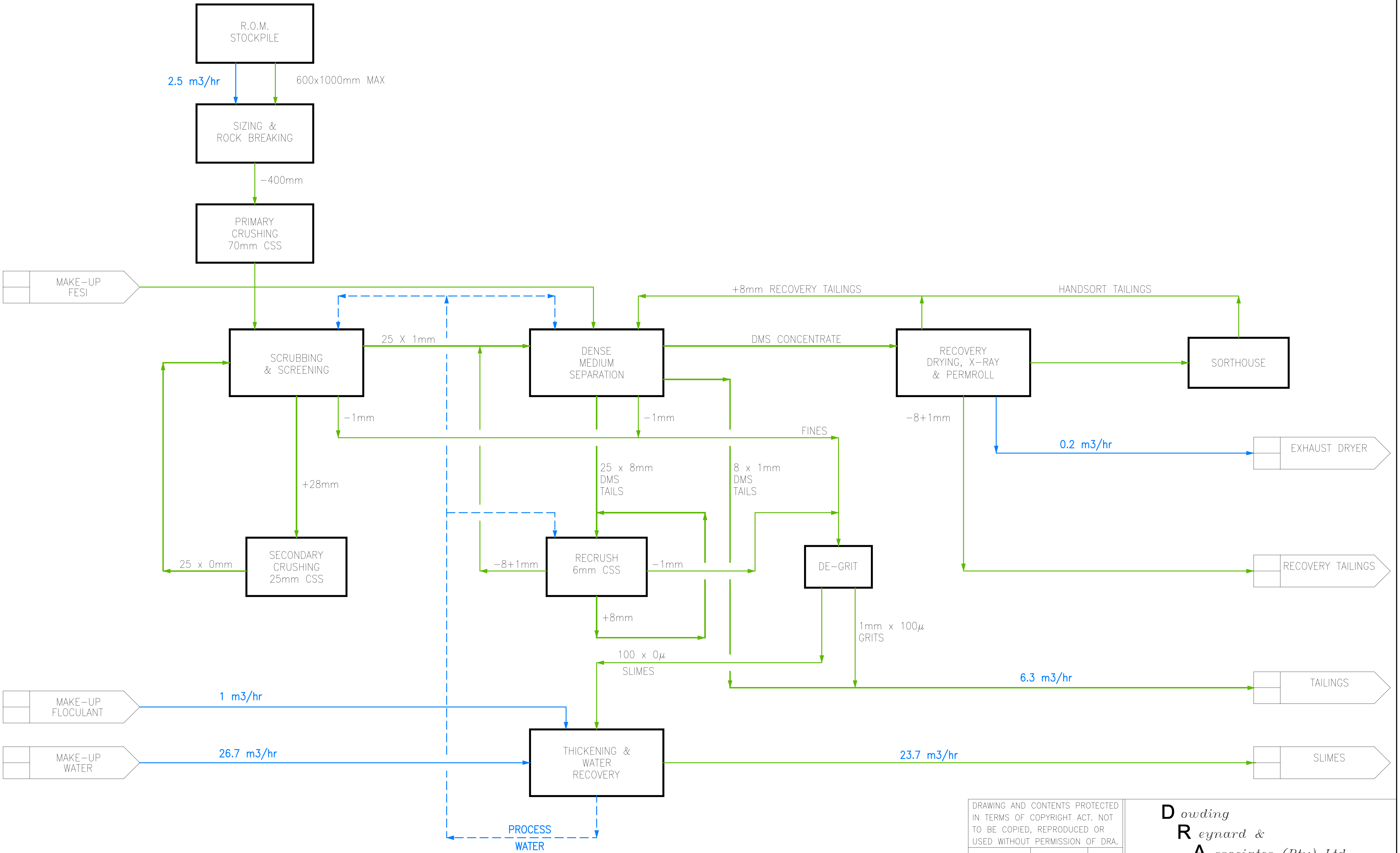
Jericho Diamond Project Feasibility Study			
Overall Underground Mine Layout **(looking Northeast)**			
PROJECT: 2CT005.06	DATE: June 2000	APPROVED:	FIGURE:: **5.15**



CLIENT		
PLANT MANAGER		
PROJECT MAN.		
PROCESS ENG.	I. GILBERT	MAY '99
CHECKED		
DRAWN	MMK	30/04/99
APPROVAL	SIGNATURE	DATE

D *owding*
R *eynard &*
A *ssociates (Pty) Ltd.*

TITLE
TAHERA CORPORATION
BLOCK DIAGRAM SHOWING MAJOR
PROCESSES AND MATERIAL FLOW

DRAWING No.	CAD REF		
FIGURE 6.1	1420B001−3 21/06/2000		
	REVISION 0	SIZE A3	SCALE N.T.S.

LEGEND

- Town
- City

Major Deposits

- Gold
- Base Metals
- Diamonds
- Jericho Pipe

- YK to Jericho Winter Road
- Other Winter Roads
- All Weather Road or Hwy
- Nunavut Boundary

Tahera Corporation

Fig. 8.1

Date:13/1/2000

Author: MJ

Office: VAN

Winter Roads
Yellowknife to Jericho

Scale: 1:3000000

Projection: Longitude / Latitude (NAD 27 for Canada)

0 200 400 800

kilometres

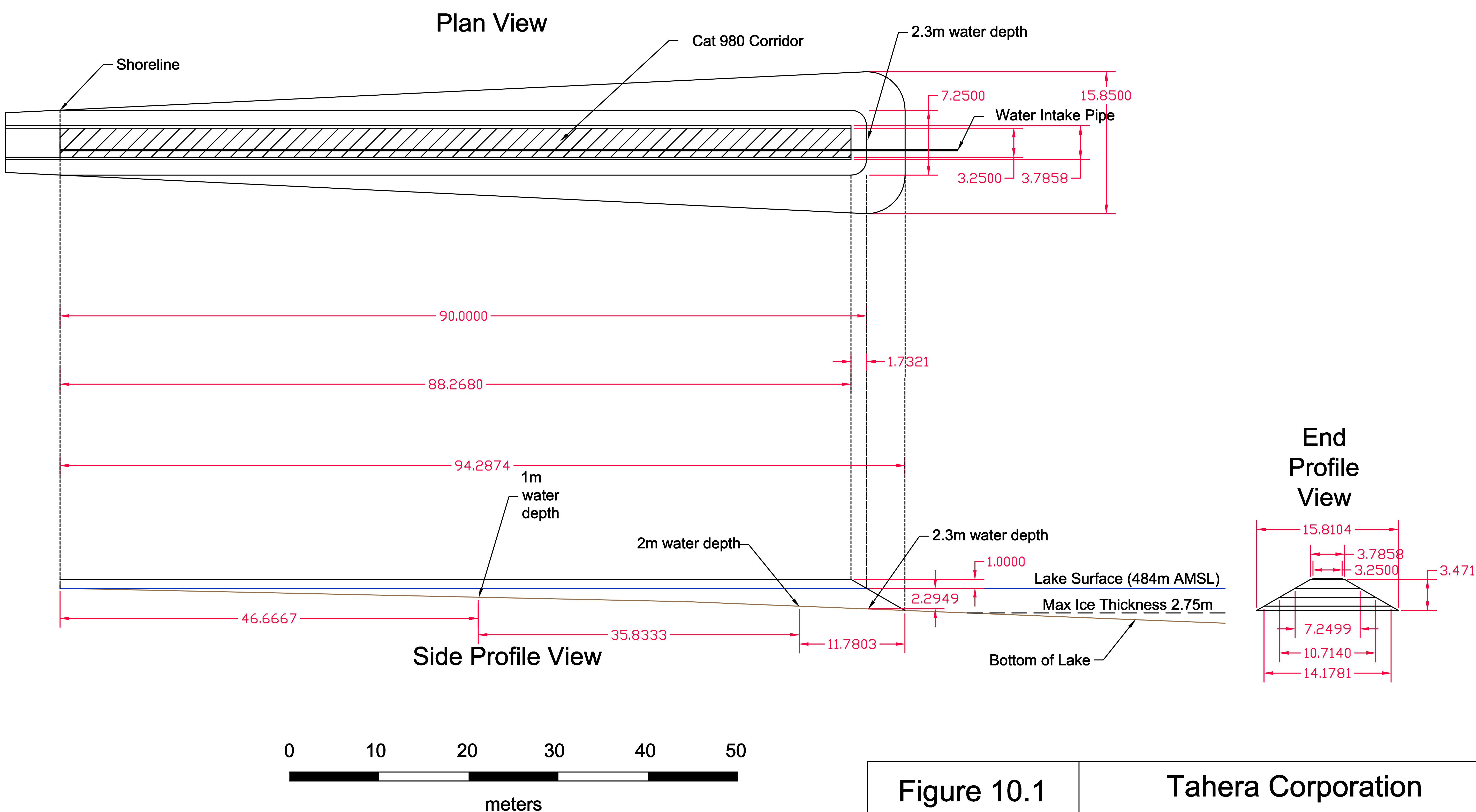
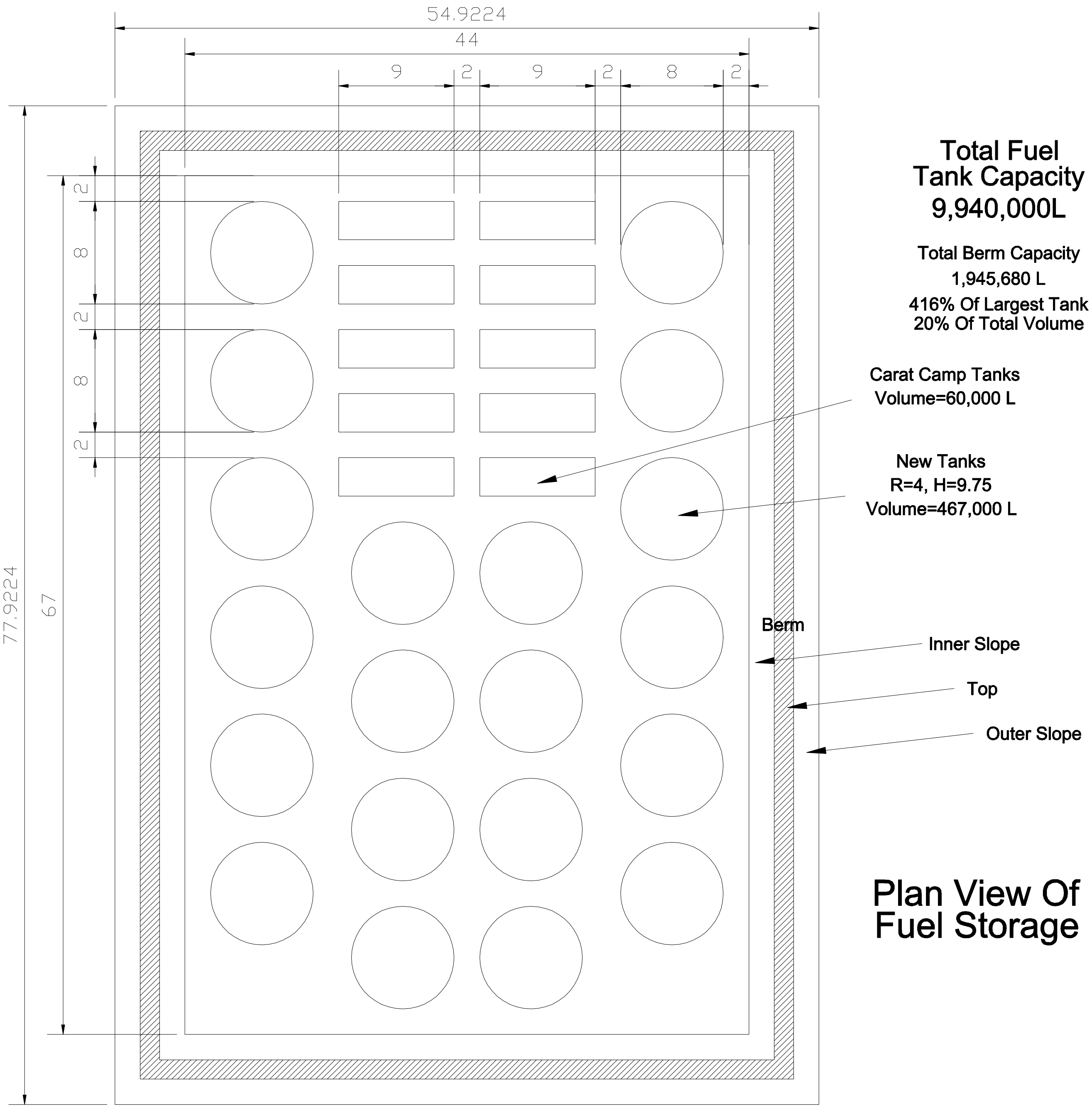
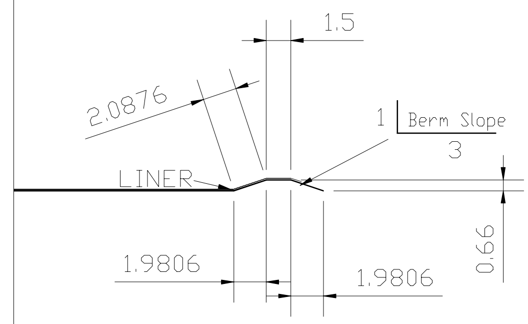


Figure 10.1	Tahera Corporation
Jericho Diamond Project January 2003	Water Intake Causeway Design



Plan View Of
Fuel Storage



Cross Section Of Berm

Jericho Project
Fuel Storage Layout

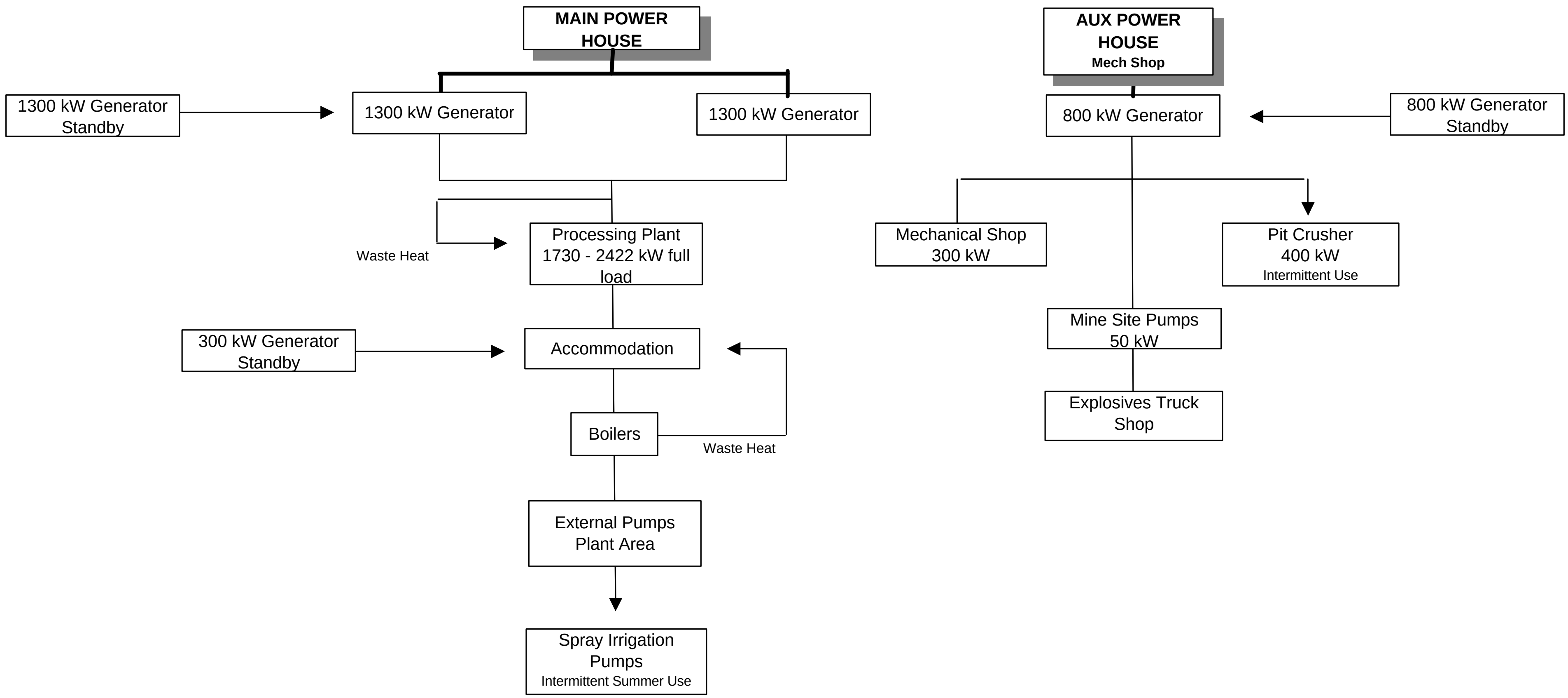
Figure: 11.1

TAHERA CORPORATION

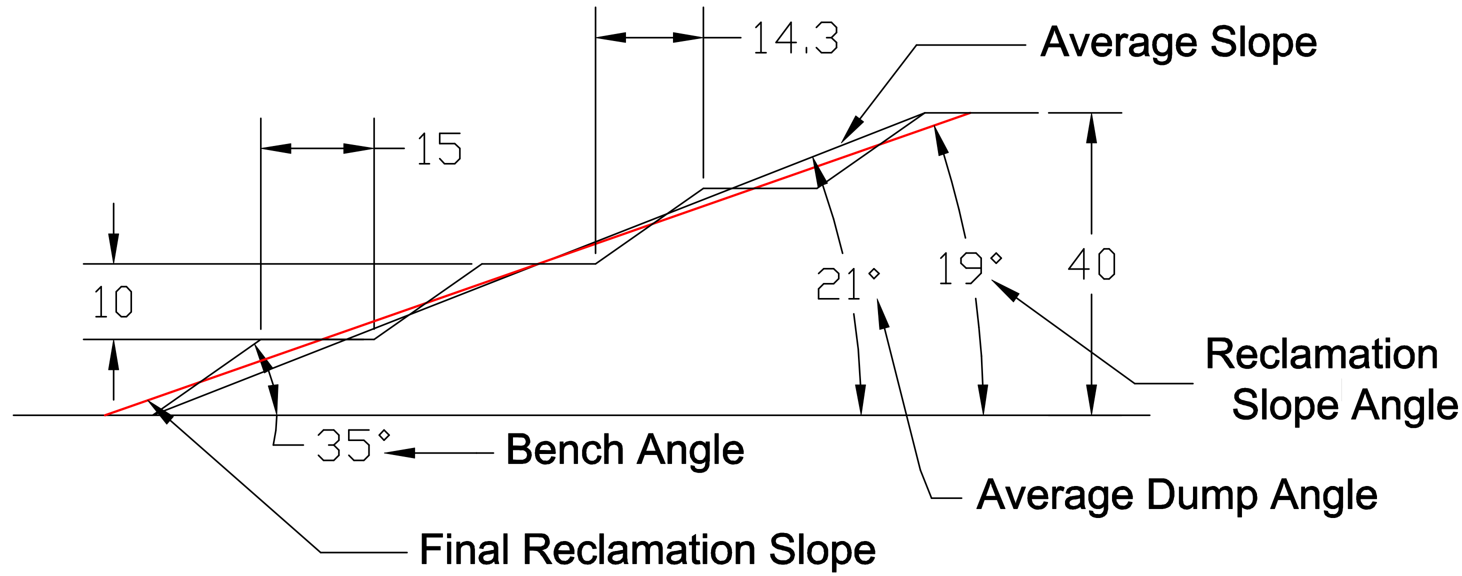
Date:
June 24, 2000

Drafted By:
M. Johnson

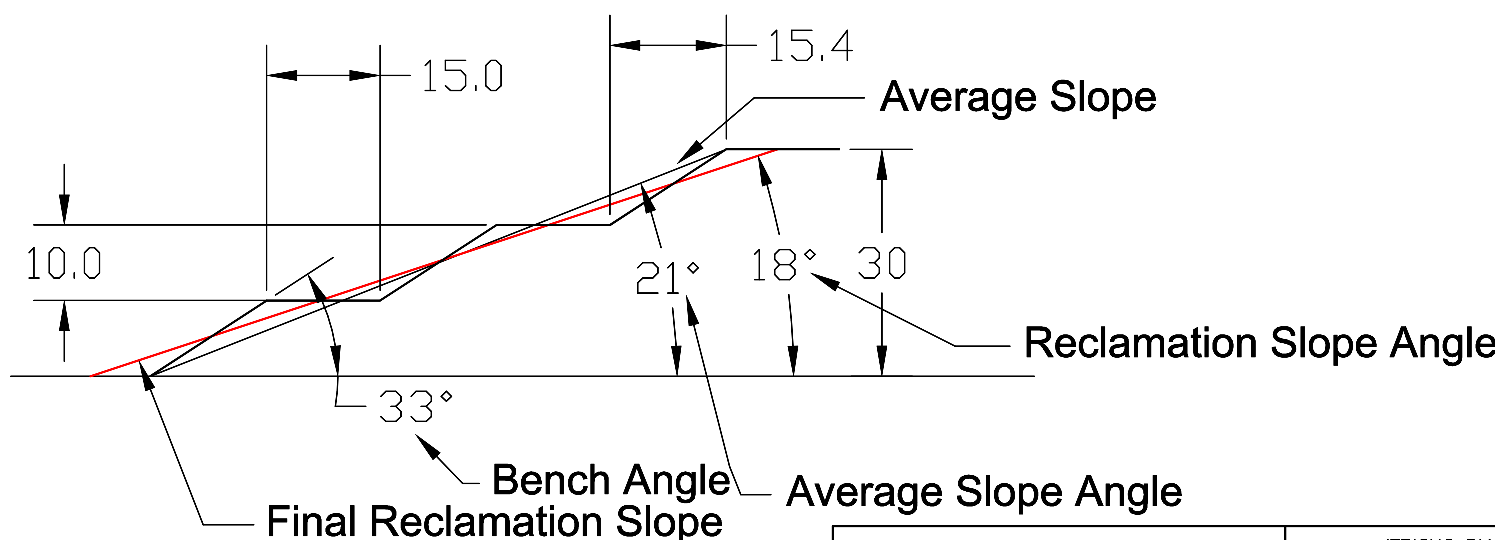
FIGURE 12.1
JERICHO PROJECT POWER USE



Cross Section Of Waste Dumps



Cross Section Of Coarse PK and Low Grade Stockpile

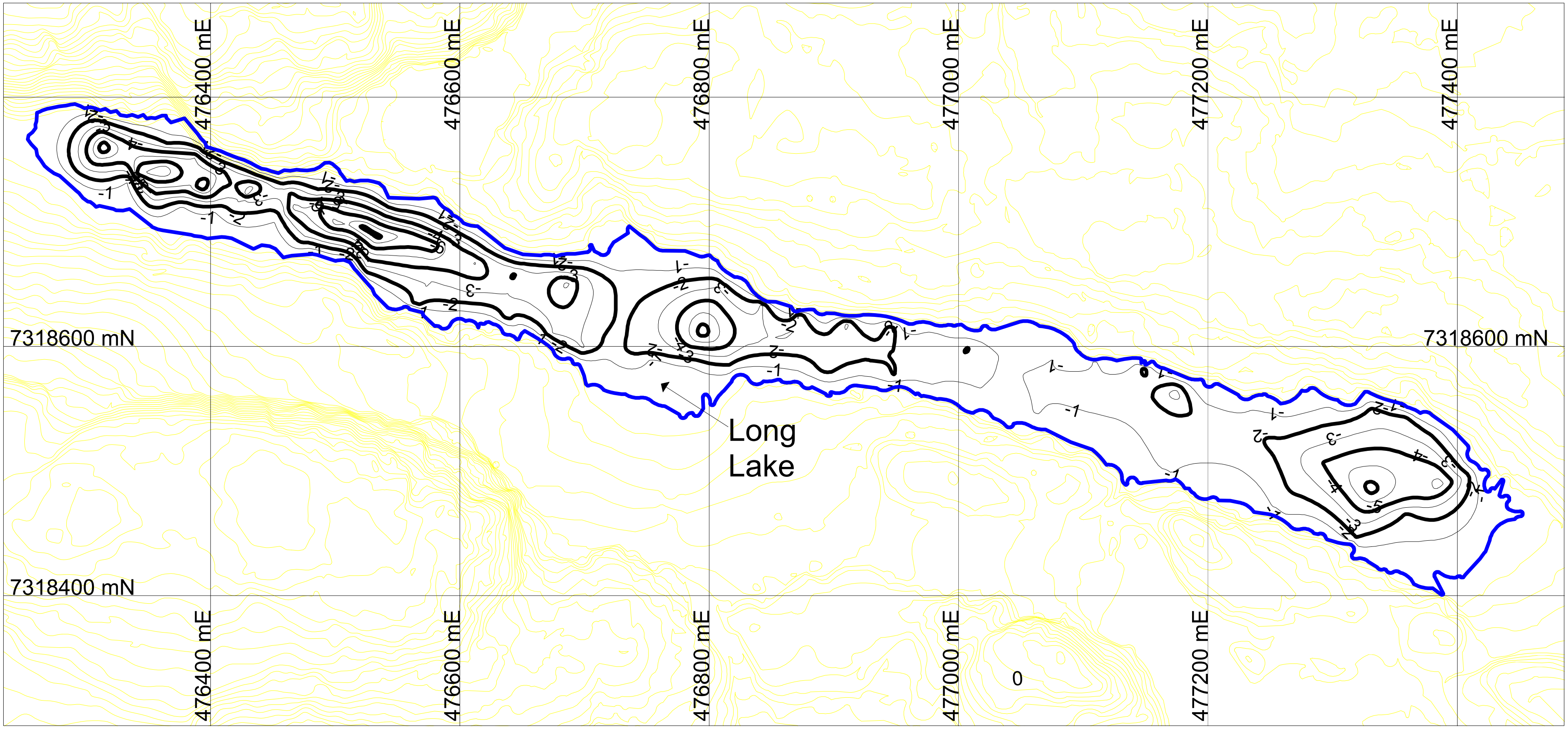


FILE REF: SECTIONSA_B.DWG

ALL ANGLES GIVEN IN DEGREES
ALL DISTANCES GIVEN IN METRES

SRK CONSULTING Consulting Engineers	JERICHO DIAMOND PROJECT
TAHERA CORPORATION	TYPICAL CROSS SECTION OF DUMP AND STOCKPILE FACES

PROJECT NO.	DATE	APPROVED	FIGURE
	JUNE 2000		14.1



LEGEND

Lakeshore

Bathymetry Contours

Even Contours

Odd Contours

N
W E
S

Tahera Corporation

Fig. 15.1

Date:30/06/00

Author: MJ

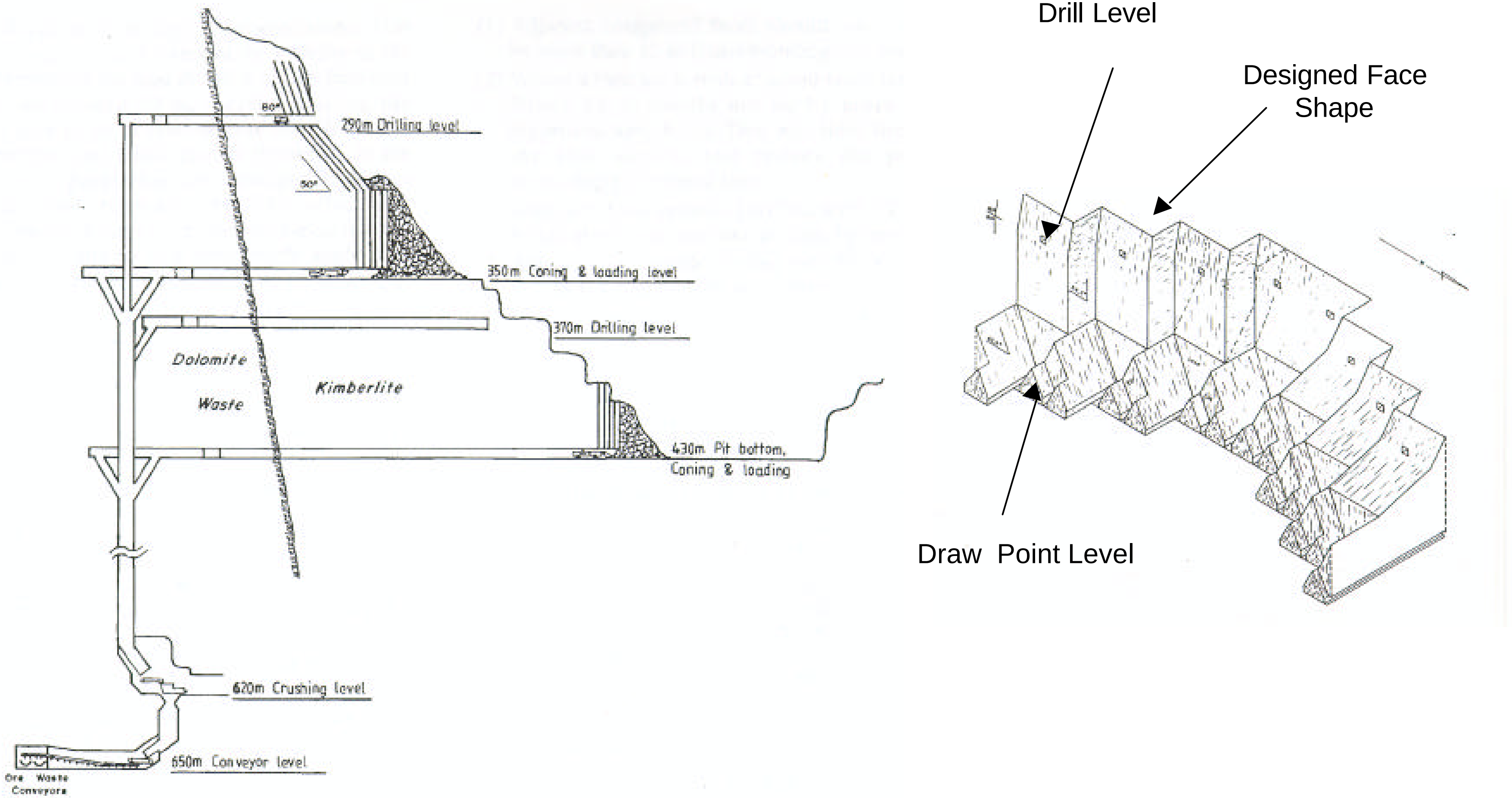
Office: Van.

Bathymetric Survey Of Long Lake

Scale: 1 : 5000

Projection: UTM Zone 12 (NAD 27)

0 metres 100



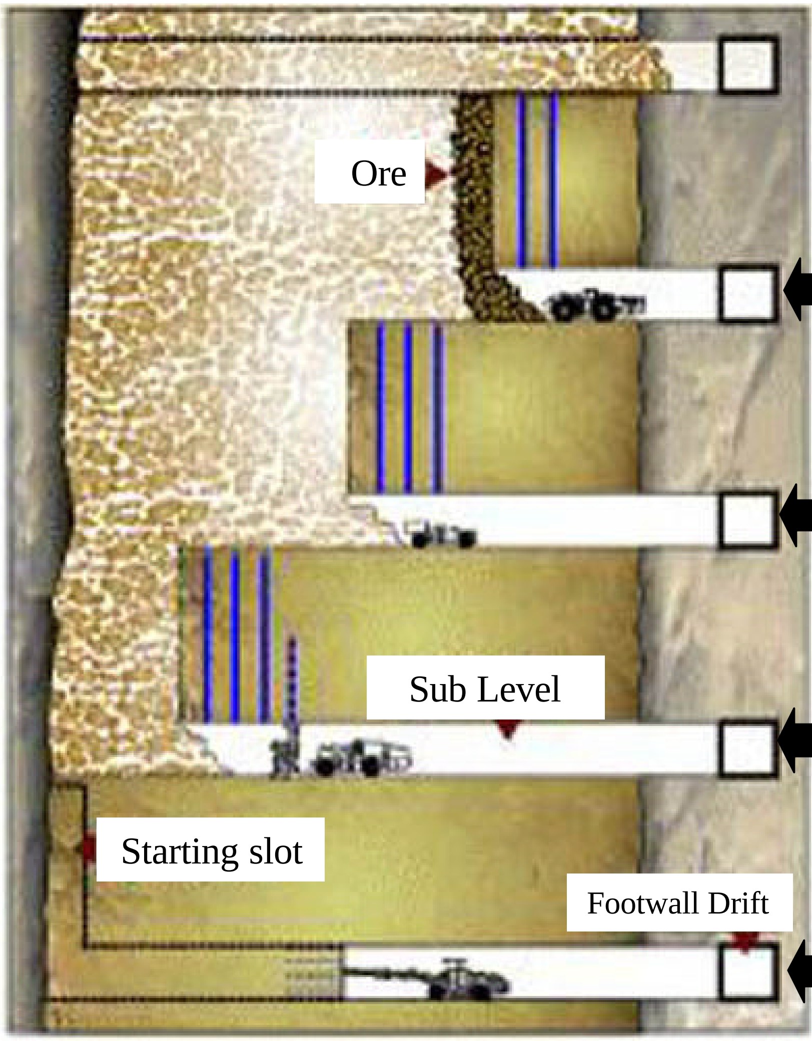
SRK Consulting
Engineers and Scientists

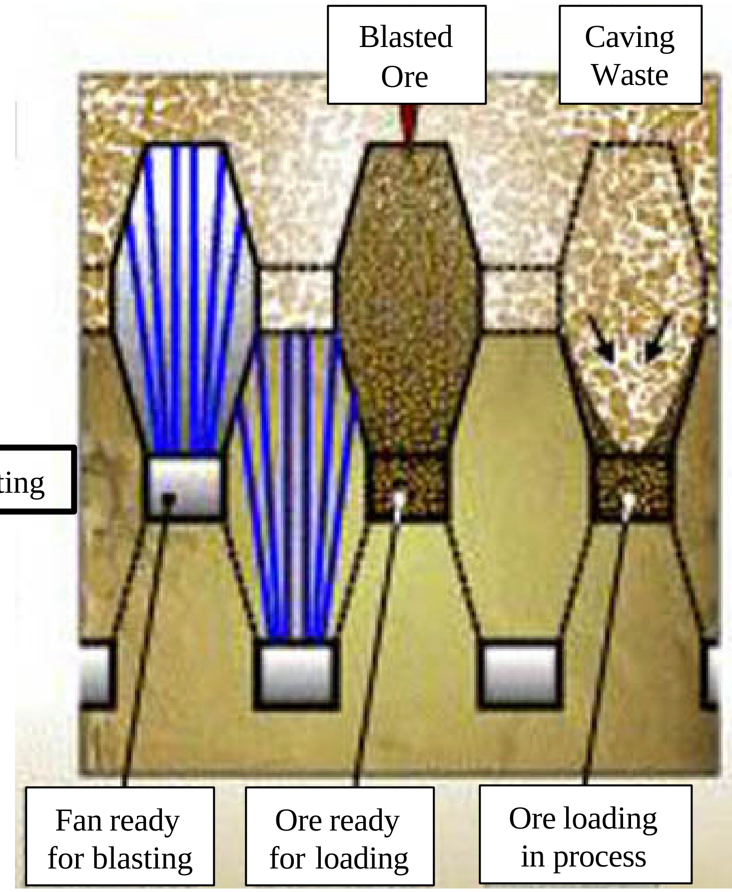
TAHERA CORPORATION

Jericho Diamond Project
Feasibility Study

Schematic of Open Benching at Finsch Mine

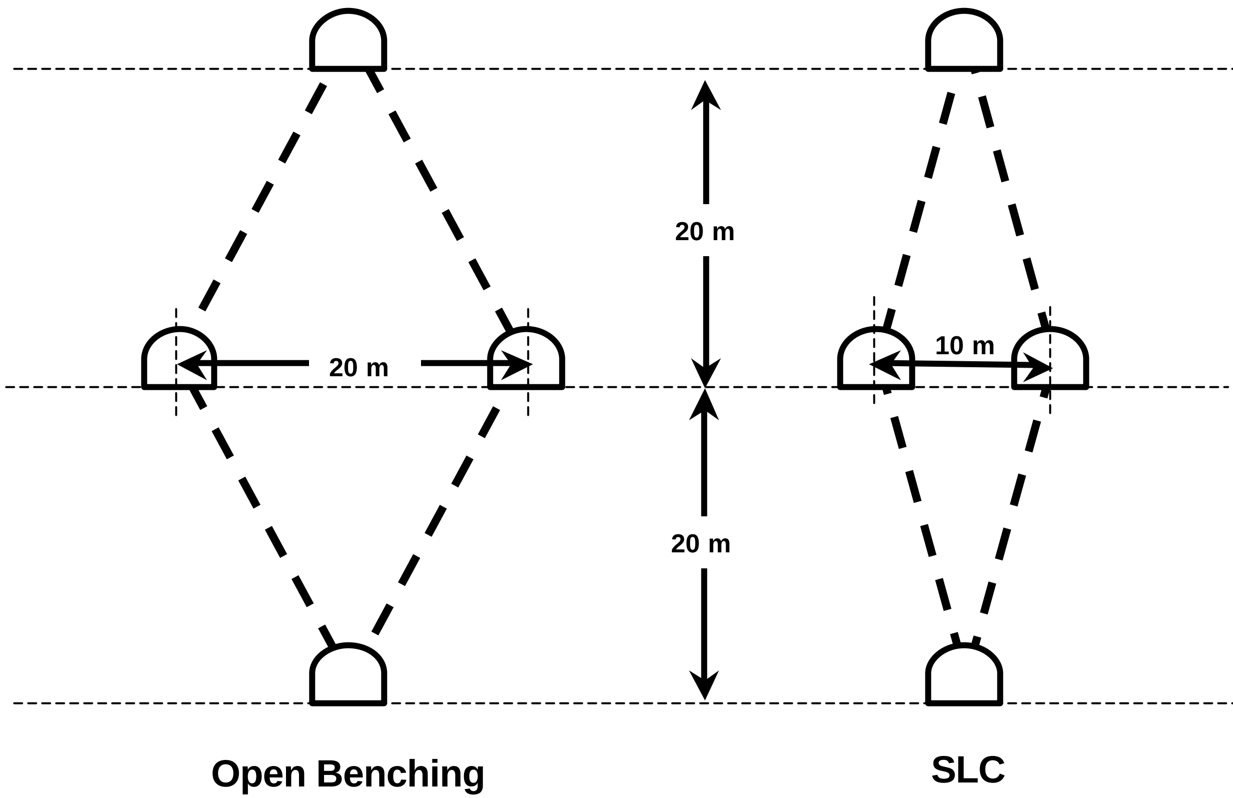
PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		**22.1**





SRK Consulting Engineers and Scientists	Jericho Diamond Project Feasibility Study
TAHERA CORPORATION	**Schematic of Sub-Level Caving**

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		**22.2**



SRK Consulting
Engineers and Scientists

TAHERA CORPORATION

Jericho Diamond Project
Feasibility Study

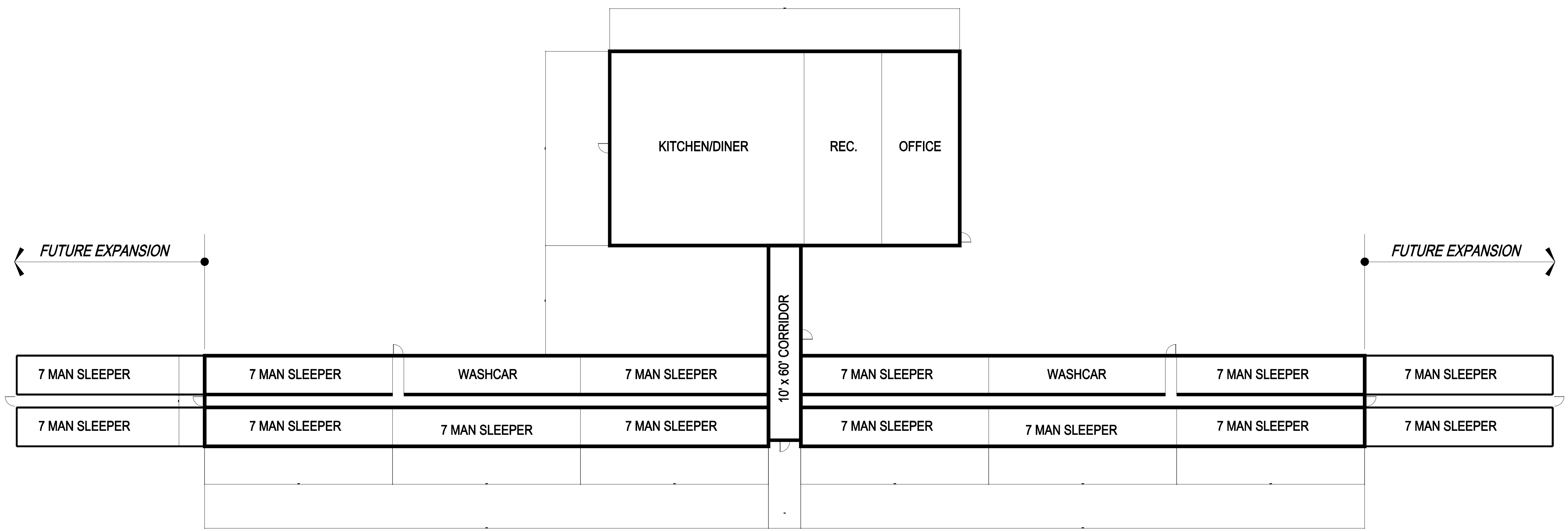
Comparison of Open Benching and SLC Layouts

PROJECT:	DATE:	APPROVED:	FIGURE::
2CT005.06	June 2000		22.3

ATTACHMENTS

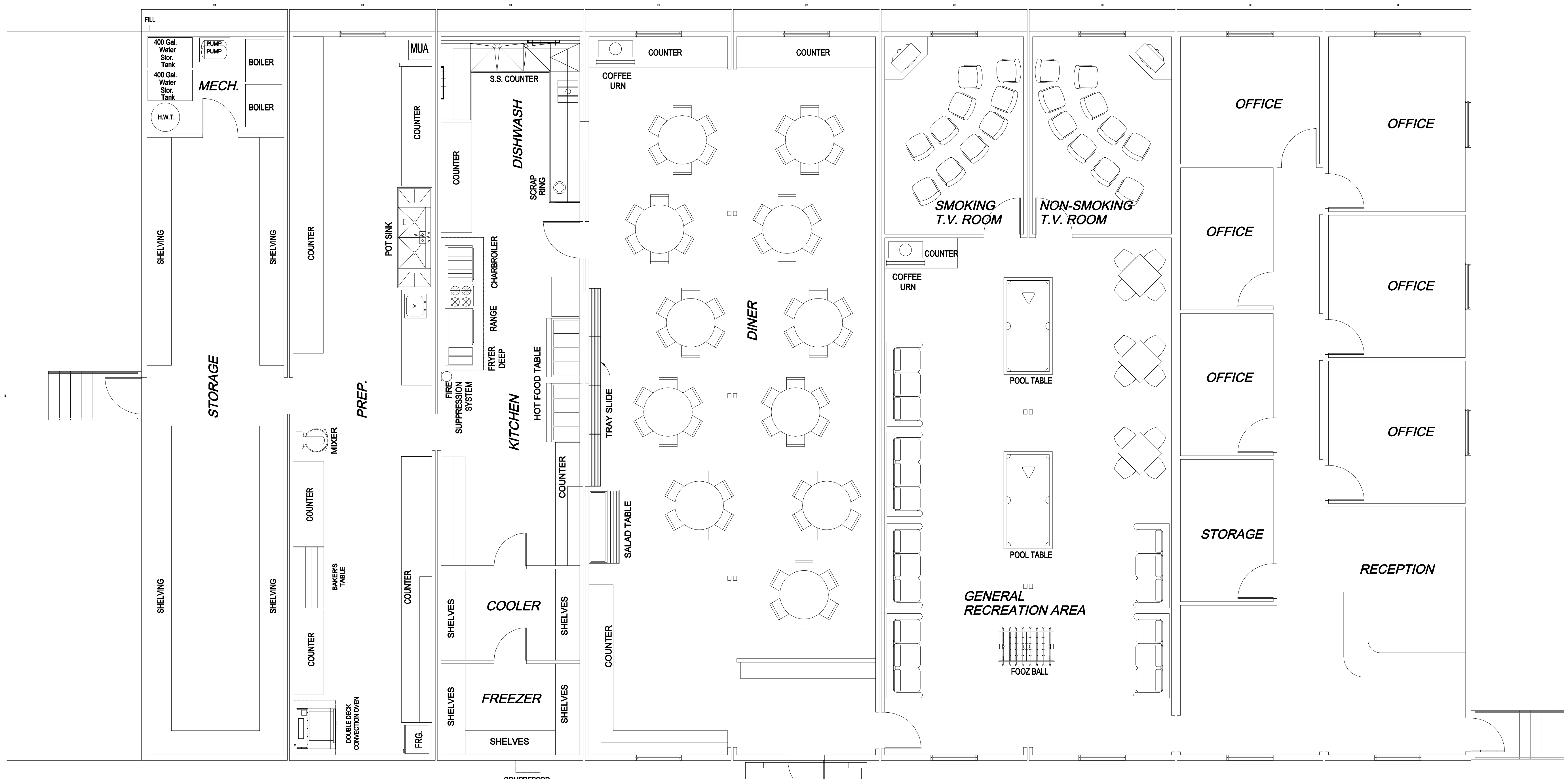
ATTACHMENT 9.1

CAMP LAYOUT ALTERNATIVES



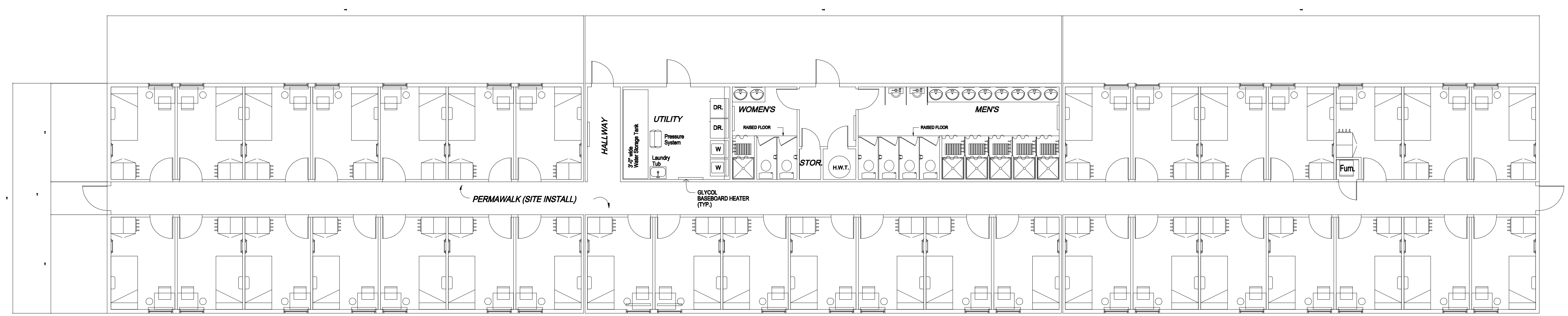
SITE PLAN

SHANCO CAMP SERVICES LTD.	by VIC scale	date 11-05-2000 sheet number F-110A
project	70 MAN BASE CAMP COMPLEX	
drawing	SITE PLAN	



COMPLEX FLOOR PLAN

SHANCO CAMP SERVICES LTD.	by VIC scale	date 11-05-2000 sheet number F-110C
project: 9 UNIT SIDE BY SIDE KIT./DINER/REC/OFFICE COMPLEX		
drawing: COMPLEX FLOOR PLAN		



COMPLEX FLOOR PLAN

(TYPICAL OF TWO - ONE MIRRORED IMAGE)

SHANCO CAMP SERVICES LTD.	by VIC	date 04-05-2000
	scale	sheet number F-110D

project

drawing

70 MAN CAMP COMPLEX
35 MAN SLEEPER WING
COMPLEX FLOOR PLAN

ATTACHMENT 21.1

NUNA LOGISTICS' RECLAMATION COST ESTIMATE

ADDENDUM TO NUNA LOGISTICS RECLAMATION COST ESTIMATE

The attached document contains dates that are no longer valid. However, the sequence of events discussed will still take place. The table below lists the old and new dates together with mining activity. New dates should be substituted for the dates in the report, i.e., 2005 = 2007 and 2011 = 2013. Costs are in 2000 dollars.

JERICHO DEVELOPMENT SCHEDULE			
Old Date	**New Date**	**Mining Year**	**Activity**
2002	2004	1	Construction – open pit, plant
2003	2005	2	Open pit mining
2004	2006	3	Open pit mining
2005	2007	4	Open pit mining
2006	2008	5	Underground Development & Mining
2007	2009	6	Underground Mining
2008	2010	7	Underground Mining
2009	2011	8	Processing only
2010	2012	9	Processing only
2011	2013	10	Closure

NUNA LOGISTICS

340 Park Place, 666 Burrard Street, Vancouver, BC V6C 2X8
Tel.: (604) 682-4667 Fax: (604) 682-4473

3010 Calgary Trail South, Edmonton, AB T6J 6V4 Tel.: (780) 434-9114 Fax (780) 434-7758

June 27, 2000

Delivered Via Telecopier: 604-904-9877

Bruce Ott, Ph.D.
Environmental Scientist
Tahera Corporation
1408 Crown Street
North Vancouver, BC
V7J 1G5

Dear Sirs:

RE: <u>REVISION #2 - RECLAMATION COST ESTIMATE - JERICHO PROJECT - CONTRACT MINING</u>

The cost of reclaiming the Jericho site is expected to be a follows:

YEAR 2005 – RECLAIM THE TWO WASTE DUMPS **$531,200**

This work would be incremental to the final year of open pit mining. Most of the work would occur during the summer season, with the final completion of Dump 2 to coincide with the finish of the open pit mining.

The work would require the following heavy equipment time:

D10 Dozer	617 Hours
992 Loader	282 Hours
777 Trucks	846 Hours
16G Grader	141 Hours

The work will require 140,000 litres of fuel.

YEAR 2011 – RECLAIM THE REMAINING SITE **$4,670,200**

This work would require a dedicated crew after the completion of all mining activities. We have included the costs for the use of Tahera's equipment and facilities assuming Nuna Logistics' rates for similar equipment and facilities. We have assumed that the full dismantling of facilities and equipment can be completed in three months.

Aside from the Tahera equipment, Nuna Logistics would mobilize a D10 dozer, two D300 trucks, 16G grader, and a large all-terrain crane.

The cost breakdown is as follows:

Earthworks	$698,068	
Mobilization and Demobilization	285,000	(Nuna Equipment)
Facilities Disassembly	2,173,712	
Administration	673,384	
Support Equipment	150,000	
Tails Dam Liner	150,000	
Support Facilities	540,000	
	$4,670,164	

The work will require the following heavy equipment time:

D10 Dozer	585 Hours
777 Trucks	1,283 Hours
992 loader	428 Hours
16G Grader	214 Hours

The work will require 900,000 litres of fuel.

YEAR 2012 – OUTBOUND FREIGHT **$928,800**

The outbound freight is based on 129 loads and was estimated based on a verbal conversation with Serge Benoit.

The above estimate summarizes a calculation sheet prepared by Serge Benoit of Tahera Corporation. The calculation sheet uses a revised quantities schedule prepared by Tahera Corporation and uses similar assumptions, productivities, etc. to the original calculation sheet prepared by Nuna Logistics.

We have made no allowance for contingency or risk items. The estimated costs are expressed in Year 2000 Canadian Dollars.

Yours truly,

NUNA LOGISTICS LIMITED

Courtland Smith, P.Eng.
Vice President
CS/jm

cc: Roy Meade, Deputy Chairman – Tahera Corporation
Serge Benoit, Plant Metallurgist – Tahera Corporation

NUNA LOGISTICS

Assumptions

Waste Dump 1 and 2
- Reclaimed at the end of open pit 2005
- Edges would be sloped, Upper bench would be cover with 0.3m of overburden

Tailings area
- A liner would be installed to cover the fine kimberlite rejects 1$/m
- East Cell would be reclaimed prior to the end of operation
- West Cell 2 would be reclaimed at the end of the processing
- 0.5 m of coarse tailing would be deposited on the liner
- 0.3 m of overburden would be deposited on the coarse kimberlite rejects

Pads
- Edges dozed by 5m, no cover on flat portion only scarified

Roads
- 5 m on shoulder dozed and covered, flat area scarified only

Coarse Kimberlite Rejects Dump
- Doze down slopes
- Cover slopes and upper bench with 0.3m of overburden

Low Grade Stockpile
- Doze down edges
- Cover upper bench

Reclaim Waste Dumps at the end of Open Pit using Mining Contractor Equipment

Contour					Total
Assume D10 can countour	1000	m2/hr			
Waste Dump #1					
Waste Dump #1 Slope Length	123	m			
Waste Dump #1 Perimeter	1993	m			
Waste Dump #1 Final Contour	245139	m2	to contour the side of the dumps		
Approximate time to contour	245	hrs			245
Waste Dump #2					
Waste Dump #2 Slope Length	123	m			
Waste Dump #2 Perimeter	1919	m			
Waste Dump #2 Final Contour	236037	m2	to contour the side of the dumps		
Approximate time to contour	236	hrs			236
Cover with Overburden					
Assume D10 Pushes	1000	m3/hr			
Waste Dump 1					
Cap over Waste dumps	0.3	m			
Last Bench Area Waste Dump 1	229013	m2			
To spread overburden over dump 1	68,704	m3	top of last bench only		
Approximate time to spread	69	hrs			69
Waste Dump 2					
Last Bench Area Waste Dump 2	222,430	m2			
To spread overburden over dump 2	66,729	m3	top of last bench only		
Approximate time to spread	67	hrs			67
					617
Load and Haul Overburden					
Cycle time 992 loader	4	min/load			
Cycle time 992 loader	12	load/hr	loader	1	992
Cycle 777 Highway truck	15	min/trip	Off higway truck	3	777
Cycle per 777	4	trip/hr			
Cycle 3 -777	12	trip/hr			
777 Capacity	50	m3/hr			
Capacity per hour	600	LCMs/hour			
Capacity per hour	480	ECMs/hour			
Qty Waste Dump 1	68,704				
Qty Waste Dump 2	66,729				
Hrs required	282				

Equipment	Fuel/hr litres	Total Hrs	Machine Cost /hr	Operator Cost/hr	Total	Total Fuel	
D10	84	617	250	56.83	189,194	51,795	
992	78	282	273	56.83	93,062	22,008	
777	70	846	211	53.7	224,057	59,252	
16G	44	141	120	56.23	24,848	6,204	
Sub Total					531,161	139,259	531,161

Rates for equipment all inclusive

Incremental operation 12 more people,

Reclaim Remaining

Disturbance Areas Contour		
Tails Dams		
Tails Dam Distance to Contour	1,000	m
Tails Dam Area to Contour	60	m2/m
Total Arrea to Contour	**60,000**	m2
Roads		
Road Distance to Contour	15,000	m
Road Area to Contour	10	m2/m
Total Road Area to Contour	**150,000**	m2
Pads		
Total Distance to Contour	5,000	m
Area to contour	5	m2/m
Total area to Contour	**25,000**	m2
Coarse tailings waste dump	**75,000**	m2
Low Grade Stockpile	**70,000**	m2
Total area to Contour	380,000	m2
D10 hrs to Contour at 1000m2/m	380	

Disturbance Areas to Cover				
Tails Dams Overburden				
Area to Cover (overburden)	150,000	m2		West & East Cell
Volume to Cover @ 0.3m	45,000	m3		
Tails Dams Coarse rejects				
Area to Cover (coarse kim. rejects)	150,000	m2		West Cell
Volume at 0.5m	75,000	m3		
Total volume tailings	**120,000**	m3	120,000	
Roads				
Distance 6 m roads	3,500	m		
Area to cover / meter	10	m2		
Area to cover	35,000	m2		
Volume of Overburden @ 0.3m	**10,500**	m3		
Distance 9 m roads	9,000	m		
Area to cover / meter	10	m2		
Area to cover	90,000	m2		
Volume of Overburden @ 0.3m	**27,000**	m3		
Distance 18 m roads	2,000	m		
Area to cover / meter	10	m2		
Area to cover	20,000	m2		
Volume of Overburden @ 0.3m	**6,000**	m3		
Total Roads	**43,500**	m3	43,500	
Pads				
Total area contoured	25,000	m2		
Volume @ 0.3m	**7,500**	m3		
Coarse tailings waste dump				
Total area to Cover	99,400	m2		
Volume @ 0.3m	**29,820**	m3		
Low Grade Stockpile		m2		
Total area to Cover	15,000	m2		
Volume @ 0.3m	**4,500**	m3		
Total volume to contour	**41,820**	m3	41,820	
Total Volume to Cover			205,320 m3	

Tahera Corporation

RECLAMATION COSTS

6/29/00

Equipment	Fuel/hr litres	Total Hrs	Machine Cost /hr	Operator Cost/hr	Total	Total Fuel	
D10	84	585	250	56.83	179,594	49,167	
992	78	428	273	56.83	141,085	33,365	
777	70	1283	211	53.7	339,676	89,828	
16G	44	214	120	56.23	37,713	9,416	
Sub Total					698,068	181,775	698,068

Mobilization and Demobilazation				
D10	30000	2	60000	
16G	15000	2	30000	
777	45000	3	135000	
992	60000	1	60000	
Sub Total			285000	285,000

Tails Dam Liner Cover			
Total Area	110,000	m2	
Cost per m2	1		
Cost for material	110000	$	
Cost to install	40000	$	
SubTotal	150,000 $		150,000

Disassembly of Facilities	#	months	Days/mont	Hrs/day	Rental $/hr	Operator $/hr	Total	
Crane		6	30.4	16	200	61.53	763249	
Welders & Rigers	4	6	30.4	12		63.1	552453	
Labourers	8	6	30.4	12		49	858010	
							2173712	2,173,712

Note Assembly costs = 3,425,000
Proposed di-assembly cost = 2,137,712 or 68% of assembly costs

Nuna Overheads	#	months	Days/mont	Hrs/day	Rental $/hr	Operator $/hr	Total	
Site Supervisors	1	6	30.4	12		78.76	172390	
Foreman	2	6	30.4	12		61.53	269354	
Safety	1	6	30.4	12		56.83	124390	
Administrator	1	6	30.4	12		49	107251	
							673384	673,384

Support Equipment	150000	150,000

Support Facilities		
Cost per month	90000	
Cost for 6 months	540000	540,000

Transport South Bound 2011		
Plant Building	51	
Plant	24	
Gen Sets	4	
Fuel Tanks	10	
Mobile Equipment	10	
Misc.	30	
# Loads	129	
Cost per load	7200	
Total Transport	928800	928,800

Total Reclamation 2010 4,670,164
Total Reclamation 2011 928,800

Total Reclamation 6,130,126