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Technical Report (NI 43-101) on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia

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Prepared for
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3.1 Executive Summary

The purpose of this Report is to present the results of the Definitive Feasibility Study (DFS) compiled by Aker Metals, a division of Aker Solutions Canada Inc. (Aker Solutions) and numerous consultants for Khan Resources Inc. (Khan). The DFS was commissioned by Khan to update and augment the Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA) Technical Reported dated September 27, 2007. This update provides an evaluation of the economics of establishing underground and open-pit mining and mineral processing facilities at the Dornod Project site in northeastern Mongolia. The DFS assumes a production rate of 1 225 000 t of ore per year (3500 t/d, 350 d/a).

The Dornod Project comprises several uranium deposits and some infrastructure. There are two deposits for which mineral resources and reserves have been estimated.

- An open-pit mine at the No. 2 Deposit. From 1988 to 1995, Priargunsky Industrial Mining and Chemical Enterprise (Priargunsky) extracted some 590 000 t of material at an average grade of 0.118% U_3O_8. Currently, the open pit is full of water.

- An underground uranium deposit (No. 7) which remains partially developed by two shafts and approximately 20 000 m of development drifts. Some of this development is also related to the nearby Nos. 4 and 5 Deposits. Currently, the underground workings are flooded.

Khan is a Canadian reporting issuer with a corporate office in Toronto. Khan, in joint venture with Priargunsky (a Russian government entity, based in Krasnokamensk, Eastern Siberia), and Mongol Erdene (a division of the Ministry of Energy, Geology and Mining of Mongolia), plans to bring the Dornod Project into production.

3.2 Conclusions and Recommendations

The DFS commissioned by Khan for the Dornod Project shows a positive economic outcome, including the following key results:

(a) Mineral Resources

At the 0.040% U_3O_8 cutoff grade and 5-m minimum vertical thickness of mineralization, the No. 7 Deposit contains 14.36 Mt of Indicated mineral resources at an average grade of 0.154% U_3O_8.

At the 0.025% U_3O_8 cutoff grade and 2-m minimum vertical thickness of mineralization, the No. 2 Deposit contains 10.95 Mt of Indicated mineral resources at an average grade of 0.065% U_3O_8 and 2.18 Mt of Inferred mineral resources at an average grade of 0.050% U_3O_8.

Several additional uranium deposits and showings have been discovered in the general Dornod area. In particular, the No. 5 Deposit is situated within the Additional Dornod Property (Mineral Licence 9282X). Two other deposits, Nos. 8 and 9, are situated outside the present property.

Past and recent exploration work has been carried out in a systematic manner and is well documented. These data are acceptable to estimate mineral resources.

(b) Mineral Reserves

The proven and probable reserve estimate for the No. 2 Deposit open-pit mine, at 0.028% U_3O_8 cutoff grade, is 7 407 000 t grading 0.074% U_3O_8. Mining dilution of 15% at a 0.018% U_3O_8 grade is included.

The proven and probable reserve estimate for the No. 7 Deposit at a 0.061% U_3O_8 cutoff is 10 634 000 t grading 0.174% U_3O_8. Underground mining recovery of 88% and dilution of 10% at 0% U_3O_8 grade is forecast.

(c) Mining

Underground and open-pit mines are planned, producing a total of approximately 1 225 000 t of ore per year, at a rate of 3500 t/d.

A total of 18.04 Mt of ore at an average grade of 0.133% U_3O_8 will be mined from the Nos. 7 and 2 Deposits over a period of 15 years.

(d) Processing

Uranium mineralization of the No. 7 Deposit is refractory. This is presumed to be due to the presence of brannerite, (a uranium titanate mineral), zircon, and the high carbonate content (4% to 7%) associated with the mineralization.

In order to liberate the uranium, it is necessary that a significant amount of silica in the ore be dissolved. The presence of the dissolved silica causes a gel to form and hinder the filtering of uranium. To overcome these problems, a Resin in Pulp (RIP) method of removing the uranium from the ore has been selected.

A metallurgical recovery of 84.86% has been used for No. 7 Deposit and 89.28% has been used for No. 2 Deposit.

Uranium mineralization of the No. 2 Deposit is free milling. This is based on previous testwork and results by Priargunsky.

A milling rate of 3500 t/d is planned for the combined production from the Nos. 7 and 2 Deposits.

(e) Water Management

There are no perennial rivers in the vicinity of the Project site. Fresh water requirements for the operation of the processing plant will have to be supplied either from the harvesting of surface water runoff (from occasional rainfall events or from seasonal thaw), or from groundwater. Surface water runoff will be highly intermittent and relatively unreliable; therefore, groundwater will have to be the primary source.

The water currently in the open pit represents a source of water which can be used for the start up of operations. The open pit can also be used as a source of water on an ongoing basis. Historical observations of pit water levels suggest that it may be possible to withdraw up to about 500 000 m^3 annually, providing that the pit water level is fully drawn down to stimulate groundwater inflow and to reduce evaporative losses. It has not been demonstrated that such large yields can be sustained on a year-to-year basis. The long-term sustainable yield from the open pit will depend on the size of the drawdown cone and the rate of recharge. Hydrogeologic studies should be undertaken as part of future studies to allow estimation of the long-term sustainable yield of the open pit.

It is anticipated that the Project will be operated such that it does not produce any liquid effluent. Inflows and outflows can be kept in balance by controlling the open-pit water level.

(f) Closure Plan

Golder Associates Ltd. (Golder) has prepared a conceptual closure plan to ensure long-term physical and chemical stability of the Project components remaining on-site at closure, to minimise long-term care and maintenance requirements, and to minimise the health and safety hazards posed by the site with regard to local residents and their livestock.

The principal closure measures that will be employed include:

- Construction of a boulder-berm around the open-pit rim and placement of a lockable swing gate at the entrance to the pit ramp
- Regrading of waste rock stockpile slopes to 2.5 H:1 V and placement of revegetated cover over the dump footprints
- Placement of a cover on the surface of the Residue Management Area (RMA) to provide clean surface runoff
- Decommissioning and removal of Water Collection Pond and Polishing Pond
- Caping of all shafts and ventilation raises and the backfilling of the production ramp and portal, and the return air raises
- Decommissioning and demolition / removal of the processing facility and other surface infrastructure and equipment.

Long-term care and maintenance will consist of the following actions.

- Local labour will be employed to ensure site security is maintained during closure implementation
- Periodic site inspections and maintenance will be carried out for the RMA and drainage work in the long term.
- Quarterly surface water quality sampling will be performed during Years 1 to 5 at the open-pit lake, the RMA Pond, and at locations upstream and downstream until stable trends are established; sampling will occur annually thereafter

- Quarterly groundwater quality sampling will be performed during Years 1 to 5 at one location downstream of the RMA, two locations upstream of the RMA, one location upstream of the Waste Rock Storage Facilities, and one location downstream of the Project site, until settable trends are established, reducing to annually thereafter.

3.3 Economic Analysis

A financial analysis has been completed for the Project. This evaluation has been done from the perspective of the joint venture.

(a) Capital Cost

The capital cost for mining and surface facilities as described in this Report is USD 332,786,000 in fourth quarter 2008 United States dollars, with no allowance for escalation, interest or financing during construction.

The direct costs (Items D0 to D9, Table 3-1) are all the costs associated with permanent facilities. This includes equipment and material costs, as well as construction and installation costs.

The indirect costs (Items IA to IQ, Table 3-1) cover all the costs associated with temporary construction facilities and services, construction support, freight, Vendor representatives, spare parts, initial fills and inventory, Owner's costs, Engineering, Procurement and Construction Management (EPCM), commissioning and start up.

The contingency allowance of 11.4% of process plant and infrastructure direct and indirect costs has been included in the estimate. P&E, based on their experience, has allowed a 15% contingency on the mining portion. The overall contingency, therefore, is 13.3% of total direct and indirect costs, exclusive of Owner's costs.

The capital cost estimate is presented in Table 3-1.

(b) Operating Cost Estimates

Operating cost estimates reflect fourth quarter 2008 US dollars. The DFS operating cost estimates are prepared by major area – Mining, Plant, General and Administration, and consider the mine plan and processing schedule.

Life-of-mine total operating costs are presented in Table 3-2. Note that Years 2009 to 2011 are considered as preproduction and their costs are included in mine capital cost estimates.

Table 3-1 - Capital Cost Estimate

Aker Solutions	Dornod Project	EST6-04/20/2009-11:24:01
1838 00	Khan Resources	PAGE 1
	COST IN USD	Final - March 05, 2009

	DESCRIPTION	HOURS	LABOR	EQP USAGE	MATERIAL	SUBCONTR	EQUIPMENT	TOTAL
D0	Mining	0	0	0	0	66,752,600	0	66,752,600
D1	Earthworks	327067	4,716,107	579,081	1,462,257	16,208,405	0	22,965,851
D2	Concrete	311860	4,585,069	3,113,608	6,295,400	0	0	13,994,077
D3	Structural Steel	173110	2,850,089	2,685,044	7,056,601	0	0	12,591,735
D4	Architectural	182548	2,667,975	381,531	2,164,376	2,270,000	0	7,483,883
D5	Mechanical	381620	5,888,396	1,908,100	1,250,000	600,000	38,609,684	48,256,180
D6	Electrical	200846	3,054,873	401,692	4,955,850	581,500	8,711,550	17,705,466
D7	Instrumentation	73989	1,125,386	147,979	794,058	0	2,048,101	4,115,525
D8	Piping	212334	3,516,272	1,061,703	4,100,192	0	0	8,678,167
D9	Plant Mobile Equipment	0	0	0	0	0	2,892,000	2,892,000
IA	Temporary Building & Facilities	0	0	0	0	2,500,000	0	2,500,000
IB	Temporary Construction Utility Services	0	0	0	0	884,000	0	884,000
IC	Winter Work and Lost Productivity	0	0	0	0	387,000	0	387,000
ID	Construction Site Support & Operations	0	0	0	0	2,972,000	0	2,972,000
IE	Construction Camp and Catering	0	0	0	0	6,225,560	0	6,225,560
IF	Power During Construction	0	0	0	0	8,160,000	0	8,160,000
IG	Spare Parts	0	0	0	0	2,410,389	0	2,410,389
IH	Initial Fills	0	0	0	0	3,816,836	0	3,816,836
IJ	Freight and Insurance	0	0	0	0	5,431,361	0	5,431,361
IK	Vendor Representative	0	0	0	0	964,000	0	964,000
IL	Owner Costs	0	0	0	0	12,420,000	0	12,420,000
IN	EPCM	0	0	0	0	38,747,944	0	38,747,944
IP	Commisioning and Startup	0	0	0	0	4,714,316	0	4,714,316
IQ	Contingency	0	0	0	0	37,717,106	0	37,717,106
REPORT TOTALS		1863378	28,404,169	10,278,741	28,078,737	213,763,017	52,261,335	332,786,000

Table 3-2
Life-of-Mine Operating Costs

Year		Tonne Milled (x '000)	Mining (USD)	Plant (USD)	G&A (USD)	Total (USD)	Cost/Tonne Milled (USD)
2009							
2010							
2011							
2012	1	854	32,976,454	20,443,546	7,040,000	60,460,000	70.83
2013	2	1,225	44,664,514	31,246,486	7,040,000	82,951,000	67.72
2014	3	1,225	43,142,514	31,246,486	7,040,000	81,429,000	66.47
2015	4	1,225	44,169,514	31,246,486	7,040,000	82,456,000	67.31
2016	5	1,225	47,345,714	30,880,286	6,300,000	84,526,000	69.00
2017	6	1,228	46,680,714	30,880,286	6,300,000	83,861,000	68.29
2018	7	1,225	44,334,714	30,880,286	6,160,000	81,375,000	66.43
2019	8	1,225	50,113,714	30,880,286	6,160,000	87,154,000	71.15
2020	9	1,225	52,096,714	30,880,286	6,160,000	89,137,000	72.76
2021	10	1,225	31,863,386	22,334,614	4,977,000	59,175,000	48.31
2022	11	1,225	28,903,738	20,930,262	4,977,000	54,811,000	44.74
2023	12	1,225	29,184,738	20,930,262	4,977,000	55,092,000	44.97
2024	13	1,225	27,133,738	20,930,262	4,977,000	53,041,000	43.30
2025	14	1,225	29,708,738	20,930,262	4,977,000	55,616,000	45.40
2026	15	1,262	20,756,000	14,626,000	4,977,000	40,359,000	31.98
TOTAL		**18,044**	**573,074,904**	**389,266,096**	**89,102,000**	**1,051,443,000**	**58.26**
Cost/lb U3O8		45,279,000	12.71	8.60	1.97	23.22	
Cost/Tonne Milled			31.76	21.56	4.94	58.26	

Note that the above amounts do not include VAT or the interest costs associated with the leasing of mining equipment. The interest on the leased equipment is shown in the Project Cash Flow, Table 20-34.

(c) Manpower

A total of 933 people will be employed during an average year. A breakdown of the workforce is presented in Table 3-3.

Table 3-3
Total Manpower – Average Year

	Staff	Hourly	Total
Mine	46	665	711
Mill	22	127	144
G&A	27	36	63
Camp	8	2	10
TOTAL	**103**	**830**	**933**

The percentage of expatriates to total labour complement in the average years of the mine life is 2.5%.

(d) Financial Analysis

A financial model for the underground and open-pit mine with an annual production rate of 1 225 000 t was prepared. Key production and financial parameters are summarised in Table 3-4.

(e) General Parameters

The financial analysis model covers the time span from Year -3 through Year +16. The preproduction years are Years -3, -2 and -1. Production years are from +1 to +16. Underground mining is from Years +1 to +9, whilst open-pit mining will commence from Years +10 to +16. Year 16 is allowed for Project closure.

The mill feed rate from the mine is 1 225 000 t/a, with first year of production at 854 000 t, thus allowing the mill to ramp up to full production. The total ore mined over the life of mine is 10 634 000 t. The average head grade over the life of mine is 0.133% U_3O_8. The average head grade for underground mining is 0.174% and for the open pit 0.074%.

The process recovery for uranium (U_3O_8) is 84.5% for the underground and 89.28% for the open pit. Over the life of mine, the total production of U_3O_8 is 20 538 t (45 279 000 lb).

Product pricing is based on the recommendation of Khan and is assumed to be on an FOB mine site basis.

Table 3-4
Financial and Production Data

Annual mine throughput	1 225 000 t
Mine life	15 years
Average grade	0.133% U_3O_8
Recovered U_3O_8	45,279,000 lb
Average value	USD 65/lb

Tables 3-5 and 3-6 summarise the financial analysis model. NPV is calculated on end-year basis.

Table 3-5
Financial Data
(USD '000)

	TOTAL
Revenue	2,943,111
Operating Costs, Mine Site	1,051,443
Other Operating Costs including Royalties	158,109
Total Operating Costs	1,175,028
Total Initial Capital Investment Costs	371,174[1]
Nett Initial Capital Investment Costs	332,786
Sustaining Capital Investment Costs	154,706
Pretax Cumulative Cash flow	1,242,203
Taxes, Income	317,273
After Tax Cumulative Cash flow	924,929

[1]Initial capital investment plus VAT.

Table 3-6
IRR and NPV Values
(USD '000)

	End of Year	
	Pre-tax	**After Tax**
IRR	36.4%	29.1%
NPV @ 0%	1,242,203	924,929
NPV @ 10%	406,827	275,993
Payback Period, Years	1.9	2.3

The Project is subject to graduated levels of taxation and flat rate royalty based on gross revenue. Income tax is payable at a rate of 10% for initial income of 3,000,000,000 tugriks (USD 1.94 million) and below and at a rate of 25% for income over the 3,000,000,000 tugriks threshold. Royalty is payable at 5% of gross revenue.

The close-out cost is estimated at USD 37.4 million. USD 1.4 million is for close-out engineering and is applied in Year +15, whilst the close-out cost is applied in Year +16.

Chart 3.1 revolves around the after tax NPV @ 10% of USD 275,993,000 calculated on the end of year basis.

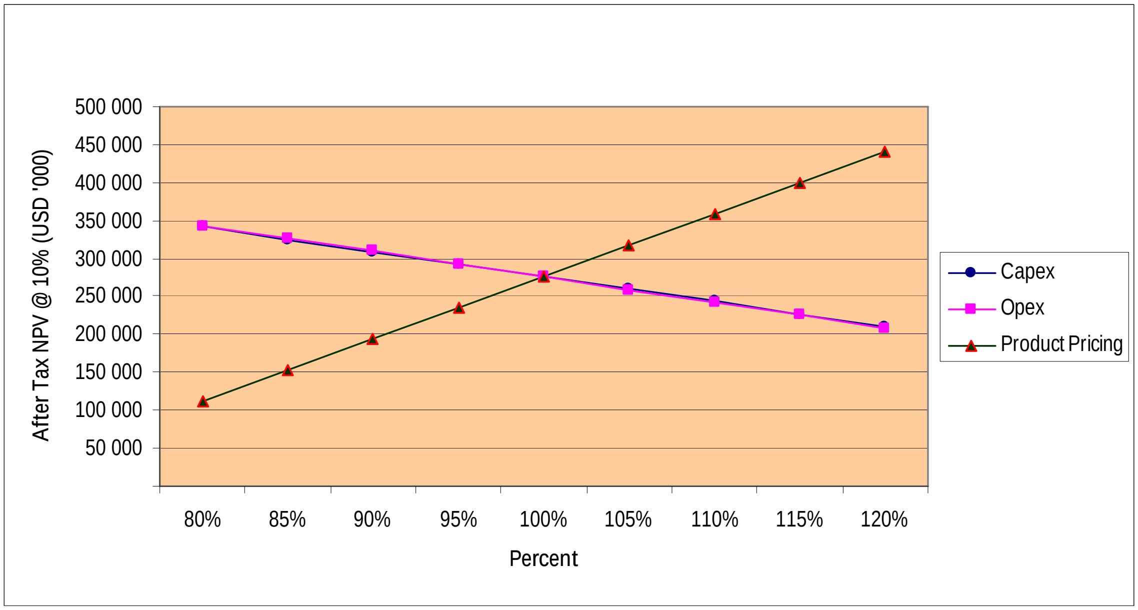


Chart 3.1
Sensitivity Graph

(f) Project Implementation

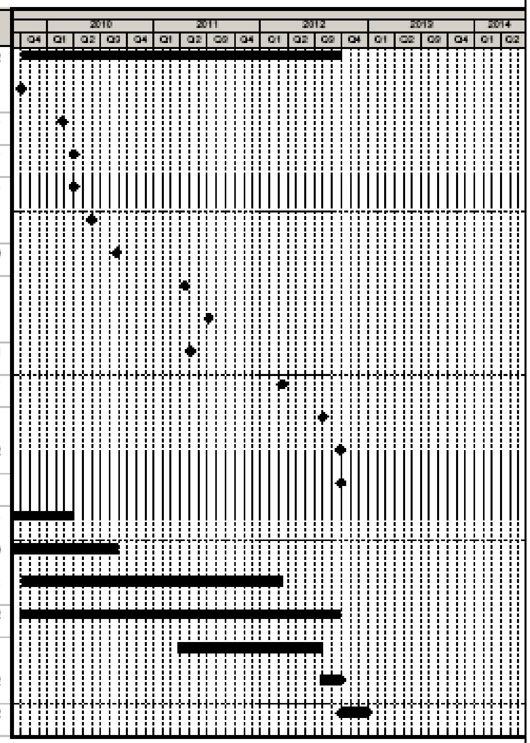
The Project Execution Plan, Figure 3.1 outlines the summary of major activities leading to successful completion of the Project. The major activities are grouped into major categories: Agreements, Environmental Assessment, Engineering, Mining, Construction, Commissioning and Ramp up.

The scheduled start of the EPCM activities is October 9, 2009, dependent on receiving Government of Mongolia approval for the Project. The schedule identifies activities occurring during the first half of 2010 necessary to maintain the planned completion date.

The overall duration from March 2010 to achieving full production is 33 months. From the start of detail engineering to completion of precommissioning is 28 months. The construction duration of the surface facilities is 18 months. A 3-month duration for production ramp-up is planned.

Khan Resources Inc

Activity ID	Activity Name	Original Duration	Start	Finish
MILESTONES		781	02-Oct-09	30-Sep-12
M0020	Award Basic Engineering Contract	0		02-Oct-09
M0030	Long Lead Equipment Orders Placed	0		18-Feb-10
M0040	Complete Basic Engineering	0		31-Mar-10
M0050	Award Detailed Engineering	0		30-Mar-10
M0060	Environmental Assessment Complete	0		01-Jun-10
M0070	Land Use Permit & Water Licenses Granted	0		24-Aug-10
M0080	Start Construction	0	14-Apr-11	
M0090	Camp Construction Complete	0		07-Jul-11
M0100	Complete Detailed Engineering	0		05-May-11
M0110	All Long Lead Equipment Orders Received on Site	0		15-Mar-12
M0120	Mechanical Completion	0		31-Jul-12
M0130	Commissioning Complete	0		30-Sep-12
M0140	Mine Production	0	30-Sep-12	
AGREEMENTS		176	27-Jul-09	30-Mar-10
ENVIRONMETAL ASSESSMENT		409	29-Jan-09	24-Aug-10
ENGINEERING		639	02-Oct-09	15-Mar-12
MINING		782	01-Oct-09	28-Sep-12
CONSTRUCTION		353	24-Mar-11	31-Jul-12
COMMISSIONING		50	22-Jul-12	30-Sep-12
RAMP UP		65	01-Oct-12	31-Dec-12



Milestone
Summary

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Dornod Uranium Project

Figure 3.1
Project Execution Schedule

(g) Key Project Dates

The following activity key dates are identified.

- October 2, 2009 — Award Basic Engineering Contract
- February 18, 2010 — Long-Lead Equipment Orders Placed
- March 30, 2010 — Complete Basic Engineering and Award Detailed Engineering
- August 24, 2010 — Land Use Permit and Water Licenses Granted
- April 14, 2011 — Start of Construction
- May 5, 2011 — Complete Detailed Engineering
- July 7, 2011 — Construction Camp Completed
- March 15, 2012 — All long-lead equipment orders received on-site
- July 31, 2012 — Mechanical Completion
- September 30, 2012 — Commissioning Complete

3.4 Technical Summary

3.4.1 Property Location and Description

The Dornod Project is located in northeastern Mongolia, approximately 125-km north of Choibalsan, capital of the Dornod Aimag (province). The population of Choibalsan is about 15,000, and it is situated along a major east-west road connecting the town with Ulaanbaatar, the capital of Mongolia, some 650 km to the west. The abandoned settlement of Mardai, built for Russian mineral exploration crews, is 14-km west of the Project.

3.4.2 Land Tenure and Ownership

The Dornod Property consists of two mineral licences, a Mining Licence (237A, originally U-27) and an Exploration Licence (9282X). Mining Licence 237A, known as the Main Dornod Property, was granted by the Office of Geological and Mining Cadastre (OGMC), of the Minerals Resources and Petroleum (MRPAM) Authority of Mongolia, to Central Asian Uranium Corporation (CAUC), a limited liability company organised under the laws of Mongolia. Khan, through a subsidiary corporation, holds 58% of the issued and outstanding common shares of CAUC.

An application to convert the exploration license to a mining license was submitted in September 2007. The application included the August 2007 Pre-Feasibility Study. Exploration License 9282X, known as the additional Dornod Property, has an area of 243 ha and is contiguous with the Main Dornod Property. It is registered through a wholly owned subsidiary of the Corporation, and was renewed for a 3-yr period in February 2008. The corporation is currently taking all necessary steps to convert the exploration license into a mining license, in accordance with the Revised Minerals Law of Mongolia (RMLM). To this end, the Corporation has recently submitted the reserve calculation and environmental impact assessment for the Additional Dornod Property, prepared in accordance with Mongolian standards and requirements. These are necessary preconditions in the process of converting an exploration license to a mining license in accordance with the RMLM.

3.4.3 Permitting

The Mineral Resources and Petroleum Authority of Mongolia (MRPAM) is the authority that oversees mining and exploration licensing in Mongolia. To change a license from exploration to mining, the company must submit:

- Mineral resource / reserve approved by the Minerals Council
- Feasibility study approved by the Mining Department of MRPAM
- Mongolian Environmental Impact Assessment (EIA) approved by the Ministry of Nature, Environment and Tourism.

Khan expects that the DFS will satisfy the requirements for a feasibility study as it includes a Life of Mine Plan.

To date, all permits and licenses are in place for the program presently underway. All licenses for the properties are in good standing.

The Project status and schedule is dependent on the company obtaining an investment agreement from the Mongolian Government. Khan expects that government review will commence in the third quarter of 2009 and this process will be finished and approved by the end of the fourth quarter of 2009. It is not known at this time what impact these negotiations will have on the existing ownership structure.

3.4.4 Access

Access to the Dornod Property is by paved road, about 100-km east from Ulaanbaatar to the coal mining town of Baganoor, then 550-km east by dirt road from Baganoor to Choibalsan in northeastern Mongolia and then about 125-km north by dirt road from Choibalsan to Mardai. The main access road to the mine, from the town of Choibalsan, is presently an unimproved dirt road and will have to be graded and maintained to provide year-round access.

3.4.5 Infrastructure

Infrastructure near the Project is limited. Power is generated at Choibalsan. A power line is presently under construction and is scheduled to be completed in May 2009. Telephone service is available at the site. Water is available from wells near the property. Some mining equipment and personnel are available at Choibalsan, Ulaanbaatar, and in northern Mongolia, where a few open-pit gold deposits are being developed.

3.4.6 History

Historic mining and prospecting activities in the Dornod Uranium District of northeastern Mongolia, which hosts the Dornod deposits, date back to the 1940s. Early prospecting work led to the discovery of the Dornod No. 2 uranium deposit and production started from an open pit in 1988. The area is host to numerous undeveloped uranium occurrences. From 1988 to 1995, some 590 000 t of material at an average grade of 0.118% U_3O_8 were mined from the No. 2 Deposit of the Dornod site. The advent of Perestroika in 1985 and the dissolution of the Soviet Union in 1991 led to cessation of mining activity.

In 1995, Priargunsky - on behalf of World Wide Minerals Ltd. (World Wide), a predecessor company to Khan - commenced stripping and mining operations at the No. 2 Deposit as an open-pit mine. Due to low uranium prices, however, the mine was shut down in 1995. Until 2005, the Project had been maintained on a care and maintenance basis. In early 2005, Khan became operator and began a confirmation drilling program on the areas of the Nos. 2 and 7 Deposits. Results of this program confirmed earlier Priargunsky results and established the continuity of uranium mineralization at the two deposits. Khan commissioned a Scoping Study on Dornod in 2005, followed by a PFS in 2006, and a DFS in 2008 which is the subject of this report.

3.4.7 Geology

Mongolia is within the Central Asian branch of the Ural-Mongolian Mobile Belt. The Main Mongolian Lineament, an arcuate series of deep-seated faults that extend generally east-west through the mid-section of the country, divides Mongolia into Northern and Southern Megablocks. The Dornod uranium district is within the North Choibalsan mineral region in extreme northeast Mongolia, in the Northern Megablock at the eastern end of the Central Mongolian Fold System.

Although uranium mineralization is common throughout the Dornod Complex, economic concentrations of uranium mineralization occur in a narrow stratigraphic interval in the lower part of the Complex. Mineralization is most extensive in horizons of porous sedimentary and volcanic rocks usually enriched with organic or sulphide minerals. Deposits are controlled by major zones of steeply dipping fractures of the northerly and northeasterly faults and their junctures with northwesterly faults.

The area of the Dornod Property is underlain by Jurassic volcanic and sedimentary rocks. The volcanic rocks are comprised of amygdaloidal basalt, andesite, ignimbrite, rhyolite and tuff. The sedimentary rocks are predominantly sandstone and conglomerate containing interbed carbonaceous partings.

Uranium mineralization in the Dornod district is found at depths of 30 m to 700 m and is concentrated within a 30-km^2 area. Thirteen deposits have been identified in the Dornod district, of which five have been explored in detail. The No. 7 Deposit, which is the largest, has been partially developed for underground exploration. The No. 2 Deposit, which is closer to surface, has been partially mined by open pit methods.

Uranium mineralization occurs as pitchblende-coffinite assemblages associated with carbonaceous partings and fragments in areas of structural preparation. The uranium mineralization occurs as "blanket-like" horizons from less than 1-m thick to greater than 30-m thick within the volcano-sedimentary succession at depths from 30 m to greater than 450 m below surface. A number of uranium deposits and target areas have been outlined in the Dornod area by systematic exploration work.

The No. 7 Deposit is situated at the northern end of the Dornod uranium district and occupies the southern half of the area covered by Mining Licence 237A. The Deposit is situated approximately 1-km south of the No. 2 Deposit. The No. 7 Deposit comprises a number of separate, flat-lying uraniferous horizons spread over an area measuring 1000 m by 500 m. The most continuous zone is a 30- to 40-m-thick tabular body of high-grade uranium mineralization occurring at vertical depths between 410 and 450 m below surface.

The No. 2 Deposit comprises a number of separate uraniferous horizons spread over an area measuring approximately 1800 m by 1500 m. There are at least five horizons of sedimentary rocks hosting uranium mineralization, which are interlayered with felsic to intermediate volcanic rocks. The most continuous zone (Layer 3) is a 6- to 10-m-thick layer of low-grade uranium mineralization which is stratabound and defines the broad southwest trending synform in the area. This layer occurs at vertical depths between 75 and 225 m below surface, and was the target of most past mining activity.

Russian exploration of the No. 7 Deposit included 123 surface diamond drill holes, 143 underground diamond drill holes and approximately 20 000 m of underground development including drifts, cross-cuts, and three shafts, which extend to the No. 5 Deposit area. Russian exploration of the No. 2 Deposit included 450 surface diamond drill holes.

From August 2005 to April 2007, Khan completed a program of confirmation drilling in both deposits, totalling 5885 m in 23 vertical diamond drill holes.

In 2007, Khan continued to test the area between the Nos. 2 and 7 Deposits, as well as the area southeast of the No. 2 open pit, by drilling. In total, some 1987 m of drilling was completed in eight diamond drill holes.

In late 2007, Khan completed two large diameter diamond drill holes and sampled the central part of the No. 7 Deposit for metallurgical testwork.

3.4.8 Mineral Resources

Scott Wilson RPA updated the mineral resources of the Nos. 7 and 2 Deposits, based on a new digital database of previous results, and additional confirmation drilling results. The Scott Wilson RPA mineral resource estimate is in accordance with the Mineral Resource / Reserve Classification as recommended by the CIM Committee on Mineral Resources / Reserves. The mineral resources are presented in Table 3-7.

Table 3-7
Mineral Resource Estimate

Location	**Category**	**Tonnes** (million)	**% U_3O_8**	**lbs U_3O_8 (million)**
No. 7 Deposit	Indicated	14.36	0.154	48.6
No. 2 Deposit	Indicated	10.95	0.065	15.7
TOTAL	**Indicated**	**25.31**	**0.116**	**64.3**
No. 2 Deposit	Inferred	2.18	0.050	2.4

Notes:

1. CIM definitions were followed for mineral resources.
2. Mineral resources were estimated using a U_3O_8 price of USD 55/lb.
3. Mineral resources were estimated using a cutoff grade of 0.04% U_3O_8 for No. 7 Deposit, and 0.025% U_3O_8 for No. 2 Deposit.
4. No. 7 Deposit was modeled at a minimum of 5- m-vertical thickness, No. 2 Deposit was modeled at a minimum of 2-m-vertical thickness.
5. Mineral resources are inclusive of, not in addition to, mineral reserves.
6. The numbers for tonnage, % U_3O_8 and contained lbs U_3O_8 are rounded figures.

Systematic density measurements, made on drill core by staff of Priargunsky, and confirmed by more recent testing, result in an average density of 2.60 g/cc for the host rock siltstones.

Interpretation of mineralization was done at a threshold of approximately 0.015% U_3O_8 for the No. 7 Deposit, and approximately 0.010% U_3O_8 for the No. 2 Deposit. Separate block models were evaluated for each deposit, within the interpreted wireframes. Blocks in the models were compared to higher cutoff grades, calculated using operating costs, metallurgical recoveries, and the uranium price.

Scott Wilson RPA classified the mineral resources in the Nos. 7 and 2 Deposits into the Indicated category based on drill-hole spacing, apparent continuity of mineralization, and the results of the recent confirmation drilling. A small additional part of the No. 2 Deposit has been classified as Inferred mineral resources, in an area extending both inside and outside (north) of the current boundary of Mineral Licence 237A.

In plan view, the No. 7 Deposit block model shows a high-grade central core, with a large halo of mineralization in which the grade declines smoothly towards the edges. The No. 2 Deposit block model shows several areas of higher-grade (>0.10% U_3O_8) mineralization, with the largest area concentrated underneath the current pit, and another area to the southeast. West of the current pit, grades start below 0.10% U_3O_8, and decrease gradually.

3.4.9 Mineral Reserves

Mineral reserves were estimated by P&E Mining Consultants Inc. (P&E) for the DFS assuming underground longhole open stoping methods with cemented and uncemented waste rock backfill for the No. 7 Deposit, with stope sizes and pillar layouts as described in a geotechnical study by Golder Associates. Mineral reserves for the No. 2 Deposit assume open-pit mining. Mineral reserves are summarised in Table 3-8.

Table 3-8
Mineral Reserve Estimate

Location	Category	Tonnes (million)	% U_3O_8	lbs U_3O_8 (million)
No. 7 Deposit	Probable	10.63	0.174	40.8
No. 2 Deposit	Probable	7.41	0.074	12.1
TOTAL	**Probable**	**18.04**	**0.133**	**52.9**

Notes:

1. CIM definitions were followed for mineral reserves.
2. Mineral reserves were estimated using a U_3O_8 price of USD 55/lb.
3. Mineral reserves were estimated using an underground cutoff grade of 0.061% U_3O_8 for No. 7 Deposit, and an open-pit cutoff grade of 0.028% U_3O_8 for No. 2 Deposit.
4. The numbers for tonnage, % U_3O_8 and contained lbs U_3O_8 are rounded figures.

(a) Dilution – No. 7 Deposit

External dilution for No. 7 Deposit stopes is estimated to average 10% at zero grade, including hanging wall and backfill dilution.

(b) Dilution – No. 2 Deposit

Examination of the block model for the No. 2 Deposit shows the gently-dipping mineralized layers angling into, and out of, successive benches. Open-pit grade control will have to be applied to each bench, in order to determine boundaries for ore definition, on a scale that matches the selectivity of the mining equipment. A dilution allowance of 15% at a grade of 0.018% was factored into bench grades to account for this problem.

(c) Resource Extraction – No. 7 Deposit

Mineral reserve tonnage (exclusive of dilution) totals 74% of mineral resource tonnage for the No. 7 Deposit. Metal content in mineral reserves (40.8 million pounds) totals 84% of mineral resource metal. Extraction was assessed in two stages; first, by application of stope outlines, with some resources rejected for being too thin or scattered to form stopes; and second, by application of expected recoveries for various stope configurations.

(d) Resource Extraction – No. 2 Deposit

Portions of mineralized layers will be rejected by open-pit grade control, where dilution within an ore bench is too high or mineralization lies under too much waste stripping cover. A resource extraction factor of 68% was calculated.. Actual mining extraction within the open-pit design was determined to be 95%.

3.4.10 Mining Operations

The DFS outlined mining of the Nos. 2 and 7 Deposits, at a production rate of 3500 t/d, or 1.225 Mt/a. Mining of all Mineral Reserves is expected to require slightly more than 15 years.

The No. 7 Deposit was partially developed for exploration, with two shafts, and development drifting on 550 Level. The exploration drifting was extended southwards to test other potential deposits (Nos. 4 and 5 Deposits), with another ventilation shaft (No. 2 Shaft) serving that area. Currently, the underground workings are flooded and the mine needs to be dewatered before a full evaluation of their condition can be completed. For the most part, the mine infrastructure, which supported the original exploration, has been destroyed or removed and has to be replaced.

Underground mining is proposed for No. 7 Deposit, using Longhole Open Stoping with cemented and uncemented waste rockfill backfill. Production at the full rate of 3500 t/d for the first 8 years will come from the No. 7 Deposit.

The No. 2 Deposit was mined as an open-pit operation from 1988 to 1995 by Priargunsky. The open pit is currently partially flooded, and is expected to serve as a reservoir for process water during the early years of operation. As production from the No. 7 Deposit decreases. Phase 1 open pit mining will begin. Two additional phases are proposed, with total open-pit mining expected to last just over 7 years.

(a) Underground Mine Design – No. 7 Deposit

Golder completed a geotechnical review entitled “Mine Geotechnical Underground Design for Dornod Project Mongolia,” dated September 2006. Golder’s review, based on evaluation of drill core, included recommendations for stope dimensions and ground support requirements, which are used for the DFS.

Access to the underground mineralized zones and old development areas will be by an inclined ramp from surface. The ramp portal is situated near the processing plant. This ramp will also facilitate truck haulage of ore to the processing plant.

The mining method is Longhole Open Stoping will mainly use longholes drilled in a downhole fan pattern. In areas near the top of the orebody, to minimise development, stopes with heights of less than 15 m will be mined using upholes drilled in a parallel pattern. Stopes will be nominally 15-m wide by 18-m long and a maximum of 30-m height (floor to floor).

The orebody geometry, with a length of approximately 600 m and a width of approximately 500 m, requires that the stopes be combined into mining blocks with barrier pillars left between mining blocks, to provide regional stability as mining progresses. This divides the orebody into a chequer board of blocks with each mining block having dimensions of 150 m in the west-east direction and 108 m north-south. The regional pillars between mining blocks will be 38-m wide. Each mining block between levels is subdivided into individual stopes having nominal dimensions of 15-m wide by 18-m long. A mining block will therefore consist of 60 stopes.

Barrier pillars between mining blocks are oriented north-south and east-west. The east-west pillars are called primary pillars and the north-south secondary pillars.

All Primary Access Drift and Secondary Access Crosscut headings will be 5 m by 5 m to accommodate haul trucks and ventilations requirements. Truck loading areas will be developed at all remucks by taking down the backs to a height which will accommodate truck loading by load, haul, dump (LHD) vehicles.

A slot raise will be developed at the far end (north) of each stope. The stope will be drilled off in a fan pattern. The first stope blast will break into the slot raise and subsequent blasts into the mucked-out void. Each stope will be ring blasted in three blasts.

Broken ore will be loaded in the undercut sill crosscuts into 6.1-m^3 LHDs and transported to the closest orepass. Orepasses deliver ore to the 480 Level for loading into the haul trucks for haulage to surface.

Within each mining bloc, stopes will be mined in a primary / secondary sequence, where primary stopes on either side of a secondary stope are mined and backfilled, after which the secondary stope is mined. In addition, each north-south line of primary stopes (six stopes per line) in a mining block will be retreated from north to south, ahead of the retreating lines of secondary stopes. The same sequence will also be extended vertically, where primary and secondary stopes below must be completed, before primary or secondary stopes above are mined.

With the primary and secondary sequencing of stopes, backfilling will use a combination of cemented waste rock backfill in primary stopes and 2/3 of secondary stopes with the remaining stopes backfilled with uncemented waste rock.

Mining block sequencing is dictated by ventilation and pillar recovery requirements. Stope sequencing uses the primary / secondary sequence for mining individual longhole stopes.

Mining blocks will be mined in a sequence to ensure one time use of ventilation air which has been in contact with ore. Mining blocks, in general, will be mined from the northwest to the southeast. When all mining blocks around a primary and secondary pillars are mined out, the pillars will be recovered immediately afterwards to minimise mining problems and allow for areas to be permanently abandoned.

Pillars between mined-out blocks will be recovered by longhole mining as well, with stopes developed at right angles to the pillar drifts and crosscuts. The stopes will be mined with widths of 10 m and lengths of 17.5 m on one side and 12.5 m on the other. The longer stope would be mined and backfilled first, followed by the shorter stope. Stopes will retreat from west to east and north to south of pillar drifts and crosscuts, respectively. All pillar recovery stopes will be backfilled with cemented waste rock.

Due to stress shedding to the pillars, mining conditions will be more difficult, requiring rehabilitation of the sill drifts and extra cable bolting to maintain stope stability.

Waste rock backfill will be delivered by truck to the stopes. The waste rock will be delivered to the 453 and 435 Levels via backfill raises from surface. The bottom of the backfill raise will be equipped with a truck loading chute and slurry addition system. This will produce a cemented waste rock backfill with approximately 4% cement content. The truck will transport the resulting backfill to the stope being backfilled.

Backfill raises will be located in the centre of four mining blocks to provide optimum backfill distribution to the different mining block areas. A total of three backfill raises is planned.

(b) No. 7 Deposit Ventilation

Detailed ventilation design and modeling were undertaken by Intergen Safety and Environment Solutions Inc. of Saskatoon, Saskatchewan, Canada.

The underground ventilation system is required to provide airflow volumes and distribution that will provide wholesome air for all underground workers. Specifically for this Project, the system is designed to control airborne radiation, airborne respirable silica concentrations, and diesel exhaust fume concentrations in the workplace.

The following specific design criteria were adopted for the Project.

(i) The system will be designed to control airborne radiation concentrations to levels that, together with other radiation exposure management measures, are conductive to maintaining radiation exposures consistent with As Low As Reasonably Achievable (ALARA) principle.

(ii) The system will be designed to provide at least 0.05 m^3/s (100 ft^3/min) per brake horsepower (BHP) of diesel equipment operating underground.

Air distribution is dependent on the radiation protection requirements and the manner in which diesel equipment is deployed throughout the mine. Achieving adequate radiation protection requires that the air be moved from the fresh air source to the exhaust in an expedient manner with the controlled reuse of air minimised.

The Fresh Air Raise (FAR), in parallel with Shaft No. 3, will convey the bulk of the intake air to the mine workings. Vitiated air will be removed from the mine to surface via two Return Air Raises (RARs). Intake air will flow from the bottom of FAR on the south side of levels in a northerly direction to the RARs on the north side of the mine. The proposed main ventilation system will consist of a 6-m-dia intake vent raise (FAR) and a 6-m-diameter downcast shaft (Shaft No. 3) on the south side of the orebody, and two 4-m-diameter exhaust vent raises on the north side of the orebody. The bulk of the fresh air will downcast the Fresh Air Raise (FAR) and a smaller amount will downcast the shaft.

The aim of the ventilation distribution system is to provide fresh air to workers in their workplaces, minimise work in areas that may be upstream of other active working areas, and ensure careful monitoring. Excessive airborne radiation, diesel emission or silica contamination may require localised ventilation arrangements to avoid unnecessary exposure of workers.

(c) Underground Preproduction Development

Preproduction mine development and construction, including initial mining blocks, requires approximately 3 years (Table 3-10). All preproduction development and construction will be performed by a mining contractor. Work during the preproduction period will include:

- Dewatering of existing underground workings and discharge to existing No 2 open pit
- Developing the main ramp from surface to the 483 Level
- Sinking and lining the FAR No. 1 (near No. 3 Shaft) and RAR No. 1 and RAR No. 2
- Constructing and installing main surface ventilation fans on raises and No. 3 Shaft
- Constructing miscellaneous surface facilities related to the mine
- Completing the northwest internal ramp and lateral development on the 483, 453, 435 and 405 Levels
- Installing 483 Level infrastructure (maintenance shop, refuge station, fuel bay, explosives and detonator magazines, sumps, etc.)
- Developing initial internal ventilation raises
- Installing and commissioning all required mine services.

The underground mine development schedule for the preproduction period is shown in Table 3-9.

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Table 3-9
Preproduction Development Schedule

Component	Quantity	Units	Dimensions	Year -3 Q4	Year -3 Q3	Year -3 Q2	Year -3 Q1	Total Year -3	Year -2 Q4	Year -2 Q3	Year -2 Q2	Year -2 Q1	Total Year -2	Year -1 Q4	Year -1 Q3	Year -1 Q2	Year -1 Q1	Total Year -1	TOTAL
Underground Infrastructure Development																			
Main Ramp Surface to 510 Level	3,860	metres	5m W X 5m H		420	420	420	1,260	420	420	420	420	1,680	420	420	80		920	**3,860**
Lateral Development								0					0					0	
Internal Ramp 482 to 435		metres	5 m W x 5 m H					0					0	96	700			796	**796**
405 Level Main Accesses	115	metres	5m W X 5m H					0					0			62		62	**62**
435 Level Main Accesses	2,515	metres	5m W X 5m H					0					0	193	360	704	118	1,375	**1,375**
453 Level - Main Accesses	633	metres	5m W X 5m H					0					0					0	**0**
483 Level Main Accesses	2,811	metres	5m W X 5m H					0				722	722	155			422	577	**1,299**
Truck Loading Stations	320	metres	5m W X 10m H					0					0				60	60	**60**
Raises																			
Ventilation Raises	832	metres	4m X 4m					0				42	42	664	29		97	790	**832**
Backfill Raise	1,000	metres	2.4m X 2.4m					0					0		500			500	**500**
Mine Services								0					0					0	
483 Trackless Maintenance Shop	18,234	cu.m.						0					0			18,234		18,234	**18,234**
453 Explosives Magazine	803	cu.m.						0					0		803			803	**803**
453 Detonators Magazine	57	cu.m.						0					0		57			57	**57**
483 & 510 Refuge Stations	1,606	cu.m.						0		803			803	803				803	**1,606**
483 and 510 Latrines	148	cu.m.						0		74		74	148					0	**148**
483 Fuel Bay	439	cu.m.						0					0	439				439	**439**
510 Fuel Bay	439	cu.m.						0					0	439				439	**439**
483 & 453 Storage Areas	60	metres	6m X 5m H					0			30		30		30			30	**60**
510 Main Dewatering Sump	705	cu.m.	7 m dia.					0			705		705					0	**705**

All raise development work during preproduction and production period will be performing by the mining contractor.

(d) Underground Mining

The mine production schedule is based on mining 3500 t of ore per day for 350 d/a, or 1 225 000 t of reserves per year.

Each stope produces approximately 1000 t/d during the mucking cycle. A stope is drilled blasted, mucked out and backfilled in a total of approximately 73 days for Longhole Open Stoping – Downholes and 23 days for Longhole Open Sloping – Upholes.

Production requirements will be met with an average of five to six stopes loading, blasting and mucking, six stopes drilling and one stope backfilling per shift. Backfilled stopes will require approximately 30 days curing time before adjacent mining can take place.

It should be noted that all production ore will be transported by 50-t trucks traveling up the ramp to surface.

(e) Open-pit Mining – No. 2 Deposit

The proposed Dornod open pit will be developed at the site of the former uranium open pit. The historic pit will be dewatered and further developed to create the proposed Dornod open pit. It is envisaged that the open pit will be developed concurrent with the last year of underground mining (Year 8), and that the historic pit will be dewatered as part of the underground mining and ore processing operations.

The Dornod open pit will be developed by Khan using its own equipment and workforce. They will have responsibility for: the dewatering of the historic pit and re-establishment of the pit haulage roads; production drilling and blasting; the excavation of ore to the primary crusher and waste rock to the waste rock management area; oversize breakage; haul road maintenance; and equipment maintenance. Khan will provide the open-pit equipment, supervision, operator training, the mine consumables, the pit operations and maintenance facilities, and a pit technical and health and safety program including radiation monitoring and dose assessments.

The open-pit operation will make use of the following site infrastructure components that will have been constructed to service the underground mining operation:

- Surface shops and warehouse facilities
- Dry, camp and office facilities
- Explosive and detonator magazines on surface
- Electrical power distribution system
- Ore crusher on surface.

The site infrastructure will be expanded to include:

- An open-pit equipment maintenance shop
- The addition of a grizzly and rock breaker at the hopper feeding the ROM ore conveyor, grizzly and jaw crusher. The addition of a metal detector and interlocks on the feeder to the ore grizzly and primary crusher to assist in detecting / removing scrap steel including drill bits from the run of pit ore.

(i) Preproduction Development

The preproduction development work consists of prestripping 11 Mt of waste rock.

(ii) Open-pit Production Schedule

The open-pit production schedule includes a preproduction period (Year 8), and the pit operations phase. The pit is scheduled to be developed and readied for production concurrent with the last year of underground mining. It is projected that the pit will produce 7.4 Mt of ore in slightly over 6 years. The pit will supply 1.225-Mt/a ore to the processing plant. The open-pit production schedule is shown in Table 3-10.

Table 3-10
Open-Pit Production Schedule

Year	Ore Tonnage (kt)	Waste Tonnage (kt)	Total Tonnage (k)	Tonnes Per Day (kt/d rock)	Waste / Ore Ratio (t waste:t ore)
8					
9	26	11 249	11 275	55	439.6
10	1 225	18 025	19 250	55	14.7
11	1 225	18 025	19 250	55	14.7
12	1 225	18 025	19 250	55	14.7
13	1 225	18 025	19 250	55	14.7
14	1 225	18 025	19 250	55	14.7
15	1 225	12 577	13 802	39	10.3
16	31	591	32	4	0.02
TOTAL	**7 407**	**113 952**	**121 359**		**15.4**

The open pit will be developed in three phases. The waste rock will be disposed in four waste rock piles to be constructed adjacent to the pit.

(iii) Open-pit Mining Method

The geology of the open pit includes at least five horizons hosting uraniferous mineralization that are interlayered with felsic to intermediate volcanic rocks. These layers dip and angle in and out of the successive pit benches and are flat or near horizontal on some elevations. The ore interceptions, varying ore thicknesses and the need to control dilution and ore losses necessitate that the mining method provide operational flexibility and include ore grade control and survey control programs; as such as follows.

- The pit will use conventional mining equipment and a combination of 10- and 5-m bench heights and flexible mining practices. Most of the waste rock will be mined using 10-m-high benches and conventional open-pit drilling, blasting, excavating and haulage methods. Ore layers that are horizontal or near-flat dipping will be mined using 10-m-high benches or 5-m-high split benches, depending on the ore thickness and ore grade control requirements.

- In parts of the pit, the ore is relatively thinner with gentle to steep sloping surfaces. These areas are not amenable to mining with 10-m-high benches. A combination of 5-m-high split benches and 5-m-high split benches with flitch mining will be used to mine the ore in these areas. The flitch mining will involve the selective removal of waste rock over the ore layer, followed by the selective mining of the ore layer. A portion of the ore in these areas will be rejected by the ore grade control program or otherwise not recovered by the mining operations.

- The main pit production equipment fleet has been sized for mining 10-m-high benches. The pit will also have a fleet of smaller mobile equipment for mining 5-m-high benches. The smaller equipment will include a hydraulic excavator that will provide improved selectively, in comparison to the loading units to be used to excavate the 10-m benches.

- The pit will have a radiometric ore grade control program to determine the boundaries for ore definition and finalise the blast plans, and a survey control program.

(iv) Open-pit Operations

The open-pit mining operations will be carried out on a two 12-hr shifts per day basis with 2 weeks on, 2 weeks off rotations.

Ten-Metre Benching

The 10-m bench blastholes will be drilled off using two Sandvik model D245S drills. This drill is a diesel-powered self-propelled crawler-mounted blasthole drill that is equipped as a rotary drill for 127-mm to 203-mm (5 in. to 8 in.) diameter holes to a depth of up to 45 m (148 ft).

The drilling and blasting parameters for the 10-m benches are shown in Table 3-11.

Table 3-11
Ten-Metre Bench Drilling and BlastIng Parameters

	Parameter	
Item	**Ore**	**Waste**
Bench height	10 m	10 m
Blasthole diameter	172 mm (6-3/4 in.)	172 mm (6-3/4 in.)
Burden	5.25 m	5.25 m
Spacing	5.25 m	6.4 m
Subdrill	1.7 m	1.7 m
Stemming	2 m	2 m
Blasting agent	ANFO at 1.05 g/cc	ANFO at 1.05 g/cc
Tonnage factor	2.6 t/m^3 in-situ	2.6 t/m^3 in-situ
Powder factor	0.32 kg/t (0.85 kg/m^3)	0.27 kg/t (0.70 kg/m^3)

One Caterpillar RH12OE diesel-hydraulic shovel, one Caterpillar 994F wheel loader, and a fleet of Caterpillar 785C haulage trucks were selected for the purposes of this study and are well suited to the Project. The RH 120E hydraulic shovel has a 16.5-m^3 capacity (2:1 heap) bucket. The Caterpillar 994F wheel loader will be equipped with a nominal 16-m^3 bucket.

The Caterpillar 785C haul truck has a nominal payload capacity of 136 t. The number of Caterpillar 785C haul trucks in the equipment fleet in each year of the pit life is shown in Table 3-12.

Table 3-12
Caterpillar 785C Haulage Truck Fleet

Year	Number of Caterpillar 785C Trucks Purchased	Number of Caterpillar 785C Trucks On-site
8	4	4
9	6	10
10	2	12
11		12
12		12
13		12
14		12
15		12

Five-metre Split Benches

The 5-m-high split benches will be mined using a combination of smaller mobile equipment and the main pit production equipment fleet depending upon three general field conditions as shown in Table 3-13. Based upon a review of proposed bench elevations and ore layer geometry and thicknesses, the small equipment fleet will be utilised to mine thinner ore layers. It is assumed that the smaller equipment fleet will be utilised to mine 10% of the ore and waste.

The smaller mine equipment fleet will include the following.

- Two Sandvik DP800 drills – This drill is a self-propelled, crawler-based top hammer drill equipped with a climate-controlled operators cabin, dust collector and a rod changer. It is designed to drill 76 to 127 mm (3 to 5-/2 in.) vertical, inclined or horizontal holes.

- One Caterpillar 345D diesel hydraulic excavator - The 345D is a crawler-mounted excavator equipped with a nominal 1.8-m^3 bucket.

- Two Caterpillar D9T bulldozers – They will be equipped with a single shank ripper, and nominal 13.5-m^3 capacity blade and blade tilt cylinder.

Table 3-13
Flexible Mining Approach

Field Condition	General Approach
1. The ore layer is generally horizontal and ore control allows it to be mined as a 10-m-high bench or a 5-m-high split bench.	Ore is drilled off using the main blasthole drills. The blasted ore is excavated using the main loading and haulage equipment. It is assumed that approximately 90% of the ore and waste will be mined using this approach.
2. The ore layer is generally horizontal within a 5-m-high split bench. Reduce dilution.	The local bench elevation is adjusted and the 5-m-split bench is mined using the main loading and haulage equipment; or smaller track-mounted drills are used to drill off the waste or ore. The blasted waste or ore is removed using a bulldozer or excavator to a nearby location, where it is rehandled by the main loading and haulage equipment.
3. The ore layer dips and angles within a 5-m-high split bench. Ore control requires the selective mining of ore and waste.	Smaller track-mounted drills are used to drill off the waste or ore. The blasted material is removed using a smaller hydraulic excavator and bulldozers and stockpiled nearby for reclaim by the main loading and haulage equipment.

(f) Life-of-Mine Plan

The life-of-mine production plan for both the underground and open-pit mining operations is presented in Table 3-14.

Table 3-14
Dornod Life-of-Mine Production Schedule

		Underground		Open Pit		Total Mined		Mill Feed	
Year	Source	Ore Mined (Tonnes)	Grade (% U3O8)	Ore Tonnes	U3O8 %	Ore Tonnes	U3O8 %	Ore Tonnes	U3O8 %
-2	UG	2,000	0.062			2,000	0.062		
-1	UG	97,000	0.181			97,000	0.181		
1	UG	755,000	0.230			755,000	0.230	854,000	0.224
2	UG	1,228,000	0.234			1,228,000	0.234	1,225,000	0.234
3	UG	1,226,000	0.183			1,226,000	0.183	1,225,000	0.183
4	UG	1,226,000	0.208			1,226,000	0.208	1,225,000	0.208
5	UG	1,226,000	0.166			1,226,000	0.166	1,225,000	0.166
6	UG	1,229,000	0.136			1,229,000	0.136	1,225,000	0.136
7	UG	1,225,000	0.115			1,225,000	0.115	1,225,000	0.115
8	UG	1,225,000	0.149			1,225,000	0.149	1,225,000	0.149
9	UG & Pit	1,195,000	0.167	26,000	0.068	1,221,000	0.164	1,225,000	0.164
10	Pit-Ph-1			1,225,000	0.093	1,225,000	0.093	1,225,000	0.093
11	Pit-Ph-1			1,225,000	0.082	1,225,000	0.082	1,225,000	0.082
12	Pit-Ph-1&2			1,225,000	0.075	1,225,000	0.075	1,225,000	0.075
13	Pit-Ph-1,2&3			1,225,000	0.070	1,225,000	0.070	1,225,000	0.070
14	Pit-Ph-2&3			1,225,000	0.058	1,225,000	0.058	1,225,000	0.058
15	Pit-Ph-3			1,225,000	0.066	1,225,000	0.066	1,225,000	0.066
16	Pit-Ph-3			31,000	0.086	31,000	0.086	37,000	0.086
Total		**10,634,000**	**0.174**	**7,407,000**	**0.074**	**18,041,000**	**0.133**	**18,041,000**	**0.133**

3.4.11 Surface Infrastructure

(a) Water

The water balance calculated for the DFS indicates that about 179 m^3/h of process water will be required for the plant. Process water will be reclaimed from the mine and pit, which should be capable of supply up to 60 m^3/h of water once dewatering is complete. Currently, there is approximately 1.56 Mm^3 of water available in the flooded pit as per last survey performed on-site (Oyu Survey LLC, 2008). An allocation for the drilling of a well at the plant site has been made.

(b) Power

Khan has been informed by both the Aimag Business Development Manager and the Power Plant Manager that the power plant in Choibalsan has been refurbished to consistently produce over 30 MW and spare capacity presently exists within the system to meet Project needs. This will be enhanced once the system is connected to the Mongolian national grid. Power (16 MW) will be brought to the site via an overhead power line currently being constructed by Khan for mine dewatering.

3.4.12 Mineral Processing and Metallurgical Testing

The Dornod claims area contains several known ore deposits. This DFS provides for the mining and processing of Nos. 2 and 7 Deposits. Due to its higher grade, the No. 7 Deposit will be developed first. This is expected to take up to about 9.8 years. After about 9 years, it will become difficult to extract 3500 t/d from the No. 7 Deposit. At this time, the tonnage will be replaced with lower grade No. 2 Deposit ore.

The No. 7 orebody, after dewatering the mine, will be accessed via a new ramp to be sunk adjacent to the richest part of the deposit. The existing No. 3 shaft will become the primary ventilation shaft. The No. 2 Deposit will be developed as an open-pit mine.

A milling rate of 3500 t/d is planned. In Years 1 to 9, treating only No. 7, the ore head grade will be typically 0.2% U_3O_8 for Years 1 to 4 and 0.1 in Years 5 to 7. After Year 9, once No. 2 ore is added to the mix, grade will gradually decrease until it reaches average grade for No. 2 ore only after about Year 10 of 0.08% U_3O_8, dropping to 0.07 in Years 11 and 12 and to 0.06 through the end of mine life at Year 16.

The No. 7 Deposit has proven to be refractory. This is presumed to be as a result of the presence of brannerite, a uranium titanate mineral, due to the ore's high in-situ carbonate content and because the uranium minerals are very fine and are closely associated with gangue particles . These effects result in high acid consumption if acceptable recoveries are to be achieved. The difficulty experienced in the leaching seems to vary throughout the deposit. Although the uranium mineralization has been found to exist as very small and intergrown crystals, it has not been necessary to grind the ore to very fine particle size. It is, however, necessary that a significant amount of silica in the ore be dissolved, in order to liberate the uranium. The presence of this dissolved silica causes a gel to form, making the ore difficult to settle or filter. To overcome these problems, a Resin in Pulp (RIP) method of removing the uranium from the ore has been selected to recover the dissolved uranium.

An average leach recovery of 88% has been achieved in testwork to date on the No. 7 Deposit ore. This recovery, with a precipitation yield of 96%, is used in the financial analysis.

The No. 2 Deposit is free milling and, based on the Russian experience, a leach recovery of 93% has been assumed for this ore. This assumption needs to be confirmed in the laboratory. Reagent consumptions for this material also need to be confirmed at the detailed engineering stage.

The No. 7 ore will be brought to surface through a new ramp in 50-t trucks and dumped into a communal dump hopper. A bypass is provided to stockpile ore should the dump hopper be full. This stockpiled material, along with ore from the No. 2 Deposit surface stockpile, will be fed back to the feed hopper using a front-end loader.

After about 10 years of the mine life, the No. 2 Deposit ore will be transported to the stockpile or the dump hopper using 140-t ore trucks.

The dump hopper is provided with a 300-mm grizzly. The grizzly oversize will be crushed to –300 mm in an open-circuit jaw crusher. This crusher is able to handle the larger ore from the No. 2 orebody pit.

The –300-mm material will be fed to an open-circuit 20-ft-diameter by 12-ft-long (6.1 m by 3.7 m) semi-autogenous grinding (SAG) mill. This will produce an 80% passing 2-mm feed to a 16-ft-diameter by 21-ft-long closed-circuit ball mill. The SAG mill will be equipped with a

2200-kW motor, while the ball mill will be powered with a 1750-kW motor. The grinding circuit will produce 80% passing 75 micron material.

Testwork has indicated that the ore is relatively hard and will produce a critical size which will not break down in the SAG mill. For this reason, a 4-ft pebble crusher has been included in the design. This will crush oversize material scavenged from the SAG mill discharge trommel.

The milled material, before acidification, settles well and will be thickened to a density of 50% solids in a high-rate 7-m-dia thickener. In order to save on acid costs, a portion of the thickener underflow material will be further dewatered on a 10-disk vacuum disk filter. This dewatered material will be mixed with unfiltered thickener underflow and repulped to produce a 58% solids feed stream to feed the leach section.

Some of the residual heat in the leach discharge stream will be used to preheat the leach tank feed. The lowering of the leach discharge pulp temperature is required to protect the integrity of the ion exchange resin in the uranium recovery section.

A conventional sulphuric acid leach section has been designed to treat the two ores. After thickening and preheating, the pulp will be leached in a series of 18 pachuca tanks. A residence time of 42 hours was used in the design. The free acid in the leach section will be maintained at about 25 g/L and the pulp will be heated to 80ºC. This will be done by the injection of live steam produced in the acid plant. Oxygen, produced in a dedicated oxygen plant, will be injected into the leach tanks to maintain the EMF at approximately 480 mV. Each of the tanks will be agitated using a 260-kW agitator.

In order to protect the resin from osmotic shock, after leaching and before the heat exchange, the leached pulp will be partially neutralised to a pH of 2 to 2.5 by the addition of lime.

The dissolved uranium will be removed from the leached pulp by adsorbing the uranium onto anion exchange resin (Purolite A660 or equivalent). The resin and the pulp will flow countercurrenly to each other in an eight-stage KEMIX carousel type resin-in-pulp circuit. At the end of the process, the loaded resin will be separated from the pulp stream by screening the pulp on a vibrating screen. The barren pulp will be sent to neutralisation and then to disposal in the tailings dam.

The loaded resin will be washed before being eluted with sulphuric acid in a batch type elution circuit. Provision has been made to periodically wash the stripped resin with a caustic solution to remove any silica that may have adhered to the resin.

Before uranium precipitation from the pregnant liquor, impurities will be removed by adjusting the pH to approximately 3.2. In this way iron, arsenic and sulphates will be removed by the addition of lime and ferric sulphate in an oxidising environment. The resulting solids, mainly gypsum, will be removed on a belt filter. The resulting filtrate will be further clarified by passing it through sand filter clarifiers.

Yellowcake will be precipitated from the clarified solution by the addition of magnesia and hydrogen peroxide to form insoluble uranium oxide. This will be dewatered in a thickener and a centrifuge before being dried in a multi-hearth drier.

Leached pulp from the resin-in-leach (RIL) circuit will be neutralised with lime and treated with ferric sulphate and barium chloride before thickening and sending the material to tailings. This will precipitate heavy metals, radium 226 and arsenic ions into the solid tailing.

An extensive water treatment system has been designed. This system includes neutralisation, clarification and reverses osmosis treatment. All tailings dam return water, underground and open-pit mine water, and surface runoff will report to a surface surge pond before treatment and disposal, or being pumped to the mill process water tank.

Potable water will be produced from open-pit supernatant water by reverse osmosis.

Metallurgical Testwork

The metallurgical testwork that underpins the DFS design is in three parts:

(a) Early work conducted by the Russians

(b) Work in preparation for the PFS that was conducted in 2007 / 2008 and was reported in the PFS

(c) Additional work that was conducted in 2008 for the DFS.

3.4.13 Environmental and Geotechnical Considerations

Water Management

Golder assumes that the open-pit lake will have an available volume of 1.0 Mm^3 of water at start up and will operate as a water storage facility for a period of 7 years before the open-pit prestripping starts in Year 8 under mean annual precipitation conditions.

The main objectives of the water management plan are to collect and manage all water on the site; maximize flow and design for zero discharge to the environment under normal operating conditions.

Three water collection ponds will operate at the site: the RMA Pond; the Water Collection Pond; and the Polishing Pond.

Water from the RMA Pond will be pumped directly to the processing plant. Additional water required for processing will be pumped from the open-pit lake for the first 7 years, and then from the Water Collection Pond after Year 7, when the open pit will be prestripped and mined.

Runoff from adjacent lands, from the surface waste rock dumps, ore stockpiles and overburden stockpiles will be collected in ditches and pumped or directed to either the open-pit lake (first 7 years) or to the RMA or the Water Collection Pond.

Residue Disposal

The process will produce several waste streams, as follows.

(a) Leach residue will be discharged at the end of the RIL section. It will be neutralised and treated with ferric sulphate and barium chloride prior to disposal.

(b) A gypsum stream that results from the neutralisation, with lime, of residual acid in the eluate pregnant solution. This will contain insoluble metal hydroxide ions.

(c) A very small intermittent stream of material similar to the described in Item (b) above which originates in the water treatment section.

(d) Solvent extraction (SX) crud will comprise a small volume of waste from the solvent extraction. It is assumed that, because of the organic content, the SX crud will be disposed of separately from the leach residue.

Prior to disposal, the waste streams will be treated with lime, so that their pH is neutral to slightly basic. In addition to the above process streams, a relatively small volume of ash from coal burning boilers will also be disposed of in a lined RMA, which will be located in the southwest corner of the land use permit area.

Containment for the residue will be provided partially by the surrounding topography and partially by the construction of three perimeter dams. The dams will be constructed in two or more stages. The first stage (i.e., the Starter Dam Stage) was designed to contain approximately 2 years of residue production.

3.4.14 Environmental and Social Impact Assessment

(a) Introduction

An internationally recognised Environmental and Social Impact Assessment (ESIA) for the Dornod Uranium Mining Project (the Project) was prepared by AATA International, Inc., based in Denver, Colorado, U.S.A.

The ESIA provides: comprehensive information about the key environmental and social characteristics of the Project; data on the current or baseline (predevelopment) environmental and social conditions at the Project site based on recent studies at the site and historical information; evaluations of potential impacts of the Project; and, recommendations for impact mitigation measures. It also includes a comprehensive document, the Environmental and Social Management Program (ESMP), which provides detailed information on the policies, practices and procedures that will be implemented by Khan at the Dornod Project to comply with applicable Mongolian regulatory requirements, as well as, conform to international guidelines and standards, to which Khan is committed.

The ESIA was developed in accordance with good international industry practice (GIIP) including those specifically defined by the Performance Standards on Social and Environmental Sustainability of the International Finance Corporation (IFC - a unit of the World Bank) and by the Equator Principles.

The study methodology was comprised of the following activities.

- Obtaining all pertinent historical information on the Project from local and national sources, including mine plans and documents, aerial photography images, government reports and other pertinent documents

- Conducting a review of existing literature and data for the Project area

- Identifying Khan's corporate environmental and social policies and guidelines; Mongolian environmental and social regulations and legislative framework; and, international environmental and social guidelines and standards with which the Project must comply or conform

- Performing field baseline studies to collect Project site-specific data on current environmental and social conditions

- Describing the overall Project with an emphasis on processes that may potentially impact the environmental and social conditions

- Characterising the physical, chemical, biological, and social and radiological components of the environment potentially affected by Project development

- Identifying and ranking environmental and social risks and impacts for each Project component for each phase of the Project

- Developing an environmental and social management program that describes mitigation measures designed to eliminate or minimise environmental and social impacts

- Identifying net Project impacts.

The ESIA report includes an Executive Summary, Introduction, Project Description, Project Alternatives, Regulatory Framework, Description of the Baseline (Existing) Environmental and Social Conditions (including Geology and Mineral Resources), Analysis of Potential Impacts and Mitigation Measures, Waste Management, Occupational Health and Safety, Radiation Protection, Emergency Response and Hazard Prevention, Decommissioning and Reclamation (i.e., Project Closure), and Net Environmental and Social Impacts.

The ESMP has been prepared to satisfy Mongolian laws, international guidelines and standards of environmental and social practice, and standards of industry practice that meet Khan's corporate environmental and social policies.

(b) Net Environmental and Social Impacts

The predicted net environmental and social impacts for the Project are based on an impact analysis conducted for the ESIA with the following assumptions.

- Mongolian laws and regulations applicable to the Project will be complied with in the design, construction, operation and closure of the Project;

- Internationally recognised criteria and standards (e.g., IFC Performance Standards, Equator Principles, WHO guidelines, etc.) will be adopted in the design, construction, operation and closure phases of the Project; and,

- Proper mitigation measures, employing GIIP as defined by the IFC, will be implemented during all phases of the Project.

Many adverse effects that could occur from the Project will be eliminated or minimised by proper design, maintenance, management, and mitigation measures. The net environmental and social analysis assumes that the environmental and social management, monitoring, and reclamation measures will be implemented as discussed in both the ESIA and ESMP.

A table summarising the potential net environmental and social impacts is presented. Net impacts were calculated based on worst-case impact scenarios (i.e., gross impacts), minus the effects of all proposed prevention and mitigation measures.

This analysis indicates that implementation of the environmental and social management, mitigation, monitoring, and reclamation measures that have been proposed by Khan will eliminate or minimise the potential negative environmental and social impacts of the Project; and, will provide economic and social benefits to the region.

4 Introduction and Terms of Reference

This Report reflects the contents of a DFS compiled by Aker Solutions and numerous consultants for Khan Resources Inc. (Khan). This Report is intended to be used by Khan to further the development of the Dornod uranium property by providing estimates of resources and reserves, classification of resources and reserves, in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Classification System) and an economic evaluation of the Dornod Uranium Project (Project) located in northeastern Mongolia.

The DFS was commissioned by Khan to update and augment the Scott Wilson RPA Technical Report dated September 27, 2007. This update provides an evaluation of the economics of establishing an underground and open-pit mining and mineral processing operation of the deposit. This DFS assumes an annual production rate of 1 225 000 t of ore.

Khan may use the Report for any lawful purposes to which it is suited. The Report has been prepared in general accordance with the guidelines provided in NI 43-101 Standards of Disclosure for Mineral Projects.

The economic evaluation contained in this Report specifically pertains to the mineral resource and reserves contained within Mining Licence 237A and Exploration Licence 9282X.

This Report provides an updated estimate of mineral reserves from underground and open-pit mining and mineral processing based on metallurgical testing, reference to comparable projects, and Aker Solutions' in-house expertise.

4.1 Use of Report

This Report was prepared for Khan by Aker Solutions pursuant to the contract agreement (Agreement) between Khan and Aker Solutions.

The Report is based in whole or in part on information and data provided to Aker Solutions by Khan and / or third parties. Aker Solutions represents that it exercised reasonable care in the preparation of this Report and that the Report complies with published industry standards for such reports, to the extent such published industry standards exist and are applicable. However, Khan agrees that, except to the extent specifically stated in writing in the Agreement, Aker Solutions is not responsible for confirming the accuracy of information and data supplied by Khan or third parties and that Aker Solutions does not attest to or assume responsibility for the accuracy of such information or data. Aker Solutions also does not attest to or assume responsibility for the accuracy of any recommendations or opinions contained in this Report or otherwise expressed by Aker Solutions or its employees or agents, which recommendations or opinions rely upon the accuracy of such information or data.

The recommendations and opinions contained in this Report assume that unknown, unforeseeable, or unavoidable events, which may adversely affect the cost, progress, scheduling or ultimate success of the Project, will not occur.

Any discussion of legal issues contained in this Report merely reflects technical analysis by Aker Solutions and does not constitute legal opinions or the advice of legal counsel.

Aker Solutions makes no representations, guarantees, or warranties except as expressly stated herein or in the Agreement and all other representations, guarantees, or warranties, whether express or implied, are specifically disclaimed.

Except to the limited extent that may be required for this Report to qualify as a "technical report" by a "qualified person" in accordance with National Instrument 43-101 as adopted by rulemaking authority of the Ontario Securities Commission and entering into force on December 30, 2005, the use of this Report or the information contained herein is at the users sole risk. Khan specifically agrees to release, defend, indemnify and hold Aker Solutions, its affiliated companies, and its/their officers, directors, employees and agents harmless from any and all liability, damages, or losses of any type, including consequential and punitive damages, suffered by Khan or any third party, even if such damages or losses are contributed to or caused by the sole or concurrent fault or negligence of Aker Solutions, its affiliated companies, or its/their officers, directors, employees, or agents; provided, however, such release, limitation and indemnity provisions shall be effective to, and only to, the maximum extent allowable by law.

4.2 Terms of Reference

In preparing this report, Aker Solutions was responsible for those items listed below:

- Study Management
- Processing Plant Design
- Normalization of Capital and Operating Costs prepared by others
- Economic Evaluation.

Capital cost estimates are expressed in first quarter 2009 United States dollars (USD) with no allowance for escalation, interest costs or financing during construction. Cost estimates and factors were solicited from suppliers for all major pieces of equipment and the plant. The capital costs are based on designs presented in the DFS and have an overall level of accuracy of ±15%.

Operating cost estimates were prepared for each phase of the operation and include operating labour, fuel, replacement parts, operating supplies, maintenance labour and supplies, plant consumables, power and shipping.

4.3 Sources of Information

Information on resources and reserve determination used in the preparation of this Report was obtained from Khan officials and technical staff for the major part. The information was taken from company documents or obtained from Khan staff in personal communication.

A complete list of the consultants and contributors who have provided supporting reports and data for the Report and for the related DFS is provided in Item 5, Reliance on Other Experts.

4.4 Site Visits

Site visits were completed by the following personnel associated with the preparation of this Report.

- Hrayr Agnerian, M.Sc. (applied) P.Geo.: In December 2006 and July 2008.
- Eugene Puritch, P.Eng., in January 2008 and November 2006
- Malcolm Buck, P.Eng., in January 2005.

4.5 List of Abbreviations

Units of measurement used in this report conform to the SI (metric) system. All currency in this report is US dollars (USD), unless otherwise noted. Uranium grades are presented as "% U_3O_8", which involved conversion from "% U" for some data, such as laboratory assays.

μ	micron(s)
°C	degree Celsius
°F	degree Fahrenheit
μg	microgram
A	ampere
a	annum
ALARA	As Low As Reasonably Achievable
asl	above sea level
bbl	barrels
BHP	brake horsepower
Bq	Becquerel
Btu	British thermal units
BWI	Bond Work Index
CAD	Canadian dollars
cal	calorie
CAUC	Central Asian Uranium Corporation
CCD	countercurrent decantation
cfm	cubic feet per minute
cm	centimeter
cm^2	square centimeter
CNSC	Canadian Nuclear Safety Commission
CO	carbon monoxide
CO_2	carbon dioxide
d	day

DCN	Distributed Communication Network
DCS	Distributed Control System
DFS	Definitive Feasibility Study
dia	diameter
dmt	dry metric tonne
dwt	dry-weight ton
EIA	Environmental Impact Assessment
EPCM	Engineering, Procurement and Construction Management
ERP	Emergency Response Plan
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Program
FAR	Fresh Air Raise
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram(s)
G	giga (billion)
gal	gallon
GIIP	good international industry practice
g/L	gallon per litre
g/t	gram per tonne
GKP	Gauss Kruger-Posgar
Golder	Golder Associated Ltd.
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
hr	hour
ha	hectare
hp	horsepower
gr/ft^3	grain per cubic foot
IAEA	International Atomic Energy Authority
IFC	International Finance Corporation
in.	inch

in.^{2}	square inches
IUCN	International Union for Conservation of Nature and Natural Resources
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
Khan	Khan Resources Inc.
km	kilometer
km/h	kilometre per hour
km^2	square kilometer
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre(s)
lb	pound(s)
L/s	litres per second
LHD	long, haul, dump
m	metre
M	mega (million)
m^2	square metres
m^3	cubic metres
m^3/h	cubic metres per hour
min	minute
mm	millimetre
MNE	The Mongolian Ministry of Nature and Environment
MNT	Mongolian National Togrog
mph	miles per hour
MRPAM	Mineral Resources and Petroleum Authority of Mongolia
msal	metres above sea level
MSDS	Material Safety Data Sheets
MSHA	Mine Safety and Health Administration
mSv	milliSievert

Mt	million tonne(s)
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
NAA	Neutron Activation analysis
NGOs	non-governmental organizations
NO_2	nitrogen dioxide
O_3	ozone
OGMC	Office of Geological and Mining Cadastre
OHSP	Occupational Health and Safety Plan
OPAs	overburden placement areas
opt, oz/st	ounce per short ton
OSHA	Occupational Safety and Health Administration
oz	Troy ounce (31.1035g)
oz/dmt	ounce per dry metric tonne
PCDP	Public Consultation and Disclosure Plan
P&E	P&E Engineering
PM	particulate matter
PPE	personal protective equipment
ppm	part per million
PLC	Programmable Logic Controller
Project	Dornod Uranium Project
psia	pound per square inch absolute
psig	pound per square inch gauge
QA	quality assurance
QC	quality control
RAR	Return Air Raise
RC	reinforced concrete
RIL	resin-in-leach
RL	relative elevation
RMA	Residue Management Area
RMLM	Revised Minerals Law of Mongolia
RIP	Resin in Pulp
RPP	Radiation Protection Plan

RQD	Rock Quality Designation
s	second
Scott Wilson RPA	Scott Wilson Roscoe Postle Associates Inc.
SO_2	sulphur dioxide
SPI	SAG Power Index
st	short ton
stpa	short ton per year
stpd	short ton per day
SX	solvent extraction
t	metric tonne
T&E	Threatened and Endangered
t/a	metric tonne per year
t/d	metric tonne per day
TDS	total dissolved solids
USD	United States dollars
USg	United States gallon
USgpm	US gallon per minute
UTM	Universal Transverse Mercator
V	volt
VAT	Value Added Tax
W	watt
wmt	wet metric tonne
WHMIS	Workplace Hazardous Materials Information System
Western Prospector	Western Prospector Group Ltd.
WHO	World Health Organization
WMP	Water Management Plan
World Wide	World Wide Minerals Ltd.
WRSF	Waste Rock Storage Facility
yd^3	cubic yard
yr	year

5 Reliance on Other Experts

5.1 Consultants and Contributors

In the compilation of this Study, Aker Solutions has relied on the contributions of a variety of specialist consultants who have provided reports and studies for the Report and for the related DFS. Aker Solutions has not audited these reports.

The financial analysis that Aker Solutions have prepared is based, in part, on commodity prices and Owner's cost estimates provided by Khan which have not been audited by Aker Solutions.

Permitting status and the present status of mining rights and areas cited in this document have been provided by Khan and which have not been audited by Aker Solutions.

Assistance and information was obtained from the following experts.

Table 5-1
List of Consultants and Contributors

Scott Wilson Roscoe Postle Associates	Geological database validation and estimation of Mineral Resources
Golder Associates Ltd.	Geotechnical studies, water management plan, residue (tailings) management plan, conceptual closure plan
P&E Mining Consultants Inc.	Open-pit and underground mine design, estimation of Mineral Reserves
Khan Resources Inc.	U_3O_8 pricing, Owner's cost estimates, tax information, land title status, current permitting status, overview of uranium industry, royalties
SGS Minerals Services	Metallurgical testing, mineralogical studies
AATA	Environmental and social baseline conditions and impact assessment

5.2 Qualifications of Consultants

The individuals who have provided input of this Report and who are listed in Table 5-2 have extensive experience in the mining industry or in supporting capacities in the industry.

Table 5-2
Key Project Personnel

Hrayr Agnerian, P.Geo.	Geology, resource estimate
Eugene Puritch, P.Eng.	Open-pit mining, reserve estimate
Malcolm Buck, P.Eng.	Underground mining, reserve estimate
Les Heymann, P.Eng.	Mineral processing

6 Property Description and Location

The Dornod Project is located in northeastern Mongolia, approximately 125-km north of Choibalsan, capital of the Dornod Aimag (province) (Figure 6.1). The population of Choibalsan is about 15,000, and it is situated along a major east-west road connecting the town with Ulaanbaatar, the capital of Mongolia, some 650 km to the west. The abandoned settlement of Mardai, built for Russian mineral exploration crews, is 14-km west of the Project.

The Dornod Property consists of two mineral licences, a Mining Licence (237A, originally U-27) and an Exploration Licence (9282X). Mining Licence 237A was granted by the OGMC, of the Minerals Resources Authority of Mongolia, to CAUC, a limited liability company organized under the laws of Mongolia. Khan, through a subsidiary corporation, holds 58% of the issued and outstanding common shares of CAUC (Lynch, 2004).

An application to convert the exploration license to a mining license was submitted in September 2007. The application included the August 2007 Pre-Feasibility Study. Exploration License 9282X, known as the additional Dornod Property, has an area of 243 ha and is contiguous with the Main Dornod Property. It is registered through a wholly owned subsidiary of the Corporation, and was renewed for a 3-yr period in February 2008. The corporation is currently taking all necessary steps to convert the exploration license into a mining license, in accordance with the Revised Minerals Law of Mongolia (RMLM). To this end, the Corporation has recently submitted the reserve calculation and environmental impact assessment for the Additional Dornod Property, prepared in accordance with Mongolian standards and requirements. These are necessary preconditions in the process of converting an exploration license to a mining license in accordance with the RMLM.

Mining Licence 237A is a rectangular block (1.5 km by 1.74 km) and Exploration Licence 9282X is a narrow strip of land adjacent to the north, east and south of Mining Licence 237A. Together, they cover an area of approximately 504 ha (Figure 6.2). On January 27, 2005, Khan agreed to acquire Exploration Licence 9282X from Western Prospector Group Ltd. (Western Prospector) by paying 400,000 shares in the capital of Khan to Western Prospector. In March 2005, Khan's wholly owned Mongolian subsidiary, Khan Resources Ltd. (Khan Mongolia) acquired the Mineral Licence 9282X (Additional Dornod Property), which is adjacent to the existing Dornod Property. The Additional Dornod Property contains approximately one-third of the No. 7 Deposit and contains part of the No. 5 Deposit with reported Mineral Resources. As part of the agreement, Western Prospector also retained a 3% Gross Royalty on any discovery made in the area of Exploration Licence 9282X. Both licences are situated within a larger original block (U27) of 2.65 km (east-west) by 3.63 km (north-south). The latter block covers an area of approximately 962.5 ha. The geographic co-ordinates of the property are listed in Table 6-1.

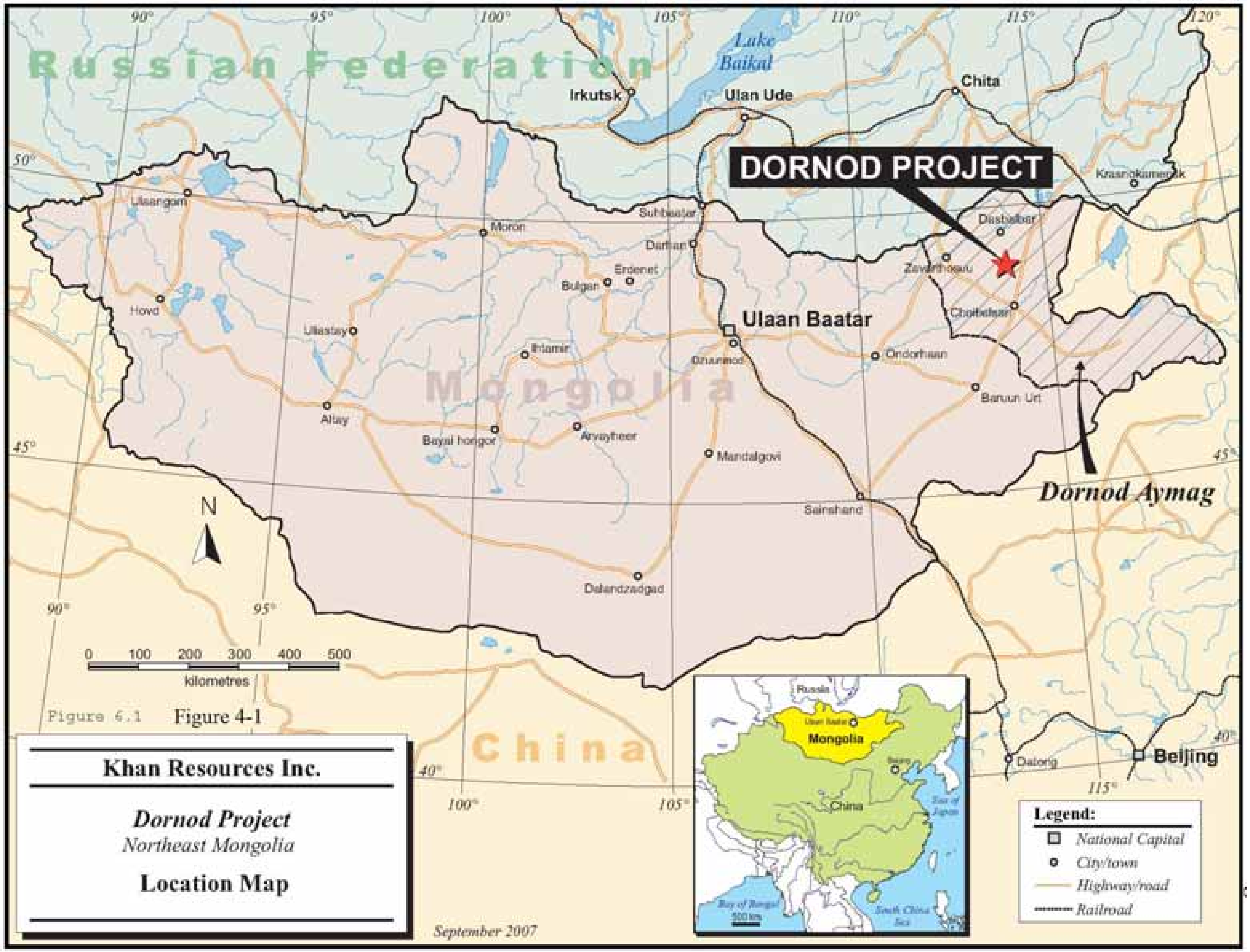


Figure 6.1 - Location Map

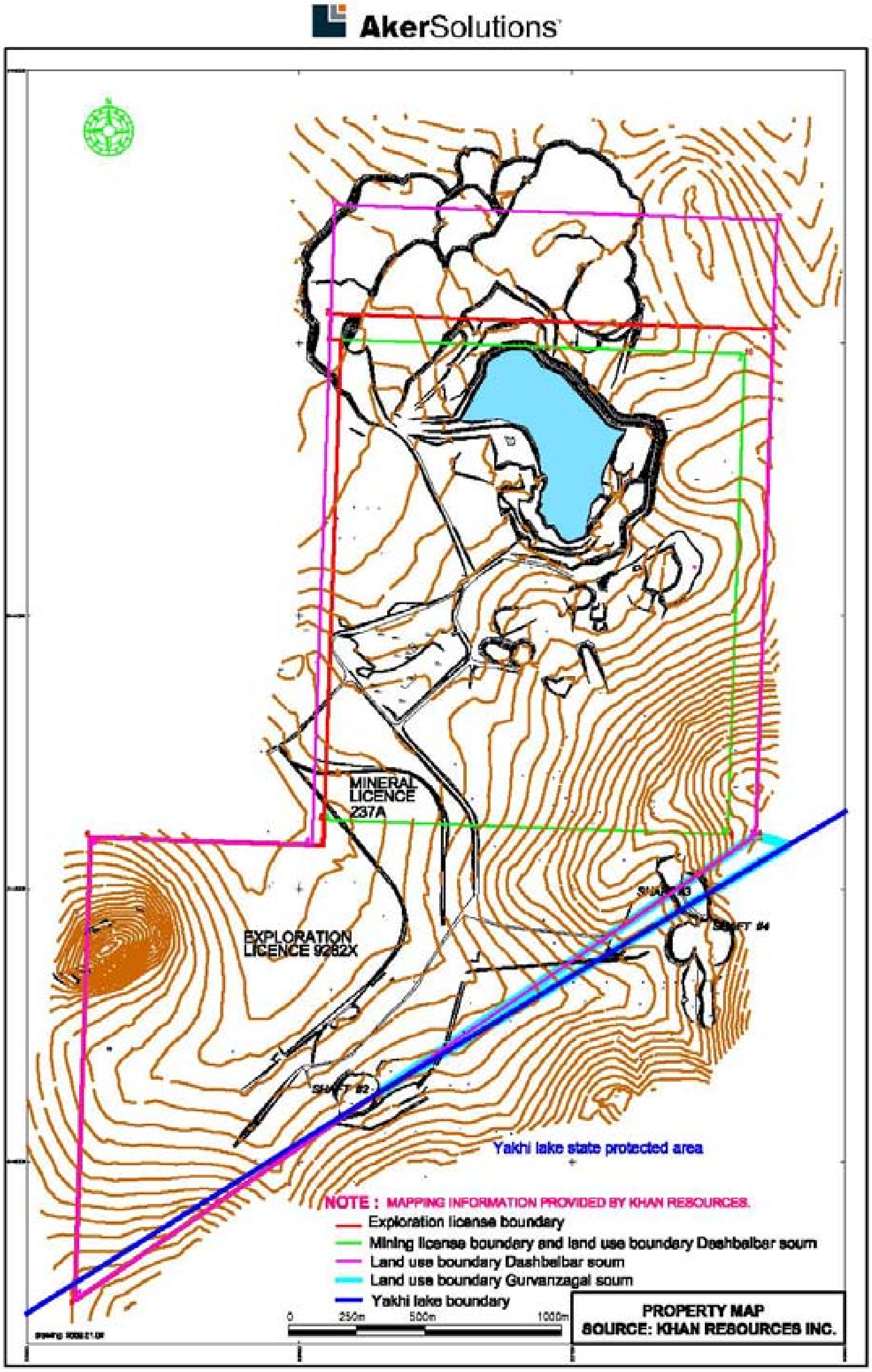


Figure 6.2 - Property Map and Location of Mineralized Zones

Table 6-1
Geographic Coordinates of Mineral Licences

Mineral Licence 237A

Corner	Latitude	Longitude
1 (NW)	49°06’25” N	114°20’42” E
2 (NE)	49°06’25” N	114°21’55” E
3 (SE)	49°05’28” N	114°21’55” E
4 (SW)	49°05’28” N	114°20’42” E

Exploration Licence 9282X

Corner	Latitude	Longitude
1 (NW)	49°06’28” N	114°20’40” E
2 (NE)	49°06’28” N	114°22’00” E
3 (SE)	49°05’28” N	114°22’00” E
4 (SW1)	49°04’49” N	114°20’40” E
5 (SW2)	49°04’30” N	114°20’00” E
6 (SW3)	49°05’25” N	114°20’00” E
7 (SW4)	49°05’25” N	114°20’40” E

Source: Mineral Resources Authority of Mongolia, 2004.

The original Licence (U-27) was granted to CAUC by the Minister of Agriculture and Industry on June 12, 1997, for an initial period of 15 years. The Licence was reregistered as of September 26, 1997, as Licence 237A following the enactment of the Minerals Law in July 1997. Pursuant to the Minerals Law, a mining licence is granted for an initial period of 60 years and the holder may apply for one extension of the licence for an additional period of 40 years (Mineral Resources Authority of Mongolia, 2004 and Lynch, 2004). Therefore, Licence 237A is in good standing until July 2057, provided that licence fees are paid.

On August 31, 2003, Khan acquired all of the issued shares of Khan Resources Bermuda Ltd. (Khan Bermuda), which had earlier (effective July 31, 2003) acquired all of the issued shares of CAUC Holding Company Limited (CAUC Holdings, formerly World Wide Mongolia Mining Inc.) that owns 58% of the issued shares of CAUC. Khan reports that Khan Bermuda, by acquiring 100% of the issued shares of CAUC Holdings, has legally and effectively acquired indirect ownership of a 58% of the issued shares of CAUC, which had retained its 58% interest in the Dornod Project (Blake, 2004).

6.1 Regulatory Framework

This description of the regulatory framework for mining in Mongolia was originally prepared by Golder for the Scott Wilson RPA Technical Report dated September 27, 2007, and has been reviewed by Khan for the DFS.

6.1.1 Permitting

The permitting process in Mongolia consists of the following main components.

- Review and approval by various Mongolian ministries of the general mine plan prior to mine construction

- Review and approval of all aspects of the mine construction and operation by responsible parties representing the Company, such as the environmental consultants, the Chief Design Engineer, and the Manager of Construction, during mine construction and following mine start up

- Review and approval of design and construction field verification by various state (Aimag) agencies, such as Environment and Ecology Regulator, the Fire Safety Inspector, and the Building Inspector, during mine construction and following mine start up (WWUH, 1998b)

- Mongolian ministerial review

- Top-level Government approval of the mine occurs in five ministries:

  - Ministry of Agriculture and Industry
  - Environmental and Ecology Ministry
  - Infrastructure Ministry
  - Ministry of Finance
  - Health and Labour Ministry.

6.1.2 Exploration to Mining

The MRPAM is the authority that oversees mining and exploration licensing in Mongolia. To change a license from exploration to mining, the company must submit:

- Mineral resource / reserve approved by the Minerals Council
- Feasibility study approved by the Mining Department of MRPAM
- Mongolian EIA approved by the Ministry of the Environment.

6.1.3 Company Technical, Engineering and Environmental Review

Prior to final mine approval by the Aimag (state) offices, all of the technical consulting, engineering, and construction management representing the mine must certify that their studies and design work are complete and correct, and that the facilities have been constructed in conformance with the plans. This group will include the Mine Engineering Capital Works Manager, the Design Engineering Manager, the Mechanical Engineering Manager, the Chief Power Engineer, and the environmental consultants used by Khan. The state office representatives will rely on the certifications by these technical experts in making their review and approval of the mine.

6.1.4 Aimag Approvals

During mine construction, but prior to mine start up, the following Aimag reviews and approvals will be completed to ensure that the mine meets all applicable regulations.

- Department of Construction and Architecture will approve designs of all construction and subsequently approve as-builts projects.
- Department of Registration, Supervision and Standards will approve designs for technical and engineering criteria.
- Aimag Agriculture and Industry Regulator will approve the general mine plan.
- Aimag Environmental and Ecology Regulator will approve the EIA and the environmental protection program.
- Aimag Infrastructure Regulator will approve the infrastructure operations plan.
- Aimag Health and Labour Regulator will approve the employee health and safety plan, the standard operating procedures, job descriptions and personnel protection program.
- Aimag Fire Safety Inspector will approve the fire protection plan.
- Aimag Emergency Preparedness Inspector will approve the emergency preparedness plan for dealing with industrial and natural disasters.
- Aimag Building Inspector will approve the on-site construction.
- Aimag Doctor of Industrial Health will approve the medical programs, certify the medical personnel and the medical facilities.

In addition, the local Soum Government has land use approval rights. Any consumables such as water, gravel, sand, etc., used in the on-site construction, as well as off-site construction (roads, power lines, etc.), will require a permit from the local Soum (county) authority.

Although each approval is required, the two most significant reviews will be from the Environmental and Ecology Inspector, and the Health and Labour Regulator. Plans for the safe handling and containment of industrial chemicals and for worker protection around the facilities using these chemicals will be required.

As well, any off-site construction requires special permits and environmental approvals. Extension to the proposed power line will require a site specific EIA, as well as approval by the local Soum Government(s) and the central Energy Management Authority. Road and rail construction similarly require site-specific EIAs, as well as approvals by the local Soums and Aimags, as well as the respective authority for roads and railways.

Prior to mine start up, but as part of the operations, approvals must be received for the proper storage, transportation to the site and handling on the site of the following consumables:

- Explosives: special licenses required by employees who handle explosives; State Police agency will also assume an inspectorate role with respect to explosives
- Fuels
- All chemicals used for mill operation including tailings operations.

Although a Material Safety Data Sheets (MSDS) program does not presently exist in Mongolia per se, one will be introduced at the mine, as well as an overall Workplace Hazardous Materials Information System (WHMIS) program consistent with programs in Canada. WHMIS training will be part of mine operations. It is expected the introduction of this program will be adopted by Mongolian authorities for future mine and industrial operations. As a result, future permitting will depend on maintaining MSDS and satisfying WHMIS.

6.1.5 Radiation Monitoring and Yellowcake Transportation

As part of underground activities and eventual mining, the operations will be subject to meeting requirements for operations within a uranium environment. At present, Mongolia is in the process of developing standards for mining uranium. It is expected the Nuclear Regulatory Authority and its inspectorate branch will require the operator to include in its mine plan detail the recovery, handling and processing methods, which will ensure exposure levels meeting international accepted standards to humans.

As for the safe handling and transportation of yellowcake, it is expected Mongolia will adopt the standards established by the International Atomic Energy Authority (IAEA) for the safe handling and transportation of yellowcake between the mine and its next destination.

6.1.6 Applying for a Mining Permit

According to the Minerals Law renewed in July 2006, only an area licensed with a special exploration permit can be subject to an application for a special mining permit by the special exploration permit holder. Such holder is entitled to apply for a special mining permit which can cover any portion of the area secured by its special exploration permit though the area that is applied for shall not exceed 25 ha and meet the requirements provided in laws. (Article 24, Minerals Law of Mongolia).

A special exploration permit holder needs to file the following documents and copies of documents in support of its application for a special mining permit. (Article 25, Minerals Law).

- A notarised copy of Company Registry Certificate (if a foreign invested company, a notarized copy of the certificate of foreign invested legal entity)
- A filled-in form specifying the name and address of the company as well as the name of competent person in decision-making position (the forms are provided by the Cadastre Office)

- Map of the area applied for (description of the location and the number of hectares)
- A document confirming the payment for services (USD 1,000)
- Protocol of a meeting of Minerals Council and a decision of state administration agency as to the approval by the said authorities of the results of exploration efforts in the area
- An EIA study
- A document proving proper execution of the environmental protection plan during exploration works (it is given by the governor of sub-provincial authority, Soum Governor).

According to the law, a special mining permit for an area explored through State budget funding or an area legally surrendered is granted through tender. A legal entity who secured an area through tender is required to attach to its application for a permit the decision of the organisation that conducted the tender, in addition to the documents specified above.

In clarifying the issue on EIA, Article 4 of Law on Environmental Impact Assessment requires that a general EIA be conducted prior to any production at a mining permit area. In order to realise this, necessary documents such as a feasibility study, a work plan and document on technological solution must be submitted to the Ministry of Environment, which is responsible for carrying out general EIAs. If a qualified expert commissioned by the Ministry finds that detailed assessment is necessary after his/her initial assessment, the Ministry will have an authorised legal entity conduct a detailed environmental impact assessment.

An environmental scoping report was submitted by Khan to The Mongolian Ministry of Nature and Environment (MNE) in February 2007. The Ministry issued their Screening Decision on April 2007. This decision concludes that a comprehensive ESIA has to be undertaken and the summary report is to be submitted to the MNE by the end of 2007.

6.1.7 Status

To date, all permits and licenses are in place for the program presently underway. All licenses for the properties are in good standing.

The Project status and schedule is dependent on the company obtaining an investment agreement from the Mongolian Government. At present, the company has hired a Mongolian legal firm and a Mongolian Country Manager to support the senior management activities in pursuit of this agreement. The process of obtaining an investment agreement involves a formal request submitted to the Minister of Finance. A working committee will then be established consisting of representatives of the Minister of Finance, Environment and Industry and Trade. It is expected there will also be a representative from the Nuclear Regulatory Authority as this is a uranium mine. Once in place, formal negotiations begin to draft an agreement. The approval route for this agreement is dependent on the amount of capital necessary to develop the Project. It is expected this process will commence in the fourth quarter of 2009. Khan expects this process will be finished and approved by the first quarter of 2010. It is not known at this time what impact these negotiations will have on the existing ownership structure.

Renewal of the Mining License requires the submission of a report of planned activities for the upcoming calendar year to be submitted by September of the previous year. This year, the report will be based on the Preliminary Feasibility Study schedule translated into Mongolian. The detail will reflect the expectations of the company to obtain an Investment Agreement in a timely manner. Renewal of the license is subject to timely submission of this report and prompt payment of all fees. To date, the company has met all of its commitments in these matters.

6.1.8 Documents to Construct and Commission a Uranium Mine

According to the Article 35.4 of the Minerals Law, a special mining permit holder can commence mining activities after the new mine has been accepted by a commission appointed by the State central administrative agency in charge of geology and mining. The commission consists of relevant officers from the State Professional Inspection Agency and the Ministry of Environment and geological and mining specialists from the Ministry of Trade and Industry and the Minerals Reserve Authority, relevant officers from local governments and, in case of activities related to radioactivity and activities that use toxic chemicals or dangerous substances, relevant officers from the Ministry of Construction and Urban Development, Atomic Energy Commission, and health and emergency organisations.

A special mining permit holder is to keep the following documents at the mine site after the mine is commissioned.

- Feasibility Study and a mine work plan, the latter must be approved by the Minerals Reserve Agency and the State Professional Inspection Agency
- EIA
- Environmental Protection Plan
- An act or document on borders and border marks of mining claim area
- Contract on land and water usage
- Contracts on capital rent and on sales of products
- A notarised copy of the Company Registry Certificate.

Within 3 months following the registration of special mining permit in the registry of permits, the permit holder is obliged to mark the mine area and deliver an act or documents to the State Professional Inspection Agency.

In commissioning a uranium mine, the following additional documents are required to be submitted to the State Professional Inspection Agency as provided in Article 16.1.1 of the Law on Special Permits for Legal Entities and Articles 8.1.2 and 8.1.2.9 of the Law on Radioactivity Control, in order to obtain special permits to explore, mine, process, enrich, import, export, transport radioactive materials, bury their wastes and rehabilitate mined lands.

- Application for a special permit setting forward the type of activity and duration
- A letter on the purpose and type of activities related to radioactive materials, a Feasibility Study, and professional human reserves
- Specifications and certificates of the equipment to be used to handle radioactive materials
- Document of evaluation by the Atomic Energy Commission on whether or not the facilities meet the standards of radioactivity protection and safety operation.
- Inference of relevant labour safety and health inspection offices as to the adequacy of labour conditions and safety operations at the buildings and facilities where activities will be undertaken
- Inference of Environmental Impact Assessment conducted in accordance with relevant laws on the project that involves activities related to radioactive materials
- A plan on measures to be taken in case of radioactivity emergency
- A decision to appoint a staffer responsible for radioactive safety operations
- Internal rules on radioactive safety operations approved by the Commission
- Document showing that Stamp Duty has been paid.

7 Accessibility, Climate, Local Resources, Infrastructure and Physiography

7.1 Accessibility

Access to the Dornod Property is by paved road, about 100-km east from Ulaanbaatar to the coal mining town of Baganoor, then 550-km east by dirt road from Baganoor to Choibalsan in northeastern Mongolia, and then about 125-km north by dirt road from Choibalsan to Mardai. The main access road to the mine, from the town of Choibalsan, is presently an unimproved dirt road and will have to be graded and maintained to provide year-round access.

Air service between Choibalsan and Ulaanbaatar is available.

A rail line connecting Choibalsan to the Russian Trans Siberian railway had a spur line to the Dornod site, which is now defunct.

7.2 Climate

The climate in the Project area is continental. The average annual air temperature is -1ºC. The average temperature of the coldest month, January, is -20.7ºC and that of the warmest month, July, is 18.7ºC. The maximum air temperature on record is +38ºC; the minimum air temperature on record is -38ºC. On average, there are 189 d/a with the mean temperature above zero. They typically occur between April 10 and October 15. The winter season is approximately 7 months long. The frost depth reaches 1.2 m.

Most of the precipitation occurs in the warm season. On average, there are 52 rainy days in a warm season, of which 16 days have precipitation greater than 5 mm. Average annual precipitation is 250 mm to 300 mm, of which approximately 70% occurs in summer and the remaining 30% occurs in winter.

The predominant wind direction (35% to 40% of the time) is from the north, northeast and northwest. The typical wind speed is 3 m/s to 5 m/s in January and 4 to 6 m/s through the rest of the year. Wind gusts as high as 40 m/s have been recorded.

7.3 Local Resources

The land in the mining area is used mainly by local Mongolian nomads for pasturing of domesticated animals, such as horses, cattle, sheep, goats and camels. Most parts of the Mardai area remain uninhabited or largely unpopulated due to a shortage of water.

Mining is a major contributor to the economy in Dornod Aimag. Currently, there are 24 operating mines in the Dornod Aimag, with additional exploration activities. Existing mining activities include fluorspar, oil, tin, uranium, gold and silver mines (Eco-Trade, 2006).

7.4 Infrastructure

Power is generated at Choibalsan. A power line is presently under construction and is scheduled to be completed in May 2009. Telephone service is not available at the site. Water is available from wells near the property. Some mining equipment and personnel are available at Choibalsan, Ulaanbaatar, and in northern Mongolia, where a few open-pit gold deposits are being developed. A high-voltage power line connecting Ulaanbaatar and Choibalsan has been proposed by the Government of Mongolia, which should increase the capacity of the local grid to support multiple mining projects in the Dornod area.

7.5 Physiography

The Project lies in a sparsely populated and remote region. The landscape is characteristic of the semi arid high steppe that is typical of the Eastern Steppe. Small conical hills and gently sloping plains, with various species of grasses and rare stands of birch and aspen are common. Permanent surface water bodies, such as lakes, stream or springs are rare to absent. However, seasonal streams and ponds may appear after rainy periods.

The Project is located within a northwest tending valley surrounded by gently sloping hills. The area has a low to moderate topographic relief, with elevations between 900 to 1100 m above sea level (asl). Other than the site features and the remaining supporting transportation infrastructure, the area is largely in a natural and undisturbed state.

7.6 Soils

Soils are comprised mainly of the carboniferous brown soils typical of the Eastern Steppe. At higher elevations, soils are shallow and poorly developed, with little organic matter. At lower elevations and in flatter topography, soils are better developed and contain organic-rich surface layers.

7.7 Seismicity

Based on historical records, the Project area is not likely to suffer from strong earthquake activity. Therefore, it is not anticipated that the underground mine or any surface geotechnical structures will have to be designed to resist seismic events.

8 History

Historic mining and prospecting activities in the Mardai district of northeastern Mongolia, which hosts the Dornod deposit, date back to the 1940s. Early prospecting work led to the discovery of the Dornod uranium deposit and production started from an open pit in 1988. The area is host to numerous undeveloped uranium occurrences. From 1988 to 1995, some 590 000 t of material at an average grade of 0.118% U_3O_8 were mined from the No. 2 Deposit of the Dornod site. The advent of Perestroika in 1985 and the dissolution of the Soviet Union in 1991 led to cessation of mining activity.

In 1995, Priargunsky - on behalf of World Wide Minerals Ltd. (World Wide), a predecessor company to Khan - commenced stripping and mining operations at the No. 2 Deposit as an open-pit mine. Due to low uranium prices, however, the mine was shut down in 1995. Until 2005, the Project had been maintained on a care and maintenance basis. In early 2005, Khan became operator and began a confirmation drilling program on the areas of the No. 2 and 7 Deposits. Results of this program confirmed earlier Priargunsky results and established the continuity of uranium mineralization at the two deposits. Khan commissioned a Scoping Study on Dornod in 2005, followed by a Prefeasibility Study starting in 2006.

For a detailed account of past exploration and development activities, the reader should refer to the 2006 Scoping Study, available on SEDAR.

9 Geological Setting

9.1 Regional Geology

Mongolia is within the Central Asian branch of the Ural-Mongolian Mobile Belt. The Main Mongolian Lineament, an arcuate series of deep-seated faults that extend generally east-west through the mid-section of the country, divides Mongolia into Northern and Southern Megablocks. The Northern Megablock contains four regions of geosynclinal structures. These are:

- The Northern Mongolian Fold System of early Cambrian age
- The Mongol-Altai Fold System of early Paleozoic age
- The Mongol-Transbaikalian Fold System of late Paleozoic to early Mesozoic age
- The Central Mongolian Fold System of late Paleozoic to early Mesozoic volcanic-plutonic intrusive complexes as well as late Mesozoic tectono-magmatic activity.

The Southern Megablock includes the Southern Mongolian Fold System of late Paleozoic metamorphosed eugeosynclinal sediments, the South Gobian Fold System of metamorphosed Precambrian deposits among Paleozoic geosynclinal formations, and the Inner Mongolian Fold System of late Paleozoic volcanogenic eugeosynclinal formations (World Wide, 2002).

The Dornod uranium district is within the North Choibalsan mineral region in extreme northeast Mongolia, in the Northern Megablock at the eastern end of the Central Mongolian Fold System (Figure 9.1).

In the North Choibalsan mineral region, geosynclinal subsidence in the late Precambrian resulted in the accumulation of the continental-volcanic deposits (sandstones, shales and diabase sheets) of the Erdenedavaa Formation. Continued tectonic-magmatic activity during the late Paleozoic era formed plutons of granite, granodiorite, monzonite, syenite, and gabbro-diorite in the region (Figure 9.2).

The Dornod ore district is in the central portion of the Choibalsan-Onon volcanic belt on the north flank of the Dornod volcanic structure. The significant geologic formation in the district is the late Jurassic Dornod Complex, a series of volcanic-sedimentary strata, 1000-m to 1500-m thick. Extensive northeast, northwest, and north-trending faulting created ore-controlling and ore-containing structures throughout the Dornod area (Figure 9.2). The regional stratigraphy is presented in Figure 9.3.

Although uranium mineralization is common throughout the Dornod Complex, economic concentrations of uranium mineralization occur in a narrow stratigraphic interval in the lower part of the complex. Mineralization is most extensive in horizons of porous sedimentary and volcanic rocks usually enriched with organic or sulphide minerals. Deposits are controlled by major zones of steeply dipping fractures of the northerly and northeasterly faults and their junctures with northwesterly faults.

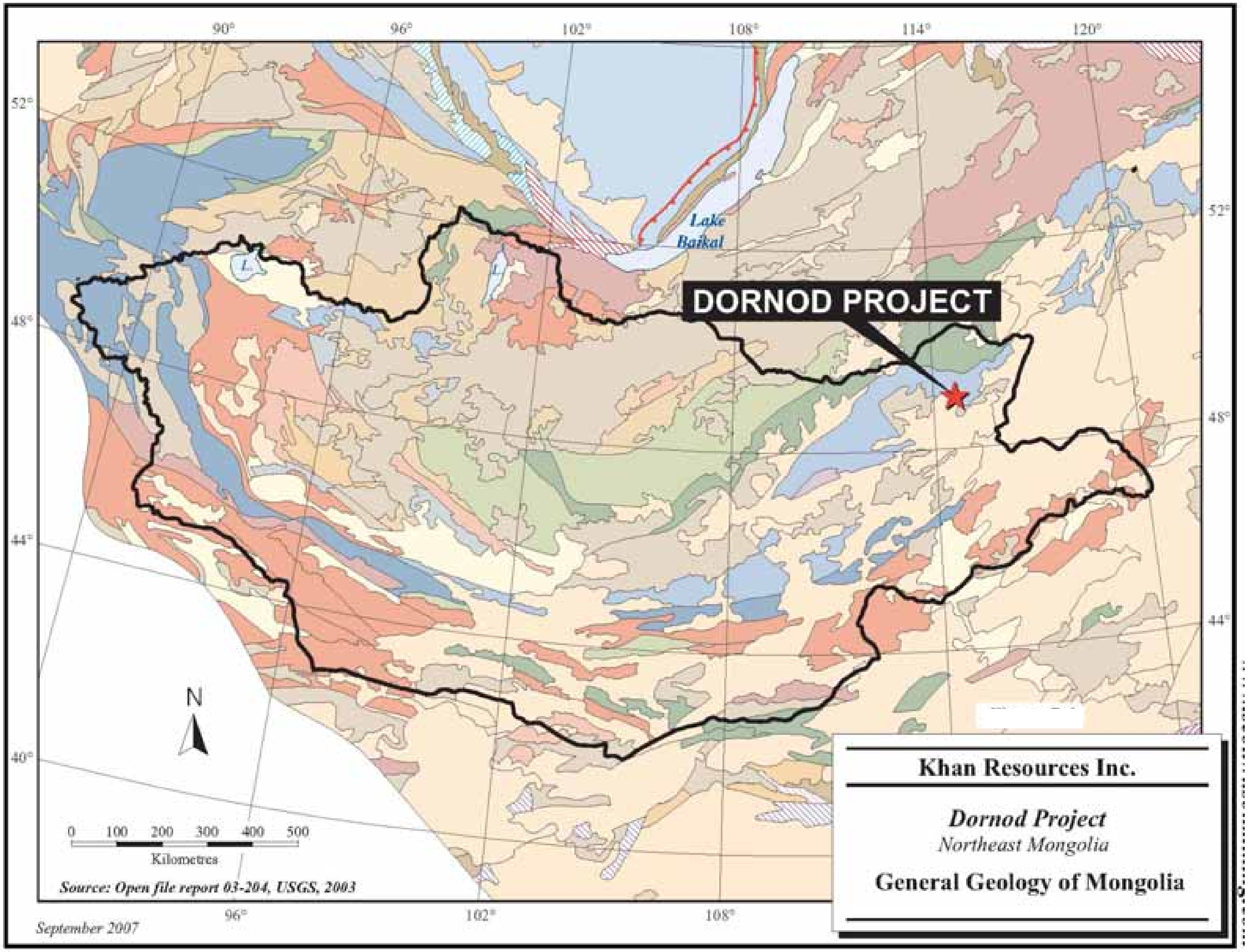


Figure 9.1 – General Geology of Mongolia

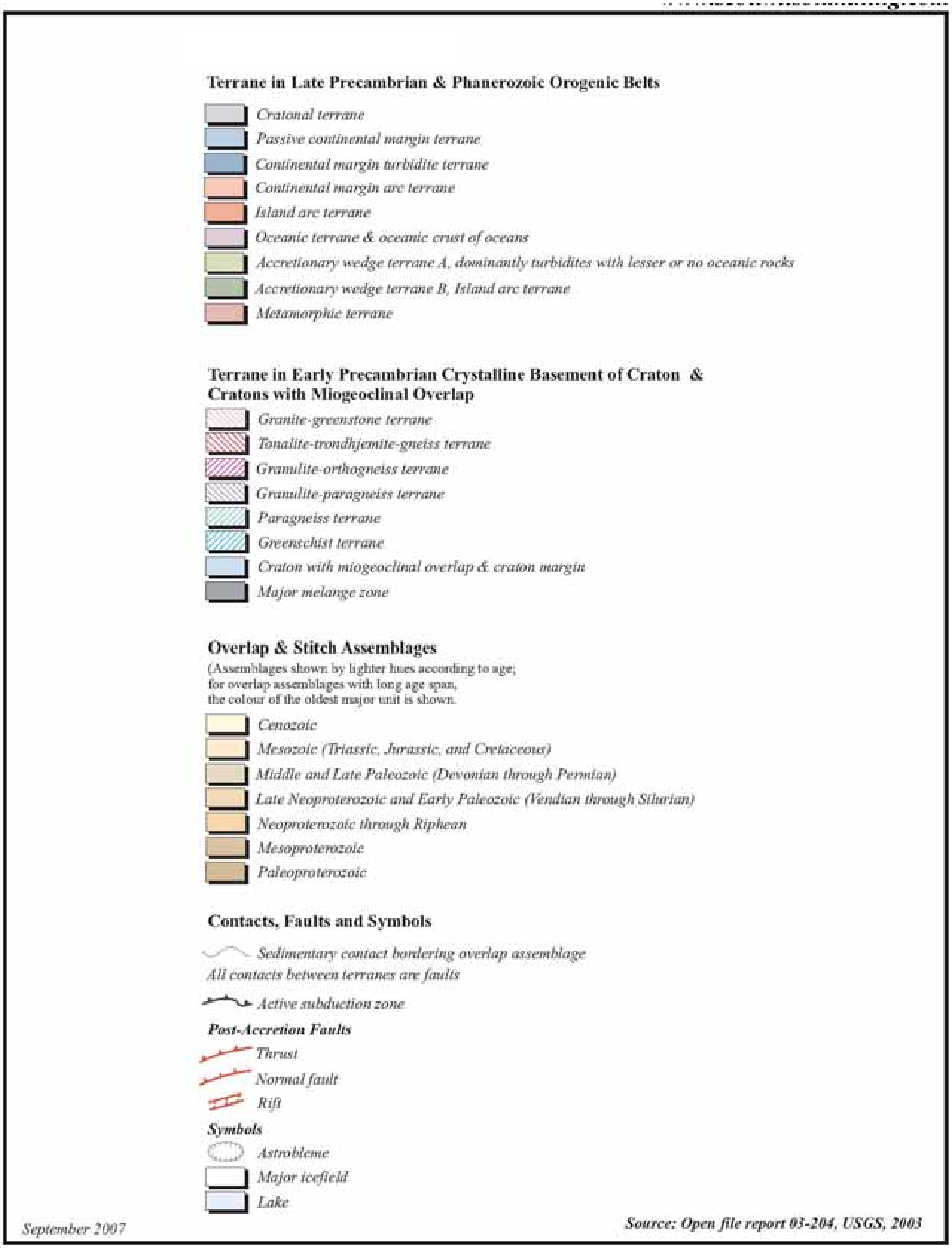


Figure 9.1 – Legend

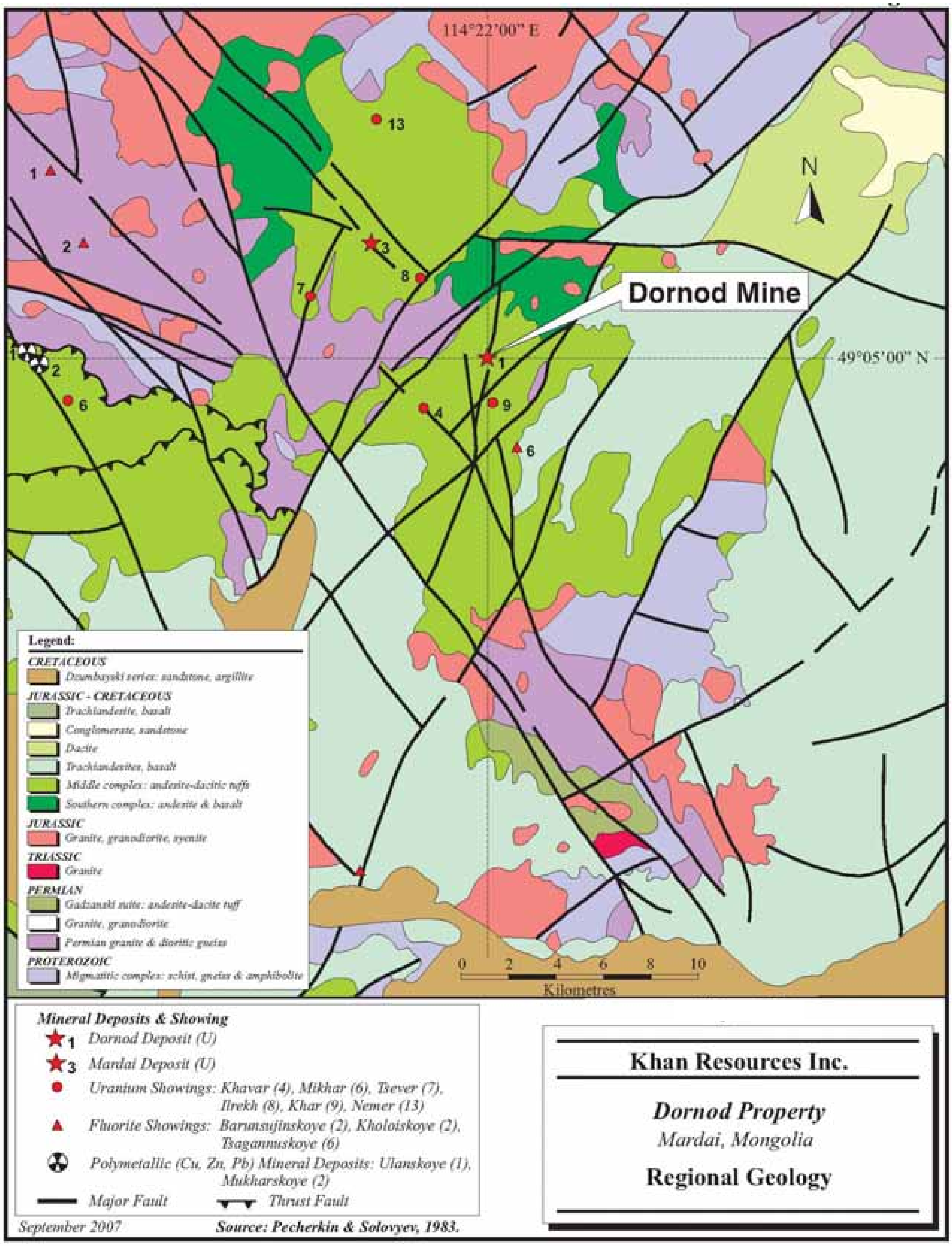


Figure 9.2 – Regional Geology

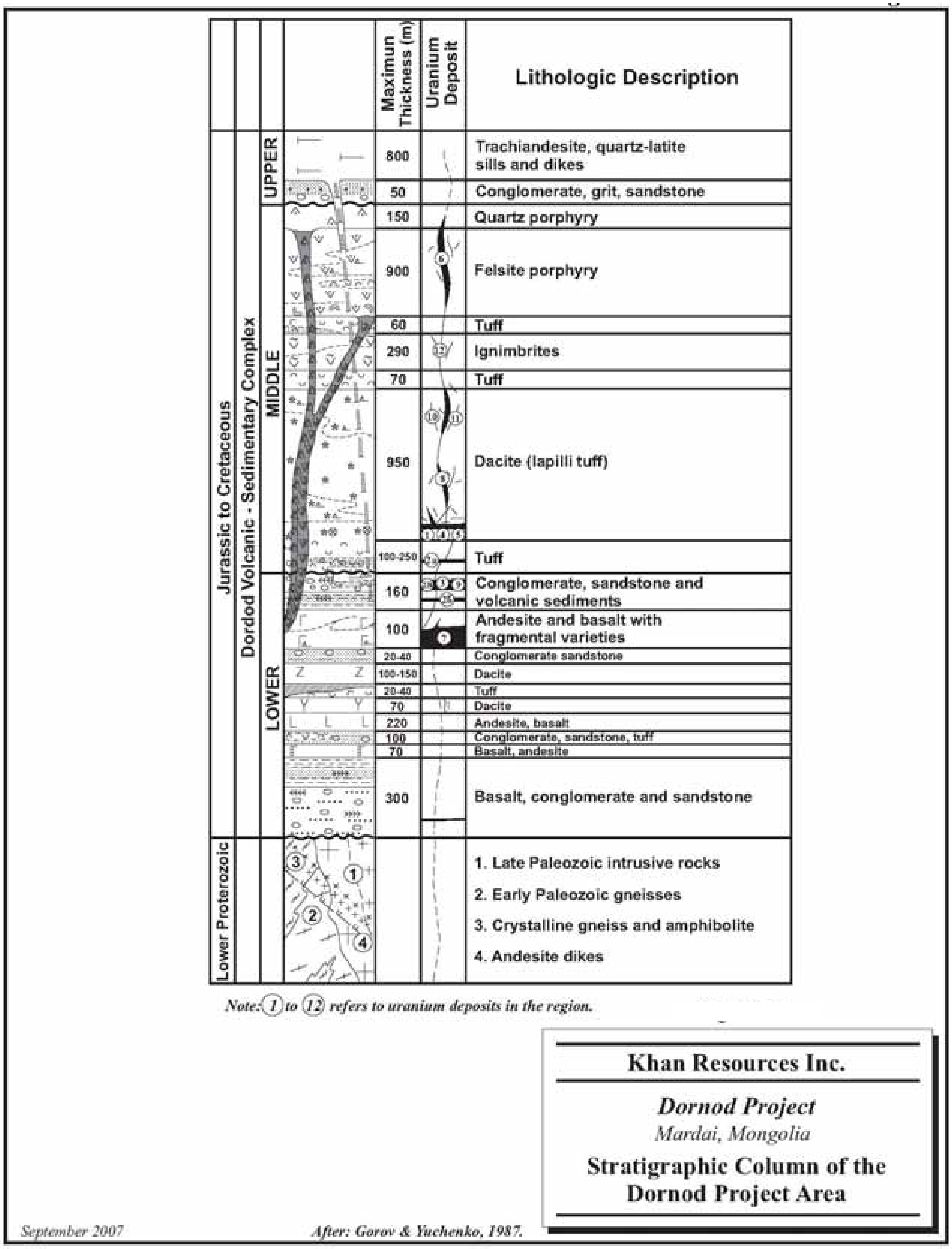


Figure 9.3 - Stratigraphic Column of the Dornod Project Area

Uranium mineralization in the Dornod district is found at depths of 30 to 700 m and is concentrated within a 30-km^2 area. Thirteen deposits have been identified in the Dornod district, of which five have been explored in detail. The No. 7 Deposit, which is the largest, has been partially developed for underground exploration. The No. 2 Deposit has been mined by open pit methods from 1988 to 1995.

9.2 Local and Property Geology

The area of the Dornod Property is underlain by Jurassic volcanic and sedimentary rocks. The volcanic rocks comprise amygdaloidal basalt, andesite, ignimbrite, rhyolite and tuff. The sedimentary rocks are predominantly sandstone and conglomerate containing interbedded carbonaceous partings.

For their September 27, 2007 report, Scott Wilson RPA carried out a new interpretation of the Nos. 2 and 7 Deposit areas, based on the new digital lithologic database provided by Khan. This interpretation is provided below.

9.3 No. 7 Deposit

The flat-lying No. 7 Deposit is situated at the northern end of the Dornod uranium district and occupies the southern half of the area covered by Mining Licence 237A. The deposit is situated approximately 1-km south of the No. 2 Deposit. As with the latter, the No. 7 Deposit was discovered during the 1970s, as a result of a large-scale uranium exploration program jointly conducted by Russian and Mongolian Geological Expeditions.

The No. 7 Deposit is hosted by amygdaloidal and brecciated basalts and andesites within a volcano-sedimentary assemblage as much as 1500-m thick. In the area of the deposit, this assemblage is represented by a series of subhorizontal flow lavas, pyroclastic and sedimentary rocks. The volcanic rocks comprise by far the largest component to the stratigraphic assemblage and include amygdaloidal basalt, andesite, felsites, quartz porphyries, rhyolite breccia and tuff. Sedimentary rocks are predominantly sandstone and conglomerates (Figure 9.3).

The No. 7 Deposit comprises a number of separate uraniferous horizons spread over an area measuring 1000 m by 500 m. The most continuous zone is a 30- to 40-m thick tabular body of high-grade uranium mineralization occurring at vertical depths between 410 m and 450 m below surface. Lithologic logging indicates an approximately 450-m to 680-m-thick stratigraphic sequence of rocks (Figure 9.3). From top to bottom, these are as follows:

- Upper rhyolitic layer, 50-m to 100-m thick
- Upper andesitic layer, 35-m to 40-m thick
- Main rhyolite, 200-m to 350-m thick
- A layer of interbedded mudstone and sandstone, up to 20-m thick
- A layer of basaltic rocks, 20-m to 40-m thick
- Main mineralized zone: A 30-m to 40-m-thick zone of interlayered sandstones and siltstones

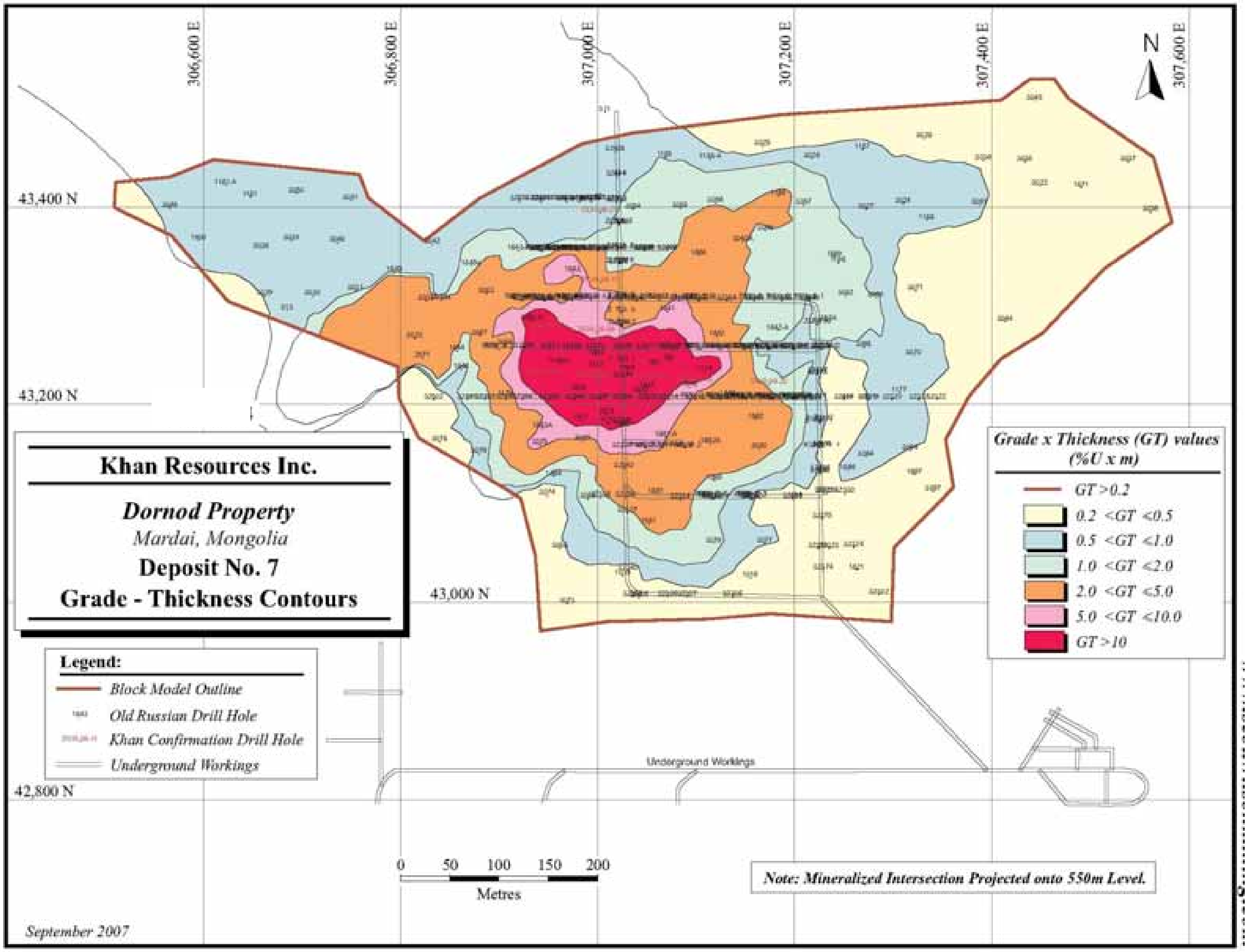
- A layer of pebble conglomerate, up to 10-m thick
- Lower mudstone layer, up to 60-m thick
- Lower conglomerate unit, up to 20-m thick.

Figure 9.4 shows the grade x thickness (% U*m) values of the mineralized intersections and Figure 9.5 is a generalised cross-section of the No. 7 Deposit. At the northern part of the deposit, the No. 7 Deposit is approximately 1000-m wide and, at the southern part, it is approximately 400-m wide, with a high-grade core centered around grid coordinates 307,000 E and 43,225 N (Figure 9.6).

9.4 No. 2 Deposit

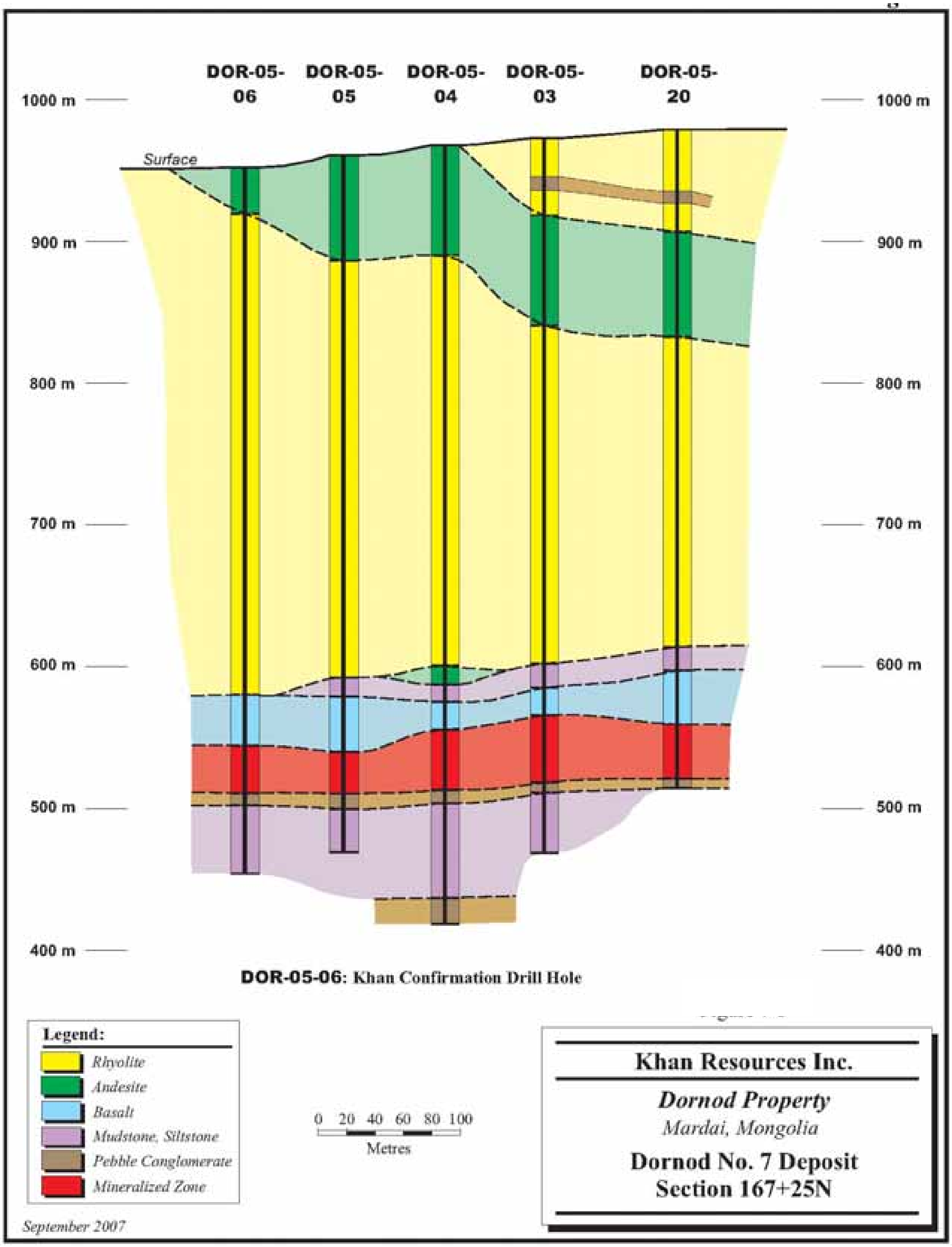
The No. 2 Deposit comprises a number of separate uraniferous horizons spread over an area measuring approximately 1800 m by 1500 m. There are at least five horizons of sedimentary rocks hosting uranium mineralization, which are interlayered with felsic to intermediate volcanic rocks. The most continuous zone (Layer 3) is a 6-m to 10-m-thick layer of low-grade uranium mineralization which is stratabound and defines the broad southwest trending synform in the area. This layer occurs at vertical depths between 75 and 225 m below surface. Figure 9.7 is a generalised cross-section of the No. 2 Deposit. Lithologic logging indicates an approximately 120-m to 250-m-thick stratigraphic sequence of rocks. From top to bottom, these are as follows:

- Rhyolitic layer, 5-m to 50-m thick
- Tuff, 2-m to 15-m thick
- Brecciated rhyolite, 50-m to 85-m thick
- Tuff, 10-m to 25-m thick
- Sandstone, 0-m to 2-m thick; in places it hosts the No. 1 mineralized layer, which is discontinuous
- Massive rhyolite, 10-m to 25-m thick
- Tuff, 15-m to 30-m thick; lower contact is generally mineralized and hosts No. 2 mineralized layer
- Mudstone / siltstone, 5-m to 10-m thick
- Main (No. 3) mineralized layer, generally hosted by sandstones, 2-m to 40-m thick, in places also includes conglomeratic layer
- Sandstone, 2-m to 5-m thick, generally hosts the No. 5 mineralized layer
- Andesite, 20-m to 25-m thick
- Mudstone / siltstone, 0-m to 30-m thick; in places includes No. 7 mineralized layer
- Granite.



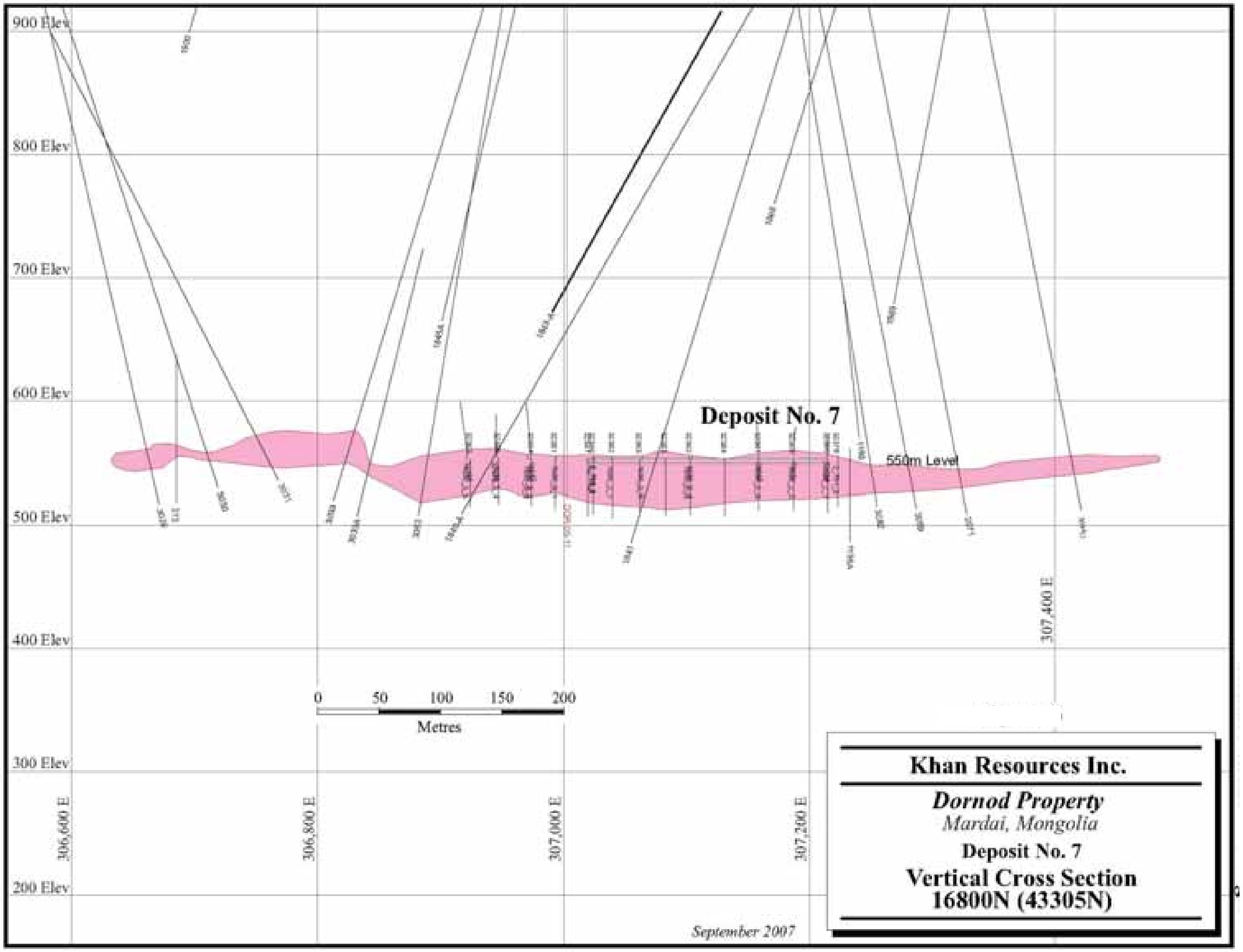
Source: Scott Wilson RPA, September 2007.

Figure 9.4 – Deposit No. 7 Grade – Thickness Contours



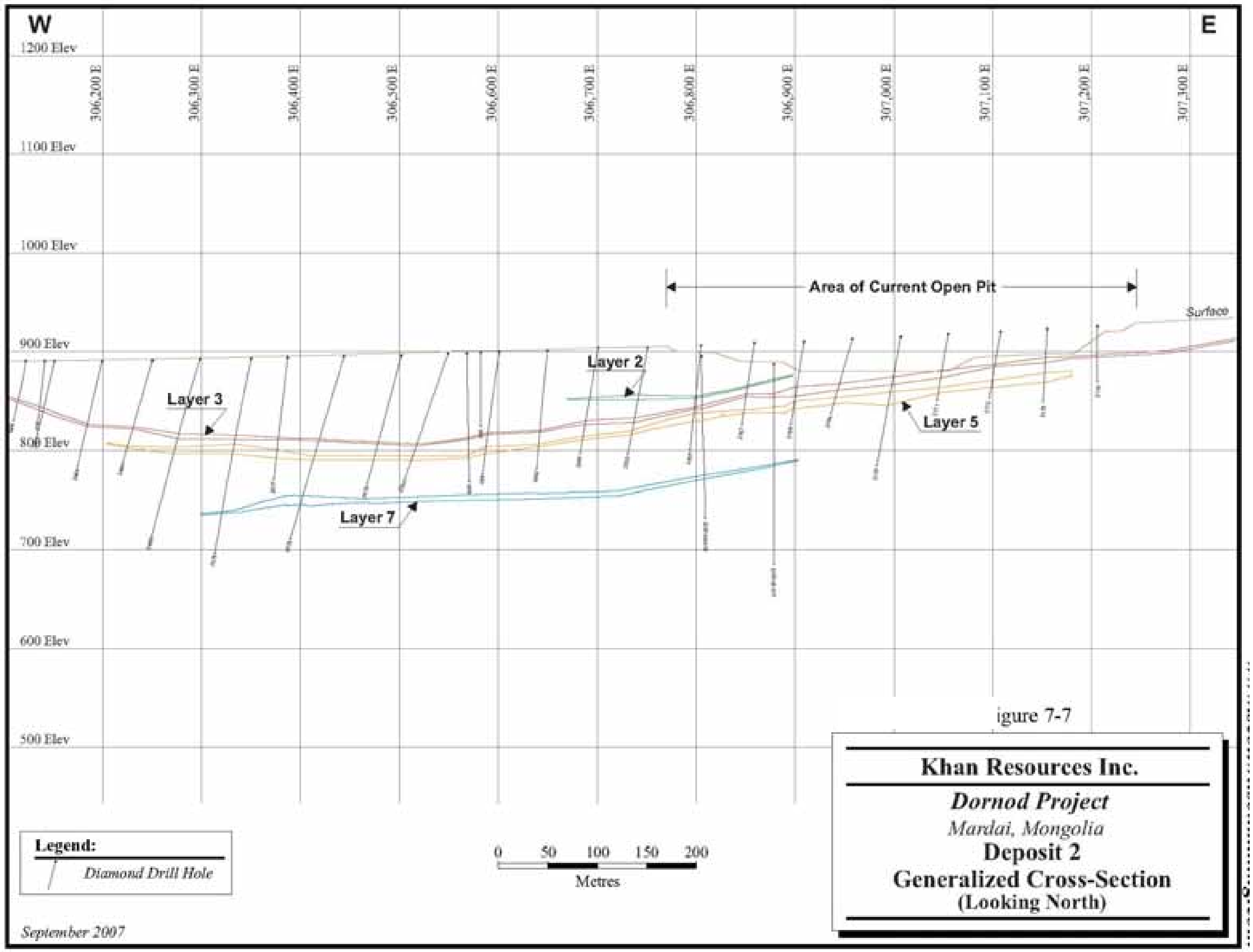
Source: Scott Wilson RPA, September 2007.

Figure 9.5 – Dornod No. 7 Deposit, Section 167+25 N



Source: Scott Wilson RPA, September 2007.

Figure 9.6 – Deposit No. 7 Vertical Cross-Section 16800N (43300 N)

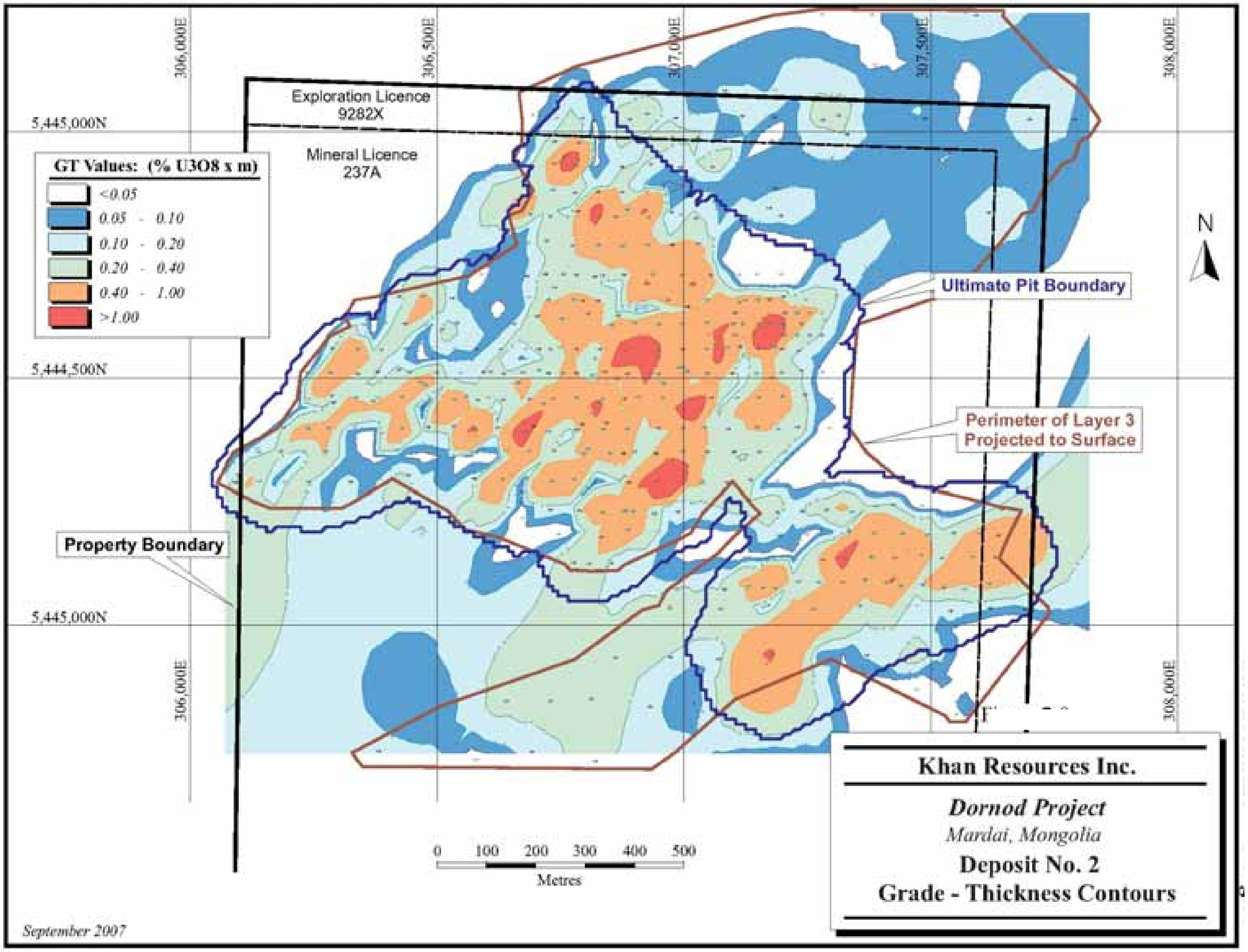


Source: Scott Wilson RPA, September 2007.

Figure 9.7 – Deposit No. 2 Generalized Cross-Section (Looking North)

The No. 2 Deposit has been mined in the past as an open pit, which is currently full of water. Most of the mining activity in the past has been directed towards Layer 3. Since 1995, there has been little work completed on this deposit, except for the recent confirmation drilling by Khan. Occasional normal faults cut the sequence of rocks and the mineralized layers, such as one interpreted to be present southeast of the existing open pit.

Drill results indicate that the various mineralized layers reflect the general basement topography in the area, as a wide topographic low (basin) which has been filled by Jurassic sedimentary rocks, except at the margin of the basin where these layers are at an angular disconformity with the underlying granitic rocks (Figure 9.8).



Source: Scott Wilson RPA, September 2007.

Figure 9.8 – Deposit No. 2 – Grade – Thickness Contours

10 Deposit Types

The Dornod Nos. 2 and 7 Deposits are hosted by Jurassic conglomerates and sandstones interlayed with andesitic and basaltic volcanic rocks as well as mudstones. As such, they are classified as sedimentary uranium deposits.

11 Mineralization

Uranium mineralization in the Dornod Mine area is hosted by Jurassic volcanic and sedimentary rocks. Mineralization occurs as pitchblende-coffinite assemblages associated with carbonaceous partings and fragments in areas of structural preparation. The uranium mineralization occurs as "blanket-like" horizons from less than 1-m thick to greater than 30-m thick within the volcano-sedimentary succession at depths from 30 m to greater than 450 m below surface. A number of uranium deposits and target areas have been outlined in the Dornod area by systematic exploration work.

Mineralization within the No. 7 Deposit is confined to a 30-m to 40-m-thick zone of interlayered sandstones and siltstones within brecciated andesite and basalts that form the lower horizon to a sheet-like andesite-basalt. Within the brecciated andesite, uranium mineralization is commonly associated with thin slivers and discontinuous seams of carbonaceous material that is incorporated in the quartz-carbonate cement and as disseminations and small impregnations within highly strained and cracked volcanic blocks and fragments.

A zone of high-grade uranium mineralization (main mineralized unit), surrounded by a halo of lower-grade mineralization, has been identified in a single thick horizon at the No. 7 Deposit. This horizon is up to 40-m thick and is continuous for about 400 m along strike (north-south) and up to 400 m in an east-west direction.

The No. 2 Deposit comprises a number of separate uraniferous horizons spread over an area measuring 1800 m by 1500 m. There are at least five horizons of sedimentary rocks hosting uranium mineralization, which are interlayered with felsic to intermediate volcanic rocks. The most continuous zone (Layer 3) is a 6- to 10-m-thick layer of low-grade uranium mineralization which is stratabound and defines the broad southwest trending synform in the area. This layer occurs at vertical depths between 75 m and 225 m below surface, as noted in Item 9, Figure 9.7.

12 Exploration

12.1 Previous Work

Early exploration on the Dornod Property was carried out by the Russians, as discussed in Item 8, History and in the 2006 Scoping Study.

12.2 Recent Work

In 2007, Khan carried out a combined ground magnetometer and gravity survey over the entire Dornod Property. The objective of this survey was to identify geophysical characteristics of the mineralized horizons and detect hitherto untested target areas. Results of this survey indicate the following:

- In general, the area of Mineral Licence 237A is underlain by rocks exhibiting relatively high magnetic susceptibilities, which also correlates with an area of gravity high (not corrected for terrain).

- In general, the area of the Exploration Licence (Additional Dornod Property 9282X) is underlain by rocks exhibiting relatively low magnetic susceptibilities, which also correlates with an area of gravity low (not corrected for terrain).

- The No. 2 Deposit area coincides with a magnetic low within the large magnetic high.

- An area of magnetic low of similar size is situated northeast of the No. 2 open pit. This area corresponds with the 2B-9-C1 area identified in the old Russian reports. It is likely that the magnetic low of the No. 2 Deposit and the 2B-9-C1 area are part of a southwest to south-southwest trending axis.

- The No. 7 Deposit area is situated near the boundary between the magnetic low (in the south) and magnetic high (in the north). However, it contains three small "thumbprint" magnetic highs.

- A number of small magnetic lows are situated within the large magnetic low which covers Exploration Licence 9282X. One of these magnetic lows coincides with the No. 5 Deposit area.

13 Drilling

13.1 Previous Work

The No. 7 Deposit has been explored by some 123 surface diamond drill holes, 143 underground diamond drill holes and some 20 000 m of underground development including drifts, cross-cuts, and three shafts, which extend to the No. 5 Deposit area.

The No. 2 Deposit at Dornod has been explored by some 450 surface diamond drill holes. Systematic testing of the uranium bearing zones was started by Priargunsky in 1977 and continued until 1988.

The State Geological Survey (Russia) carried out lithologic logging on drill core and down hole radiometric logging. Drill-hole collars were surveyed and inclinations recorded at regular intervals (MacCormack, 1998). Detailed logging procedures are not available at this time, but exploration data, such as radiometric logs (digital as well as analog), indicate that the work was comparable to Western industry standards.

13.2 Recent Work

From August 2005 to April 2007, Khan completed a program of confirmation drilling of 5885 m in 23 diamond drill holes. All of the holes were vertical holes. Twelve of these holes were in the area of the No. 7 Deposit, ranging in total depth from 357 to 537 m, and 11 of the holes were in the area of the No. 2 Deposit area, ranging in depth from 145 to 170 m. Scott Wilson RPA notes that these are not twinned holes, rather, drill holes which have tested the areas of mineralization that represent the bulk of the remaining mineral resources of the No. 2 Deposit, and both high-grade and medium-grade areas of the No. 7 Deposit. In general, the new drill-hole collars are within 25 m of the old drill holes completed by the Russians (Figures 13.1 and 13.2). Scott Wilson RPA understands that most of the old drill-hole collars could not be located. Consequently, the new drill holes are located based on coordinates of old drill holes derived from plans, and not from digital data. Scott Wilson RPA also understands that the new drill-hole collars are based on both Universal Transverse Mercator (UTM) coordinate system, as well as the Gauss Kruger-Posgar (GKP**)** coordinate system. This was verified in the field by Mr. John Kita, P.Geo., Khan Chief Geologist, during the 2006 / 2007 field program.

In 2007, Khan continued to test the area between the Nos. 2 and 7 Deposits, as well as the area southeast of the No. 2 open pit, by drilling. In total, some 1987 m of drilling was completed in eight diamond drill holes.

In late 2007, Khan completed two large diameter diamond drill holes and sampled the central part of the No. 7 Deposit for metallurgical testwork.

The procedures used during the diamond drilling programs are as follows.

- Holes are drilled to produce HQ- or NQ-sized core
- The collar locations of all drill holes are surveyed and marked in the field with azimuth. All holes were vertical at the collar.

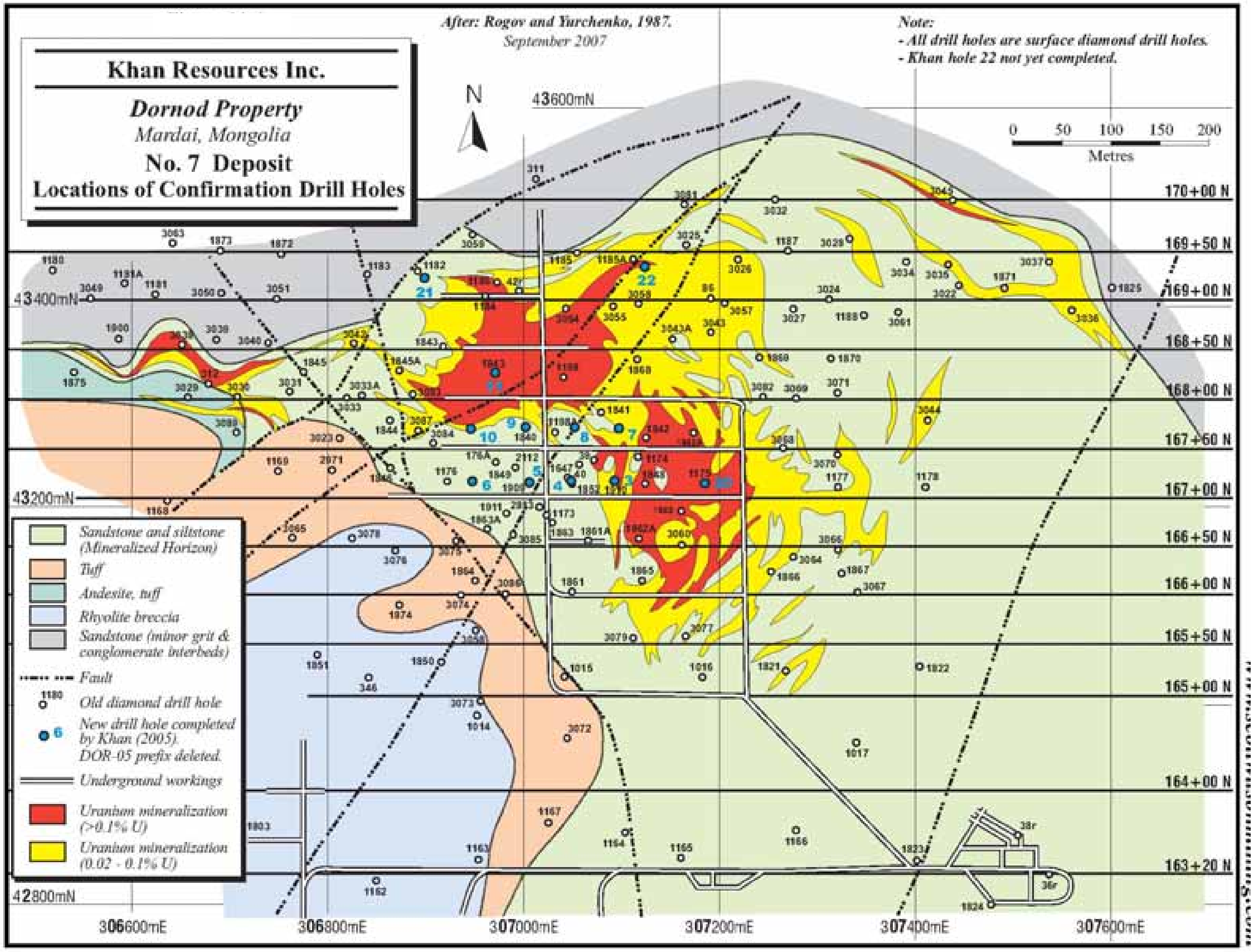


Figure 13.1 – No. 7 Deposit – Locations of Confirmation Drill Holes

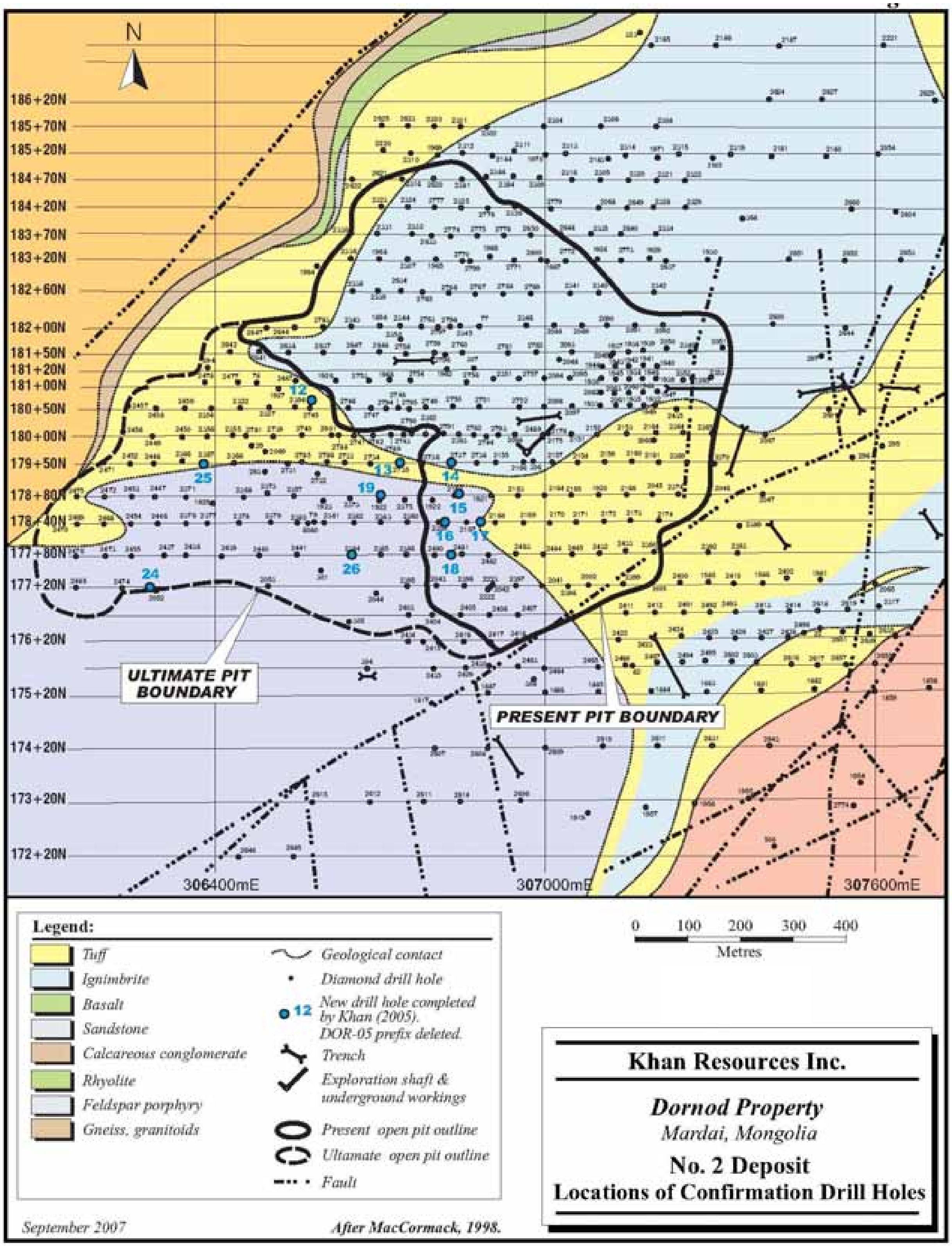


Figure 13.2 – No. 2 Deposit – Locations of Confirmation Drill Holes

- Control information on the directional deviation (both azimuth and change in inclination) is recorded for the drill holes, by use of a Flexit Smarttool Single Shot. Scott Wilson RPA noted that a number of measurements are taken per hole, depending on the hole length, and included in the drill hole data, at a minimum of one measurement at the bottom and top of each hole, and for each 50-m interval down the hole (Agnerian et all, 2007).

- Lithologic logging is done on drill core and geotechnical observations are made by company geologists. This includes marking lithologic contacts, descriptive geology, core angles, core diameter, percent core recovery record, and graphic log depicting all downhole data including assay values. All information is recorded on handwritten logs. Currently, key information is summarised in a digital database.

- Systematic measurements of Rock Quality Designation (RQD) are also included as part of the drill-hole logging.

In August 2007, Khan completed a Prefeasibility Study on the Dornod deposit covering the above issues.

14 Sampling Method and Approach

14.1 Previous Work

Early diamond drilling was completed by Priargunsky. Drill core was logged by company geologists. This included marking lithologic contacts, descriptive geology, core angles, core diameter, down-hole inclination and azimuth readings, percent core recovery record, true thickness calculations and graphic log depicting all down hole radiometric surveys. The methodology of sampling of the drill core and the procedures for sample preparation and determination of the uranium in the rock are described below.

- Continuous radiometric sampling was carried out on all surface drill holes at intervals of 10 cm down the hole. All assays were converted to percent U_3O_8 (from percent U) and consecutive assays of significance were averaged into intercepts above a cutoff 0.013% U (0.015% U_3O_8) over a minimum vertical thickness of 1.2 m. Detailed gamma logs were included with all drill logs.

- Chemical assaying has been completed on core samples from a total of 39 drill holes in the No. 2 Deposit to provide a check on similar sections radiometrically assayed. These analyses were completed at the assay laboratory at the Priargunsky processing facility in Krasnokamensk, Siberia. While there appears to be significant deviation when comparing specific samples, a comparison of radiometric values and chemical assays averaged for the total number of samples checked reveals 0.140% U_3O_8 (radiometric) versus 0.144% U_3O_8 (chemical assay). Results of the check assays in general indicate good agreement between the two sets of data.

14.2 Recent Work

Lithologic logging of the 2006 / 2007 drill holes was done by Derek McBride, Ph.D., P.Eng., Khan Project Geologist, and / or Ms Demchig Oyuntungalag (Tunga), Khan Field Geologist. Field procedures also included:

- Recording of alteration patterns and structural features on the core

- Radiometric logging of the core by a handheld Exploranium SPP2 scintillometer

- Calculation of the RQD value

- Sampling of mineralized intersections of drill core at 1-m intervals.

Mineralized drill core intervals to be sampled were identified and marked by the geologist. Sample lengths varied from 50 cm to 1.0 m. Visual indicators of the intervals to be sampled include lithologic contacts and clay altered rock. The sampling procedure is as follows.

- Sample intervals were marked by sample tags stapled into the core box, and were normally extended for 2 to 5 m into unmineralized rock.

- Flagging tape was used to mark sample intervals prior to sampling. This was because, in places, the unsawed core may mask the sample tags at the bottom of the row in the core box, and thus may cause mixing of samples.

- Prior to sampling, drill core was marked by a line drawn along the core, so that systematically one side of the core was sampled.

- Sample tags were inserted at the beginning of each sample.
- Sample tags were inserted only after the samples have been collected.
- Sample bags were numbered prior to sampling.
- Marked sample intervals were split in half using a diamond saw. A technician collected the sawed core.
- Sample tags were placed into the core box at the end of each sample.
- Permanent marker was used to mark sample intervals on the core boxes, i.e., in addition to the flagging tape.
- Samples were collected in medium sized 20-cm by 30-cm clear polyethylene bags and sealed.

The drill-hole sampling procedures employed by Khan conform to industry standards, in Scott Wilson RPA's view. Nevertheless, Scott Wilson RPA recommends that detailed gamma-ray logging be part of future exploration programs. Scott Wilson RPA was of the opinion that down-hole radiometric logging provides important complementary information to the assay database, and may even substitute for assay values, by calculating the equivalent uranium grade (U_3O_8) of deposits with uranium disequilibrium, as discussed below.

As part of the field program, Khan personnel also collected a suite of mineralized samples for metallurgical testwork. Details of this sampling are discussed in Item 18, Mineral Processing and Metallurgical Testing.

In August 2007, Khan completed a Prefeasibility Study on the Dornod deposit covering the above issues.

14.3 Radiometric Logging and Uranium Disequilibrium

Early diamond drill holes by Priargunsky were radiometrically logged, and regular chemical checks also were done on drill core, as noted above. The following is a discussion on radiometric logging and the general state of disequilibrium at most uranium deposits.

Heavy, long-lived radioactive elements, such as U^{238} decay naturally, producing a series of daughter products, and end up as a stable element, such as lead (Pb^{206}). Since the members of the decay series are different chemical elements, they may be selectively separated from the original element (parent isotope) by geochemical processes. Radioactive equilibrium is attained when all the daughter products disintegrate at the same rate as they are produced by the parent isotopes and all nuclides remain together (Schmeling, 1982). In nature, however, this almost never occurs, as explained below.

Since long-lived nuclides disintegrate at a slower rate than short-lived ones, it is necessary to have more of the slower disintegrating daughters in order to have equilibrium. An ideal state of equilibrium, however, is never attained, because the parent isotopes are subject to decay without replacement; but if the decay constant of the parent is small (the half-life is large) a state of relative or "secular" equilibrium may be attained. Since most detection methods do not measure the parent material, the amount (or quantity) of the parent material is inferred by measuring the radiation from the daughter products. It is important to determine the state of "secular" equilibrium when one estimates the amount of uranium from gamma ray logs. The main sources of the gamma energy from the U^{238} decay series are the daughter products Pb^{214} and Bi^{214}.

Radioactive disequilibrium happens if one or more of the daughter products, or the parent isotope, is completely or partially absent. The various disequilibrium states may be caused by the following.

- Radon, the gaseous member of the uranium series, is easily separated from the rest of the elements in the decay series. Since some of the elements which emit radioactivity are produced after the occurrence of radon, a disequilibrium results which will negatively bias the inferred quantity of the parent U^{238}.

- Recent deposition of parent material either by initial deposition or by remobilisation, i.e., little or no daughter products. This will also cause an underestimation of the quantity of the parent material.

- Estimation of parent material based on measurements on remobilised daughter products, with little or no parent material present. This will result in an overestimation of the parent material.

It is important to note that sometimes disequilibrium may be masked by higher emissions of gamma rays from the daughter products of the Thorium series, especially Th^{208}.

When there is disequilibrium in the uranium series, and when the absent nuclides are short lived, approximately 350,000 years are required for the uranium series to regain equilibrium. Normally, if the series is disturbed at the beginning of the chain, then it can take up to 2.5 million years to regain equilibrium. To calculate the time required to regain equilibrium, one considers the longest half life of the daughters which have been mobilised and multiply it by 10. For example, if radon is lost, the time to regain equilibrium is 3.8 days by 10, or approximately 1 month. For the long lived U^{234}, with a half life of 2.5 by 105, the time to regain equilibrium is 2.5 by 105 by 10, or 2.5 million years (Schmeling, 1982).

Based on some 100 chemical checks on the calculated uranium grades from radiometric logs, Priargunsky concluded that the two sets of values were very similar. Therefore, the uranium disequilibrium in the samples was insignificant.

15 Sample Preparation, Analysis and Security

15.1 Previous Work

Sample preparation, assaying and quality control / quality assurance (QA / QC) procedures used by Priargunsky were not available to Scott Wilson RPA. Scott Wilson RPA noted that the procedures used during the exploration and production phase of the Project were similar to Western industry standards (Tserenmur, 2004).

15.2 Recent Work

Sampling of drill core was done at 1-m intervals. Samples were sent to Alex Stewart Assayers Mongolia LLC, where sample preparation was carried out. Thereafter, samples were sent to Activation Laboratories (Actlabs) in Ancaster, Ontario, for uranium assays by the Delayed Neutron Counting (DNC) method. Details of the Actlabs analytical method were provided in the 2006 Scoping Study and the August 2007 Prefeasibility Study.

16 Data Verification

16.1 Verification of Historical Data

During the Priargunsky drilling campaign on the Project area, data verification and quality control was done by Priargunsky personnel.

16.2 Verification of Recent Exploration Data

During the 2006 / 2007 confirmation drilling program field data, as well as assay, data were verified first by Dr. Derek McBride, P.Eng., Project Geologist, and later by Mr. John Kita, P.Geo., Chief Geologist of Khan. Both Messrs. Kita and McBride are Qualified Persons under the definition of NI 43-101. Mr Kita also supervised a compilation program on all previous exploration data (Kita, 2006).

16.3 Confirmation Drilling by Khan 2005 / 2006

A comparison of results from the old Russian database with the new results from the confirmation drilling was presented in the 2006 Scoping Study, and is reproduced in Table 16-1. This information was included in the August 2007 Prefeasibility Study.

Table 16-1
Comparison of Diamond Drilling Results

	Khan Results				Previous Results				
DDH No.	**From (m)**	**To (m)**	**Interval (m)**	**% U_3O_8**	**DDH No.**	**From (m)**	**To (m)**	**Interval (m)**	**% U_3O_8**
DOR-05-03	503	533	30	0.510	1910	35.1	71.8	36.7	0.303
DOR-05-04	423	454	31	0.540	1852	18.0	51.9	33.9	0.433
					1847	17.5	48.2	30.7	0.531
DOR-05-05	394	429	35	0.748	1909	9.8	46.6	36.8	0.423
DOR-05-06	397	427	30	0.631	1176	41.8	74.8	33.0	0.068
					1176A	28.8	65.1	36.3	0.496
DOR-05-09	405	439	34	0.332	1840	7.7	45.5	37.8	0.244
DOR-05-10	393	424	31	0.553	3083	12.6	45.8	33.2	0.150
DOR-05-11	391	427	36	0.350	1843	25.2	59.4	34.2	0.293
DOR-05-20	423	457	34	0.154	1175	19.8	65.6	45.8	0.109
DOR-05-21	378	414	36	0.026	1184	34.2	56.1	21.9	0.099
					1186	21.5	60.1	38.6	0.062
TOTAL			**300**					**380.3**	
Average			**33.3**	**0.416**				**31.7**	**0.287**

Notes:

1. Average grades are weighted average grade over the combined intervals of all holes in the group.

2. Mineralized intervals for the old holes are calculated intersections marked on cross-sections and not from the surface.

Table 16-1 shows that there is a good correlation between the two sets of results. They also indicate that the average grade of the group of holes from the new Khan drilling is approximately 45% higher than the Russian drilling results. In Scott Wilson RPA's opinion, the reason for this difference is that the Russian data are based predominantly on radiometric logs, and the reported uranium contents are converted from these data, and not from actual chemical assays. Khan data, on the other hand, are based on actual assays. Scott Wilson RPA understands that Khan is currently carrying out a compilation program to convert the previous drill results.

During the recent compilation program, Khan carried out data verification including a study and comparison of the lithologic logs with the down-hole radiometric logs to detect possible errors, as recommended in the 2006 Scoping Study. In addition, Khan instituted standardised radiometric logging of all accessible drill holes from Priargunsky's offices in Krasnokamensk, also recommended by Scott Wilson RPA.

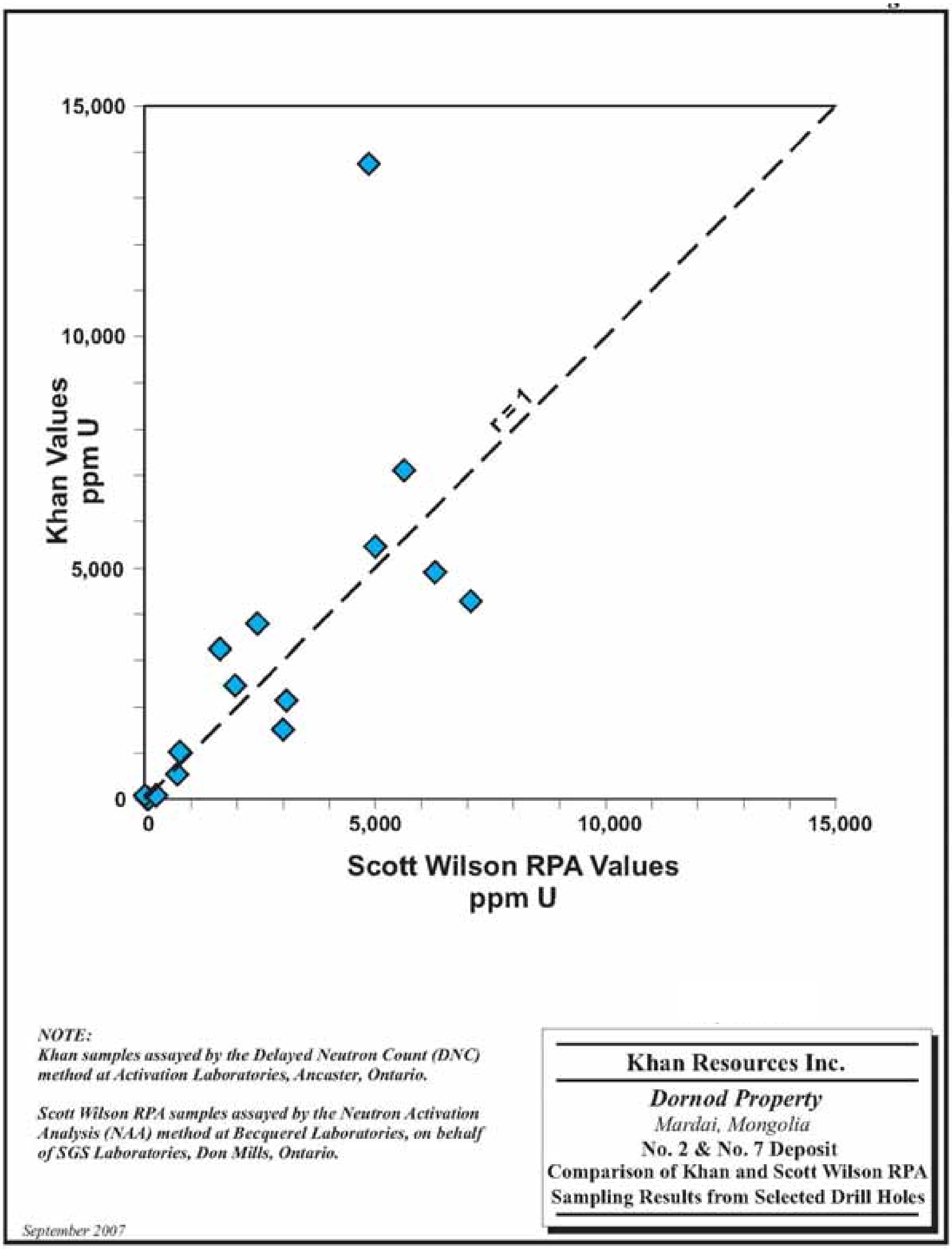
16.4 Independent Sampling by Scott Wilson RPA

During the second site visit, Scott Wilson RPA collected a total of 15 independent samples from the recently completed diamond drill holes, and sent them to SGS Laboratories, Don Mills, Ontario, for independent assays. These included six samples (three each) from holes DOR-05-14 and DOR-05-16 testing the Dornod No. 2 Deposit, and nine samples from the Dornod No. 7 Deposit; four samples from hole DOR-05-06 and five samples from hole DOR-05-05. The uranium determinations were done at the Becquerel Laboratories, Hamilton, Ontario, on behalf of SGS, using the Neutron Activation Analysis (NAA) method. Table 16-2 provides the sample intervals and assay results. Details of the sample preparation and analytical methods used at Becquerel Laboratories are provided in the 2006 Scoping Study.

Table 16-2
Independent Sampling Results

DDH No.	Scott Wilson RPA Sample No.	Khan Sample No.	From (m)	To (m)	Interval (m)	Scott Wilson RPA ppm U	Khan ppm U
DOR-05-14	**71023**	179	43.2	44.2	1.0	**275**	106
	71024	180	44.2	45.2	1.0	**3,010**	1,510
	71025	181	45.2	46.2	1.0	**5,020**	5,510
DOR-05-16	**71026**	151	55.0	56.0	1.0	**74.3**	43
	71027	152	56.0	57.0	1.0	**33.4**	81
	71028	154	58.2	59.2	1.0	**788**	1,040
DOR-05-06	**71029**	122	415.0	416.0	1.0	**730**	589
	71030	123	416.0	417.0	1.0	**1,980**	2,480
	71031	124	417.0	418.0	1.0	**1,670**	3,270
	71032	125	418.0	419.0	1.0	**2,460**	3,850
DOR-05-05	**71033**	26	428.5	429.5	1.0	**3,100**	2,210
	71034	27	429.5	430.5	1.0	**4,860**	13,800
	71035	28	430.5	431.5	1.0	**5,650**	7,120
	71036	29	431.5	432.5	1.0	**6,290**	4,900
	71037	30	432.5	433.5	1.0	**7,090**	4,280

In general, the Scott Wilson RPA samples confirm the presence of uranium values at similar levels and orders of magnitude as the Khan assays, with one exception. Of the 15 samples, 8 of the Khan samples had higher values than the Scott Wilson RPA (SGS) assays and 7 of the Khan samples had values lower than Scott Wilson RPA assays (Figure 16.1). The differences are considered to be due to the different assay methodologies at the two laboratories, Actlabs and SGS, and are not cause for concern, in Scott Wilson RPA's view. Actlabs used the DNC method on 1-g samples and Becquerel used the NAA method using 2-g to 3-g samples, as noted above.



Source: Scott Wilson RPA, September 2007.

Figure 16.1 – No. 2 and No. 7 Deposit – Comparison of Khan and Scott Wilson RPA Sampling Results from Selected Drill Holes

17 Adjacent Properties

A number of deposits in the Mardai area were explored by the Russian-Mongolian team that carried out the work on the Dornod Project.

Western Prospector Group Ltd. (Western Prospector) operates the Saddle Hills Project, which is adjacent to the Dornod Uranium Project. The Saddle Hills Project includes the Gurvanbulag Deposit, which is reported to contain some 4.2 Mt of Inferred mineral resources at an average grade of 0.25% U_3O_8. In addition, the deposit is reported to contain "historic" Russian C2 category resources totalling some 6.3 Mt at an average grade of 0.14% U_3O_8 (Western Prospector Press Release, March 7, 2006). Western Prospector is currently carrying out diamond drilling to upgrade the existing mineral resources.

Ulaanbaatar Xin-Xin Corporation Ltd. has an operating mine at the Ulan Pb-Zn deposit situated approximately 10-km south of the Dornod Project. At the present time, Scott Wilson RPA does not have any information regarding the mineral resources at that deposit.

Scott Wilson RPA is not aware of any other exploration work currently being carried out on lands outside of the Dornod Property.

18 Mineral Processing and Metallurgical Testing

18.1 Introduction

The Dornod claims area contains several known ore deposits. This DFS provides for the mining and processing of Nos. 2 and 7 Deposits. Due to its higher grade, the No. 7 Deposit will be developed first. This is expected to take up to about 9.8 years. After about 9 years, it will become difficult to extract 3500 t/d from the No. 7 Deposit. At this time, the tonnage will be replaced with lower grade No. 2 Deposit ore.

The No. 7 Deposit after dewatering the mine, will be accessed via a new ramp to be sunk adjacent to the richest part of the deposit. The existing No. 3 shaft will become the primary ventilation shaft. The No. 2 Deposit will be developed as an open-pit mine.

A milling rate of 3500 t/d is planned. In Years 1 to 9.8, treating only No. 7 ore head grade will be typically 0.2% U_3O_8 for Years 1 to 4 and 0.1 in Years 5 to 7. After Year 9, once No. 2 ore is added to the mix, grade will gradually decrease until it reaches average grade for No. 2 ore only after about Year 10 of 0.08% U_3O_8, dropping to 0.07 U_3O_8 in Years 11 and 12 and to 0.06 U_3O_8 through the end of mine life at Year 16.

The No. 7 Deposit has proven to be refractory. This is presumed to be as a result of the presence of brannerite, a uranium titanate mineral, due to the ore's high in-situ carbonate content and because the uranium minerals are very fine and are closely associated with gangue particles . These effects result in high acid consumption if acceptable recoveries are to be achieved. The difficulty experienced in the leaching seems to vary throughout the deposit. Although the uranium mineralization has been found to exist as very small and intergrown crystals, it has not been necessary to grind the ore to very fine particle size. It is however necessary that a significant amount of silica in the ore be dissolved in order to liberate the uranium. The presence of this dissolved silica causes a gel to form, making the ore difficult to settle or filter. To overcome these problems, a Resin in Pulp (RIP) method of removing the uranium from the ore has been selected to recover the dissolved uranium.

An average leach recovery of 88%% has been achieved in testwork to date on the No. 7 Deposit ore. Precipitation recovery of 96% results in an overall recovery of 84.5%. This recovery will be used in the financial analysis.

The No. 2 Deposit is free milling and, based on the Russian experience, a leach recovery of 93% has been assumed for this ore. This assumption needs to be confirmed in the laboratory. Reagent consumptions for this material also need to be confirmed at the detailed engineering stage.

The No. 7 ore will be brought to surface through a new ramp in 50-t trucks and dumped into a communal dump hopper. A bypass is provided to stockpile ore should the dump hopper be full. This stockpiled material, along with ore from the No. 2 Deposit surface stockpile, will be fed back to the feed hopper using a front-end loader.

After about 10 years of the mine life, the No. 2 Deposit ore will be transported to the same pile or the dump hopper using 140-t ore trucks.

The dump hopper is provided with a 300-mm grizzly. The grizzly oversize will be crushed to –300 mm in an open-circuit jaw crusher. This crusher is able to handle the larger ore from the open pit of the No. 2 Deposit.

The –300-mm material will be fed to an open-circuit 20-ft-dia by 12-ft-long (6.1 m by 3.7 m) SAG mill. This will produce an 80% passing 2-mm feed to a 16-ft diameter by 21-ft long closed-circuit ball mill. The SAG mill will be equipped with a 2200-kW motor, while the ball mill will be powered with a 1750-kW motor. The grinding circuit will produce 80% passing 75 micron material.

Testwork has indicated that the ore is relatively hard and will produce a critical size which will not break down in the SAG mill. For this reason, a 4-ft pebble crusher has been included in the design. This will crush oversize material scavenged from the SAG mill discharge trommel.

The milled material, before acidification, settles well and will be thickened to a density of 50% solids in a high-rate 7-m diameter thickener. In order to save on acid costs, a portion of the thickener underflow material will be further dewatered on a 10-disk vacuum disk filter. This dewatered material will be mixed with unfiltered thickener underflow and repulped to produce a 58% solids feed stream to feed the leach section.

Some of the residual heat in the leach discharge stream will be used to preheat the leach tank feed. The lowering of the leach discharge pulp temperature is required to protect the integrity of the ion exchange resin in the uranium recovery section.

A conventional sulphuric acid leach section has been designed to treat the two ores. After thickening and preheating, the pulp will be leached in a series of 18 pachuca tanks. A residence time of 42 hours was used in the design. The free acid in the leach section will be maintained at about 25 g/L and the pulp will be heated to 80ºC. This will be done by the injection of live steam produced in the acid plant. Oxygen, produced in a dedicated oxygen plant, will be injected into the leach tanks to maintain the EMF at about -480 mV. Each of the tanks will be agitated using a 260-kW agitator.

In order to protect the RIL resin from osmotic shock, after leaching and before the heat exchange, the leached pulp will be partially neutralized to a pH of 2 to 2.5 by the addition of lime.

The dissolved uranium will be removed from the leached pulp by adsorbing the uranium onto anion exchange resin (Purolite A660 or equivalent). The resin and the pulp will flow countercurrenly to each other in an eight-stage KEMIX carousel type resin-in-pulp circuit. At the end of the process, the loaded resin will be separated from the pulp stream by screening the pulp on a vibrating screen. The barren pulp will be sent to neutralisation and then to disposal in the tailings dam.

The loaded resin will be washed before being eluted with sulphuric acid in a batch type elution circuit. Provision has been made to periodically wash the stripped resin with a caustic solution to remove any silica that may have adhered to the resin.

Before uranium precipitation from the pregnant liquor, impurities will be removed by adjusting the pH to approximately 3.2. In this way iron, arsenic and sulphates will be removed by the addition of lime and ferric sulphate in an oxidising environment. The resulting solids, mainly gypsum, will be removed on a belt filter. The resulting filtrate will be further clarified by passing it through sand filter clarifiers.

Yellowcake will be precipitated from the clarified solution by the addition of magnesia and hydrogen peroxide to form insoluble uranium oxide. This will be dewatered in a thickener and a centrifuge before being dried in a multi-hearth drier.

Leached pulp from the RIL circuit will be neutralised with lime and treated with ferric sulphate and barium chloride before thickening and sending the material to tailings. This will precipitate heavy metals, radium 226 (Ra^{226}) and arsenic ions into the solid tailing.

An extensive water treatment system has been designed. This system includes neutralisation, clarification and reverses osmosis treatment. All tailings dam return water, underground and open-pit mine water, and surface runoff will report to a surface surge pond before treatment and disposal, or being pumped to the mill process water tank.

Potable water will be produced from open-pit supernatant water by reverse osmosis.

18.2 Metallurgical Testwork

The metallurgical testwork that underpins the DFS design is in three parts:

(a) Early work conducted by the Russians

(b) Work in preparation for the PFS that was conducted in 2007 / 2008 and was reported in the PFS

(c) Additional work that was conducted in 2008 for the DFS.

18.2.1 Early Russian Work

The Russian metallurgical test reports supplied by Khan are briefly summarised below.

The early work on the testing of No. 7 ore deposit was done at the Russian Research Institute of Chemical Technology (VNIIChT) in the Ministry of Atomic Energy. Only a portion of the reports issued were made available to Aker Solutions. These documents provide information on the various process options investigated, and, unfortunately, conventional Pachuca leaching was not extensively evaluated. The authors concluded that the tested samples from the No. 7 Deposit were refractory. This was stated as being due to its fine crystalline structure, the presence of brannerite (a titanium mineral), zircon, and greater than normal amounts of carbonate (4% to 7%). (The carbonate would consume excessive amounts of acid.) VNIIChT recommended the pressure leaching of this ore in an autoclave. It was concluded that this approach would yield higher recoveries (perhaps as high as 90%).

18.2.2 The 2007 Lakefield Testwork

For the purposes of the PFS, it was decided to further explore the conventional atmospheric pressure leaching route before accepting the autoclave option. (Pressure leaching of uranium ore has not been extensively used in the industry.)

Khan consequently commissioned a mineralogical study by Dr. Anthony Mariano to try to further understand the nature of the leaching problems. Thin sections of the ore and of the leached tailings were examined. This work showed that the unleached uranium existed in very finely divided and intergrown crystalline structures within the ore matrix. No brannerite was observed. The report entitled “Characterization of Uranium bearing minerals in the

Dornod No. 7 Deposit in Mongolia" is referenced in Item 23, References, and included in the DFS report.

During 2007 / 2008, additional leaching studies on No. 7 borehole samples from the No. 7 Deposit were conducted at SGS. The work showed that recoveries in excess of 90% could be achieved with high acid consumptions in a conventional leach, provided that the following conditions were employed:

- a grind size of 80% passing 75 micron
- pre-aeration for 12 hours
- leach temperatures of 80ºC
- the used of sodium chlorate as an oxidiser.

Several other options including alkaline leaching, oxygen assisted leaching and low pressure autoclaving were also tested. These methods produced less beneficial results. The results of this work are included in the SGS report referenced in Item 23, References, and included in the DFS report.

18.2.3 The 2008 Testwork Summary

This work was also conducted at SGS Minerals Services in Lakefield, Ontario. The report on this work is referenced in Item 23, References, and included in the DFS report.

The testwork included composite preparation, grindability testing, bench scale leach tests, solid-liquid separation and recovery of uranium from leach solutions by solvent extraction and by ion exchange.

Five composites were prepared from two drill holes of the Dornod deposit, each being unique rock types as identified by Khan Resources' consulting geologist. The head grades for each composite, and the overall composite are presented in Figure 18.1.

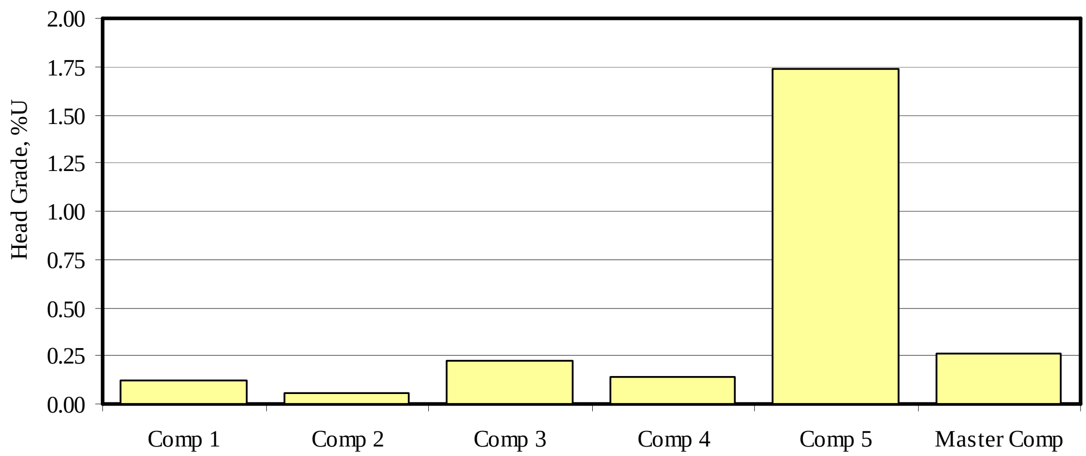


Figure 18.1
Uranium Composite Sample Head Grades

Each composite was subjected to Bond Work Index (BWI) and SAG Power Index (SPI) testing. Results are presented in Table 18-1.

Table 18-1
Grindability Test Summary

Sample Name	BWI kWh/t	SPI min
Comp 1	17.7	118.2
Comp 2	18.7	154.6
Comp 3	16.5	104.7
Comp 4	15.1	89.6
Comp 5	13.8	98.3
Master Comp	16.3	122.1

Acid leaching resulted in extractions of about 88% after 48 hours with acid additions of about 225 kg/t and manganese dioxide additions of roughly 12 kg/t.

The leach discharge slurry was subjected to standard static thickening testwork. Settling behaviour was not favourable with high flocculant additions, low settling rates, large thickener unit areas and low ultimate settled solids density.

The uranium loaded well onto a common solvent extractant, Alamine 336. A strong base anion exchange resin also extracted uranium effectively from the leach discharge solution. This indicated that RIP could be an attractive process for the Dornod Project, due the unfavourable settling characteristics of the ore.

The general conclusions arising from this testwork are as follows.

- The ore in the deposit appears to vary in terms of its leachability. The master composite of the 2007 samples, that from the two 2008 drill-hole samples, three additional samples comprising geotechnical drill holes, and the variability samples (five) from the 2008 campaign all gave different uranium recoveries and acid consumptions. On average, over all of the samples tested under standard leach conditions, a recovery of 88% may be anticipated.
- The standard leach conditions that worked on the 2008 composite are as follows:
  - A grind size of 80% passing 120 μ
  - Sulphuric acid consumption of 225 kg/t
  - Oxidant in the form of manganese dioxide to give an EMF of 480 mV (approximately 12 kg/t.). The addition of oxygen or sulphur dioxide was also shown to work
  - A leach temperature of 80ºC
  - Kinetic studies indicated that a leach residence time of 42 hours was sufficient to ensure the 88% recovery.

- The leach recovery did not appear to depend upon grind size on the 2008 composite over a range of 80% passing 50 μ to 200 μ.
- Recovery quickly dropped off if the free acid concentration fell below 25 g/L.
- An EMF of approximately -500 mV is required to ensure recoveries of between 85% and 90%.
- Most of the leach tests were conducted at 50% solids. Problems were encountered with the SGS test equipment for samples with higher densities. The opportunity of leaching at higher density needs to be investigated at the Basic Engineering stage.
- Analysis of the pregnant solutions arising out of the leaching work gave fairly standard results. Iron values were in the 10 000-mg/L to 20 000-mg/L range. This material will need to be precipitated before the uranium, in order to ensure product quality.
- The leached pulp filtered and settled poorly. This is thought to be as a result of silica gel formation.
- The uranium adsorbed well onto both alamine 336 solvent extraction organic and onto strong base ion exchange resin (Purolite A660 / 4750). In each case, a three- or four-theoretical stage adsorption process is anticipated. The uranium is essentially completely adsorbed after four stages. No competition was observed from iron. Resin loading was in the range from 40-g/L to 45-g/L range.
- Work index determination on the ore tested indicated that the ore was in the fairly hard range. A Bond Work index in the range 13.8 to 18.7 was observed.

18.3 Discussion of Recoveries

In the Russian work, only limited results were found to indicate the recovery that might be expected from the No. 7 deposit using acid leaching under atmospheric conditions. These are summarised in Table 18-2.

Table 18-2
Pachuca Leaching of No. 7 Deposit

H_2SO_4 Concentration (kg/t)	Head Grade (% U)	Residue (% U)	Residence Time (h)	Percent Extraction (%)
260	0.402	0.029	5.3	92.8
550	0.402	0.026	4.9	93.6
373	0.13	0.024	3.2	81.5
635	0.13	0.019	6.7	85.9

Aker Solutions notes that the tests were performed at 88ºC, and that this elevated temperature should result in a higher percent extraction. No liberation size was noted. It is also evident that better recoveries are obtained from the higher-grade ores. The No. 7 Deposit average head grade is 0.276% U_3O_8, somewhere between the two head grades reported. Longer residence times are known to improve the extraction. For these reasons, the following elements have been included in the design of the Dornod flow sheet design.

- The more difficult ore from the No. 7 Deposit will be given higher free-acid conditions
- A grind of 80% passing 120 microns has been assumed
- A leach temperature of 80ºC will be employed.

The 2007 work at SGS has confirmed that leach recoveries of 90% might be anticipated from the ore tested at that time at the expense of relatively high-acid consumptions (180 kg/t) and a finer grind size under the above conditions. The ore tested in 2008 gave a lower recovery of about 88% under similar conditions. Both ore samples were chosen to be representative of the first 5 years of the mine life.

A recovery of 88% will be used in the economic analysis.

A sensitivity analysis on recovery will be performed to try to evaluate the effect of higher or lower recoveries on financial outcome.

18.4 Process Selection and Description

(Refer to Figure 18.2)

A conventional sulphuric acid leach plant has been designed to treat the two ores. At this time, a liberation size of 80% passing 120 microns has been selected for both ores. This may change for the No. 2 ore, but this will need to be confirmed in future testwork.

The No. 7 ore will be trucked in 50-t ore trucks from underground to surface via a new ramp to be built. It will be fed to a dump hopper large enough to accommodate 140-t trucks. A bypass is provided to stockpile ore should the dump hopper be full. This stockpiled material, along with ore from the No. 2 Deposit surface stockpile, will be fed back to the feed hopper using a front-end loader.

The ore from the No. 2 Deposit will be delivered to the dump hopper or to the ore stockpile in 140-t ore trucks. Reclaimed materials will be moved with a 988C loader from the stockpiles to the feed hopper.

Oversize material will be scalped on a grizzly and crushed in a jaw crusher. The grizzly undersize and the jaw crusher product will be fed to a crushed ore stockpile. From here, the SAG mill will be fed via a variable speed belt feeder.

Size reduction will be achieved in an 20-ft-diameter by 12-ft-long open-circuit SAG mill and a 13-ft-diameter by 22-ft-long conventional ball mill in closed-circuit with a cyclone pack. A grind size of 80% passing 120 μ will liberate the uranium minerals. The SAG mill will be equipped with a 2200-kW motor, while the ball mill will be powered with a 1750-kW motor.

Testwork has indicated that the ore is relatively hard and will produce a critical size which will not break down in the SAG mill. For this reason, a 4-ft pebble crusher has been included in the design. This will crush oversize material scavenged from the SAG mill discharge trommel. The crushed material will be returned to the SAG mill feed belt.

The milled ore will be thickened to 50% solids in a 7-m-diameter high-rate thickener, prior to leaching. A portion of the thickener underflow will be further dewatered on a disk filter to about 90% solids. This material will be mixed back with additional thickener underflow to produce a 55% solids leach feed.

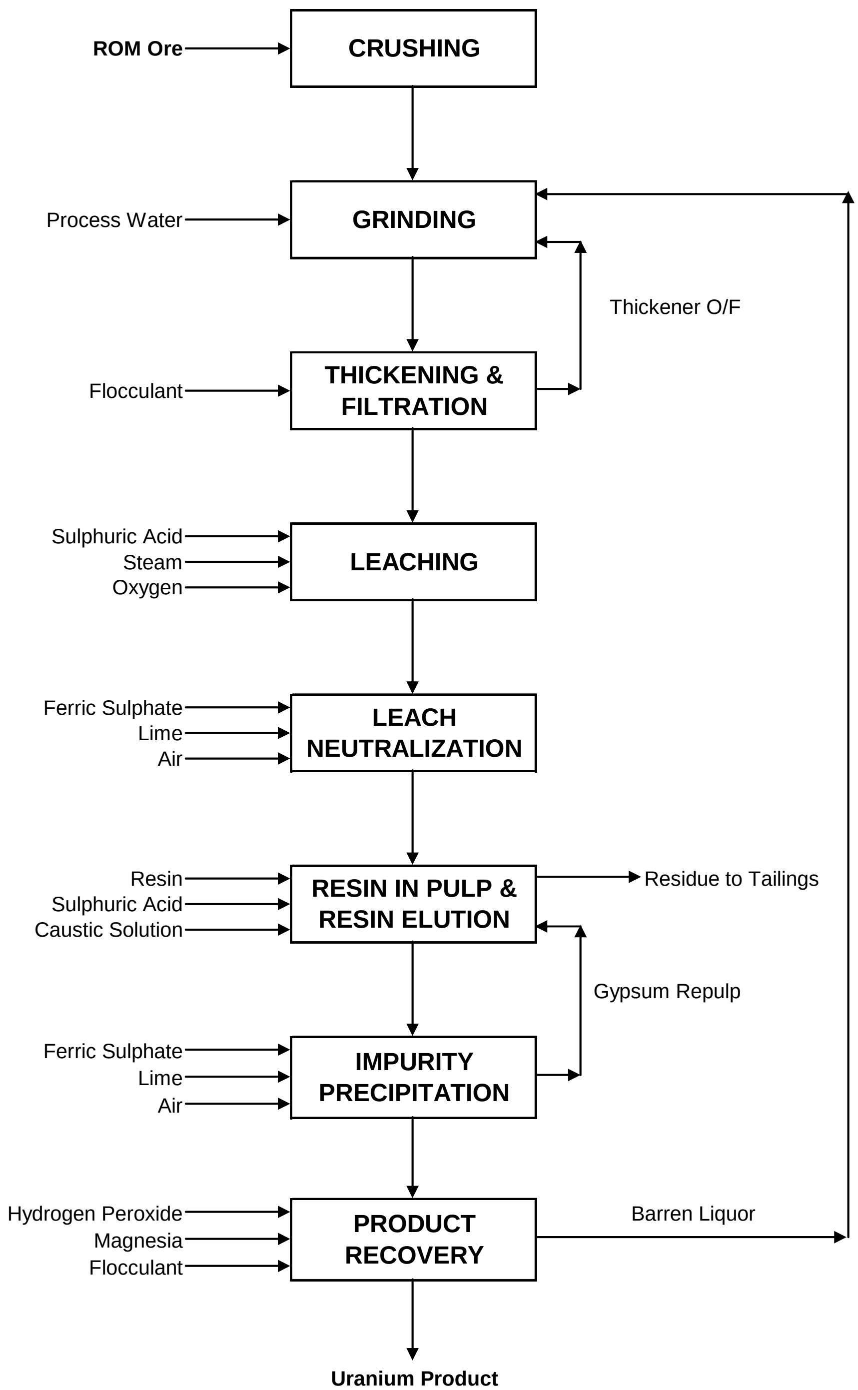


Figure 18.2
Process Block Diagram

A conventional sulphuric acid leach section has been designed to treat the two ores. After thickening and preheating, the pulp will be leached in a series of 18 pachuca tanks. A residence time of 42 hours was used in the design. The free acid in the leach section will be maintained at about 25 g/L and the pulp will be heated to 80ºC. This will be done by the injection of live steam normally produced in the acid plant. Oxygen, produced in a dedicated oxygen plant, will be injected into the leach tanks to maintain the EMF at approximately 480 mV. Each of the tanks will be agitated using a 260-kW agitator.

Provisions have been made to add reagents at several points down the leach train, but it is anticipated that most of the acid and oxidant will be added with the ore at the head of the leach train. The 11-m-diameter by 20-m-high rubber-lined tanks will be mechanically agitated. All reagents will be added via downpipes. Gravity flow between tanks will be employed. The ability to bypass tanks has also been provided.

The leached pulp will discarded from the leach section at 80ºC and at very low pH. Before entering the RIP section, the pH of the leached ore will be adjusted by the addition of lime and the temperature will be lowered by recovering some of the heat in a shell and tube heat exchanger. The heat recovered will be used to heat the unleached pulp entering the leach section.

The dissolved uranium will be removed from the leached pulp by adsorbing the uranium onto a weak base ion exchange resin. (Purolite A660 or equivalent).

The resin and the pulp will be contacted with each other in an eight-stage KEMIX carousel RIP circuit. Each RIL tank will be charge with 50 L of resin and will have a pulp volume of 200 m^3**.** In order to protect the resin against degradation by abrasion, the resin charge will be retained in its leach tank using proprietary wedge wire stainless steal screens. The strained pulp will be moved to the next adsorption stage using pump cell pumps. The residual uranium in the train will be monitored and the first tank in the series will be removed when breakthrough takes place. This is expected to happen about every 5 hours. At this point, the second tank in the series becomes the first tank and a fresh tank with eluted resin joins the end of the train.

At the end of the process, the loaded resin, from the first tank, will be separated from the pulp stream by screening the pulp on a vibrating screen.

The barren pulp will be sent to neutralisation section where the pH is adjusted to about 8, the arsenic is fixed with ferric sulphate, and the RA^{226} is precipitated by the addition of barium chloride. The neutralised pulp is then sent to the tailings dam for disposal.

The loaded resin will be stripped of uranium in a batch operated elution section. The loaded resin will be water washed before being eluted with weak eluate and then dilute sulphuric acid. (The weak eluate is the solution which results from the dilute acid washing of the previous batch.) In this way, the uranium concentration in the pregnant liquor is increased, helping with the rejection of iron and other cations.

Provision has been made to periodically wash the stripped resin with a caustic solution to remove any silica that may have adhered to the resin.

The stripped resin is returned, in batches, to the RIP circuit.

Before uranium precipitation from the pregnant liquor, impurities will be removed by adjusting the pH to about 3.2. This will be done in a series of four stirred precipitation tanks. The neutralisation will result in the formation of gypsum. Some of the gypsum formed will be circulated back to the first tank as seed to promote the formation of large gypsum crystals. In this section, iron, arsenic and sulphates will also be removed by the addition of lime and ferric sulphate in an oxidising environment. The resulting solids, mainly gypsum, will be removed on a belt filter. The filtrate will be further clarified by passing it batch wise through one of three sand filters. The clarified pregnant solution is sent to the uranium precipitation section.

"Yellowcake" will be precipitated from the clarified solution by the addition of magnesia and hydrogen peroxide in a series of two precipitation tanks to form insoluble uranium oxide. This will be dewatered in a 7-m-diameter yellowcake thickener and a centrifuge before being dried in a diesel-fired, multi-hearth drier.

The dried product will be stored in a yellowcake storage bin capable of holding 4 weeks worth of peak plant output. Product drums will be filled via a dedicated screw conveyor. A baghouse-type dust collector services this area of the plant.

Golder has calculated that the mine will be short of water during the later years of the mine life. During the early years, this deficit will be made up by using the water currently being stored in the open pit. This water will be consumed, or will be unavailable, by the time the open pit needs to be dewatered. For this reason, it is important that all water is conserved and recycled within the plant and tailings areas.

All tailings dam return water, underground and open-pit mine water and surface runoff water will report to a surface surge pond before treatment or return to the mill process water tank.

An extensive water treatment system has been designed to remove impurities and make the water reusable. This system includes neutralisation, clarification and reverse osmosis treatment.

Since there is currently only limited potable water available at the site (and these wells will eventually be covered by waste rock), the reverse osmosis plant has been sized to also produce drinking water.

18.5 Plant Services Design

Individual unit operations for the processing facilities will be modularised, wherever possible, to minimise construction installation time at the site, and housed within the "clear-span" pre-engineered processing building. Given the severe winter conditions at the Project site, mill support facilities will be housed in the same structure or located close by, and covered and insulated, where necessary, to protect from freezing.

18.5.1 Utilities

Water

Process water will be reclaimed from the tailings area, the mine, the pit and from surface runoff. The water will be stored either in the pit or in the later years in a surge pond near the tailings area and then pumped to the process water tank. Open-pit water may also be sent to the reverse osmosis plant and then to the fresh water tank. This tank will be constructed so that the bottom portion will be unavailable to the process. This water will be reserved for fire fighting. The fire-fighting equipment will comprise an electrically-driven pump, as well as a diesel pump. The fresh water tank will provide water to gland service, flocculent makeup and potable water. There will also be a fresh water bleed to the process water tank.

Power

Electrical power to the underground mine and surface facilities will be provided from the Mongolian grid by a new line to be built by the Mongolian Power Utility as described in Item 20, Surface Infrastructure. Standby diesel generator units are provided to ensure the running of key processing units. An emergency generator will also be provided for the man camp and the ventilation fan and hoist in the mine.

A central fuel storage and distribution system will be provided with capacity to store a 4-wk fuel supply on-site. Fuel supplies will be delivered by rail.

Heating

The acid plant will provide 25 MBTU/h of steam for the heating of theleach tanks. The acid plant will also produce 225 MBTU of energy in the form of hot water at 65ºC. This will be used for mine air heating, general heating of the offices building and heating in the process plant and the service buildings. This will be done by means of an insulated hot water distribution piping system and hot water radiators.

The uranium oxide drier will be diesel fired.

Site Facilities

The processing facilities will be located to the south, and in close proximity to the Ramp Portal. Ore stockpiles are located close to the crusher location. The power substation will be located as close as practical to the grinding section of the plant, since this section consumes the most power. Other mine infrastructure is also located in close proximity to the mill. This included assay laboratory, general offices, mill workshops and dry facilities. This would be the most favorable and cost-effective arrangement. Figure 18.3 is a preliminary layout showing the approximate location of the plant facilities. Final plant layout should be available during the feasibility phase of the Project development.

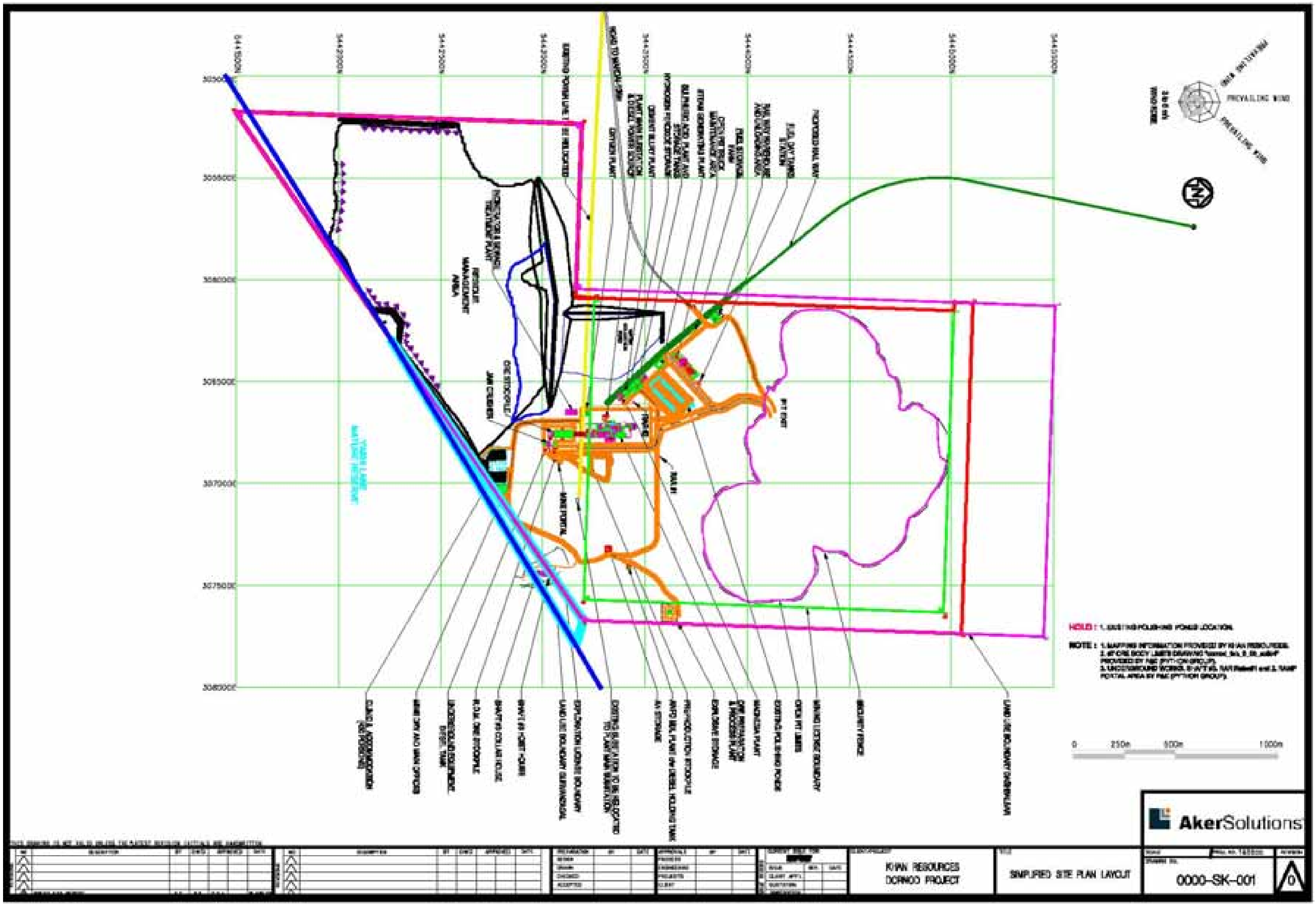


Figure 18.3 – Site Plan

19 Mineral Resource and Mineral Reserve Estimates

19.1 Mineral Resources

Scott Wilson RPA updated the mineral resources of the Nos. 7 and 2 Deposits, based on a new digital database of previous results, and additional confirmation drilling results. The Scott Wilson RPA resource estimate is in accordance with the Mineral Resource/Reserve Classification as recommended by the CIM Committee on Mineral Resources/Reserves.

Note: The mineral resources reported in this Report are the same as those reported in the Prefeasibility Study, August 2007.

Table 19-1
Mineral Resource Estimate

Location	Category	Tonnes (million)	% U_3O_8	lbs U_3O_8 (million)
No. 7 Deposit	Indicated	14.36	0.154	48.6
No. 2 Deposit	Indicated	10.95	0.065	15.7
TOTAL	**Indicated**	**25.31**	**0.116**	**64.3**
No. 2 Deposit	Inferred	2.18	0.050	2.4

Notes:

1. CIM definitions were followed for mineral resources.
2. Mineral resources were estimated using a U_3O_8 price of USD 55/lb.
3. Mineral resources were estimated using a cutoff grade of 0.04% U_3O_8 for No. 7 Deposit, and 0.025% U_3O_8 for No. 2 Deposit.
4. No. 7 Deposit was modeled at a minimum of 5-m-vertical thickness, No. 2 Deposit was modeled at a minimum of 2-m-vertical thickness.
5. Mineral resources are inclusive of, not in addition to, mineral reserves.
6. The numbers for tonnage, percentage U_3O_8 and contained lbs U_3O_8 are rounded figures.

19.1.1 Database

No. 7 Deposit

Scott Wilson RPA received a new digital database of assay results from some 266 old drill holes (123 surface drill holes and 143 underground drill holes), underground vertical channel sampling results and cross-sections. Scott Wilson RPA also received a digital database of assay results and lithologic logs for the 12 recent drill holes completed by Khan. Since the results of recent drilling were similar to the old (Russian) results, Scott Wilson RPA combined then into a single database. Scott Wilson RPA considered the vertical channels of the old database as short holes using Gemcom software to enable independent interpretation of geology and mineralized units. Scott Wilson RPA notes that, with few exceptions, data entry is of good quality. In a number of cases, where drill-hole intervals did not have assay results, Scott Wilson RPA assigned them the average of the assays of the adjacent intervals, generally a value of 0.015% U_3O_8. This value is based on the average grade of the zone peripheral to the No. 7 Deposit in the new database.

The mineral resource estimate of the No. 7 Deposit is based on surface diamond drilling completed on a 50-m by 50-m to 50-m by 100-m drill-hole spacing and on underground drilling on the 550 Level (named for metres above sea level, with surface at approximately 950). A total of 150 surface drill holes have tested the No. 7 Deposit, while underground drilling included 143 diamond drill holes. Some of the old underground drill holes were probed by gamma-logging only. Underground hole spacing is at 20- to 30-m intervals along the drifts and crosscuts. In the past, drill-hole collars were surveyed by Priargunsky and inclinations recorded at regular intervals for all surface holes. Core recovery in surface holes is reported to average 78%, while underground core recovery is reported as 75% (MacCormack, 1998).

During the recent confirmation drilling campaign, core recovery generally ranged from 90% to 95% and field procedures included:

- Recording of alteration patterns and structural features on the core
- Radiometric logging of the core boxes by a handheld Exploranium SPP2 scintillometer
- Down-hole radiometric logging of the drill holes using a Mount Sopris instrument
- Calculation of the RQD value.

Scott Wilson RPA is of the opinion that the quality of the No. 7 Deposit database is acceptable to estimate and report mineral resources.

No. 2 Deposit

Scott Wilson RPA received a new digital database of assay results from some 450 old surface drill holes and cross-sections. Scott Wilson RPA also received a digital database of assay results and lithologic logs for the 11 recent drill holes completed by Khan. As with the No. 7 Deposit data, results from recent drilling and from Russian drilling were combined into a single database. Scott Wilson RPA notes that, with few exceptions, data entry is of good quality. In a number of cases, where drill-hole intervals did not have assay results, Scott Wilson RPA assigned them the average of the assays of the adjacent intervals, generally a value of 0.010% U_3O_8. This value is based on the average grade of the low-grade material within Layer 3 of the No. 2 Deposit, in the new Khan database.

The mineral resource estimate of the No. 2 Deposit is based on surface diamond drilling completed on a 50-m by 50-m to 50-m by 100-m drill-hole spacing. Similar to the No. 7 Deposit, in the past, drill-hole collars were surveyed by Priargunsky and inclinations recorded at regular intervals for all surface holes. Core recovery in surface holes is reported to average 78% (MacCormack, 1998).

During the recent confirmation drilling campaign, core recovery generally ranged from 90% to 95% and field procedures were the same as for the No. 7 Deposit. Scott Wilson RPA is of the opinion that the quality of the No. 2 Deposit database also is acceptable to estimate and report mineral resources.

19.1.2 Density Measurements

Historical Density Measurements

Scott Wilson RPA understands that systematic density measurements were made on drill core by staff of Priargunsky, and the average value used in estimation of mineral resources of the No. 7 Deposit (the 7a horizon in the old terminology) is 2.60 g/cc (Rogov and Yurchenko, 1987).

Recent Density Measurements

In preparation for the recently completed geophysical (gravity) survey, Khan carried out a number of density measurements on various rock types in the Dornod area. The average value of two measurements of siltstones, the rock type which hosts the uranium mineralization at Dornod, is 2.575 g/cc, which is less than a 1% variance from the historical reported value. Scott Wilson RPA has retained the old value of 2.60 g/cc and used it in the current resource update.

19.1.3 Geological Interpretation and 3D Solids

No. 7 Deposit

The drill holes in the new No. 7 Deposit database were plotted on east-west drill sections at 50-m intervals. Scott Wilson RPA reviewed the previous interpretation of the mineralized zones, based on uranium assay levels, with a threshold of approximately 0.015% U_3O_8. The main mineralized unit occurs within an uraniferous horizon spread over an area measuring 450 m by 400 m shows good continuity along strike, and is almost flat lying. This mineralized zone is situated at the base of interlayered andesite and basalt overlying a conglomerate and sandstone unit.

No. 2 Deposit

The procedures for the geological interpretation and continuity of mineralization at the No. 2 Deposit were the same as for the No. 7 Deposit, except for the threshold level, which is 0.010% U_3O_8. This is lower than the threshold used in the 2006 Scoping Study. Consequently, additional mineralized layers (1, 2, 3, 5, and 7) have been interpreted compared to the earlier interpretation which included only two mineralized layers (2B and 2б). Furthermore, low-grade mineralization is interpreted to extend much farther along strike (beyond the existing open pit) than presented in earlier Scott Wilson RPA reports. All of the No. 2 Deposit mineralization is hosted by sandstone to conglomeratic units interlayered with felsic volcanic rocks.

Wireframe Models

Scott Wilson RPA developed 3D solids using Gemcom software from the mineralized lens outlines on the cross-sections. Scott Wilson RPA constructed 3D wireframe models using 3D wobbly polylines that were snapped on to the drill-hole intervals. Polylines were created both on cross-sections and on level plans. The polylines were joined together using tie lines. At model extremities, polylines were extrapolated for approximately 50 m beyond the last drill-hole intercept. As a final check, the wireframe solids were validated.

Cutting of High Values

Since there are some high-grade assays in the drill-hole database of the No. 7 Deposit, and the assays have a strong positive skewed distribution, Scott Wilson RPA considered it necessary to cut the high uranium values to 1.70% U_3O_8. This represents the 98th percentile of the total assay population. There was no cutting done for the assay database of the No. 2 Deposit.

Univariate Statistics

Statistics for the drill-hole assay data set within the mineralized zone outlined is presented in Table 19-2.

Table 19-2
Statistics of Drill-Hole Assays

	No. 7 Deposit		No. 2 Deposit
Statistic	**% U_3O_8 (Uncut)**	**% U_33O_8 (Cut)**	**% U_33O_8 (Uncut)**
Mean (length-weighted)	0.199	0.199	0.062
Median	0.073	0.073	0.024
Max. Value	2.210	1.700	1.032
Standard Deviation	0.258	0.253	0.092
Coefficient of Variance	0.692	0.703	0.673
TOTAL ASSAYS	**1,230**	**1,230**	**1,683**

Compositing and Statistics

Scott Wilson RPA composited the assays of the No. 7 Deposit into 2-m intervals down hole, and the assays of the No. 2 Deposit into 1-m intervals, for intervals inside the mineralized layers. For the No. 7 Deposit, composites less than 0.5-m long were excluded from the composite database, and for the No. 2 Deposit, composites 0.25-m long were excluded. There are a total of 3,297 drill-hole composites within the main mineralized zone of the No. 7 Deposit, and 3,510 drill-hole composites within the mineralized Layers 2, 3, 5 and 7 of the No. 2 Deposit. Overall, compositing involved 338 drill holes and face samples for the No. 7 Deposit and 447 drill holes for the No. 2 Deposit. Statistics for the drill hole composite data set are in Table 19-3.

Table 19-3
Statistics of Drill-Hole Composites

	No. 7 Deposit		No. 2 Deposit
Statistic	**% U_3O_8** (Uncut)	**% U_33O_8** (Cut)	**% U_33O_8** (Uncut)
Mean	0.187	0.187	0.054
Median	0.107	0.107	0.022
Max. Value	1.555	1.450	1.032
Standard Deviation	0.202	0.202	0.075
Coefficient of Variance	0.923	0.926	0.717
TOTAL COMPOSITES	**3,297**	**3,297**	**3,510**

Note: 2-m composites for No. 7 Deposit and 1-m composites for No. 2 Deposit.

19.1.4 Block Model and Validation

Two separate 3D block models were constructed in Gemcom based on the Gauss Kruger-Posgar coordinate system used for the Dornod No. 7 and 2 Deposits. For the No. 7 Deposit, the block size is 10 m (E-W) by 10 m (N-S) by 2 m (vertical). For the No. 2 Deposit, the block size is 10 m (E-W) by 10 m (N-S) by 1 m (vertical). The blocks were coded as to the mineralized layers noted above, or waste, based on the location of the centroid of the block relative to the 3D solids of the zones.

Grades were interpolated into the mineralized layer using only composites within the layer. Kriging was used to interpolate the uranium grades of the blocks. The search strategy used a search ellipse with long axes oriented along the strike and dip of the layer, and short axis across the dip. A minimum of 1 and a maximum of 20 composites were required for interpolation, with a maximum of 3 from any drill hole for the No. 7 Deposit. A minimum of 2 and a maximum of 20 composites were required for interpolation, with a maximum of 4 from any drill hole for the No. 2 Deposit. The limits of the block model are shown in Tables 19-4 and 19-5.

Table 19-4
Description of Block Model, No. 7 Deposit

	Easting	Northing	Elevation
Block Size (m)	10	10	2
Block Origin	306500	42905	610
No. of Blocks	120	70	60
Minimum (all)	306540	42985	502
Maximum (all)	307610	43515	598
Minimum (mineralized blocks)	306540	42985	502
Maximum (mineralized blocks)	307610	43515	598

Table 19-5
Description of Block Model, No. 2 Deposit

	Easting	Northing	Elevation
Block Size (m)	10	10	1
Block Origin	306800	43205	1050
No. of Blocks	240	230	350
Minimum (all)	306050	43705	700
Maximum (all)	307850	45255	918
Minimum (mineralized blocks)	306050	43705	700
Maximum (mineralized blocks)	307850	45255	918

Search Strategy and Grade Interpolation

For the No. 7 Deposit, a search ellipsoid using a minimum of 1 and a maximum of 20 composites was used to interpolate cut uranium grades into blocks, and the search ellipse was oriented with a grid northeast strike along the average dip of the zone, in most cases approximately 5° to the southwest. For the No. 7 Deposit, the search ellipsoid used had 135-m radius along strike, 110-m radius down dip, and 30-m radius for across strike in the vertical dimension.

For the No. 2 Deposit, the search ellipsoid used had a 135-m radius along strike, 110-m radius down dip, and a 30-m radius across strike in the vertical dimension, using the Gemcom Unwrinkled Method. This method uses special routines to convert the gently folded layers into horizontal layers, which is done before variography and block grade interpolation by the kriging technique. Scott Wilson RPA used a two-pass approach; the first search ellipsoid had a horizontal radius of 65 m and a vertical radius of 4 m. The second search ellipsoid had a horizontal radius of 150 m and a vertical radius of 10 m.

Block Model Validation

Scott Wilson RPA used three methods to validate the block model mineral resource estimate. These were:

- Visual inspection and comparison of block grades with composite grades
- Statistical comparison of composite and block grade distributions
- Comparison of composite and block grades by section.

There were no discrepancies in the above validation methods. Scott Wilson RPA, therefore, concludes that the Dornod Nos. 7 and 2 Deposits block models are valid, reasonable, and appropriate for supporting the mineral resource estimate.

19.1.5 Classification of Mineral Resources

Scott Wilson RPA classified the mineral resources in the Nos. 7 and 2 Deposits into the Indicated category, based on drill-hole spacing and apparent continuity of mineralized layers at a 0.015% U_3O_8 grade (for the No. 7 Deposit) and 0.010% U_3O_8 grade (for the No. 2 Deposit), and the results of the recent confirmation drilling.

All of the mineral resources of the No. 7 Deposit are considered as Indicated mineral resources.

The bulk of the mineral resources of the No. 2 Deposit are considered as Indicated mineral resources. A small amount has been classified as Inferred mineral resources, in an area extending both inside and outside (north) of the current boundary of Mineral Licence 237A.

19.1.6 Cutoff Grades

Scott Wilson RPA has estimated cutoff grades for both the Nos. 7 and 2 Deposits based on an average uranium price forecast, production cost, and expected recovery in the resource models. A price forecast of USD 55/lb of U_3O_8 was used, the same number used in the Prefeasibility Study.

No. 7 Deposit

For the underground No. 7 Deposit, the cutoff grade is based on:

- Metallurgical recovery of 90%
- Cash operating costs estimated to be USD 49.21/t, including USD 23.04/t for mining, USD 22.89/t for processing, and USD 3.28/t for administration
- Price of USD 55.00/lb of U_3O_8.

Based on the above, the cutoff grade for the Dornod No. 7 Deposit resource estimate is calculated as:

Cutoff = cost/(value x recovery) = USD 49.21/t /[(USD 55/lb U_3O_8) x 90%]=0.045% U_3O_8.

Scott Wilson RPA recommends reporting of the Dornod No. 7 resources at a cutoff grade of 0.04% U_3O_8. For comparison, mineral resources are presented at a range of cutoff grades in Table 19-6.

Table 19-6
No. 7 Deposit Mineral Resources, Various Cutoff Grades

Cutoff Grade (% U_3O_8)	**Category**	**Tonnes** (million)	**% U_3O_8**	**lbs U_3O_8** (million)
0.025	Indicated	17.81	0.130	51.2
0.040	**Indicated**	**14.36**	**0.154**	**48.6**
0.050	Indicated	12.08	0.174	46.4
0.075	Indicated	8.69	0.218	41.9
0.100	Indicated	6.78	0.256	38.2
0.150	Indicated	4.47	0.325	32.0
0.200	Indicated	3.41	0.372	28.0

Notes:
1. CIM guidelines were used in estimating and classifying mineral resources.
2. Mineral resources are estimated using a price of USD 55/lb U_3O_8.
3. The numbers for tonnage, % U_3O_8 and contained lbs U_3O_8 are rounded figures.
4. The grade values (% U3O8) are converted from %U values.

In plan view, the No. 7 Deposit block model shows a high-grade central core, with a large halo of mineralization in which the grade declines smoothly towards the edges. As the cutoff grade decreases, more and more of this halo is included in the resource estimate.

No. 2 Deposit

Open-pit modeling (detailed below, under mineral reserves) was carried out on the No. 2 Deposit wireframe, which was defined by a threshold value of 0.010% U_3O_8. The open-pit analysis calculated costs (including required waste stripping) versus revenue individually for each block in the model, so it is not possible to calculate a global cutoff grade beforehand.

An open-pit discard cutoff grade, which assumes that all material within the designed pit will be mined, can be calculated. This cutoff grade determines only whether a given block should be sent to the mill or to the waste dump. It does not determine whether that block should have been mined at all; the open-pit modeling makes that determination. The discard cutoff grade is based on:

- Metallurgical recovery of 93%
- Cash operating costs of USD 28.17/t, including USD 2.00/t for open-pit mining, USD 22.89/t for processing, and USD 3.28/t for G&A
- Price of USD 55.00/lb of U_3O_8.

Cutoff = cost/(value x recovery) = USD 28.17/t/[(USD 55/lb U_3O_8) x 93%] = 0.025% U_3O_8.

Scott Wilson RPA recommends reporting mineral resources at a cutoff grade of 0.025% U_3O_8. Mineral resources at a range of cutoff grades are listed in Table 19-7, for comparison.

Table 19-7
No. 2 Deposit Mineral Resources, Various Cutoff Grades

Cutoff Grade (% U_3O_8)	**Category**	**Tonnes** (million)	**% U_3O_8**	**lbs U_3O_8** (million)
0.010	Indicated	16.66	0.049	18.0
0.025	**Indicated**	**10.95**	**0.065**	**15.7**
0.050	Indicated	5.67	0.092	11.5
0.075	Indicated	2.98	0.120	7.9
0.100	Indicated	1.58	0.150	5.2
0.150	Indicated	0.53	0.211	2.5

Notes:
1. CIM guidelines were used in estimating and classifying mineral resources.
2. Mineral resources are estimated using a price of USD 55/lb U_3O_8.
3. The numbers for tonnage, % U_3O_8 and contained lbs U_3O_8 are rounded figures.
4. The grade values (% U_3O_8) are converted from %U values.

The No. 2 Deposit block model shows several areas of higher grade (>0.10% U_3O_8) mineralization, with the largest area concentrated underneath the current pit, and another area to the southeast. West of the current pit, grades start below 0.10% U_3O_8, and decrease gradually.

19.2 Mineral Reserves Estimate

Mineral reserve estimates for underground and open pit were developed by P&E.

The probable reserve estimate for the No. 2 deposit open-pit mine, at a 0.028% U_3O_8 cutoff grade, is 7 407 000 t grading 0.074% U_3O_8. Mining dilution of 15% at a 0.018% U_3O_8 grade is included.

The probable reserve estimate for the No, 7 deposit at a 0.061% U_3O_8 cutoff grade is 10 634 000 t grading at 0.174% U_3O_8. Underground mining recovery of 88% and dilution of 10% at 0% U_3O_8 grade is forecast. A summary of the mining reserves is shown in Table 19-8.

Table 19-8
Dornod Probable Reserves

Mining Area	**Cutoff U_3O_8 %**	Tonnes	**U_3O_8**	**lbs** (million) U_3O_8
No. 2 Deposit	0.028	7 407 000	0.074	12.1
No. 7 Deposit	0.061	10 634 000	0.174	40.8
TOTAL RESERVE		18 041 000	0.133	52.9

Note: When comparing the Prefeasibility Study (August 2007) Mineral Reserves Estimate, Table 4-4, to the DFS Probable Reserves, Table 19-8, there is an increase of 3.8 lbs (millions) U_3O_8. The increase of pounds of U_3O_8 is probably due to the increased detail of the estimating methodology.

20 Other Relevant Data and Information

20.1 Mining

The Dornod No. 2 deposit will be mined by open pit, while the No. 7 deposit will be mined by underground mining techniques.

The No. 2 Deposit's known, potentially economic mineralization, extends from surface to approximately 165 m below surface. The deposit is approximately 1400 m on strike length (east-west direction) and 1100 m in width (north-south direction).

The No. 7 Deposit's known, potentially economic mineralization, extends from approximately the 400 to 480 m below surface elevations. The deposit is approximately 600 m on strike length (east-west direction) and 500 m in width (north-south direction).

20.1.1 Existing Development

The No. 2 Deposit was mined to an approximate depth of 80 m in the past by open pit. This pit is now flooded. Waste dumps are located to the north and northwest of the existing pit rim.

Access and underground development for the No. Deposit includes:

- Two 6-m-dia concrete lined Nos. 3 and 4 Shafts (not presently useable as they are located inside a conservation area) located to the southeast of the orebody to a depth of approximately 453 m below surface
- Underground lateral and raise development of approximately 2658 m on the 453-m Level.

All underground workings are presently flooded and the shafts are capped with concrete.

20.1.2 Mining

The higher grade No.7 underground deposit will be developed and mined prior to the lower grade No. 2 open-pit deposit. The preproduction development schedule is presented in Table 20-1 and the life-of-mine production schedule is presented in Table 20-2.

(a) Underground Preproduction Development

Preproduction mine development and construction, including initial mining blocks, requires approximately 3 years to complete. All preproduction development and construction will be performed by a mining contractor. Work to be completed during the preproduction period has been presented in Table 20-1 and will include:

- Dewatering of existing underground workings and discharge to existing No. 2 open pit
- Developing the main ramp from surface to the 483 Level
- Sinking and lining the FAR No. 1 (near No. 3 Shaft) and RAR Nos. 1 and 2
- Constructing and installing main surface ventilation fans on raises and No. 3 Shaft

- Constructing miscellaneous surface facilities related to the mine
- Completing the northwest internal ramp and lateral development on the 483, 453, 435 and 405 Levels
- Installing 483 Level infrastructure (maintenance shop, refuge station, fuel bay, explosives and detonator magazines, sumps, etc.)
- Developing initial internal ventilation raises
- Installing and commissioning all required mine services.

Table 20-1
Preproduction Development Schedule

Component	Quantity	Units	Dimensions	Year -3				Total	Year -2				Total	Year -1				Total	**TOTAL**
				Q4	Q3	Q2	Q1	Year -3	Q4	Q3	Q2	Q1	Year -2	Q4	Q3	Q2	Q1	Year -1	
Underground Infrastructure Development																			
Main Ramp Surface to 510 Level	3,860	metres	5m W X 5m H		420	420	420	1,260	420	420	420	420	1,680	420	420	80		920	**3,860**
Lateral Development		metres						0					0					0	
Internal Ramp 482 to 435			5 m W X 5 m H					0					0	96	700			796	**796**
405 Level Main Accesses	115	metres	5m W X 5m H					0					0			62		62	**62**
435 Level Main Accesses	2,515	metres	5m W X 5m H					0					0	193	360	704	118	1,375	**1,375**
453 Level - Main Accesses	633	metres	5m W X 5m H					0					0					0	**0**
483 Level Main Accesses	2,811	metres	5m W X 5m H					0				722	722	155			422	577	**1,299**
Truck Loading Stations	320	metres	5m W X 10m H					0					0				60	60	**60**
Raises																			
Ventilation Raises	832	metres	4m X 4m					0				42	42	664	29		97	790	**832**
Backfill Raise	1,000	metres	2.4m X 2.4m					0					0		500			500	**500**
Mine Services								0					0					0	
483 Trackless Maintenance Shop	18,234	cu.m.						0					0			18,234		18,234	**18,234**
453 Explosives Magazine	803	cu.m.						0					0		803			803	**803**
453 Detonators Magazine	57	cu.m.						0					0		57			57	**57**
483 & 510 Refuge Stations	1,606	cu.m.						0		803			803	803				803	**1,606**
483 and 510 Latrines	148	cu.m.						0		74		74	148					0	**148**
483 Fuel Bay	439	cu.m.						0					0	439				439	**439**
510 Fuel Bay	439	cu.m.						0					0	439				439	**439**
483 & 453 Storage Areas	60	metres	6m X 5m H					0			30		30		30			30	**60**
510 Main Dewatering Sump	705	cu.m.	7 m dia.					0			705		705					0	**705**

(b) Underground Mining

The No. 7 Deposit will be accessed by a main truck haulage ramp from surface. All ore will be hauled to surface by underground haulage trucks. All mining areas will be accessed by the main ramp, using mobile diesel powered mechanised equipment (see Figure 20.1).

Underground mining will utilise mobile rubber tire diesel powered equipment to produce 3500 t/d or 1 225 000 t/a of reserves.

The main proposed mining method is Longhole Open Stoping using downholes. Near the top of the orebody, Longhole Open Stoping, using upholes will also be employed. Longhole stopes will be backfilled with a combination of cemented and unconsolidated waste rock.

Stopes will have maximum nominal dimensions of 15-m wide by 18-m long and a 30-m-vertical height.

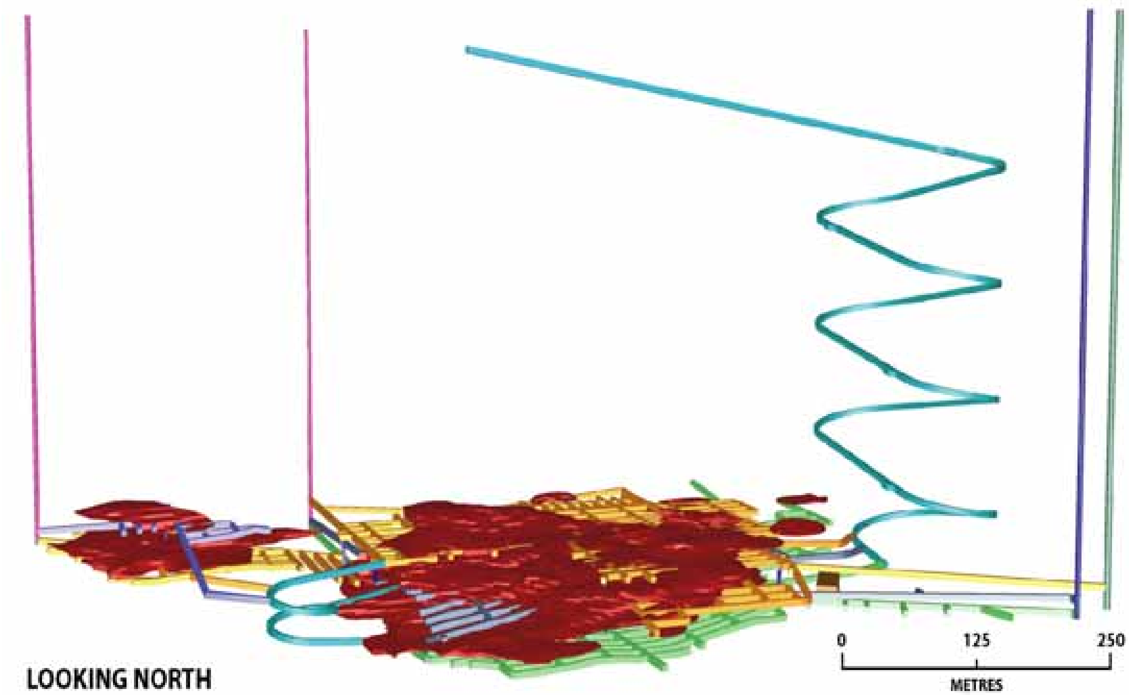


Figure 20.1
Underground Mine 3D View

(c) Open-Pit Mining

The Dornod open pit will be a conventional open-pit mining operation that will encompass the open-pit mining and processing of 3500 t/d (1 225 000 t/a) of ore to produce U_3O_8. The projected operating life of the pit is approximately 6 years. Total daily production of ore and waste will average 55 000 t/d throughout the pit life.

The proposed Dornod open pit will be developed at the site of the former open pit. The historic pit will be dewatered and further developed to expand into the proposed Dornod open pit. It is envisaged that the open pit will be developed concurrent with the last year of underground mining from the No. 7 Deposit (Year 9) and that the historic pit will be dewatered as part of the underground mining and ore processing operations (see Figure 20.2).

The Dornod open pit will be developed by Khan using its own equipment and workforce. Khan will have responsibility for: the dewatering of the historic pit and re-establishment of the pit haulage roads; production drilling and blasting; the excavation of ore to the primary crusher and waste rock to the waste rock management area; oversize breakage; haul road maintenance; and equipment maintenance. Mining will be accomplished on 10-m-high waste and 5-m-high ore benches utilising a conventional fleet of off-road haulage trucks, a front-end loader, a hydraulic shovel and ITH blasthole drills.

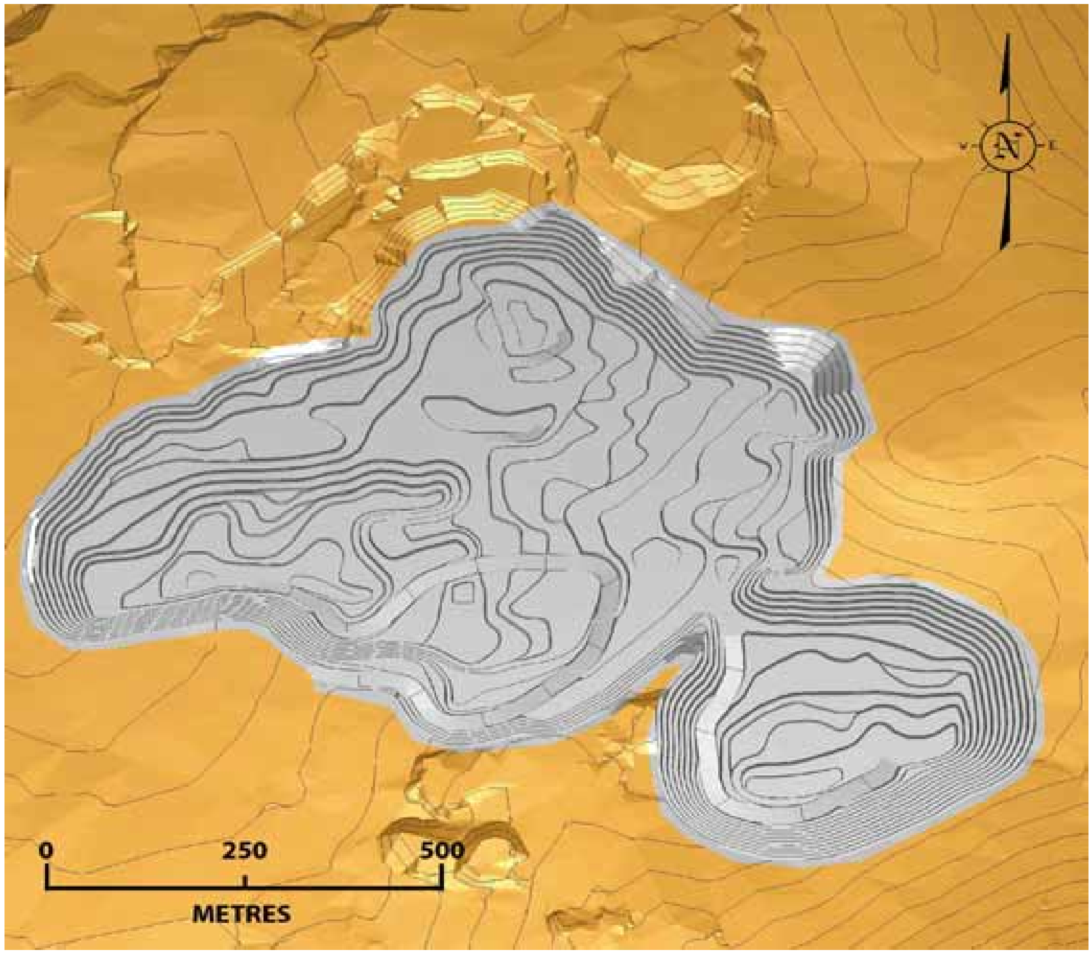


Figure 20.2
Open-Pit Mine 3D View

Table 20-2
Dornod Life-of-Mine Production Schedule

Year	Source	Underground Ore Mined (Tonnes)	Underground Grade (% U3O8)	Open Pit Ore Tonnes	Open Pit U3O8 %	Total Mined Ore Tonnes	Total Mined U3O8 %	Mill Feed Ore Tonnes	Mill Feed U3O8 %
-2	UG	2,000	0.062			2,000	0.062		
-1	UG	97,000	0.181			97,000	0.181		
1	UG	755,000	0.230			755,000	0.230	854,000	0.224
2	UG	1,228,000	0.234			1,228,000	0.234	1,225,000	0.234
3	UG	1,226,000	0.183			1,226,000	0.183	1,225,000	0.183
4	UG	1,226,000	0.208			1,226,000	0.208	1,225,000	0.208
5	UG	1,226,000	0.166			1,226,000	0.166	1,225,000	0.166
6	UG	1,229,000	0.136			1,229,000	0.136	1,225,000	0.136
7	UG	1,225,000	0.115			1,225,000	0.115	1,225,000	0.115
8	UG	1,225,000	0.149			1,225,000	0.149	1,225,000	0.149
9	UG & Pit	1,195,000	0.167	26,000	0.068	1,221,000	0.164	1,225,000	0.164
10	Pit-Ph-1			1,225,000	0.093	1,225,000	0.093	1,225,000	0.093
11	Pit-Ph-1			1,225,000	0.082	1,225,000	0.082	1,225,000	0.082
12	Pit-Ph-1&2			1,225,000	0.075	1,225,000	0.075	1,225,000	0.075
13	Pit-Ph-1,2&3			1,225,000	0.070	1,225,000	0.070	1,225,000	0.070
14	Pit-Ph-2&3			1,225,000	0.058	1,225,000	0.058	1,225,000	0.058
15	Pit-Ph-3			1,225,000	0.066	1,225,000	0.066	1,225,000	0.066
16	Pit-Ph-3			31,000	0.086	31,000	0.086	37,000	0.086
Total		**10,634,000**	**0.174**	**7,407,000**	**0.074**	**18,041,000**	**0.133**	**18,041,000**	**0.133**

20.1.3 Geotechnical

Golder performed the rock mechanics studies on the Dornod No.7 Deposit (Golder report 05-1118-041).

Design parameters were provided and incorporated into underground the mine design and cost estimates. Due to the lack of geotechnical studies having been performed on the Dornod No. 2 Deposit, very conservative pit slopes were assumed.

(a) No. 7 Underground Deposit

All permanent opening will have arched backs. Ground support will generally consist of pattern bolting using grouted rebar, welded wire mesh and shotcrete in selective areas. Where the access drifts pass through the orebody, barrier pillars on each side of the access of at least three times the planned ore mining height should be established to protect the opening.

Stope dimensions are based on rock quality determinations and expected achievable open spans in stopes. The stope dimensions and extraction ratios anticipated will not be achieved without the use of a good quality, cemented waste rock backfill and to a lesser extent noncemented waste rock backfill.

(b) No. 2 Open Pit Deposit

At the time of this report writing, no geotechnical assessment had been undertaken on the Dornod pit to ascertain recommended slope design criteria. Due to the open pit not coming into production until Year 9 of operations, it was decided that the geotechnical assessment could wait until approximately late Year 7 of production.

In lieu of any geotechnical design criteria, P&E recommended that the pit slopes be reduced to an interramp design slope of 35 degrees which is approximately 10 degrees less than the existing Dornod pit. The 35-degree slope is also below the angle of repose for the rock in the pit area and, therefore, posses virtually no risk for a slope failure. The opportunity exists for a steeper design criteria and a reduction of waste rock in the actual design pit.

20.1.4 Mine Infrastructure

Surface support facilities will include a maintenance shop, explosives magazines, mine supervision, geology, engineering and administration offices, mine rescue station, power substation, warehouse, laydown yard, and water collection ponds.

All underground mine services including compressed air, service water, electrical distribution feeders and dewatering lines will be installed in the Main Intake Ventilation Raise and will be distributed to the mining areas along the 485 Level and in the main ramp. Other underground infrastructure will include:

- 483 trackless maintenance shop
- 483 fuel bay
- 453 explosives and detonator magazine
- 483 and 453 refuge stations
- 483 and 453 main storage areas
- Latrines
- Minewide wireless communication and control system.

20.1.5 Underground Mine Ventilation

The Dornod underground ventilation design details are in a report by Intergen Safety and Environment Solutions Inc.

The underground ventilation system is required to provide airflow volumes and distribution that will provide wholesome air for all underground workers. Specifically for this Project, the system is designed to control airborne radiation and diesel exhaust fume concentrations in the workplace.

The system will be designed to control airborne radiation concentrations to levels that, together with other radiation exposure management measures, are conducive to maintaining radiation exposures consistent with the ALARA principle.

Total air volume required underground is approximately 250 m^3/s (for all phases of mining) with the majority of this volume being moved in the Main Intake Raise. Two return air raises will remove contaminated air from the mine.

20.1.6 Underground Mine Radiation Mitigation

The grade of uranium in the mine and present exposure regulations will require significant mitigation measures for underground mine workers. The work schedule and shift rotations on-site and off-site require that workers work approximately one half of the year. Mitigation measures will ensure that for the working half of the year workers are exposed to the minimum amount of radiation possible. To achieve this requires, where practical, the following.

- Physical barriers (shields, cabs, bottled air breathing apparatus, etc.), and
- Job rotation of workers within tasks which are related to direct mining and transport of ore and those partially or not related to mining and transporting of ore.

20.2 Surface Infrastructure

The infrastructure requirements summarised in this item are based on information provided by Aker Solutions (Aker Solutions Site Plan, Figure 18.3).

20.2.1 Water

Flow modeling by Golder indicates that prior to open-pit mining, the existing Open Pit Lake can provide adequate water for the Project operations under mean annual precipitation conditions. After Year 7, additional water will be required from other sources, such as groundwater, to keep the system in balance after the Open Pit Lake has been depleted. The water required from other sources to run the operations varies between 7 m^3/hr in Year 8 and 25 m^3/hr in Year 15 for mean annual precipitation conditions.

Water collected in the RMA Pond is pumped directly to the mill from a pump barge to meet makeup water requirements. Additional water for the process is pumped from the existing Open Pit Lake for the first 7 years when the mine is an underground operation, and then from the Water Collection Pond after Year 7.

Ditches along the access roads in the process plant site convey the runoff from precipitation to the Open Pit Lake for the first 7 years and to the Water Collection Pond later.

Surface runoff from precipitation that does not drain naturally to the ponds or the open pit is conveyed in ditches to the water collection facilities. Sumps to allow pumping are required for water management at locations where the runoff does not drain by gravity to any of the water management facilities.

A water treatment plant will be constructed to provide water for the camp facilities, safety showers and eyewash stations.

A separate piping system will be provided for the firewater pump package and distribution system at the process plant and the camp.

All pipelines are to be pre-insulated HDPE pipe of varying diameters. Piping will be laid at grade or in partially cut trenches. They will be buried only where required to traverse roadways.

20.2.2 Power Supply

At the time of this report, a 35-kV power line was being built to bring power from Xin Xin Substation to the site.

A main substation at the mine site was also commissioned to provided voltage step-down from 35 kV to site primary distribution level of 6 kV.

The various plant substations will be provided for power distribution to drives and other services in the processing plant, underground, camp and administration building.

Emergency power will be provided by three 1-MV, 6-kV packaged diesel generator. Two for the plant and one for underground. A 1-MW, 400-kV plant will be located at the camp.

20.2.3 Heating

All buildings will be heated using electric furnaces and forced air circulation.

All (fresh) mine air will be heated when ambient air temperature drop below 5ºC to avoid freezing of services water and drain line pipes.

20.2.4 Compressed Air

Compressed air for the mine will be supplied by three compressors located near No. 3 Shaft. Two will operate at one time with the third on standby. The compressors will deliver a maximum of 45 000 L/min of compressed air at 8 bar pressure. The compressors will supply the main 254-mm compressed air pipeline in No. 3 Shaft.

20.2.5 Sewage

Sewage from the camp will be routed to a sewage treatment plant sized for effluent from 450 employees. Septic tanks will collect sewage from the administration office, security gatehouse and truck shop. The septic tank sludge will be pumped out and transported by truck to the treatment facilities.

20.2.6 Instrumentation and Control

Modern instrumentation and remote process control systems will monitor and control the operation of all facilities. The system will provide an information database and a computerized maintenance management system.

The control for the process plant is designed around a central Distributed Control System (DCS). It will be located in the plant control / computer room and control of the plant will be via a Distributed Communication Network (DCN).

Several of the unit components, e.g., the acid and oxygen plant, sewage treatment plant, etc., will be supplied as packages.

Process control functions will be by the DCS, while drive interlock functions will be controlled by a Programmable Logic Controller (PLC).

The major control rooms will be as follows.

- Mill Central Control Room – This facility will monitor and control all concentrator functions, the main electrical substation, fresh water supply and tailings pump.

- Ore Primary Crushing Control Room.

20.2.7 Dust Suppression

Dust collection systems will be provided to met or exceed Occupational Safety and Health Administration (OSHA), Mine Safety and Health Administration (MSHA) and local laws, ordinances and regulations.

They will be designed to maintain air entrainment into enclosures containing dust from crushing, transferring, sorting and conveying. The dust from low velocity pick-up points is to be conveyed at high velocity to a bag filter / fan unit. Ductwork will be designed for abrasives, self-balancing and low turbulence (to avoid dust accumulation). Bag filters will be designed for continuous operation by ongoing self-cleaning of the filter media.

Compressed air will be used to provide a sequential timed release back pressure to clean the various filter sections. The dust collects in the filter's hopper section and is discharged by opening a rotary valve and introducing a pneumatic pulse, back into the process stream. Cleaned air passing through the filter media is ducted outside via backward inclined, nonoverloading exhaust fan.

This is typical for the primary crushing, the reclaim and pebble crushing areas.

A bag filter system in the briquetting area will collect dust laden air from the bagging area and the concentrate dryer discharge. The discharge from the bag filter fan should be below 400ºF (205ºC) to maintain filter integrity.

20.2.8 Buildings

The main buildings to be constructed at the site are described below.

(a) Mine Dry and Main Offices

The mine dry and administration offices are at the southern boundary of the site adjacent to the camp. The 53-m by 32-m building is a single-storey steel frame structure. Side walls and roof are sandwich insulated cladding panels.

(b) Clinic and Accommodation

The camp and clinic are on a 167-m by 140-m site at the south end of the property. The camp consists of 78, 6-m-dia yurts. Each yurt houses four persons providing 312 places. The yurts are positioned in an open-triangle formation with catering, recreational facilities and the clinic within the courtyard. The compound will be enclosed with a perimeter fence. These facilities will be provided locally by an independent contractor.

(c) Truck Shop

The truck shop is a 150-m by 24-m by 19-m high steel frame building northwest of the polishing ponds. it is a steel frame building with insulated roof and walls. It provides maintenance facilities for light utility trucks and heavy (150-t) haul trucks and 50-t mine trucks. This shop will be built in two phases. Initially, it will accommodate light vehicles and underground equipment. Later, it will accommodate the larger open-pit vehicles. The shop will be divided into three service bays, wash bay, one lube bay and parts warehouse. A 40-t overhead crane will also be installed.

(d) Security Gatehouse

A 77-m^2 gatehouse will house security personnel who will manage and control movement into and out of the site.

(e) Process Complex

(i) Primary Crusher and Retaining Wall

The primary crushing building is a 12-m by 13-m by 24-m high steel frame building with insulated wall and roof cladding panels. It houses the 6-m by 8-m by 8-m deep steel dump hopper. The reinforced concrete wall at the primary crusher is 78-m-long overall and 13-m maximum height.

(ii) Tailing and Agitator Building

This is a 24-m by 12-m by 16-m high steel frame building clad with insulated roof and wall panels.

(iii) Conveyor Transfer House

The conveyor transfer house is 8-m by 7-m by 15-m high insulated steel frame building linking the ore transfer conveyor out of the primary crusher to the stockpile feed tripper conveyor.

(iv) Conveyors

The ore transfer conveyor out or the primary crusher runs 40-m west to the transfer house. The conveyor gallery is steel lattice structure supported at the primary crusher, the transfer house and at two internal piers. The piers are steel frames on reinforced concrete (RC) foundation. The stockpile feed conveyor runs 100 m from the transfer house over the ore stockpile and stockpile tunnel. The feed conveyor gallery is supported on 10-m steel portal frames at 10-m spacing. The SAG mill feed conveyor runs from within the tunnel to the processing building. Overall length is approximately 75 m in the tunnel and 65 m between tunnel and processing building. It goes from near ground level at the tunnel exit to 14 m aboveground at the processing building. The elevated part of the SAG mill feed conveyor gallery is supported on steel piers of varying heights.

(v) Crushed Ore Stockpile

The stockpile tunnel is centred under the stockpile feed conveyor between the transfer house and the processing building. It is a box section RC structure 92-m long with 1.5-m by 1.5-m chute openings at 10-m centres in the roof. Tunnel bore is 4.5-m high by 4-m wide. Roof and walls are 0.5-m thick, the floor is 0.7 m. It houses the tunnel conveyor which delivers the ore to the SAG mill ore conveyor outside the tunnel. The tunnel runs roughly horizontally with floor level near ground level at the northern end.

(vi) Process Plant Building

The process plant building envelope is 207-m by 54-m by 26-m high steel frame structure. Columns are at 8-m centres longitudinally and 18-, 9-, and 27-m laterally. Roof and wall cladding are sandwiched insulated panels. It houses the ball and SAG mills, leaching tanks, freshwater and firewater tanks, effluent treatment and reverse osmosis plant and other processing equipment and facilities. The building envelope foundations are a combination of RC pad and strip footings. Foundations for heavy equipment within the building are RC pads or rafts made structurally separate from the building envelope foundation. The building is founded on predominantly on fill ranging in depth from 0 to 1.5 m.

(vii) Process Plant Tailings Thickener

The 58-m-dia tailings thickener tank will be located north of the flotation building. The thickener walls are to be supported on a concrete ring-wall. The center mechanism will be supported on a central concrete pump chamber. The tank bottom will rest on a compacted engineered fill.

There will be two concrete rectangular tunnels exiting from the pump chamber, a main access tunnel containing piping and an emergency exit tunnel. Both tunnels will exit above grade within a closed-in housing. Both tunnels and the pump chamber will bear on rock.

There will be a secondary containment HDPE liner, along with a containment dike, provided in case of accidental spillage.

(f) Core Sample Storage Building

The core storage area is to be located about 800-m south from the process complex. It will consist of the storage and core preparation areas. The building will be of steel-framed construction with sandwich panelled roof and walls.

(g) Explosive Complex

The complex will consist of the explosive emulsion plant and the explosive storage building. It will be located about 5500-m east of the process plant area. An outside Vendor will provide details of this facility at the next stage of engineering.

20.2.9 Roads

(a) Plant Site Access Roads

The plant access roads start at the site boundary at the security check point. One branch bears north towards the open mine pit exit then backs towards the eastern side of processing area, the accommodation facilities and main offices. A second branch runs south towards the truck maintenance shop, polishing ponds and western side of the processing area. The plant site access road is 10.0-m wide. Estimated total length is 4.0 km.

(b) Mining Haul Road for Heavy Trucks

The haul road is required from the pit exit ramp to the primary crusher and to the truck shop and refuelling station during Phase 2 Operation. Total length is approximately 1.5 km.

20.3 Geotechnical Considerations

This section of the report has been prepared by Golder in support of the DFS for the Project. For the DFS reporting, Golder was involved with the following items:

- Geotechnical field investigations
- Site selection for the Residue Management Area (tailings pond)
- Site-wide water balance
- Water management plan
- Plant foundations and waste rock stability
- Residue (tailings) management area layout and design
- Residue thickening tests
- Conceptual closure plan.

Golder was also involved in the early background technical, environmental and social studies and also the geotechnical studies described with the underground mine design.

Golder's findings are reported in Item 23, References, as well as the DFS.

20.3.1 Project Description

The Project is located about 600-km northeast of Ulaanbaatar near the former mining town of Mardai in northeastern Mongolia. The design ore reserve is 18.0 Mt that will be mined at a daily rate of 3500 t/d for 350 d/yr. The mine life is slightly longer than 15 years.

The Project has two orebodies, Nos. 2 and 7, which were previously mined by the Soviet company, Priargunsky, between 1988 and 1997. The majority of the No. 2 deposit was mined as an open-pit operation. Currently, the open pit is partially flooded. The No. 7 deposit was partially developed as an underground operation.

The proposal is to first mine No. 7 Deposit underground and then continue the No. 2 Deposit as an open pit. The underground mine will be operated from Years 1 to 8.5. Prestripping of the open pit will begin in Year 8. The open pit will be operated after that until the end of the operations phase.

The main components of the Project will be one of the already existing shafts (Shaft No. 3, used to access the underground workings during preproduction and then converted to the main ventilation intake raise during operations), a new production ramp and portal, two return air raises, an ore processing plant, a RMA, a Waste Rock Storage Facility (WRSF), a Water Collection Pond, and a Polishing Pond.

The milling process consists of sulphuric acid leaching and subsequent neutralisation to recover uranium oxide (yellow cake). The residue leaving the RIP circuit is neutralised and thickened before it is sent to the RMA.

The process will produce two waste streams, as follows:

- Leach residue – will comprise particles which have not been dissolved in the leaching process, together with gypsum resulting from the neutralisation of entrained sulphuric acid, and some metal hydroxides
- Effluent treatment residue – will report to the underflow of the primary and secondary effluent treatment clarifiers. This will contain gypsum and some metal hydroxides.

The waste streams will be combined and pumped to the RMA. Prior to disposal, the waste streams will be treated with lime, so that the pH is neutral to slightly basic.

Water for the Project will be obtained from the ponded water within the RMA, the so-called RMA Pond, the Water Collection Pond, which collects surface runoff from precipitation on-site, along with overflow from the Polishing Pond, which will collect water from the underground workings. Water will also be obtained from the open pit which will collect precipitation and groundwater inflow.

20.3.2 Site Description

The site lies in a remote, sparsely populated region. The landscape is characteristic of the semi-arid high steppe that is typical of the Eastern Steppe of Mongolia. It is characterised by small conical hills and gently sloping grass covered plains with rare stands of birch and aspen. Permanent surface water bodies, such as lakes, streams or springs, are rare to absent. Seasonal streams and ponds may appear following rainy periods (Golder, 2008a).

The Project is located within a northwest trending valley surrounded by gently sloping hills. The area has a low to moderate topographic relief, with elevations between 900 and 1000 masl (Golder, 2008a).

20.3.3 Operating Data

The provided operating data are summarised below:

• Ore reserves	19.4 Mt
• Nominal production rate	3500 t/d
• Operating days	350 d/a
• Nominal annual production (3500 x 350)	1 225 000 t/a
• Life of mine	15.8 years
• Residue (tailings) / ore ratio	1.2066 by mass
• Density of supernatant	1.00 t/m^3
• Discharge slurry density	48.2% solids by mass
• Moisture content of the ore	4% of total weight
• Moisture content of concentrate	1% of total weight
• Clean (not recycled) makeup water required	16% of total flow in the mill
• Water lost in the mill to evaporation and spillage	2% of total flow in the mill
• Water used for dust control in the open pit	500 m^3/d during summer
• Potable water (treated)	55 m^3/d
• Sewage	90% of potable water
• Water in cement rock underground backfill	25 m^3/d

It is important to understand nominal and design values as used in this report. Nominal values are based on the planned annual mill throughput averaged over 350 d/a. The nominal values are used to size the RMA and for flow (water balance) modelling. The design values are larger and take into account the availability of the mill (percentage of the year that the mill is available to operate) plus an appropriate factor of safety which is understood to be unity. The design values are used to size the process facilities, pipelines and pumping systems.

Based on the information provided, it is understood that the nominal production rate is 3500 t/d for 350 d/a for a nominal annual production of 1 225 000 t (3500 x 350). In other words, Khan plans to put 1 225 000 t of ore through the mill each year. The sizing of the RMA and more importantly the flow model is based on this number.

It is important to note that the residue (tailings) / ore ratio is 1.2066. In other words, the mass of residue is about 21% greater than the mass of ore. The residue discharge is considered to be a conventional slurry at a slurry density of 48.2% solids by mass and the density of liquor (supernatant) is unity.

20.3.4 Summary of Background Studies

The following are summaries of background studies that influence the design of the RMA. The details are in the reports listed in Item 23, References and the DFS. Detailed environmental and social baseline conditions of the Project region are presented in the Environmental and Social Impact Assessment (ESIA) Report by AATA.

(a) Climate

The site has a continental climate, with hot summers and cold winters. The nearest meteorological station is in Dashbalbar Soum, which is approximately 50-km north of the Project site. The station has 10 years of collected data on precipitation and temperature, between 1994 and 2004 (Golder, 2008a). The mean annual precipitation at the Project site is approximately 245 mm and the mean annual lake evaporation is approximately 700 mm (Eco-Trade, 2006). Approximately 70% of the precipitation falls as rain during the summer months (June through August) and approximately 5% of the precipitation falls as light snow during the winter months (November through March). The mean annual temperature is 7ºC with a highest monthly mean of 20ºC in July, and the lowest monthly mean of –20.4ºC in January. Freezing conditions typically occur from November to mid-April.

Prevailing winds are from the north or northwest. Mean monthly wind speeds vary from 3 to 6 m/s with a maximum monthly mean of 24 m/s (Golder, 2008a).

(b) Seismology

The seismic risk in the Project area has been evaluated (Golder, 2008b), in accordance to a site-specific deterministic seismic hazard assessment (DSHA). The preliminary assessment concluded that the Project area lies within a region of relatively low level of historical seismicity. A peak ground acceleration of 0.26 g, which corresponds to a 1-in-475-yr earthquake event plus one standard deviation (e.g., 84th percentile), is recommended for design.

More detailed site-specific DSHA and / or probabilistic seismic hazard assessment (PSHA) studies may be required for the development of design-level ground motions such as earthquake time histories and design response spectra.

(c) Geology

The Dornod uranium district is within the North Choibalsan mineral region in extreme northeast Mongolia, in the Northern Megablock at the eastern end of the Central Mongolian Fold System.

In the North Choibalsan mineral region, geosynclinal subsidence in the late Precambrian resulted in the accumulation of the continental-volcanic deposits (sandstones, shales and diabase sheets) of the Erdenedavaa Formation. Continued tectonic-magmatic activity during the late Paleozoic era formed plutons of granite, granodiorite, monzonite, syenite, and gabbro-diorite in the region.

The Dornod uranium district is in the central portion of the Choibalsan-Onon volcanic belt on the north flank of the Dornod volcanic structure. The significant geologic formation in the district is the late Jurassic Dornod Complex, a series of volcanic-sedimentary strata, from 1000- to 1500-m thick. Extensive northeast, northwest, and north trending faulting created ore-controlling and ore-containing structures throughout the Dornod area.

The area of the Project property is underlain by Jurassic volcanic and sedimentary rocks. The volcanic rocks comprise amygdaloidal basalt, andesite, ignimbrite, rhyolite and tuff. The sedimentary rocks are dominantly sandstone and conglomerate containing interbedded carbonaceous partings.

The bedrock surface geology for the Project site was provided to Golder by Khan contract geologist John Kita. Locally, at the Project site, the bedrock surface is comprised of both felsic intrusive and rhyolitic volcanic rocks that are exposed as rare outcrop or overlain by overburden.

The site lies in an area that was not glaciated. The primary controls on rock degradation are cold, dry weather, physical weathering rather than chemical weathering and water. Some local reworking (erosion and deposition) of these weathered zones has occurred around the topographic highs, resulting in shallow transported deposits (colluvium) at some locations. The geological and weathering processes in the site resulted in an overburden profile (colluvium, saprolite and transition from highly-weathered to moderately-weathered bedrock) with a thickness varying from about 2 to 12 m. A generalised overburden and rock-weathering profile is shown on Figure 20.3.

From an engineering perspective, the overburden soils are essentially silty sand to silty clay. Stratigraphic summaries of the geotechnical boreholes by study area (WRSF, processing plant site and RMA) are summarised on Table 20-3 and records of the boreholes are presented in the Geotechnical Field Investigations Report (Golder, 2009a).

(d) Background Environmental

The site is relatively barren and dry with modest relief (900 to 1100 m). The principal vegetation is grassland which supports a bird population.

(e) Social and Community

The area is sparsely populated with primarily nomadic herders. The closest settlement is the remains of the Soviet-built former mine community of Mardai which is located about 14 km from the site. It is the closest significant population with an estimated population of between 100 to 500 people, depending on the time of year (Golder, 2008a). The economic conditions are extremely poor. There is no official representative or local authority. The town's infrastructure has been abandoned and largely destroyed.

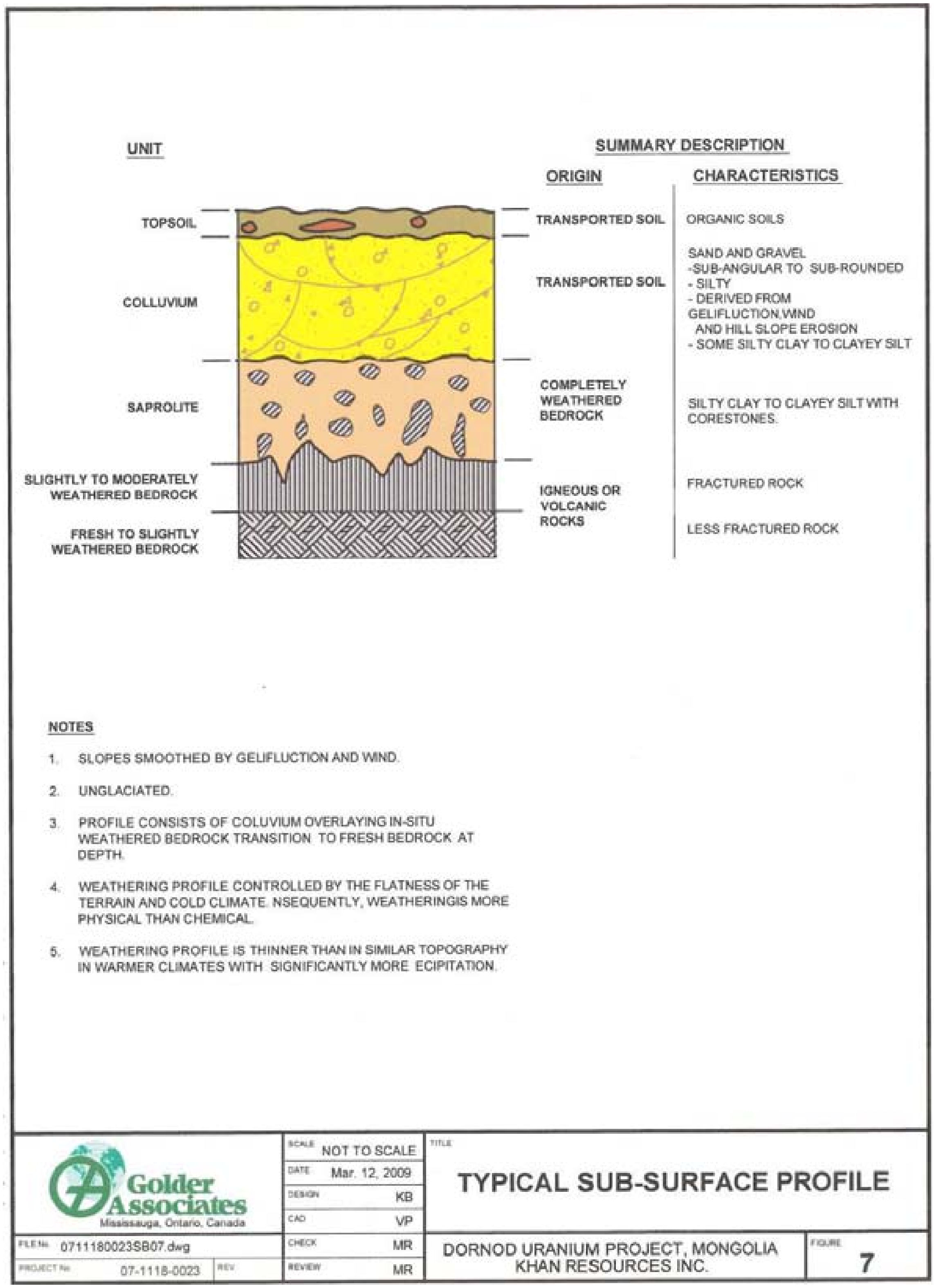


Figure 20.3 – Typical Subsurface Profile

Within a 30-km radius of the site, the Dashbalbar administration reports that there are 238 households. Aside from Mardai, none of these are grouped together. Approximately 21 households were located within a radius of 10 km of the site by AATA during a socioeconomic survey conducted in June 2008. These households are movable gers and residents interviewed said that they often travel farther from the site in winter months. Based on surveys, the average persons per household in the immediate area of the Project is 4.4.

Involuntary resettlement is not an issue.

(f) Archaeology

The investigation on the site performed in 2007 by Golder did not reveal any archaeological objects or cultural heritage assets.

Table 20-3
Summary of Subsurface Conditions Residue (Tailings) Area

Notes:

1. The details of the investigation are given in a seperate report (Golder 2009c)
2. The field investigation was carried out in April and May of 2008
3. The laboratory testing was carried out by Soil Trade in Mongolia

Stratigraphy
Colluvium
Saprolite
Weathered rock
Sound rock
Water table in weathered rock

Scope	
Boreholes	13
Test pits	29
Gradation	39
Atteberg limits	32
Permeability tests	7

Parameter		Min.	Mean	Max.
Stratum thickness (m)				
Colluvium		0.3	2.4	3.3
Saprolite		0.6	4.4	3.5
Weathered rock		2.2	7.4	22.4
Total depth	colluvium and saprolite	0.9	7.3	12.5
	colluvium,saprolite & weathered rock	8	14.8	28
Standard Penetration Test (SPT) results in boreholes (blows/ft)				
Colluvium		7	33	> 85
Saprolite		6	25	60
Weathered rock			Not applicable	
Moisture content (% moisture / dry weight of soil)				
Colluvium		1.1	8.0	27.3
Saprolite		6.4	18.7	72.0
Atterberg Limits (% moisture)				
Colluvium			Not applicable	
Saprolite	Liquid limit	19.1	33.9	55.5
	Plastic limit	9.6	13.0	18.4
Weathered rock			Not applicable	
Specific gravity				
Colluvium		2.62	2.68	2.73
Saprolite		2.69	2.72	2.74
In situ Field hydraulic conductivity (cm/sec) from falling head tests in piezometers and packer tests				
Colluvium (sandy-not tested)			none	
Saprolite (one test)			1×10^{-5}	
Weathered rock		4.5×10^{-6}	5×10^{-5}	1×10^{-4}
Sound rock		9×10^{-7}	1×10^{-5}	3.3×10^{-5}
Standard Proctor compaction of saprolite (two tests)				
Maximum density (t/m^3)			1.6	
Optimum moisture content (%)			14.7	

Gradation (mean %). Curves on Figure 8	Gravel	Sand	Silt / Clay
Colluvium	36.4	50.1	13.5
Saprolite	18.4	47.4	34.2

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20.3.5 Preliminary Foundation Recommendations Plant Site and Waste Rock Storage Facilities

(a) Assumptions

As part of the assessment of the subsurface conditions, and soil and bedrock properties in the plant site area, a geotechnical drilling and test pitting investigation was carried out by Golder in April and May 2008 at the location of the originally proposed plant site. It should be noted that based on the latest site plan provided to Golder by Aker Solutions, the proposed plant site has been relocated to an area of the site where no subsurface information was obtained during the geotechnical investigations carried out by Golder in 2008 (Golder 2009a). As such, the recommendations provided below are based on the general stratigraphic information and subsurface conditions as encountered in boreholes and test pits located near to the current plant site location.

Foundations recommendations assume that the groundwater table is located at a depth greater than 25 m below the underside of the foundations in the plant site, based on the general conditions noted during the recent geotechnical investigations. If the level is closer to ground surface than assumed, the foundation recommendations may change.

(b) Geotechnical Information

During the geotechnical investigations carried out by Golder in 2008, a total of eight test pits were excavated and six boreholes were drilled and sampled at locations within the originally proposed plant site as shown on Figure 20.4. The boreholes were advanced through the overburden soils and bedrock was cored to depths ranging from 11 to 30 m. The test pits were excavated to refusal on bedrock which occurred at depths from 1.1 to 5.5 m. Following completion of the geotechnical investigation and Golder’s demobilisation from the site, the proposed plant site was moved approximately 500 m to the west of the original location.

Records for the boreholes and test pits advanced / excavated as part of this investigation are provided in Golder (2009a).

(c) Results and Recommendations

In general, based on the available information in the closest borehole and test pit, the subsurface stratigraphy at the location of the currently proposed plant site is assumed to consist of a thin layer of surficial silty sand topsoil underlain by a deposit of colluvium (loose to dense, silty sand to sandy silt containing cobbles, boulders and organic matter), overlying a weathering profile, varying from a saprolite to weathered to fresh bedrock (Golder, 2009b). The saprolite is anticipated to be a stiff to hard silty clay with trace to some and gravel containing cobble and boulder size pieces of less weathered rock (core stones). The thickness of the overburden (colluvium and saprolite) / depth to bedrock will vary across the plant site area and is anticipated to be as shallow as about 0.5 m near the south end to as deep as about 14 m near the north end.

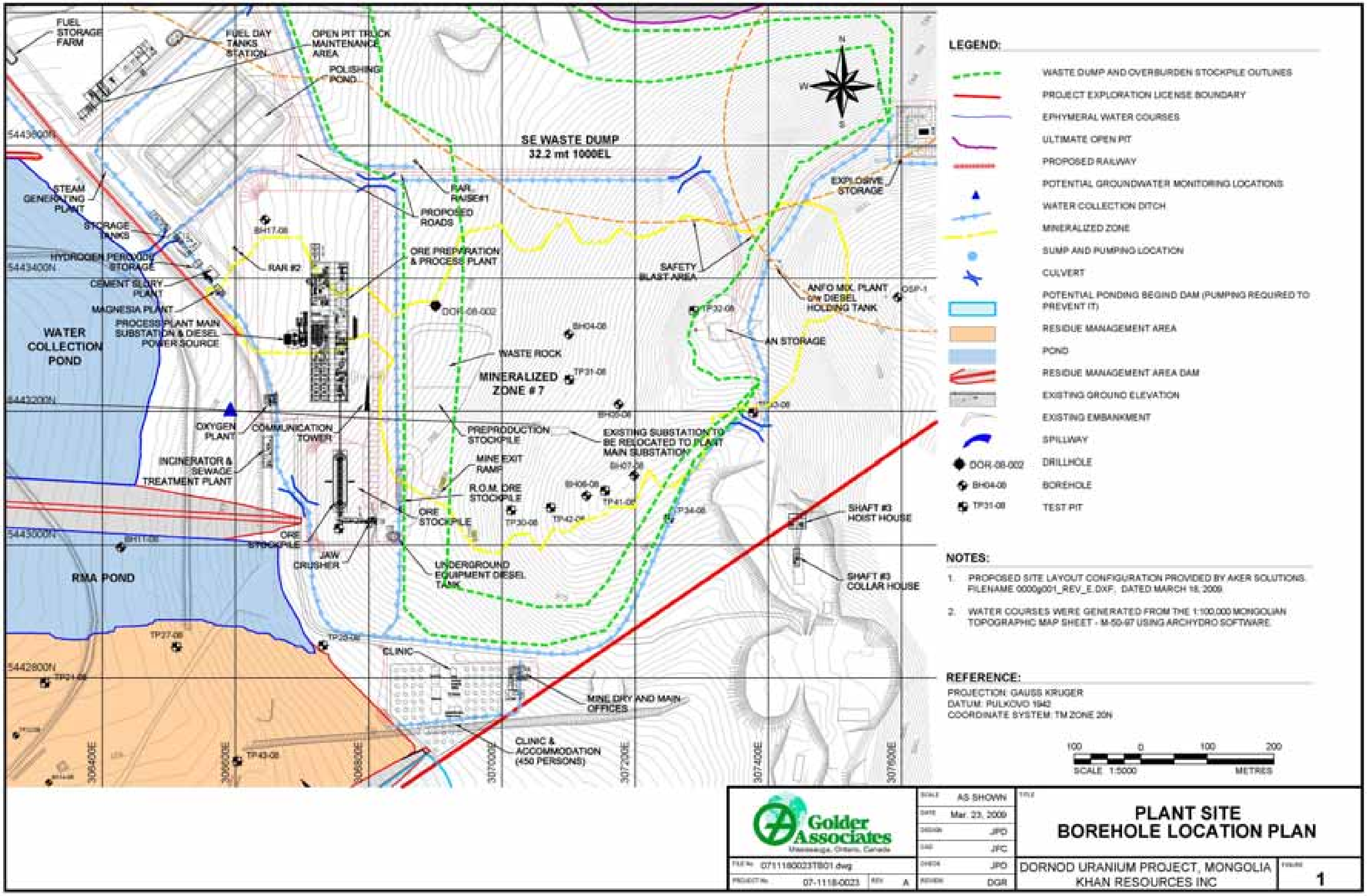


Figure 20.4 – Plant Site Borehole Location Plan

In the area northwest of the proposed ore preparation and process plant (at BH 17-08), the colluvium was found to be approximately 2.4-m thick and the underlying saprolite was found to be approximately 10.2-m thick underlain by fresh to highly weathered rhyolite bedrock.

At the south end of the proposed plant site, in the area of the ore stockpile (at TP 29-08), the overburden is comprised of thin deposits of colluvium consisting of silty sand containing some gravel and cobbles over fractured bedrock encountered at a depth of about 0.4 m.

Water levels in piezometers installed in other areas of the Project site indicate a water table that is greater than about 25 m below the elevation of the proposed plant site foundations; however, the groundwater table may be subject to seasonal variations. As such, it is important that piezometers be installed in the current proposed plant site area, as part of detail design to check the actual groundwater conditions near and around the plant site and to define the seasonal (and yearly) variation of the groundwater table.

Given the variable composition and potential for the presence of organics within the colluvium, it is recommended that the colluvium be stripped from the plan limits of the proposed plant structures prior to engineered fill placement and / or foundation construction.

Heavy proof rolling of the rough grade level should be included as part of the earthworks. Any localised loosened or softened areas identified during proof rolling should be subexcavated to expose more competent portions of the saprolite and replaced with engineered fill.

Based on the subsurface conditions encountered in the closest borehole and test pit to the proposed plant site, the majority of the equipment and structures can likely be constructed on either shallow spread footings or rafts founded on properly prepared subgrade.

For raft foundations and spread footings founded on the very stiff to hard, native saprolite soils, a maximum allowable bearing pressure of about 200 kPa is recommended for preliminary design of the structures in the plant site area assuming that all rafts / footings are founded at least 1.5 m below the final adjacent ground surface.

For raft foundations and spread footings founded on silty clay engineered fill, a maximum allowable bearing pressure of 150 kPa is recommended for design. Where foundations are constructed directly on the properly prepared bedrock, an allowable bearing pressure of 2500 kPa may be assumed for preliminary design, assuming that all loose, shattered and / or fractured rock within the footprint of the foundations is removed and replaced with mass concrete prior to construction.

Static slope stability analyses were carried out to assess the stability of the proposed ore stockpile at the south end of the plant site. The stockpiles were assumed to have a diameter of 20 m and a height of about 7 m (based on an angle of repose of 35º for the crushed ore). For this geometry, a factor of safety of about 1.2 to 1.3 is estimated for the global stability of the piles, based on the limited soils information currently available. A more detailed assessment of the stability of the ore stockpile(s) will be required as part of detail design.

Information regarding the location(s) and design (i.e., height) of the proposed WRSF has not been provided to Golder. Based on the GIS information collected by Golder, the existing waste dump piles are considered to be between about 15 and 20-m high, and constructed with a side slope angle of approximately 45º (i.e., the estimated angle of repose of the waste rock). New waste rock storage piles should be constructed with the same side slopes to no more than 15 m in height. If heights in excess of 15 m are required, a site-specific geotechnical investigation is recommended to asses the stability of the new waste rock storage piles.

20.3.6 Water Management Plan Report

(a) Assumptions

It has been assumed that the existing Open Pit Lake will have an available volume of 1.0 Mm^3 of water at start-up, and it will operate as a water storage facility for a maximum period of 7 years, before the open-pit prestripping start in Year 8.5.

It has been assumed that all the Project facilities are located within the property boundaries.

Information on anticipated groundwater inflow rates to the mine workings assumed for the DFS is limited to a report by Bruce (1998), indicating that the underground mine inflows were 25 m^3/h in the past.

No water quality constraints have been stated for water in the Residue Management Area (RMA) Pond, Water Collection Pond or Open Pit Lake to meet the process plant, underground mine backfill and dust control requirements.

The underground mine is flooded and it has been assumed that a volume of water of approximately 250 000 m^3 would be available and stored prior to start-up.

The operating data presented in Golder (2009a) has been confirmed with Khan and Aker Solutions before being used for the site wide flow model and the Water Management Plan (WMP). Nominal values have been used for all of the calculations and the design values including safety factors have been excluded form the calculations.

(b) Results and Recommendations

Flow Modelling and the Water Balance

A deterministic flow model for the Project was developed by Golder (Golder, 2009c). It has been used to simulate the site-wide flows, estimate the water collected within the Project site for water supply, and calculate maximum accumulation of water for pond sizing and maximum monthly flows for estimation of pumping rates. According to the results of the analyses, prior to open-pit mining, the existing Open Pit Lake, with a volume of approximately 1.0 Mm^3, is adequate to provide the water for Project operation requirements. It should be noted, however, that if 2 consecutive dry precipitation years were to occur in the first 7 years, additional water from other sources may be required (the 100-yr dry precipitation condition was adopted to simulate the dry years). These results are based on the assumption that 1 Mm^3 of water is available at start-up in the Open Pit Lake. It is important to note that the additional volume of water (~250 000 m^3) from the dewatering of the underground workings is not included in the calculations and, therefore, it should be considered as an additional reserve if the dry condition scenario is observed.

The flow modelling indicates that starting in Year 8, additional water would be required to keep the system in balance after the Open Pit Lake is depleted. The water required from other sources to run the operations varies between 7 m^3/h in Year 8 and 25 m^3/h in Year 15 for mean annual precipitation conditions.

(c) Surface Water Management System

Water collected within the Project site is stored in the RMA Pond, the Water Collection Pond and the Open Pit Lake to meet the operational water requirements. Water collected in the RMA Pond is pumped directly to the mill from a pump barge. The Open Pit Lake serves as a collection pond for the first 7 years. The Water Collection Pond will be built before the Open Pit Lake is depleted by the end of Year 7 and will serve as a water storage facility that will provide makeup water for the mill at times when enough water can not be recirculated from the RMA Pond to the mill, mostly during the winter months (November through March). It is also the source of water for miscellaneous flows for underground mine backfill and dust control. Surface runoff from precipitation on the open pit, the waste rock dumps and the overburden stockpile is collected and conveyed to the open pit for water management.

The overall WMP, including the proposed ditches and culverts for the start-up and final configuration of the mine, are shown in Figures 20.5 and 20.6.

20.3.7 Residue Management Area

(a) Design Basis, Drivers and Assumptions

The Residue (tailings) Management Area (RMA) has been designed to provide containment for all the residue that will be generated over the life of the mine. The acid leaching process will produce two residue streams:

- Leach residue – comprises particles which have not been dissolved in the leaching process, together with gypsum resulting from the neutralisation of entrained sulphuric acid and some metal hydroxides
- Effluent treatment residue – reports to the underflow of the primary and secondary effluent treatment clarifiers. This will contain gypsum and some metal hydroxides.

Prior to disposal, these residue streams will be combined and neutralised with lime. Due to the addition of lime, the mass of the residue will be about 21% greater than the mass of the ore. The neutralised residue will have a pH of neutral to slightly basic at the end of the process.

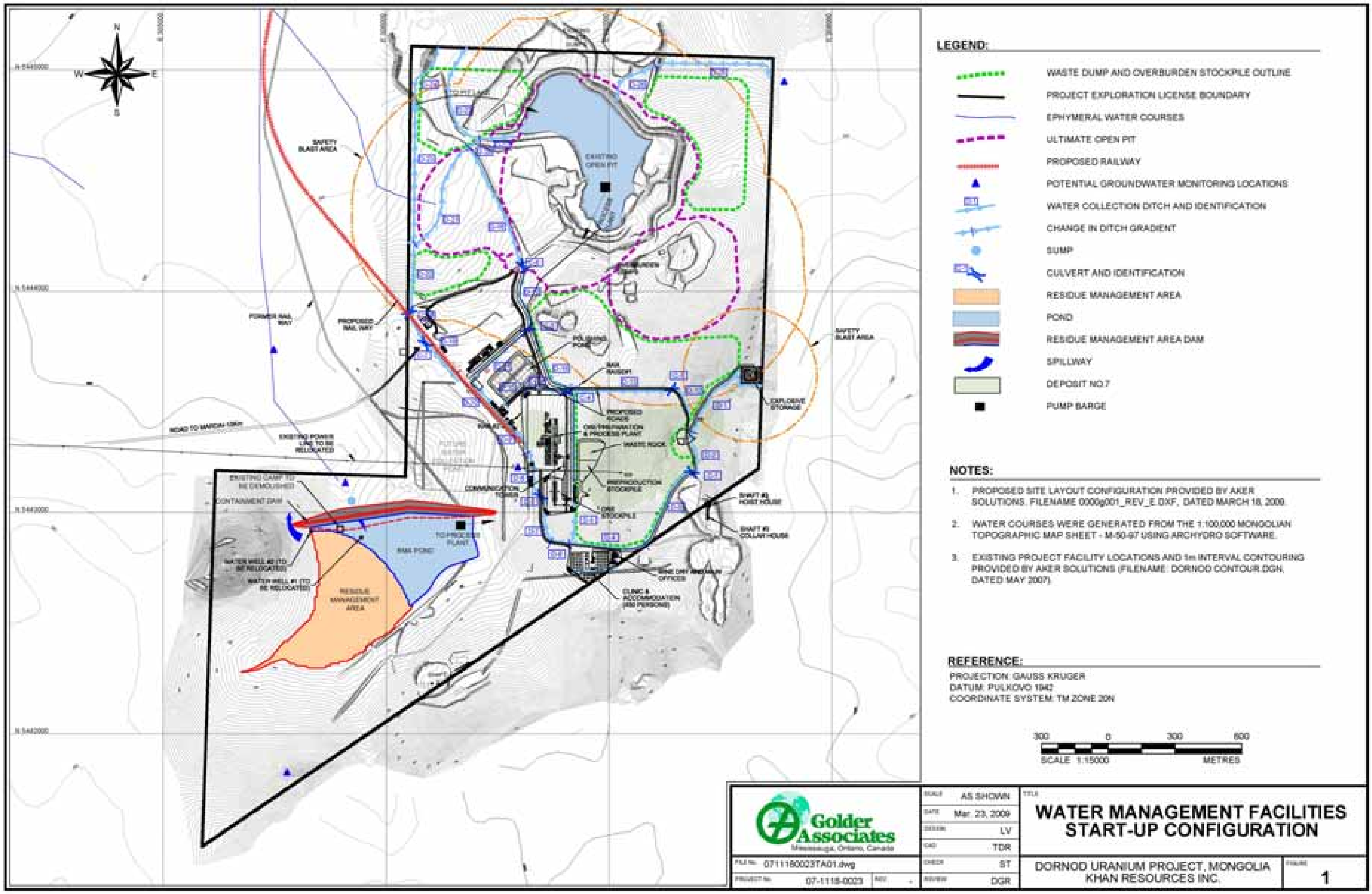


Figure 20.5 – Water Management Facilities Start-Up Configuration

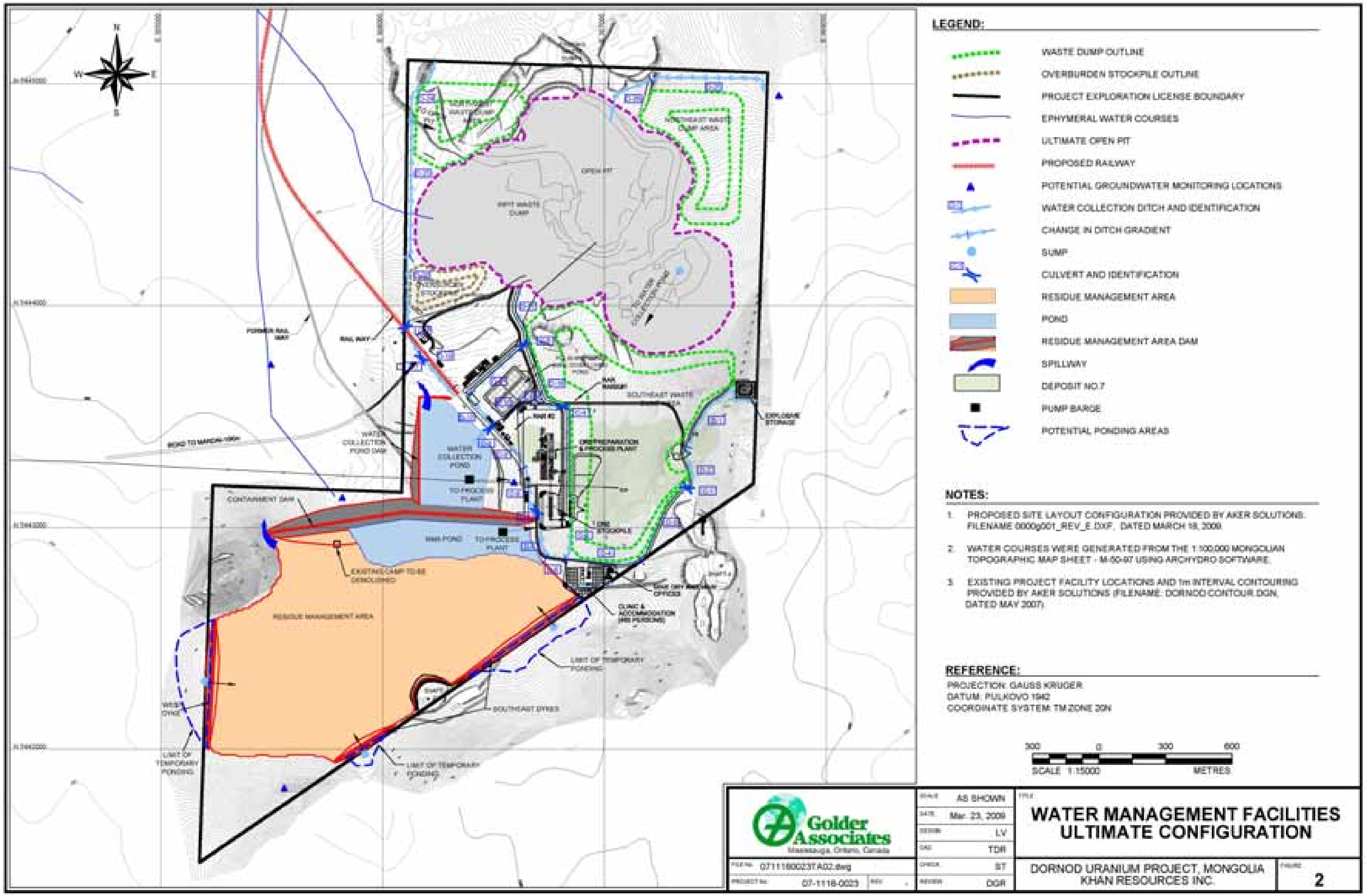


Figure 20.6 – Water Management Facilities Ultimate Configuration

The capacity of the RMA is based on the operating data listed on Table 20-4. Over the life of the mine, ore processing will generate some 23.4 Mt of residue (on dry weight basis). Assuming that the residue will consolidate to an average void ratio of 1.1 (which corresponds to a dry density of 1.21 t/m^3), the RMA has been designed to have a storage volume of 19.3 Mm^3. If the residue consolidates to a different void ratio, both the water balance and the RMA capacity will be impacted and, hence, additional confirmatory tests are recommended at the next stage of the Project.

The residue is a fine grained material with about 80% silt size particles (passing 75 microns) and 20% passing 20 µ. The measured specific gravity ranges between 2.45 and 2.55. The mineral composition is about 50% albite, 23% gypsum, 18% quartz and about 9% chlorite. Initial thickening tests performed by PasteTec (Golder, 2008c) carried out on the neutralised residue samples did not provide good results. However, subsequent small scale tests that allowed staged neutralisation of the residue showed improved dewatering properties and higher densities (Golder, 2009d). Additional large scale tests are recommended for the next stage to confirm these results.

Two of the most important design drivers for the RMA are the requirement for zero discharge to the environment under normal operational conditions and the property boundary constraints. Geochemistry is also an important driver. Other design drivers considered are subsurface conditions, availability of construction materials, conservation of water, seepage minimization, existing Shaft No. 2, design for closure and costs.

Based on a preliminary appraisal of the geochemical characteristics, there will be elevated concentrations of uranium in the residue supernatant relative to the Mongolian Drinking Water Standard (MSN 900:2005). The waste rock leachate may also be above Mongolian Drinking Water Standards for arsenic, uranium and molybdenum.

(b) Site Selection

The proposed RMA site is shown on Figure 20.7. The site selected for the design was preferred, as it is the only available site of sufficient size located with in the property controlled by Khan, which is the major selection criterion (Golder, 2008d). A number of different alignments for the containment dams were assessed (Golder, 2008e). The alignment shown was selected, because it provides the required capacity with the minimum dam volume.

After cessation of the underground operations (~Year 7), the volume of the residue reporting to the RMA could be reduced by comingling it with the waste rock (subsurface and in-pit). The feasibility of this option should be assessed at the next stage of the Project.

Table 20-4
Operating Data

<u>Nominal and design values</u>: Nominal values are based on the planned annual mill throughput averaged over 365 days per year. The <u>nominal values</u> are used to size the tailings facility and for the flow (water balance) modelling. The design values are larger

	Symbol	Source (Note 1)	Value	Unit (metric)
Ore Production				
- Ore reserve		Khan	19.408	Mt
- Nominal production rate		Aker	3,500	t/day
- Operating days per year		Aker	350	days
- Nominal annual production rate		Calculated	1,225,000	t/year
- Life of mine		Calculated	15.8	years
- Factor of safety on the design value		Aker	1.0	-
Residue Production				
- Residue / ore ratio		Aker	1.2066	-
or residue mass *(production)*			-	t/d
- Specfic gravity of tailings solids	G_s	SGS Lakefield	2.55	-
- Density of liquor *(supernatant)*	ρ_ω	Aker	1.00	t/m^3
- Discharge slurry density of the residue from the mill to thickener	S_1	Aker	48.2	% solids
- Discharge slurry density of the residue from the thickener to disposal	S_2	Aker	48.2 (no thidkening)	% solids
Deposited Residue				
- Assumed deposited void ratio *(Void volume / total volume)*	e	Golder	1.10	-
- Dry density of residue	γ_d	Caculated	1.21	t/m^3
- Total volume *(based on nominal values)*		Caculated	19.3	$M\text{-}m^3$
Flows impacting the mill water balance				
- Water content of the ore going into the mill *(% of total mass of ore)*	ω_2	Khan	7	%
- Water leaving the mill in the concentrate: Moisture content if leaving by truck *(% of total mass of concentrate)*	ω_3	Aker	1	%
- Water leaving the mill in the concentrate: **OR** slurry density if leaving by pipeline	S_3		-	% solids
- Clean (fresh) make-up water required in the mill *(% of total flow through the mill)*		Aker	30	%
- Water lost in the mill to evaporation and spillage *(% of total flow through the mill)*		Aker	2	%
Miscellaneous flows impacting the flow model				
- Water used for dust control *(taken from one of the ponds)*		Aker	500	m^3/day
- Potable water from an external source		Aker	55	m^3/day
- Sewage *(estimated as a % of potable water)*		Aker	80	%
- Water used in the power plant		Aker	none	m^3/day
- Water used for cemented rock backfill		P & E	25	m^3/day

Note: 1 The sources of the information could be either the owner / operator, contractors, Golder or other consultants.

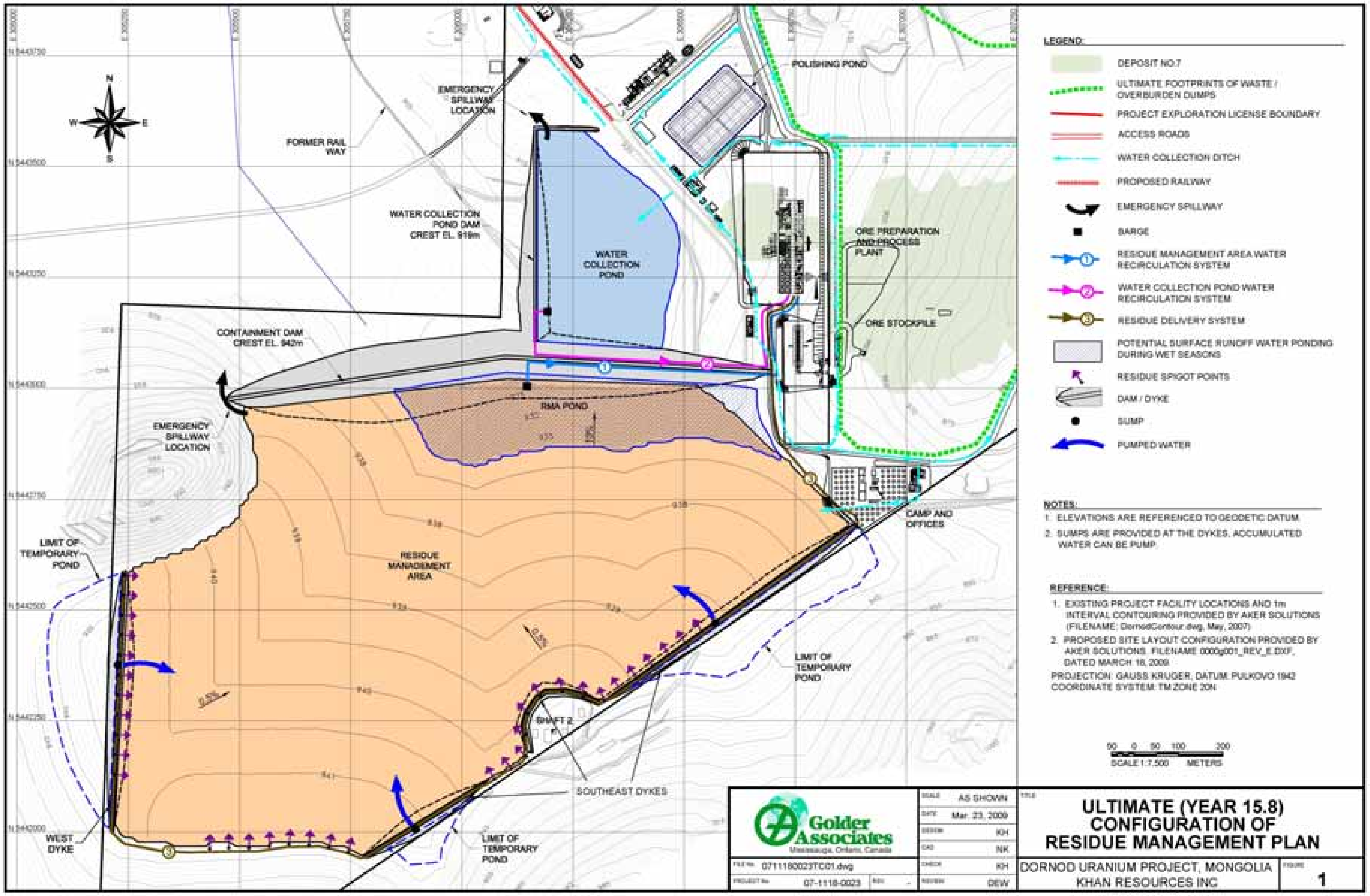


Figure 20.7 – Ultimate (Year 15.8) Configuration of Residue Management Plan

(c) Geotechnical Investigation

During the 2008 geotechnical investigation campaign (Golder, 2009a), a total of 13 boreholes and 29 test pits were drilled and dug, respectively, on the RMA site (see Figure 20.8. The field study identified the typical ground conditions underlying the RMA as:

- Colluvium, including topsoil (mean thickness ~2.4 m)
- Saprolite (mean thickness ~4.4 m)
- Slightly weathered fractured rock (mean thickness ~7.4 m)
- Sound rock underlying the weathered rock.

The depth of the profile above the sound rock varies between 8 and 24 m with an average of about 15 m. The colluvium is a transported pervious sandy soil. The saprolite is a relatively impervious clayey soil with core stones. The weathered rock is more pervious than the sound rock beneath. The groundwater level is low, located within the weathered rock or sound rock.

The typical subsurface profile along the RMA containment dam alignment is shown on Figure 20.9. The results of the filed and laboratory tests carried out on the subsurface soils and bedrock are summarised in Table 20-5.

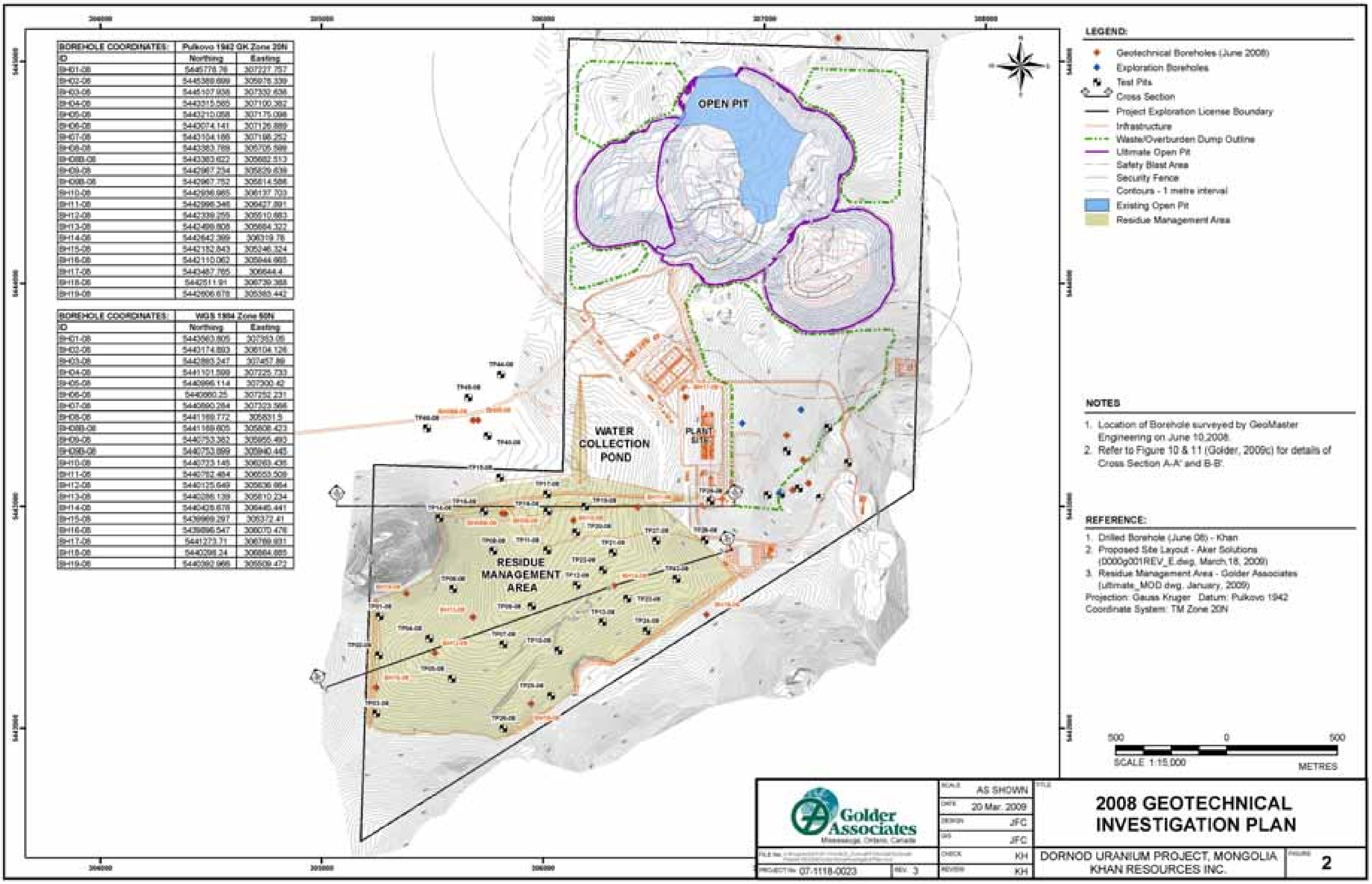
(d) Description of the Residue Management Area

Containment Dam and Dikes

As shown on Figure 20.7, the residue is partially contained by the surrounding topography partially by the containment dam. Perimeter dikes are also required on the west and south sides of the facility to keep it within the property boundary.

It is proposed to raise the containment dam in stages (Golder 2009e). The starter dam, which will provide a storage capacity for 2 years, will have crest elevation of 923 m. The ultimate dam will have crest elevation of 942 m, which includes provision to store a 24-hr, 1000-yr return environmental design storm (~2.4 m) plus a 2-m freeboard.

The containment dam and the perimeter dikes will be waste rock embankments with a composite liner on the upstream face (see Figure 20.10). The composite liner consists of a 1.5-mm (60-mil) high-density polyethylene (HDPE) geomembrane over a 4-kg/m^2 geosynthetic clay liner (GCL). The composite liner will be extended down through the colluvium layer and tie into the saprolite layer. The rock fill embankments will have 8-m crest width, an upstream slope of 2.5 H:1 V and a downstream slope of 1.5 H:1 V. The bulk of the embankment rock will be random waste rock that will be sourced from the existing waste rock stockpile and from the planned underground and open-pit operations. The upstream flank of the embankments will be dressed with a narrow strip of (~3-m thick) screened waste rock to provide bedding for the liner system. Foundation preparation for the embankments will entail removal of the topsoil and other unsuitable soils and proof rolling the foundation.



BOREHOLE COORDINATES:	Pulkovo 1942 GK Zone 20N	
ID	Northing	Easting
BH01-08	5445778.76	307227.757
BH02-08	5445389.699	305978.339
BH03-08	5445107.938	307330.636
BH04-08	5440315.565	307100.382
BH05-08	5440210.058	307175.098
BH06-08	5440074.141	307126.889
BH07-08	5440104.186	307198.252
BH08-08	5443363.789	305705.599
BH08B-08	5440383.622	305682.513
BH09-08	5442967.234	305829.639
BH09B-08	5442967.752	305814.586
BH10-08	5442936.965	306137.703
BH11-08	5442996.346	306427.891
BH12-08	5442339.255	305610.683
BH13-08	5442499.808	305684.322
BH14-08	5442642.399	306319.78
BH15-08	5442152.843	305246.324
BH16-08	5442110.062	305944.665
BH17-08	5440487.765	306644.4
BH18-08	5442511.91	306739.368
BH19-08	5442606.678	305383.442

BOREHOLE COORDINATES:	WGS 1984 Zone 50N	
ID	Northing	Easting
BH01-08	5443563.805	307353.05
BH02-08	5440174.893	306104.126
BH03-08	5442880.247	307457.89
BH04-08	5441101.599	307225.733
BH05-08	5440996.114	307300.42
BH06-08	5440660.25	307252.231
BH07-08	5440690.264	307323.566
BH08-08	5441169.772	305831.5
BH08B-08	5441169.605	305808.423
BH09-08	5440753.382	305855.493
BH09B-08	5440753.899	305840.445
BH10-08	5440723.145	306093.435
BH11-08	5440782.484	306553.509
BH12-08	5440125.649	305636.664
BH13-08	5440286.139	305810.234
BH14-08	5440428.678	306445.441
BH15-08	5439969.297	305372.41
BH16-08	5439896.547	306070.476
BH17-08	5441273.71	306789.931
BH18-08	5440298.24	306864.885
BH19-08	5440392.966	305509.472

Figure 20.8 – 2008 Geotechnical Investigation Plan

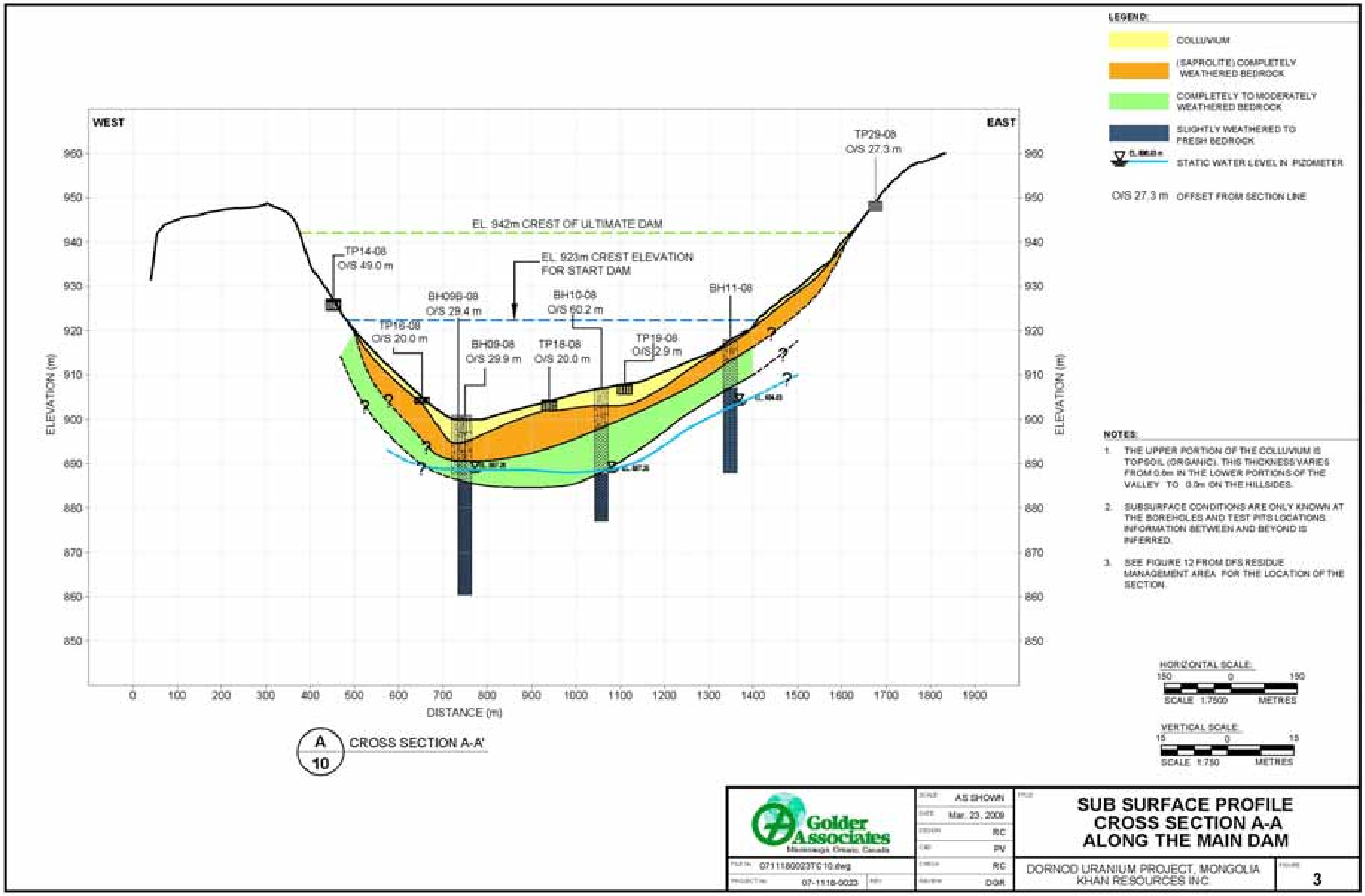


Figure 20.9 – Subsurface Profile Cross-Section A-A Along the Main Dam

Table 20-5
Summary of Subsurface Conditions

Parameter	Unit	Colluvium			Saprolite			Weathered Rock	Sound Rock
		Min.	Mean	Max.	Min.	Mean	Max.		
Thickness	m	0.3	2.4	8.3	0.6	4.4	8.5	2.2 - 22.4	-
Particle Size Distrubtion:									
Silt / Clay	%	5	14	38	14	34	87	-	-
Sand	%	19	50	85	12	47	69	-	-
Gravel	%	0	36	73	0	18	46	-	-
Moisture Content	%	1.1	8	27.3	6.4	18.7	72.0	-	-
Atterberg Limits:									
Liquid Limit	%	-			18	34	56	-	-
Plastic Limit	%	-			10	13	18	-	-
Soil / Rock Type	-	Sand and Gravel			Silty Clay to Clayey Silt			Rhyolite	Rhyolite
Standard Penetration Test (SPT- N)	# of blows	7	33	>85	6	25	60	-	-
Specific Gravity	-	2.62	2.68	2.73	2.69	2.72	2.74	-	-
Standard Proctor Compaction:									
Optimum moisture content	%	-			13	15	17	-	-
Maximum dry density	t/m^3	-			1.6			-	-
Field Hydraulic Conductivity	cm/s	-			1×10^{-5}			1×10^{-4} - 5×10^{-6}	3×10^{-5} - 9×10^{-7}

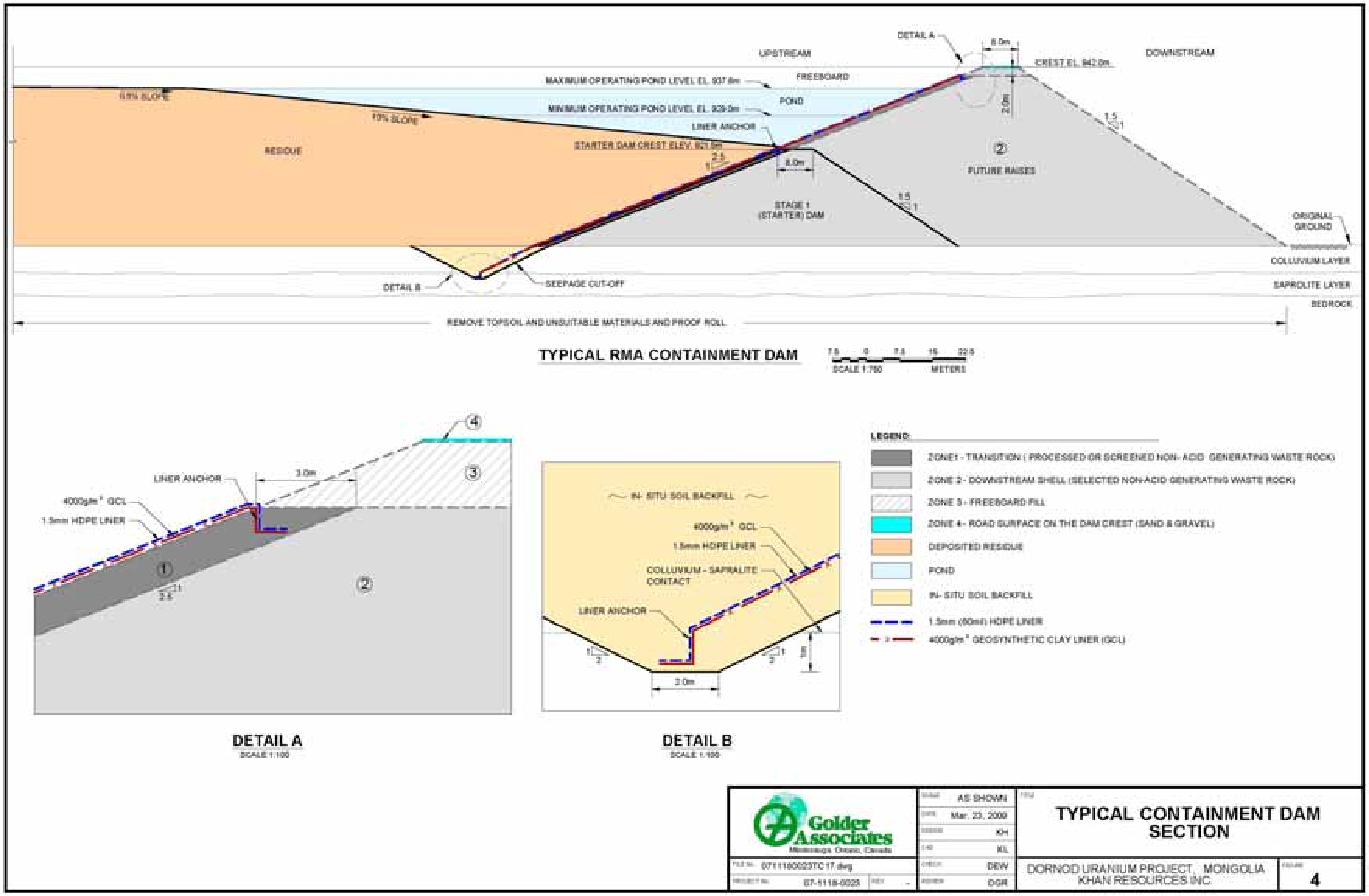


Figure 20.10 – Typical Containment Dam Section

To protect the containment dam from overtopping an emergency spillway will be constructed in the northwest corner of the facility (see Figure 20.11). The invert of the emergency spillway will be at 940 m, high enough to contain the environmental design flood. The spillway invert will be 5-m wide that will allow a safe passage of the 24-hr, probable maximum precipitation rainfall event. The invert will be lined with concrete and the outflow channel will be stepped with gabion baskets. The spillway will remain functional post closure of the RMA.

A synthetic liner is not currently planned to be used in the RMA basin. The saprolite layer will be the seepage barrier. Additional geotechnical investigation is recommended to confirm this assumption.

The residue will be pumped as a slurry at 48.2% solids through a pipeline from the process plant to the south and west sides of the RMA. In the initial years, when beaches develop, the residue will be deposited by open ended discharge. In the later years, the deposition will be from multiple spigots. The five stages of deposition are shown on Figure 20.12. The plan was developed with the assumption that the deposited residue will assume an average 0.5% beach slope and a 10% slope below the RMA Pond level.

Water Management Plan for the Residue Management Area

The inflows into the RMA include water discharged with the residue and runoff from the surrounding watersheds (Golder, 2009f). The losses are water retained in the residue and evaporation. Runoff from the adjacent property (~381 ha) that drains towards the facility is not diverted, in order to maximise water collection.

The water that accumulates in the RMA Pond will be pumped back to the process plant from a pump barge for reuse. Additional water required for the process will be pumped from the existing Open Pit Lake, at least for the first 7 years when the mine is an underground operation. After Year 7, water required for the process will be pumped from a Water Collection Pond, which will be constructed downstream of the main containment dam.

The Water Collection Pond will act as a temporary water storage facility receiving runoff water from process plant site, open-pit dewatering and freshwater from other sources. The pond basin will be lined with a 1.5 mm (60 mil) HDPE geomembrane.

Like the RMA containment dam, the Water Collection Pond dam will be constructed with waste rock and the upstream face will be lined with a composite liner that will extend down through the colluvium layer and tie into the saprolite layer. The crest elevation of the dam will be at 919 m, which includes provision to store environmental design storm plus freeboard. The dam will have a 7-m-wide emergency spillway that will allow a safe passage of the 24-hr, probable maximum precipitation rainfall event.

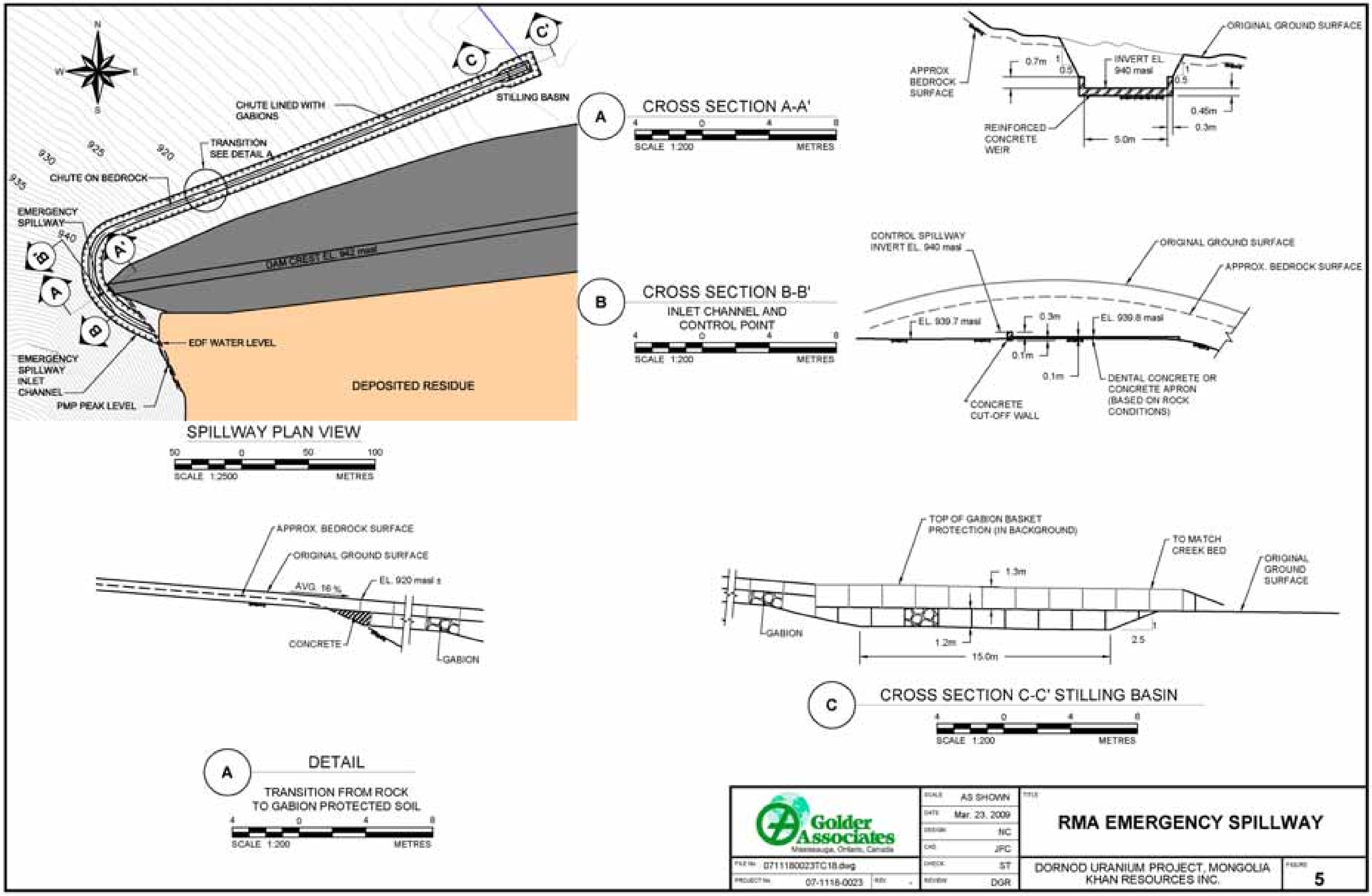


Figure 20.11 – RMA Emergency Spillway

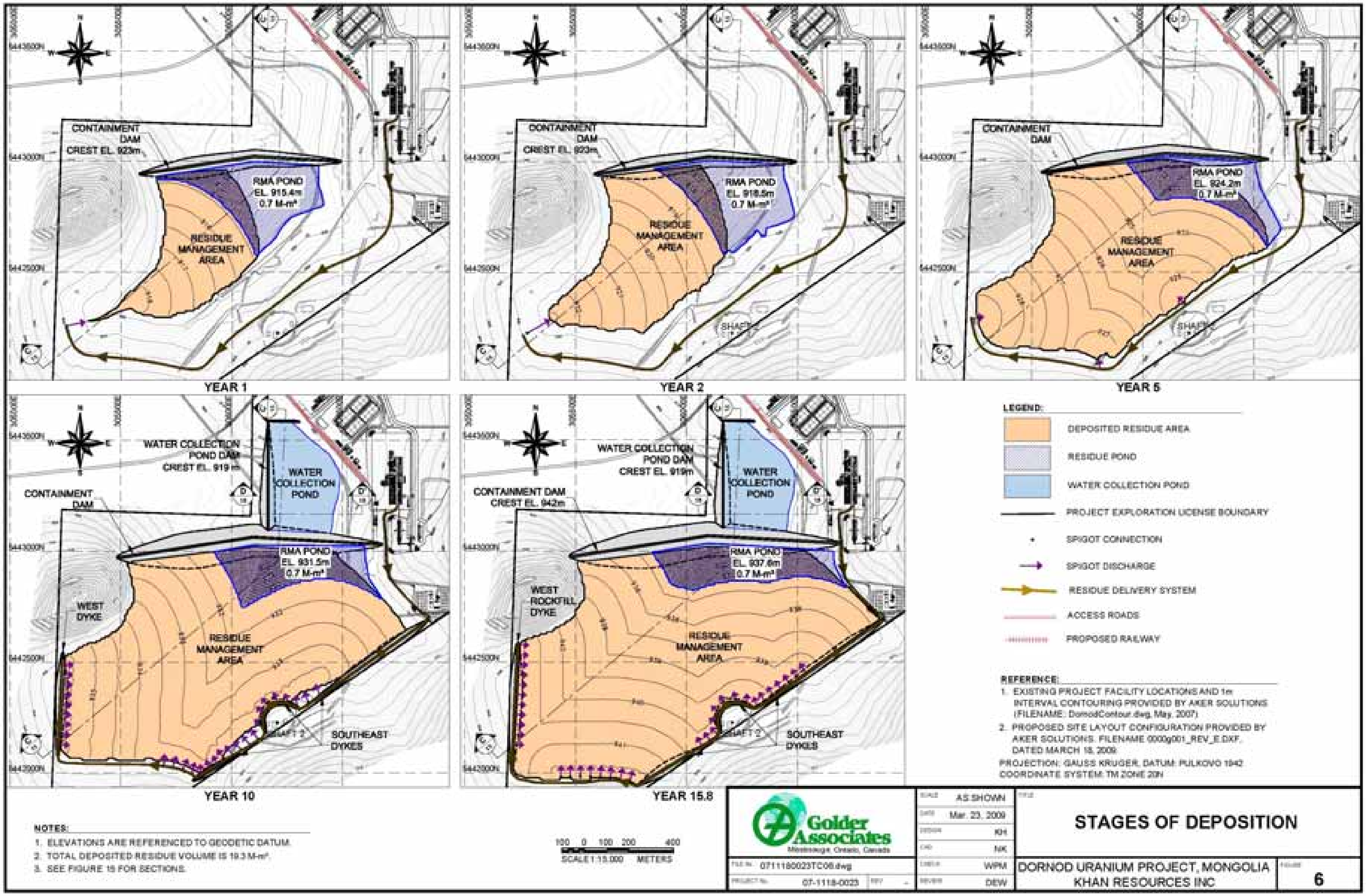


Figure 20.12 – Stages of Deposition

Closure of the Residue Management Area

The preliminary closure plan for the RMA and Water Collection Pond (Golder, 2009e) are shown on Figure 20.13.

At the end of operation, the RMA beach will be covered with a 0.3-m-thick capillary break to reduce the capillary rise of residue pore water, 0.6-m-thick clayey soil to reduce infiltration of rainwater, and 0.3-m-thick vegetated topsoil to blend the facility with the surrounding area. A herring bone drainage system will be installed on top of the soil cover to minimise seepage and convey the runoff to the RMA Pond. The quality of water in the pond will be monitored after closure. Once the quality reaches Mongolian discharge standards, a small overflow spillway will be constructed at the northeast corner of the containment dam to allow passive drainage.

The Water Collection Pond will be decommissioned and removed at cessation of the mine. The disturbed footprint area will be vegetated.

20.3.8 Conceptual Closure Plan

(a) Assumptions

Khan is committed to closing out the Project to best international practices for both the existing site infrastructure and any new proposed infrastructure (Golder, 2009).

For the purpose of this conceptual closure plan, the following guidelines and assumptions have been adopted.

- The residue and waste rock have a low potential for acid generation, though metal leaching is a potential concern.
- Given that cessation of underground mining will occur in Year 9 of operations, the decommissioning, demolition and / or removal of the underground equipment, hoist and collar house, along with the capping of shaft No. 3 and the backfilling of the ramp / portal and the return air raises will be undertaken as progressive rehabilitation during the remaining operating life of the mine.
- Given that much of underground waste rock will be used for backfill, the waste rock in the dumps on surface will primarily originate from the mining of the open pit. It is assumed that the physical and chemical properties of the waste rock will be similar to the properties of the existing open-pit walls. Therefore, the target post-closure water quality in the open pit should be comparable to the water quality before commencement of operations.
- The existing waste rock and overburden dumps will be entirely covered by the proposed southeast waste rock dump.
- The closure cover for the RMA will require a minimum thickness to limit the diffusion of radon gas and gamma radiation emissions from the residue into the atmosphere (which poses a human and ecological health risk). The minimum cover thickness has been assumed to be approximately 1 to 2 m, but this is subject to confirmation and modelling.

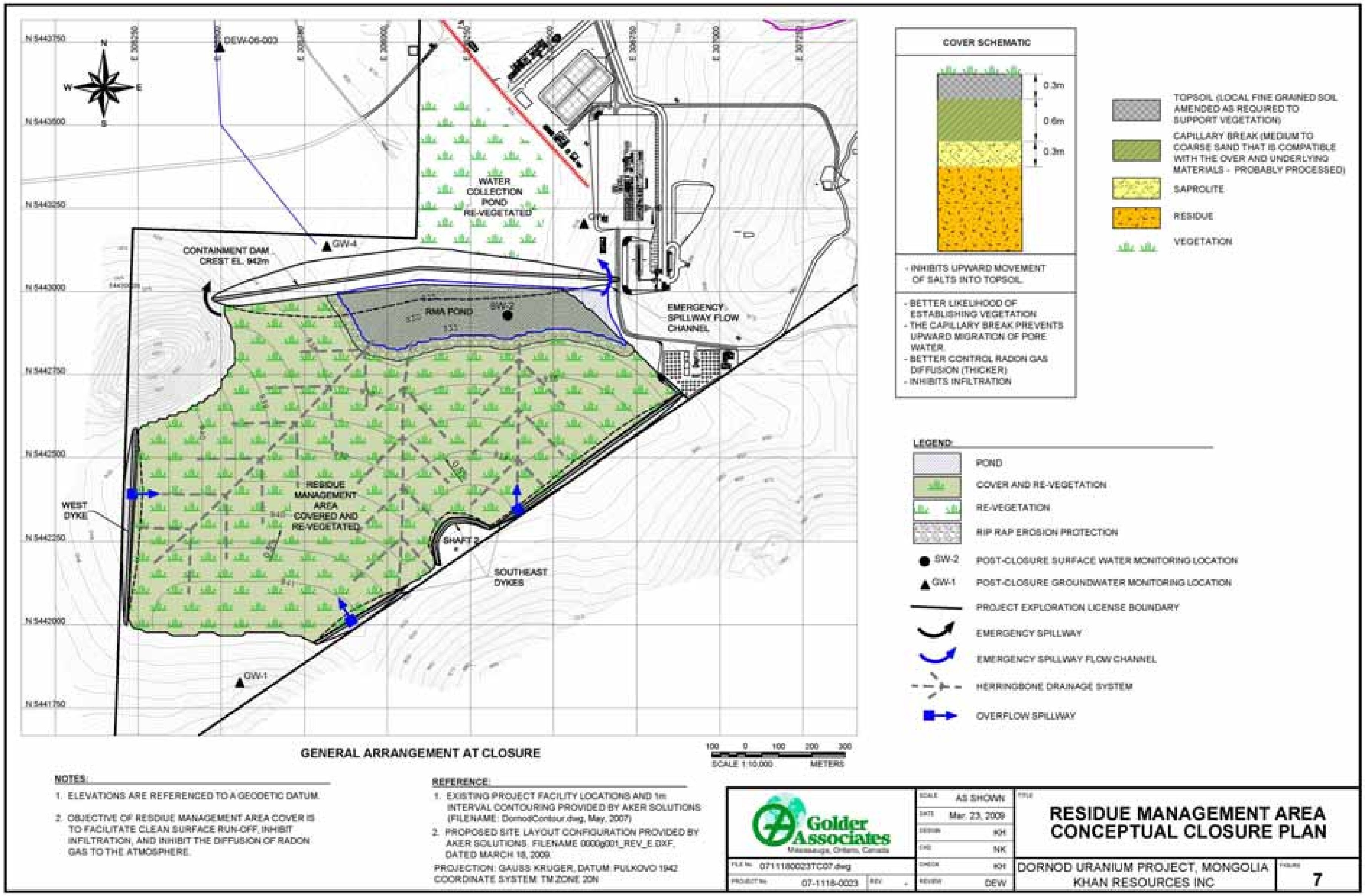


Figure 20.13 – Residue Management Area Conceptual Closure Plan

- Soils contaminated by hydrocarbons or otherwise will be excavated and placed in the RMA.
- A closure spillway will be constructed to allow for passive drainage. A small pond may be required for sediment control and to allow pumping to the treatment plant or possibly to the open pit until water quality meets discharge criteria.
- The WRSF will restrict access to a large portion of the open pit upon closure.
- To support revegetation, waste rock slopes should be no steeper than 2.5 H:1 V, where feasible. In cases where it is unfeasible to regrade the slopes, the waste rock will be pushed into the open pit or over the ore processing facility area (once the infrastructure has been demolished).
- It is assumed that only 20% of the waste rock dumps will be regraded during operations as part of progressive rehabilitation.
- Covers will not be required for In-Pit Dumps 1 and 2, and the Top Pit Dump, since they are located within the open pit.
- It is assumed that the existing mine fleet at the time of closure will be used to regrade the waste rock dumps and construct the covers both for the waste rock facilities and the RMA.

(b) Results and Recommendations

The main facilities which will be present on-site at the time of closure (see Figure 20.14) include:

- The open pit
- The Waste Rock Storage Facilities
- The Residue Management Area
- The Water Collection Pond
- The Polishing Pond
- The Ore Processing Facilities
- The main production ramp and portal, along with one ventilation intake raise (formerly shaft No. 3) and two return air raises; and
- Other infrastructure (both existing and proposed) such as the mine dry and offices, accommodation, maintenance areas, etc.

The principal closure measures that will be employed to address the facilities listed above include:

- Construction of a boulder berm around the open-pit rim and placement of a lockable swing gate at the entrance to the pit ramp
- Regrading of waste rock storage dump slopes to 2.5 H:1 V and placement of a revegetated cover over the entire dump footprints to prevent airborne dust and to minimise water infiltration to reduce the potential for metal leaching
- Placement of a cover on the surface of the RMA to provide clean surface runoff, to prevent acid rock drainage and metal leaching, and to inhibit diffusion of radon gas and gamma radiation, as well as airborne dust to the atmosphere

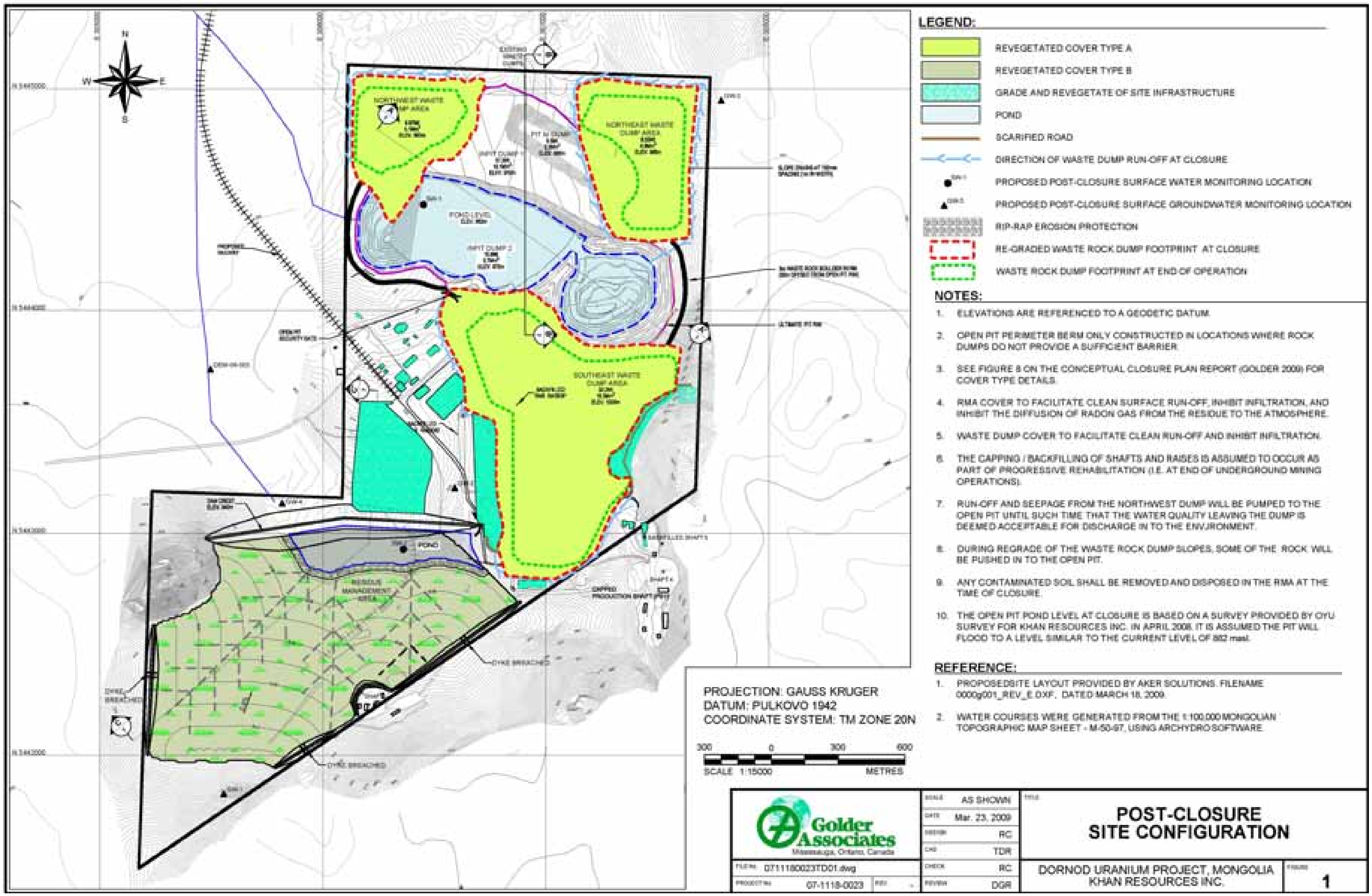


Figure 20.14 – Post-Closure Site Configuration

- Decommissioning and removal of the Water Collection Pond and the Polishing Pond
- Capping of shaft No. 3, and the backfilling of the production ramp and portal, and the return air raises
- Decommissioning and demolition / removal of the ore processing facility and other surface infrastructure and equipment.

Long-term care and maintenance for the Project will consist of the following actions:

- Local labour will be employed to ensure that site security is maintained;
- The open-pit walls, the RMA dams, shafts, and Waste Rock Storage Facility will be inspected on an annual basis by a qualified engineer to ensure their physical stability;
- Quarterly surface water quality sampling will be performed during Years 1 to 5 at the Open Pit Lake, the RMA pond, and at locations upstream and downstream of the Project site until stable trends are established; sampling will occur annually thereafter; and
- Quarterly groundwater quality sampling will be performed during Years 1 to 5 at one location downstream of the RMA, two locations upstream of the RMA, one location upstream of the Waste Rock Storage Facilities, and one location downstream of the Project site, until stable trends are established, reducing to annually thereafter.

20.4 Environmental and Social Impact Assessment

20.4.1 Introduction

An International ESIA for the Project was prepared by AATA International, Inc., based in Denver, Colorado, U.S.A. This section is a summary of the ESIA.

The Dornod Project is located 600-km east of the capital city of Ulaanbaatar, in northeastern Mongolia. Khan has current permits and licenses for exploration and extraction of uranium, which were granted by the Government of Mongolia.

The ESIA provides: comprehensive information about the key environmental and social characteristics of the Project; data on the current or baseline (predevelopment) environmental and social conditions at the Project site based on recent studies at the site and historical information; evaluations of potential impacts of the Project; and, recommendations for impact mitigation measures. It also includes a comprehensive document, the ESMP, which provides detailed information on the policies, practices and procedures that will be implemented by Khan at the Dornod Project to comply with applicable Mongolian regulatory requirements, as well as, conform to international guidelines and standards, to which Khan is committed.

The ESIA was developed in accordance with GIIPs including those specifically defined by the Performance Standards on Social and Environmental Sustainability of the International Finance Corporation (IFC- a unit of the World Bank) and by the Equator Principles.

The purpose of the ESIA is to provide Khan and potential financing agencies with: a detailed analysis of the physical, chemical, biological, and social aspects of the Project; an analysis of the potential social and environmental impacts associated with the Project; and, details on the social and environmental management and monitoring planned for the Project to protect workers, the public, and the environment. The evaluation of Project activities includes direct, indirect, cumulative and associated impact analyses.

The study methodology was comprised of the following activities.

- Obtaining all pertinent historical information on the Project from local and national sources, including mine plans and documents, aerial photography images, government reports and other pertinent documents
- Conducting a review of existing literature and data for the Project area
- Identifying Khan's corporate environmental and social policies and guidelines; Mongolian environmental and social regulations and legislative framework; and, international environmental and social guidelines and standards with which the Project must comply or conform
- Performing field baseline studies to collect Project site-specific data on current environmental and social conditions
- Describing the overall Project with an emphasis on processes that may potentially impact the environmental and social conditions
- Characterising the physical, chemical, biological, and social components of the environment potentially affected by Project development
- Identifying and ranking environmental and social risks and impacts for each Project component for each phase of the Project
- Developing an environmental and social management program that describes mitigation measures designed to eliminate or minimise environmental and social impacts
- Identifying net Project impacts.

Project description, physical environment, geology and mineral resources, and Project closure are described in other sections of this report (Items 7 and 20.3, and thus not presented here in this section).

20.4.2 Project Alternatives

Various alternatives were considered in evaluating the mine design options, locations, equipment, water and waste management, etc. Instead of the proposed operations presented in this report, the following alternatives were considered but not limited to:

- No action alternative in which the proposed Project will not be implemented
- Develop the No. 2 Deposit utilising underground mining (instead of open-pit mining)
- Construct a production shaft for the underground mine (instead of a mine decline)
- Construct a coal-fired power plant to provide electricity for the Project (instead of obtaining power from the national grid)
- Utilise groundwater and recycled water from the RMA as processing water, and release site runoff and dewatering water into the surface water drainage (instead of zero-water discharge)
- Process the tailings into a paste form for underground backfill (instead of using crushed waste rock mixed with cement and constructing an RMA retention pond)

- Utilise other material for the RMA dam construction (instead of overburden/waste rock generated by the Project).

The proposed operations (i.e., preferred alternatives) were selected based on technical and economic viability, as well as minimisation of potential environmental and social impacts.

20.4.3 Legal Framework

Since first becoming a parliamentary republic in 1990, Mongolia has adopted a new Constitution and established the principle environmental laws, policies, international agreements, and standards. The environmental laws of Mongolia generally fall into four categories: environmental protection; natural resources; natural resource use; and natural disasters. The environmental protection laws establish a legal environmental framework. In addition to the basic natural resources (e.g., air, water), the natural resource laws include those that may directly impact natural resources. Natural resource use laws were created to respond to the needs of the market economy, and the natural disaster laws outline preventative and responsive measures. The principal environmental legislation may be categorised as the following.

- Environmental Protection:
  - Mongolian Law on Environmental Protection
  - Mongolian Law on Environmental Impact Assessment
  - Mongolian Law on Land
  - Mongolian Law on Special Protected Areas
  - Mongolian Law on Buffer Zones

- Natural Resources:
  - Mongolian Law on Air
  - Mongolian Law on Water
  - Mongolian Law on Forestry
  - Mongolian Law on Subsoil
  - Mongolian Law on Mineral Resources
  - Mongolian Law on Natural Plants
  - Mongolian Law on Protection of Natural Plants
  - Mongolian Law on Hunting
  - Mongolian Law on Fauna
  - Mongolian Law on Foreign Trade of Endangered Fauna and Flora
  - Mongolian Law on Protection from Toxic and Hazardous Chemicals
  - Mongolian Law on Municipal and Industrial Waste

  - Mongolian Law on the Import, Export and Cross-Border Transport of Hazardous Wastes
- Natural Resource Use:
  - Mongolian Law on Fees for Land
  - Mongolian Law on Natural Plant Use Fees
  - Mongolian Law on Fees for Timber and Firewood Harvesting
  - Mongolian Law on Hunting and Trapping Authorization Fees
  - Mongolian Law on Hunting Reserve Use Payments and on Hunting and Trapping Authorization Fees
  - Mongolian Law on Reinvestment of Natural Resource Use Fees for the Protection of the Environment and Natural Resource Restoration
  - Mongolian Law on Water and Mineral Water Use Fees
- Natural Disasters:
  - Mongolian Law on Prevention of Steppe and Forest Fires.

The Dornod Project is designed to meet Mongolian regulatory requirements and commonly accepted international environmental, social, and consultation guidelines and standards, including IFC's Performance Standards on Social and Environmental Sustainability; IFC's General Environmental, Health, and Safety Guidelines; IFC's Environmental, Health, and Safety Guidelines for Mining; IFC's Policy on Disclosure of Information; the World Bank's Anti-Corruption Strategy; and, the Voluntary Principles on Security and Human Rights. The Project is also designed to conform to the Equator Principles, a derivative of IFC / World Bank standards.

20.4.4 Baseline Conditions

Analyses of the existing environmental and social data of the Project area were performed. Detailed surveys (and monitoring) of soils, surface water, groundwater, air quality, radiation, vegetation, wildlife, and other important environmental attributes have been carried out by Khan and its environmental contractors for several years, resulting in a comprehensive characterisation of the baseline conditions of the Project area.

Social and archaeological surveys were also conducted for the Project. All residents residing within 10 km of the Project area were identified and interviewed. Community meetings were held to inform the public about the Project and concerns were addressed. Local (soum and aimag) government agencies and non-governmental organisations (NGOs) were also interviewed.

A Mongolian EIA for the Project was also prepared and submitted to the Mongolian Government in June 2008. This Mongolian EIA is currently in the review and approval process. This International ESIA complements the Mongolian EIA and contains additional information relative to the Project's accordance with IFC's Performance Standards and the Equator Principles.

(a) Physical Conditions

The Dornod Project area is situated in a remote, sparsely populated locale in the northeastern portion of Mongolia, in the southern portion of the territory of the Dashbalbar Soum in the Dornod Aimag.

(b) Chemical Conditions

(i) Air Quality

Regional air quality is expected to be representative of global background concentrations in a remote undeveloped setting. Common air-quality parameters of concern [carbon monoxide (CO), carbon dioxide (CO_2), sulphur dioxide (SO_2), nitrogen dioxide (NO_2), ozone (O_3) and particulate matter (PM) are typically associated with industrial activities, which are not currently present within 25 km of the Project area.

The local economic activities (including grazing activities) are relatively low and are, therefore, mostly free of large air contaminant discharges to the airshed. In general, this region is a remote, undeveloped area with few anthropogenic and naturally-occurring air pollution sources; therefore, background levels of CO, CO_2, SO_2, NO_2, and O_3 are estimated to be very small (i.e., near natural background levels). Additionally, greenhouse gases are not emitted in large quantities in the region, due to the remote location with few anthropogenic and naturally-occurring air pollution sources.

PM is a mixture of small particles and liquid droplets, which may include acids, organic chemicals, metals, and soil particles. Near the Project area, PM occurs from wind erosion of disturbed areas (pre-existing mine, roads and grazed areas). PM may be measured as PM_{10}, inhalable coarse particles with diameters greater than 2.5 micrometers and less than or equal to 10 micrometers. A total of nine sets of PM_{10} samples were collected to represent conditions upwind and downwind of the Project area. Both the measured upwind and downwind PM_{10} concentrations meet the World Health Organization (WHO) ambient PM_{10} standard of 50 micrograms per cubic meter (µg/m^3). The $PM10_{10}$ concentrations ranged from 8 to 21 µg/m^3. Upwind samples had a mean $PM10_{10}$ concentration of 10.8 µg/m^3, and downwind samples had a mean $PM10_{10}$ concentration of 10.2 µg/m^3.

Noise levels of the Project area are those typical of a remote rural region with less than 40 decibels (dB) when there is little or no wind and as much as 70 dB with mild to strong winds.

(ii) Soil Chemistry

The soil chemistry of the Dornod Project site was characterised by sampling and surveying the five soil types within the Project area. A total of 27 soil samples were collected and analysed for their chemistry. All samples were generally collected from undisturbed areas or areas of historic disturbance that have been considered naturally reclaimed.

The majority of the samples had a slightly acidic pH and had a nitrogen content ranging from 0.04% to 0.34%. The soil conditions are generally adequate for agricultural purposes and, with the exception of aluminum, the metal concentrations in the soil samples fell within normal ranges as defined by Shacklette (1984).

(iii) Water Quality

Surface Water Quality

Surface water quality samples were collected from the pre-existing open-pit lake and seeps from ore stockpiles of former mining activities as well as from Daagai Spring and Hautsgait Lake, downstream receiving waters. The laboratory results were compared to the surface-water-quality standards of Mongolia, the Canadian Water-Quality Guidelines for the Protection of Agricultural Water Uses, and the United States Environmental Protection Agency's Water Quality Criteria. The majority of the analysed parameter concentrations were below the guidelines and standards. The few parameters exceeding the guidelines or standards are summarised below.

- The open-pit lake was found to be a sulphate-rich calcium-bicarbonate water body. Several parameters (e.g., arsenic, cadmium, fluorite, molybdenum, and sulphate) exceeded the applicable Mongolian and international surface-water-quality standards. Elevated uranium and radium levels were found as expected. The water currently in the open pit will be consumed (as processing water at the plant) during the first 8 years of operation. The water quality data could serve as good references for the open-pit water quality after the mine is closed.

- Stockpiles of low-grade ores were placed on a heap leach pad by previous mining operations. Elevated levels of total dissolved solids (TDS), conductivity, water hardness, and several metals are present in the seepage from the ore stockpiles.

- Daagai Spring is the nearest natural water body to the Project area. Concentrations of fluoride, nitrite, total phenolics, total arsenic, total and dissolved cadmium, total uranium, dissolved boron, and dissolved vanadium did not meet at least one of the comparison standards. The spring serves as a major regional drinking water source for cattle and horses, which may be the source of high levels of nutrients and phenolics. The rest of the parameters of the spring samples were comparable to that of the groundwater-quality samples, which would be expected as the spring is directly fed by groundwater. Daagai Spring was found to be a calcium-rich carbonate and bicarbonate source of water.

- The Hautsgait Lake samples were found to have a magnesium-sodium-potassium-rich carbonate and bicarbonate signature. Numerous parameters of the Hautsgait Lake samples were above the comparison standard concentrations. The high TDS and conductivity along with numerous parameters exceeding standards may indicate the influence / presence of animal waste, extensive runoff, or increased erosion due to land-use changes in the drainage basin.

Groundwater Quality

Groundwater samples were collected from monitoring wells, camp wells, a hand-dug well, and existing mine shafts. In general, the samples had similar chemistry in regards to metal composition and concentration levels with some outliers. Levels of pH were slightly alkaline, but generally within the commonly acceptable range of 6.5 to 8.5. Although many samples showed acceptable drinking water quality for general major ions and organics, most samples had elevated concentrations of fluoride and total phenolics. Metal concentrations are generally low, though elevated concentrations (i.e., above drinking water-quality standards) of aluminum, arsenic, cadmium, chromium and selenium were found in some samples. As expected, levels of uranium and radium are usually high and above acceptable drinking water standards.

The groundwater at the Project site is generally a sodium-bicarbonate type.

(c) Biological Conditions

The Project area lies in a steppe zone, a grassland or plains-type ecozone with limited precipitation (150 to 250 mm/a). In general, the steppe landscape is defined by flat plains and gently rolling hills with scattered mountains and sand dunes. Trees are absent, except along riparian zones, springs, lakes, sheltered regions, and human settlements. In this region, the amount of precipitation and the harsh winter temperatures are the major factors that determine the dominant vegetation cover.

The steppe zone in Eastern Mongolia is referred to as the Dornod Plain, which covers most of the Dornod Aimag with an area of approximately 250 000 km^2. The Dornod Plain is home to two national preserves and four strictly protected areas, conserving some of the largest intact grassland ecosystems in the world. The area encompasses biological features from the Siberian taiga, Manchurian flora and fauna, and Central Asian steppes. The region is also home to some of the largest remaining herds of Mongolian gazelle.

No Threatened and Endangered (T&E) flora species listed in the Redbook of Mongolia (Ministry of Nature and Environment of Mongolia, 1997) was found in the Project area; and no plant species in Mongolia are listed as extinct, critically endangered, endangered, or vulnerable according to the International Union for Conservation of Nature and Natural Resources (IUCN) Red List.

There are 23 species of mammals (9 rodents, 3 lagamorphs, 1 hedgehog, 8 carnivores, and 2 ungulates), 1 species of amphibians, and 20 species of resident and migratory birds reported in the Project area or deemed potentially occurring. There are some species deserving special attention, and a few of which have been identified as threatened, rare, or endangered by IUCN, the Redbook of Mongolia, and the World Conservation Union.

There are no aquatic species in the Project area.

(d) Social and Cultural Conditions

(i) History

As a land-locked country, Mongolia has shared much history with its neighboring countries, Russia and China. In modern history, Mongolia has transitioned through several governmental structures strongly influenced by its neighbors – from a Buddhist theocracy to a communist society to a socialist society to the present-day independent democracy.

(ii) Economics

Mongolia's economy was adversely impacted by the collapse of the Soviet Union in the early 1990s. In the Project region, several factories (bricks, carpet, meat and textiles) were closed. Soon thereafter, much hardship was endured during the initial transition to a decentralised, democratic government with a free-market economy. However, the economy has improved in recent years. Primary employment is provided by agriculture, the coal mine near Choibalsan, a heating station, and small and medium enterprises. Agriculture, usually in the form of nomadic herding, is the leading source of employment.

The socioeconomic baseline study primarily focused on the aimag and soum in which the Project area is situated, Dornod and Dashbalbar, respectively (Figure 20.15). The socioeconomics of Gurvanzagal Soum and Sergelen Soum were also assessed due to their close proximity to the Project area. These assessments provided a regional perspective of the current socioeconomic conditions.

Figure 20.15 – Geopolitical Map of the Northern Dornod Aimag

Within a 30-km radius of the Project area, the Dashbalbar administration reports the presence of 238 households with 968 people, 118 cattle sheds, and 96 wells. All but 60 of the households own livestock. In June 2007, 13 households (all practicing animal husbandry) were within 8 to 10 km of the Project area. The majority of the housing structures were gers (portable nomadic dwelling structures). These herding households generally move between different pastures seasonally. As such, the households near the Project area during the summer are often distant from the Project area during the winter.

In June 2008, AATA identified 21 households within and near 10 km of the Project area (Figure 20.16). Eighteen households were interviewed. Meetings were held with representatives of the Dornod Aimag and Dashbalbar Soum, as well as with three local NGOs.

Sixteen of the 18 families interviewed are primarily engaged in herding. Other occupations include veterinary medicine, railroad security and scrap metal collection. The households have a mean of six members, with as few as four and as many as nine members per family. With the exception of one single mother, each household has a father, a mother, and children. Usually, the father herds the livestock or performs his specific occupation. The mother milks the livestock and tends to the house, preparing meals and caring for the children. Almost all school-aged children attend school. When not attending school, the children assist their parents with herding and housework in addition to playing.

With open pastureland, herders establish relations with one another to share the grazing land and, when necessary or convenient, to assist each other with herding and livestock shearing. Strong winds, cold temperatures, harsh winter storms and droughts adversely impact herding. Families typically move two to three times a year, depending on the pastures, the water supply and the season. While 3 of the 18 interviewed households within 10 km of the Project site maintain one residence year-round, others generally stay 3 months each year. One household is within the Project area.

The livestock distribution in the Project region is fairly comparable to that of Dornod Aimag. The majority of the livestock is owned by a few. In this case, 68% the total livestock is owned by one-quarter of the herding households.

The common sources of revenue include livestock products (dairy, meat, and wool), government subsidies for children, government disability assistance, pelts and meat of game, and pensions. The annual income ranges from 270,000 to 2,830,000 Mongolian National Togrog (MNT; USD 1.00 = 1,268 MNT as of December 31, 2008) and has a median of 1,500,000 MNT. At least three-quarters of the households spend their annual income in its entirety. Since nearly all of the families practice herding, their assets are often in the form of livestock rather than finances.

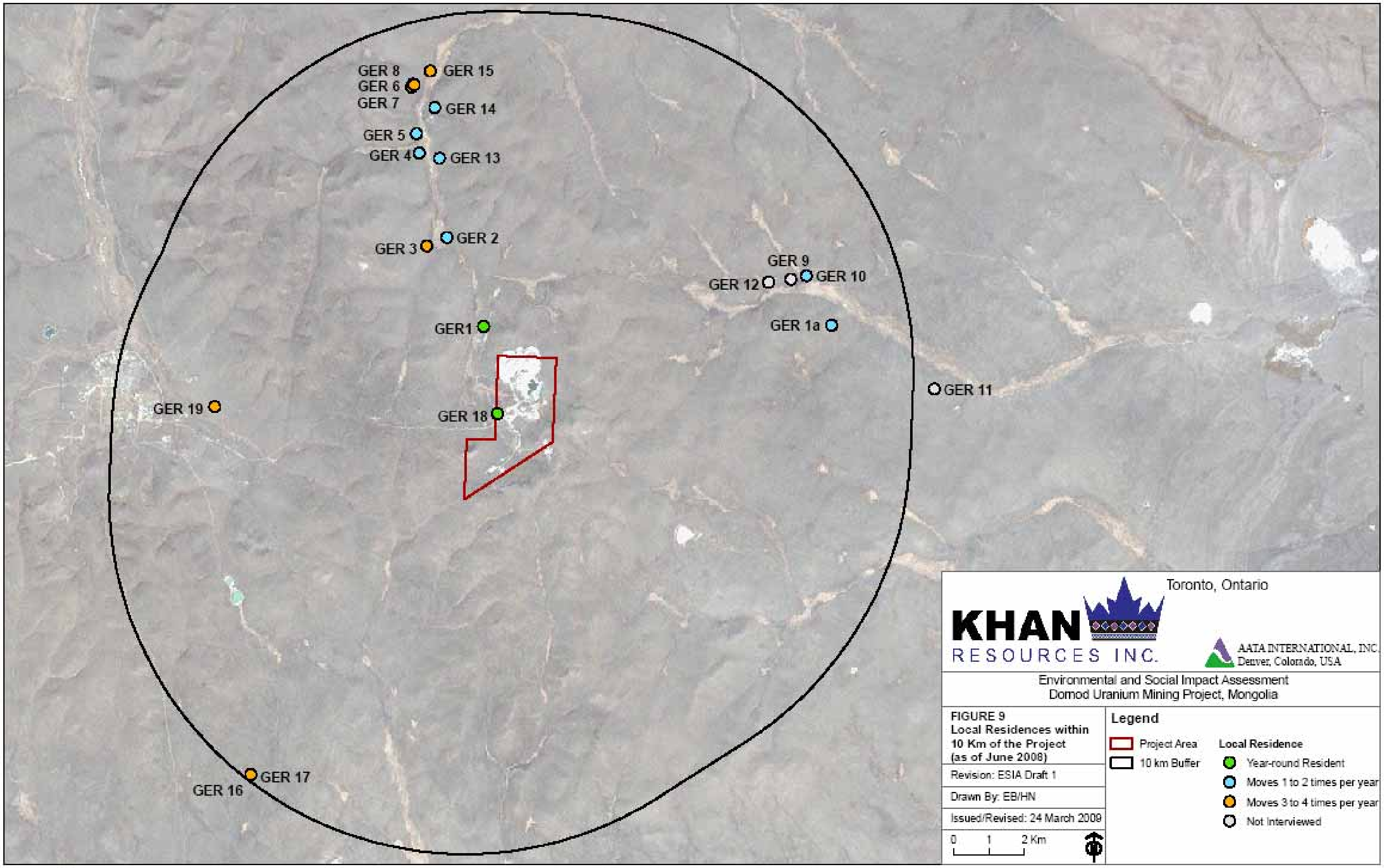


Figure 20.16 – Local Residences Within 10 km of the Project (as of June 2008)

Services and goods are purchased or exchanged at the Soum Center, Dornod Aimag Center and Ulaanbaatar. Items that are sold or exchanged include meat, dairy products, fur, leather, wool, game, and metal. As a means of transport, 16 households utilise horses, 11 have motorcycles, and 4 own a car.

Domestic and livestock water sources include wells, springs and creeks, snow and ice, and the pre-existing open pit. Snow and ice supply water in the winter whereas springs, creeks and wells supply water the remainder of the year. Ten of the 18 households have access to a well, which may be as far as 6 km from their residences. The households consume from 20 to 100 L of water per day, with a mean of 50 L/d.

(iii) Archaeology

The Dornod Project area is a brownfield industrial complex, with a recent history of geological research and mining by the Soviets. As such, the site already has been subjected to considerable disturbance. It is not known whether any archaeological resources were found at the site during previous mining development.

No archaeological objects or cultural heritage assets were discovered within the Project area; the only findings were outside the Project area. Ten units of square graves were identified 200 to 300 m east of the Project area, on top of Bayan-Erkhet Mountain. The largest of the graves is estimated to be from the 7th to 3rd century BC, representing the late Bronze Age or the early Iron Age.

(e) Radiological Conditions

Three radiological studies have been conducted in the Project area since 2005. Radiological conditions are measured in a variety of ways (types of radiation- alpha, beta, or gamma radiation; radioactive materials - uranium and radon) and from different media (e.g., ambient air, soil, water, vegetation). The environmental protections and concerns differ for each type of radiation and for each media.

(i) Air

Gamma radiation is naturally present everywhere on earth, but levels vary with location. In June and September 2008, gamma radiation was measured along 150-m transects evenly spaced in eight cardinal compass directions at the Project area. The average background radiation on-site, measured in June and September 2008, was about 0.3 microSieverts/hour (µSv/hr) or 0.8 milliSieverts per year (mSv/yr). The highest on-site measurement was about 5.5 µSv/hr (approximately 14.5 mSv/yr). The highest levels were associated with disturbed areas, such as overburden rock or low-grade ore stockpiles.

Passive radon and gamma dosimeters were installed on-site to measure average radon concentrations and gamma levels. From June to September 2008, the average radon concentration ranged from 0.02 and 0.03 becquerels per litre (Bg/L), and gamma levels averaged between 0.30 and 0.64 millirems/d

(ii) Soil

Soil samples were collected and analysed for uranium and radionuclides. In general, the radionuclide concentrations in the surface materials at the Project site reflect the differences between undisturbed and disturbed portions of the site; and, illustrate the importance of ensuring a differentiation between undisturbed and previously disturbed surface materials.

Uranium concentrations in the undisturbed topsoil ranged from about 1 to 42 parts per million (ppm), which is normal to slightly elevated compared to average soils (0.3 to 11.7 ppm; UNSC, 1993). In contrast, the uranium concentrations in the disturbed materials were over 400 ppm.

Radium-226 was evaluated in samples of undisturbed and disturbed materials collected at the Project area. Radium-226 concentrations in native soils at the Project site are less than 0.1 becquerels per gram (Bq/g), while in the overburden, the concentrations range up to 10 Bq/g.

(iii) Water

Surface Water

Radionuclides were analysed in six samples (three pairs of shallow (surface) and deep (greater than 10 m) collected during three sampling events (December 2007, June 2008 and October 2008) from the open-pit lake. The range of Lead-210 concentrations was 0.02 to 0.14 Bq/L for the shallow samples and 0.02 to 0.12 Bq/L for the samples at depth. Polonium-210 ranged from 0.005 to 0.01 Bq/L in the shallow samples and from 0.02 to 0.03 Bq/L at depth. Radium-226 was 0.06 Bq/L for all of the shallow samples and ranged between 0.06 and 0.13 Bq/L for the samples at depth. Radium-228 ranged between less than 0.04 and 0.1 for all depths. Thorium-230 and Thorium-232 were under the reporting limit. Gross alpha ranged from 70 to 99 Bq/L in the shallow samples and from 39 to 153 Bq/L in the deeper samples. Gross beta was similar at both depths, ranging from 35 to 47 Bq/L.

The majority of radionuclides measured from Daagai Spring were below laboratory detection limits. Polonium-210 had a concentration of 0.01 Bq/L for the June 2008 sample and a value of 0.007 Bq/L in October 2008. Gross alpha and gross beta values were relatively low. Gross alpha in June 2008 was 5.6 Bq/L and gross beta was 2.4 Bq/L. The October 2008 gross alpha was 0.31 Bq/L and the gross beta was 0.07 Bq/L.

Radionuclides were only analysed in December 2007 for Hautsgait Lake. Gross alpha was 4.5 Bq/L and gross beta was 8 Bq/L. Lead-210 had a value of 0.02 Bq/L; Polonium-210 was 0.07 Bq/L; Radium-226 was 0.005 Bq/L; and Radium-228 was 0.03 Bq/L.

While Radium-226, Thorium-230 and Thorium-232 were below WHO drinking water standards, concentrations of all other radionuclide parameters from the surface water in the Project region were above the WHO drinking water standards, with the open pit showing the highest concentrations, Daagai Spring in the middle range, and Hautsgait Lake at the lower end.

Groundwater

Radionuclide analyses of all of the samples collected showed a similar composition with lower concentrations of gross beta, Radium-226, Thorium-232, and Thorium-230 and elevated concentrations (i.e., above WHO drinking water standards) of gross alpha, Lead-210, Polonium-210, and Radium-228.

(iv) Vegetation

Eight samples of major vegetation types were collected from grass, poplars, willows, pine trees and cattails, and analysed for radionuclides. The results for the radionuclide concentrations in the vegetation samples show no apparent variation due to species or location. Lead-210 ranged from 0.008 to 0.1 Bq/g (dry weight); Polonium-210 ranged from 0.009 to 0.095 Bq/g; Radium-226 ranged from 0.0014 to 0.0080 Bq/g; Thorium-230 ranged from 0.0009 to 0.009 Bq/g; Uranium-235 ranged from 0.00001 to 0.009 Bq/g; and Uranium-238 ranged from 0.0002 to 0.0140 Bq/g.

20.4.5 Potential Impacts

This section discusses the potential environmental and social consequences associated with the Project. The potential impacts described herein do not consider the use of mitigation measures designed to reduce or eliminate Project impacts. The proposed prevention and mitigation measures for the Project are presented in Item 20.4.6. Net Project impacts, as a result of the use of the proposed mitigation measures, are presented in Item 20.4.11.

(a) Potential Physical Impacts

(i) Topography

The proposed mining operations in the Project area will have a localisd influence and disturbance on the current topographic conditions. Major structures and constructions that will permanently change the topography of the Project include the mine pit, the overburden placement areas, and the RMA. Open-it expansions will produce additional lower topographic areas; while the overburden placement areas and the RMA will form areas of increased elevations. Minor infrastructure components, such as the water management facilities and roadways, will have less impact on topography and geomorphology because of smaller scales with fewer relief variations.

Located at the head of a watershed, the Project area has natural visual barriers to minimise adverse visual aesthetic impacts. Much of the Project area is surrounded by small hills; therefore, the Project area is not visible from a few kilometres away on the public highway. Since no major travel routes are near the Project area and few residences stay nearby, visual sensitivity around the Project area is minimal.

(ii) Geology and Mineral Resources

The Project will change the geology configuration within the Project area as the Nos. 7 and 2 Deposits will be removed from the national inventory of mineral resources.

(iii) Seismicity

The Project area is located in a seismically inactive region. It is not anticipated that the underground mine or any surface geotechnical structures will be damaged by potential seismic activities that may occur in the Project region.

There are no major faults capable of generating an earthquake within the Project area. The Project will not change the potential seismic activities in the region nor is it expected to trigger any detectable earthquake.

(iv) Soils and Sediments

Soils will be impacted by the land disturbance activities during different phases of the Project. Potential physical impacts to soil include soil erosion, compaction, and breakdown of soil structure resulting in the loss of soil productivity. Impacts to soils in the Project area will result from the clearing of vegetation, excavating, leveling, stockpiling, compacting, and redistributing.

Mining activities have the potential to create sediments which may be transported during intense rainstorms.

(v) Surface Water

The Project is designed to operate without any discharge to the surface water drainage. No impacts to surface water quantity or flow are anticipated.

The Project itself will modify the surface-water regime within the Project boundaries by dewatering and expanding the pre-existing open-pit and underground workings, collecting all site runoff, and constructing water management, storage, treatment and processing facilities.

(vi) Groundwater

The underground and open-pit dewatering and expansion activities, as well as water consumption from wells, will temporarily alter the groundwater flow hydrology, change the hydrogeologic balance, subsurface recharge-discharge relationships, as well as groundwater elevations in the Project region. Groundwater availability is expected to decrease, due to increased water demands for ore processing and human consumption.

(b) Potential Chemical Impacts

(i) Air Quality

Fugitive Dust and Gaseous Emissions

Fugitive dust, including PM_{10}, generated from mining operations (road travel, blasting, stockpiling, etc.) could potentially impact air quality in the Project region. Emissions of greenhouse gases and other air contaminants will be generated from the combustion of diesel fuel in stationary and mobile sources, from other combustion processes (e.g., incineration, steam plant operation, etc.), from mine blasting operations, and from ore processing. Small amounts of PM_{10} emissions will be generated from combustion.

Emissions from the Project are not anticipated to have a significant impact on the surrounding environment. Dust levels, if not mitigated properly, may be a significant impact, locally and regionally, as the particulate matter (i.e., PM_{10}) may be transported a great distance away.

Noise

Local noise levels will increase especially during the construction phase. Project activities that will generate noise and vibrations are:

- Facilities construction
- Blasting
- Mining operations in the open-pit area
- Processing operations at the plant
- Transportation.

The noise and vibrations from Project activities may impact on-site and nearby off-site receptors.

(ii) Soil and Sediment Chemistry

Potential chemical impacts to the soils within the Project area include: a decrease in fertility by removal of key nutrients; acidification or alkalinization; salinization; and, contamination from wind-blown dust polluted water seepage and accidental spills.

Impacts to sediment chemistry are not anticipated, due to their scarce presence at the Project area and the zero-discharge design of the Project.

(iii) Water Quality

Surface Water

Due to the limited quantity of surface water within and near the Project area, as well as the zero-discharge design of the Project, impacts to surface water quality are not anticipated.

Groundwater

Potential impacts to groundwater quality during mining operations may arise from: seepage from the overburden / waste rock stockpiles; accidental spills of chemicals, fuels, or other mine reagents at the process facilities and / or in the pit; leakage from the RMA and other water management facilities; and, introduction of chemical additives through exploration.

(c) Potential Biological Impacts

Potential Project-related impacts to flora and fauna can be both direct and indirect in nature. Direct impacts include changes that occur as a result of actual mining operations, while indirect impacts describe changes that occur resulting from non-mining activities in, and immediately adjacent to, the Project area.

Most potential impacts on flora relate to significant landscape level changes. Potential impacts include loss of vegetation cover / removal of native steppe vegetation from roads, laydown areas, construction and other field-related activities; potential loss of species; chronic and self-perpetuating erosion prone areas owing to laydown areas, roads, and deposition of overburden and residuals; removal of critical habitat of potentially existing T&E species; introduction of invasive species from affiliated settlements and agriculture; increased human activity, pets, hunting, and gathering; and changes in vegetation structure / composition. Any mining and nonmining activity deleteriously impacting the steppe vegetation cover can potentially contribute to desertification processes.

Potential impacts on fauna include: reduction in population sizes across an undetermined number of species; increase in population sizes of an undetermined number of species; decline in individual health due to stress; and, introduction of new and alien species. Potential sources and causes of the impacts include: habitat loss in steppe and wetland areas; noise; vehicle / road kills; illegal hunting and dogs; decline in individual health due to stress as a result of disturbance, as well as increased competition for resources; and, change in population size or community structure on a local level for an undetermined number of species.

(d) Potential Social and Cultural Impacts

(i) Socioeconomics

In general, the potential impacts from the Project are both beneficial and adverse. The potential positive social impacts include, but are not limited to: employment opportunities; purchase and / or utilisation of Mongolian supplies and services; increased tax base; land and infrastructure improvements; and, community development programs.

The potential negative social impacts include, but are not limited to: decreased grazing area; decreased water resources; relocation of herders; short-term increased land disturbance; increased demand on infrastructure and services; and short-term increased risk to human health and safety.

(ii) Archaeology

Since there are no archaeological sites within the Project area, no impacts to archeological resources are anticipated. In case new archaeological sites are unexpectedly found during Project development (“chance find”), mitigation measures (see Item 2.4.6, Proposed Mitigation Measures) will be implemented to prevent the loss of cultural resources.

(e) Potential Radiological Impacts

Radiological impacts could occur due to the release of particulate material (e.g., wind-blown ore material) or due to radiation (e.g., radon emanation from the ore).

Sources and causes for radiological impacts on soil chemistry, air and water quality, flora and fauna, as well as human health include windblown residuals or other releases from the RMA (possible long-term impacts), emissions from the process plant and underground workings (through the Project life) and, spills and other accidental releases (short-term impacts).

Substances originating from uranium ore, including radionuclides, could be taken up by plants, and then ingested by herbivores, resulting in contamination of the food chain.

(f) Potential Regional and Cumulative Impacts

The Project site is located in a remote area of northeastern Mongolia. Livestock grazing is the only commercial activity in the Project area. Over 85% of the residences in the Project region are nomads who come to the area each year only during the grazing season (usually between May and August).

There is no mining activity within the Dornod Uranium Mining District at the present time. Khan will be the first developer in the District.

Over 160 ha of mine-related disturbance exists at the Project site from previous mining activities. Khan’s operation will include enlarging the existing open pit and increasing the number of overburden stockpiles, resulting in a cumulative impact on topography. Khan intends to implement a reclamation program at Project closure with recontouring and revegetation of the overburden stockpiles, some of which would otherwise remain unreclaimed from former mining.

Significant regional and cumulative effects from the Project on air quality, surface-water or groundwater resources are not anticipated. Dust and stock emissions from the site will be dispersed quickly into background levels (far before reaching any other potential emission sources in the region). Zero water discharge from the Project eliminates any potential impact to surface water in the region. Collecting all on-site surface-water runoff, recycling process water, and using water from the underground workings and open pit dewatering will significantly reduce the impact on regional groundwater resources.

Adverse regional or cumulative socio-cultural impacts are not anticipated. On the contrary, the benefits to the local community and Mongolia will be significant based upon increased employment and government revenues.

Transport of the Project supplies and the yellowcake will increase local traffic. Upgrading of the road from Choibalsan to the Project site will provide much convenience to the local communities and, thus, benefit the economic development of the region.

The Project will obtain electric power from the national grid. Further expansion of the power plant may be required for new industrial development projects in the region.

20.4.6 Proposed Mitigation Measures

A detailed discussion of the proposed prevention and mitigation measures for the Project is presented in the ESIA. The ESMP has been developed to assure that any negative environmental and social impacts are minimised or mitigated during construction and operation of the Project; and, that the site can be reclaimed to stable conditions following final decommissioning and closure. Implementation of environmental and social protection measures will enhance the capability of the Project to operate in an environmentally sound and socially responsible manner. The prevention and mitigation measures will be incorporated into the final design, construction, operation and closure of the Project.

Khan is committed to conformance with relevant international environmental, health and safety guidelines in the design, operation and eventual closure of the Dornod Project. The proposed mitigation measures are subject to change during the life of the Project, based on management and regulatory requirements, and experience gained while implementing the various phases of the Project, which can result in improved performance of these measures.

Through the implementation of a series of modern ESMP, Khan will endeavor to eliminate, reduce, or otherwise manage all areas presenting potentially significant impacts to human health, the environment, and social and cultural resources. Khan has committed to implement a comprehensive ESMP with the following components:

- Environmental Management and Monitoring Plan
- Occupational Health and Safety Plan
- Radiation Protection Plan
- Public Consultation and Disclosure Plan
- Waste Management Plan
- Emergency Response Plan.

The following is a summary of the measures that will be employed to avoid, minimise or mitigate potential environmental and social impacts associated with the Project. The preventative and mitigative measures related to physical, chemical, biological, social and radiological aspects of the Project are discussed, followed by measures related to specific Project activities.

(a) Proposed Physical Mitigation Measures

(i) Topography

The impact mitigation measures for topography will primarily be implemented during the reclamation and mine closure phase of the Project. Reclamation of disturbed areas will include, but not be limited to recontouring and revegetating the new landforms for stabilization purposes. The water collection-retention ponds will be decommissioned, backfilled and revegetated. Redundant on-site roads and other infrastructure will be decommissioned and reclaimed. All disturbed areas will be revegetated.

The long-term modification to the topography and geomorphology in the Project area will include the open pit, the RMA, and the overburden placement areas.

During construction and operations, the visual resource of the Project area may be improved by utilising building materials and paint that blend with the natural environment, building low-profile structures, and keeping a clean, well-maintained site. The proposed reclamation and decommissioning activities (e.g., recontouring, revegetating, etc.) will greatly enhance the visual resource.

(ii) Soils and Sediments

Topsoil from all major disturbed areas will be preserved and stockpiled. This material will be later used for reclamation and revegetation of the on-site facilities, overburden placement areas, and the RMA.

Erosion and sedimentation control measures will be adopted to minimise impacts on soil and reduce sedimentation, and include, but are not limited to: surface-water runoff diversion; soil stabilisation with gently-sloped stockpiles and reseeding; sediment fences; traffic minimisation; and low speed limit enforcement. Regular monitoring will be performed, and additionally, erosion and sediment control structures will be checked after major precipitation event during construction and operations. Areas no longer utilised for mining operations will be promptly reclaimed (i.e., soils replaced and revegetated) to minimize erosion. Upon mine closure, disturbed areas will be regraded, covered with topsoil and seeded for revegetation.

The water management facilities (e.g., RMA, polishing pond, and water collection pond) and runoff collection ditches constructed for the proposed operations will be used as additional measures for sediment control.

(iii) Water

Surface Water

Effort will be made to ensure zero discharge to surface-water drainages, which will eliminate any potential impacts of the Project on surface water. All site runoff will be captured by the open pit, water collection pond, and / or the RMA. Water management facilities are designed to sustain a 24-hr 1,000-yr storm.

Groundwater

Recycling of mine process water, as well as collecting and storing water, will be key mitigation measures to minimise water consumption and efficiently utilise water resources.

Upon closure of the mine facilities, the groundwater table is expected to equilibrate naturally, once the water management facilities are decommissioned and the open pit is flooded.

(b) Proposed Chemical Mitigation Measures

(i) Air Quality

Mitigation measures for air quality will require controlling and monitoring dust, gaseous emissions, and noise. Mitigative measures for controlling dust include, but are not limited to: minimising land disturbance; promptly covering or revegetating exposed soils or erodible materials; and, using dust suppression methods. Gaseous emissions in the Project area may be managed by: utilising lower-sulfur fuel; constructing an appropriate emissions stack height to avoid excessive ground level concentrations and ensure reasonable diffusion; and installing pollution control mechanisms. Noise levels may be minimised and controlled through the application of techniques, such as regularly maintaining equipment; implementing enclosure and cladding of processing facilities; and, optimizing traffic routes and speed limits to reduce reversing alarm and to maximize distances to sensitive receptors.

(ii) Soil and Sediment Chemistry

Regular monitoring of soil chemistry will be conducted to ensure effective soil mitigation. In the event of an accidental spill that results in contamination of the soil, the affected area will be surveyed and promptly remediated.

(iii) Water Quality

Surface Water

The limited amount of surface water within and near the Project area and the zero discharge operation of the Project will minimise or eliminate potential impacts to surface-water quality.

Dams, dikes, berms, and ditches will be constructed to collect runoff within the Project area.

Routine inspection and maintenance of all equipment and facilities will be conducted to minimise impacts from accidental releases of contaminants.

Groundwater

The key mitigative measure to reduce and eliminate potential impacts to the groundwater quality is proper management of the source materials, which involves containment and appropriate secondary treatment, if needed. Chemicals will be properly stored and handled. Accidental spills will be properly cleaned up in a prompt manner. Exploration / confirmation wells will be installed and abandoned properly to avoid groundwater contamination.

The RMA and water management facilities will be lined to prevent seepage. Additional groundwater monitoring wells will be installed to monitor seepage. In addition, a site-wide groundwater monitoring program will be developed to ensure groundwater quality will be comparable to premining baseline levels.

(c) Proposed Biological Mitigation Measures

A number of measures for mitigating impacts to ecosystems are being considered for the Project area including: supporting refuges in the vicinity of the Project; applying a comprehensive and effective reclamation and revegetation plan; and, supporting a periodic environmental monitoring program for certain key groups of species to make ongoing adjustments to the ecosystem management and mitigation strategy.

Revegetation of sites will use accepted technology in the interest of a cost-effective and efficient program. GIIP will be utilised to facilitate and shorten the time period necessary to stabilise the soils and potential sediment sources. To ensure successful revegetation, native seed mixtures will be selected and applied, and invasive species will be actively controlled as necessary. In addition, efforts will be made to improve habitat and other sensitive species.

(d) Proposed Social and Cultural Mitigation Measures

(i) Socioeconomics

The exclusion of the Project area from livestock grazing will minimally impact local herders, with the exception of one household that will be voluntarily relocated. Much of the Project area was previously disturbed and left unreclaimed; therefore, unsuitable for grazing. Nevertheless, Khan has been paying land use fees to the soum to compensate for the use of potential grazing land.

Using revenues from the water use fees, the soum administration maintains existing wells and installs new wells for public use.

To minimise potential adverse impacts, Khan is having open discussions with the regional government and local services to plan the expansion and / or necessary maintenance of infrastructure and services that is mutually beneficial.

Khan will eliminate or minimise risks to public health and safety. A fence will be installed around the open-pit area and the RMA. Access / egress points will be controlled, and the mine camp may be fenced. On-site security will ensure that herders and livestock maintain safe distances from exploration, operation and reclamation activities.

Employees will be provided with safety training, as well as on-site access to medical personnel, supplies, communications, and vehicle transport in case of an accident.

In addition, the PCDP has been developed as part of the ESMP to ensure that stakeholders including the local public are provided with adequate and timely information, as well as sufficient opportunity to voice their opinions and concerns. Through effective public consultation and disclosure, Khan and stakeholders may mutually benefit by pursuing environmental and social opportunities throughout the life of the Project.

(ii) Archaeology

Since no impacts to archeological resources are anticipated, mitigation measures are designed to prevent the loss of cultural resources if new resources are unexpectedly found within the Project area (“chance find”). Chance find procedures will be established to prevent any unnecessary disturbance of cultural resources. These procedures include protocols for reporting, record-keeping, excavation, protection, and / or removal under the supervision of expert archaeologists and governmental authorities. A “no-disturbance” policy for on-site and off-site archeological remains will be established and enforced.

(e) Proposed Radiological Mitigation Measures

The radiological conditions in the Project area have been impacted by previous mining activities, and the potential exists for additional impacts. The use of up-to-date techniques for mining, ore and overburden storage, and material handling and disposal, is intended to minimise any short-term impacts that may occur through the Project life. In addition, up-to-date reclamation techniques are planned to minimise any long-term radiological impacts after completion of mining.

Radiological mitigation measures for land use will primarily occur during the reclamation phase, as the land in the region is primarily used for livestock grazing and wildlife habitat. Access to potential radiation sources, such as the process plant, the ore stockpiles, the open pit / pit lake, and the RMA, will be restricted.

Site access restrictions, health and safety protocols, personnel monitoring, and proper use of personal protective equipment (PPE) will provide protection for workers, contractors, and local herders. In addition, monitoring of radiological parameters from the commencement of the Project through reclamation and closure provides information with respect to environmental media, including air, soil, water, and vegetation. Radiation from the process plant, underground workings, and the open pit will be closely monitored during operations.

Radiological mitigation for soils include properly protecting stockpiled soil, because surface material may contain elevated radionuclide concentrations due to previous mining disturbance, and protecting the soils against erosional effects.

Surface water and groundwater will be protected from potentially contaminated water. The oxidation of ore deposits allows for mobilization of parameters (e.g., uranium). Lining and monitoring of the water management facilities, including the RMA and transfer ditches will be implemented to reduce potential seepage contamination.

Routine inspections and maintenance of equipment, facilities and spill prevention / containment installations will minimise accidental discharges that may cause radiological impacts to surface water and groundwater. Monitoring of the water from underground workings, the open pit, RMA and other water management facilities will also be conducted. In addition, monitoring will be conducted to determine if leakage from the RMA is occurring.

Employees will receive training, guidance, and PPE to safely handle, store, decontaminate, and dispose of the radioactive materials in the Project area. Employees will also be trained to recognise potential hazards and to perform assigned duties in a safe and healthy manner to help reduce the possibility of accidental release. The remote nature of the site significantly reduces the possibility of radiological impacts to the general public.

With respect to the underground mine, potential radiological exposure will be reduced with ventilation using high-volume exhaust fans, PPE, and limited exposure durations. In the open-pit mine, PPE and limited exposure durations also reduce potential exposure. Within the processing areas, ventilation, PPE, and limited exposure should be used to reduce potential exposure. Protection of mine personnel in the living quarters and in the office areas will be achieved by reducing releases from adjacent sources (e.g., the RMA), implementing dust control measures, and constructing barriers where possible (e.g., a solid wall on the side of the mine camp closest to the RMA), and positive ventilation within the mine camp.

The primary radiological mitigation efforts for transportation address routes, shipping containers, and accident response.

20.4.7 Waste Treatment, Storage and Disposal

Waste management facilities will be designed to minimise impacts on air and water resources and may include gas and / or leachate control systems and proper separation distances, where appropriate. The facilities will have separate receiving and handling areas for hazardous and nonhazardous wastes. Environmentally sound and contained storage areas will be made available for materials that cannot be treated or disposed of immediately upon arrival at the facility. Waste will be composted, whenever possible.

The waste expected to be produced during the operations of the Project include the following:

- Nonhazardous solid waste
- Sewage waste
- Hazardous waste (e.g., antifreeze, motor oil, grease, paint, used batteries)
- Mill tailings
- Processing residuals.

In general, all nonhazardous wastes will initially be sorted for reusable and recyclable materials. This activity is expected to increase over the life of the mine due to planned development of local recycling markets. The remaining waste streams will be managed by a combination of incineration and landfilling.

Hazardous wastes will initially be stored temporarily on-site and incinerated as may be permissible under Mongolian law and in accordance with GIIPs. Some hazardous wastes may be transported off-site to an approved treatment/disposal facility when accumulated volumes warrant their removal.

The major effluent from the uranium processing will be the residuals (tailings). The residuals will be permanently stored in the lined RMA. The processing facility and other support facilities will produce lesser quantities of other liquid and solid wastes, which will be recycled in the various processing operations, discharged to the RMA, or discharged to a sanitary leach field. Gaseous effluent and dust generated by the Project will be released into the atmosphere with close monitoring to ensure compliance with applicable air quality standards.

20.4.8 Occupational Health and Safety Measures

As part of the ESMP, an Occupational Health and Safety Plan (OHSP) was prepared for the Dornod Project. The OHSP covers all appropriate health and safety-related issues.

A full-time medical professional will be retained on-site to address minor injuries and illness at the clinic. An Emergency Response Plan (ERP) was also prepared for the Project that outlines how serious health and medical cases will be addressed.

General safety features will be incorporated into the construction and operation of all facilities at the Project. Khan will ensure that PPE will be provided following the stipulations in the Mongolian Law on Labor and GIIPs. Workplace safety and mine worker health will be maintained. Khan will establish and maintain sanitary living conditions, clean drinking water and proper waste disposal.

Employees will meet the qualifications for the job prior to being hired. In addition, all employees will go through basic health, safety, and first-aid training upon employment and regularly thereafter, with the corresponding records duly kept. Field employees will undergo additional training as required by the job description that may include equipment training, safety briefings, and emergency responses. Great caution will be exercised before and during entry of inactive or abandoned underground workings. All underground workers will be under the supervision of an underground shift chief.

20.4.9 Radiation Protection

Similar to the overall occupational health and safety considerations, the primary responsibility for the health and safety of workers will lie with Khan and with the mine workers. Legislation related to radiation protection of the proposed Project includes international acts, regulations and codes. Mongolia is a member of the IAEA, which sets regulations and standards for occupational radiation exposures. More specific guidance is available from the Canadian Nuclear Safety Commission (CNSC), which sets standards, communicates regulatory expectations, and provides guidance for developing radiation protection policy, programs, plans, and procedures for uranium mining Projects.

Recognising international acceptance of the work of IAEA and CNSC, Khan will adopt an appropriate set of standards and practices for this Project through its Radiation Protection Plan (RPP, part of the ESMP). The RPP will set out the policy, purpose, goals, objectives, and overall performance standard of radiation protection at the Dornod Project area. The RPP will also identify corporate responsibilities and accountability for radiation protection and associated supporting policies and work instructions throughout the life of the Project. The ALARA principle will be a basis for the RPP in that every reasonable effort will be made to maintain radiation exposures as far below regulatory dose limits as practical.

Radiation exposures will be controlled by using three basic principles: time, distance, and shielding. Miniming time spent handling radioactive material; maximising the distance from radioactive material; and applying shielding material between the source and the worker. Employees responsible for implementing the RPP will be trained to ensure that all aspects of the RPP are addressed. In addition, all employees and visitors, commensurate with anticipated levels of exposure will be trained to ensure protection of health, and the environment, and ensure appropriate emergency response.

Monitoring will include a variety of approaches, such as: identification of exposure pathways and implementation of proper control and monitoring measures; and measurement of radiation levels in areas where there is a reasonable expectation of occupancy, of specific radionuclide concentrations in media that may be a potential pathway for exposure, and of health parameters. Compilation and interpretation of the monitoring results will also be performed periodically for comparison with baseline and with anticipated conditions.

20.4.10 Emergency Response and Hazard Protection

The ERP provides procedures and guidelines to follow in the event of an accidental chemical spill, equipment failure, or other emergencies. The ERP also covers emergency identification, response, and notification procedures.

A preventative maintenance schedule will be set up for each mine facility. Preventative maintenance will be conducted on a regular and frequent basis, and a record of all maintenance procedures will be maintained on-site.

Khan will establish an emergency response team to handle such incidents as fires and spills. Team members will have special training to deal with all types of emergencies.

Following Article 89 of the Mongolian Law on Labor, fire prevention measures, including installation of a fire alarm system, fire extinguishers and special equipment, will be placed at strategic locations and maintained on a regular basis at the Project site.

Khan will establish a program with a comprehensive classification and identification of all hazardous materials at each location so that appropriate management procedures can be followed, which include keeping a clear inventory and retaining an MSDS on each hazardous material. Storage areas and containers will be secured and monitored to ensure no leakage will occur.

Fuels are the most abundant material that could be hazardous if spilled; therefore, special attention will be given to fuel storage and handling. All tanks at the storage site will be underlain with impermeable material and will be surrounded by berms capable of holding 120% of the total tank(s) capacity. Tanks will be stored away from surface drainages and smoking areas. Tanks will be: equipped with lightning arresters and a grounding system; coated with mastic tar or an anti-corrosive agent to prevent tank corrosion; and connected with a common venting system controlled by a valve to ensure sufficient top space pressure to restrict the release of gases.

20.4.11 Net Environmental and Social Impacts

The predicted net environmental and social impacts for the Project presented in this section are based on an impact analysis conducted for this ESIA with the following assumptions.

- Mongolian laws and regulations applicable to the Project will be complied with in the design, construction, operation and closure of the Project
- Internationally recognized criteria and standards (e.g., IFC Performance Standards, Equator Principles, WHO guidelines, etc.) will be adopted in the design, construction, operation and closure phases of the Project
- Proper mitigation measures, employing GIIP as defined by the IFC, will be implemented during all phases of the Project.

Many adverse effects that could occur from the Project will be eliminated or minimised by proper design, maintenance, management, and mitigation measures. The net social and environmental and social analysis assumes that the environmental and social management, monitoring, and reclamation measures will be implemented as discussed in both the ESIA and ESMP.

Table 20-6 summarises the potential net environmental and social impacts of the Project. Net impacts were calculated based on worst-case impact scenarios (i.e., gross impacts), minus the effects of all proposed prevention and mitigation measures. This provides an estimate of the net impacts, both short- and long-term, that can be anticipated as a result of the Project's construction activities, operational activities and closure. The net impact analysis table is not intended to provide a detailed list of all possible impacts, but is designed to generally highlight potential risks and associated impacts in a concise manner.

This analysis indicates that implementation of the environmental and social management, mitigation, monitoring, and reclamation measures that have been proposed by Khan will eliminate or minimise the potential negative environmental and social impacts of the Project; and, will provide economic and social benefits to the region.

Table 20-6
Summary of Net Environmental and Social Impacts

Environmental Parameter	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
Topography	Construction of new land features such as the water management facilities, RMA, overburden placement areas, roads, sediment and water runoff controls, mining support facilities; and Expansion of the pre-existing open pit.	Recontour new land features at closure; Cap and revegetate RMA at closure; Revegetate disturbed areas as practical; Decommission and/or demolish mining facilities at closure; Backfill water and sediment ponds at closure; Progressively backfill underground mine and air shafts during operations; and Allow the open pit to flood naturally after closure	Short-term: Changes can be significant with newly constructed land features and mine expansion. Long-term: Except for the expanded open pit, topographic changes will be minimal due to recontouring and revegetation.
Air	Gaseous emissions from stationary and mobile sources; Fugitive dust emissions; and Increased noise levels and blasting vibrations.	Minimize land disturbance; Cover or revegetate exposed soils or erodible materials to reduce dust generation; Suppress dust with water or surfactants; Utilize low-sulphur fuel; Construct appropriate stack heights for emissions; Install pollution control features; Maintain equipment; Implement enclosure and cladding of processing facilities; Install proper noise barriers and/or noise containments at/near source equipment and at facility boundaries; and Optimize traffic routes and speed limits.	Short-term: Slight increases in dust. Increased noise levels and vibrations. No significant impact from gaseous emissions. Long-term: No significant impacts from dust after closure. Gaseous emissions, noise and vibrations will cease after closure.
Geology and Mineral Resources	Alteration of the geologic configuration; and Removal of uranium ore.	Inherent to open pit and underground mining; no mitigation measures exist.	Alteration of the geologic configuration; and Removal of uranium ore.

Table 20-6
Summary of Net Environmental and Social Impacts

Environmental Parameter (cont)	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
Soils and Sediment	Removal of topsoils; Alteration of the soil profile; Increased erosion; Decreased soil productivity; Possible contamination; and Increased sediment transport or production.	Stockpile and preserve topsoils for use in reclamation; Control surface-water runoff and maintain the zero-discharge facility as designed; Minimize traffic and enforce low speed limits; Control erosion and sedimentation with sediment fences, vegetative strips, ponds, and ditches; Properly treat contaminated soil; and Monitoring.	Short-term: Significant direct impact from soil displacement. Long-term: No significant impact due to reclamation. Net sediment impacts are not anticipated.
Surface Water	Accidental release of contaminants.	Follow standard operating procedures; Control and treat surface-water runoff as planned; Maintain the zero-discharge facility as designed; Control erosion and sedimentation; Routinely inspect and maintain equipment and water management facilities; Implement a Spill Control Plan; Properly store and handle chemicals; Immediately clean up accidental spills or release of contaminants; and Monitoring.	No short- or long-term impacts to surface water are anticipated. The Project is designed as a zero-discharge facility; and, there are no natural surface-water bodies within and near the Project area.

Table 20-6
Summary of Net Environmental and Social Impacts

Environmental Parameter (cont)	**Potential Gross Impacts**	**Mitigation Measures**	**Potential Net Impacts**
Groundwater	Alteration of subsurface recharge-discharge relationships; Alteration of groundwater flow pattern and conditions; Reduced potentiometric surface elevation; Reduced groundwater availability; and Groundwater quality alteration from accidental releases and spills, from mining and geologic exploration/ monitoring / well drilling and from flooding.	Recycle mine process water; Collect and store water resources; Decommission and reclaim on-site water management facilities at closure; Regularly monitor groundwater quantity and quality, including the cone of depression from dewatering activities; Properly store and handle chemicals; Routinely inspect and maintain equipment and water management facilities; Promptly clean up accidental spills; Line the RMA and other water management facilities; and Monitoring.	Short-term: Alteration of subsurface recharge-discharge relationships and groundwater flow conditions; and Reduced groundwater availability and groundwater resources. Long-term: No significant impacts to groundwater are anticipated.
Biological	Direct vegetation removal; Species population reduction or changes in species diversity; Chronic and self-perpetuating erosion-prone areas; Fragmentation and loss of habitat; Introduction of new or invasive species; Increased human presence and activity; Changes in vegetation structure / composition; Decline in wildlife health; and Displaced wildlife.	Update baseline environmental data; Stockpile non-commercial vegetation and slash for reclamation; Support refuges in the vicinity; Implement an effective reclamation and revegetation plan; Select native seed mixtures; Actively control invasive species as necessary; Protect sensitive species; Support a periodic environmental monitoring program; and Collaborate ecosystem management with outside stakeholders.	Short-term: Significant impact to vegetation from removal. Possible reduction in some wildlife populations in areas of disturbance. Long-term: No significant impacts; positive net gain in grassland vegetation due to reclamation and revegetation of previous industrial barrens.

Table 20-6
Summary of Net Environmental and Social Impacts

Environmental Parameter (cont)	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
Social	Increased employment opportunities; Purchase and/or utilization of Mongolian supplies and services; Increased tax base; Land improvements; Community development programs; Decreased grazing area; Decreased water resources; Relocation of herders; Short-term increased land disturbance; Increased demand on infrastructure and services; and Short-term increased risk to human health and safety.	Continue to pay land-use and water-use fees to the Soum Government; Reclaim/revegetate disturbed land; Consult regional government and local services to plan the expansion and / or necessary maintenance of infrastructure and services; Restrict access to the Project area for health and safety reasons; Provide employee safety training as well as immediate medical attention in case of an accident; and Implement the PCDP.	Short-term: Increased employment opportunities; purchase and/or utilization of Mongolian supplies and services; increased tax base; land improvements; and implementation of community development programs; minor impact to herders; increased land disturbance; increased risk to human health and safety. Long-term: Improved roads and public infrastructure; improved general economy in the region.

Table 20-6
Summary of Net Environmental and Social Impacts

Environmental Parameter (cont)	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
Radiological	Radon emanation; Release of radioactive particulates; Accidental spills; Mobilization of materials (e.g., uranium) into groundwater; Vegetation uptake of radionuclides; and Human and wildlife exposure to radionuclides.	Restrict access to the Project area; Line water facilities with appropriate materials to prevent seepage to groundwater; Monitor water management facilities; Suppress dust; Protect stockpiled topsoil for use in reclamation; Construct effective water management features; Implement RPP; Reduce and monitor employee exposure; Provide training, guidance and PPE to employees; Plan comprehensive transport, shipping containers, and accident response; Utilize up-to-date mining and reclamation techniques; Upon closure, reduce radiation exposure from the Project activities and from the previous un-reclaimed mining activities by others through soil cover and reclamation measures; and Monitor environmental parameters through the Project life.	Short-term: Restricted access; increased human radiological exposure; increased potential wildlife exposure to radionuclides; increased potential vegetation uptake of radionuclides; radon emanation; increased radioactive particulate release. Long-term: Restricted access; land improvement due to reclamation of previous mining impacts; reduced radiation exposure to wildlife due to soil cover from Khan's reclamation of previous mining activities by others at the site (although natural levels of radiation will remain in this area).

20.5 Capital Cost Estimate

20.5.1 Capital Cost Summary

The capital cost for mining and surface facilities with a capacity of 1 225 000 t/a, as described within the DFS is USD 332.8 million in fourth quarter 2008 US dollars, and is subject to the qualifications and exclusions listed below.

The capital cost is summarised in Tables 20-7 and 20-8, and is inclusive of the costs up to and including plant commissioning and start up. Sunk cost, sustaining capital cost and deferred capital costs are excluded from these estimates.

(a) Currencies

The base currency of the capital cost estimate is in United States dollars. Other currencies used and conversion rates are as follows.

Currency	**Conversion Rates**
Canadian Dollar	1.00 CAD = USD 0.81
European EURO	1.00 EUR = USD 1.35
Australian Dollar AUD	1.00 AUD = USD 0.67
South African Rand	1.00 RAND = USD 0.101
Chinese Yuan Renminbi CNY	1.00 CNY = USD 0.146
Mongolian Tugrik MNT	1.00 MNT = USD 0.00081

(b) Feasibility Study

The DFS was carried out by Aker Solutions with other consultants for Khan. Aker Solutions was responsible for both the infrastructure and process design including process plant layout and plant building design. This responsibility included, but was not limited to, process design, equipment list, capital cost estimate and operating cost estimate. Mine plan and estimates were provided by P&E. Residue Management Area (dam / pond) design and estimates were provided by Golder.

All costs are expressed in fourth quarter 2008 US dollars with no allowance for escalation, interest or financing during construction. Budgetary estimates for mining and mining equipment were provided by P&E. Eight-five percent of process equipment prices were obtained from budgetary quotation by Aker Solutions, whilst the balance was estimated from in-house historical data of similar projects. Commodity pricing for earthwork, concrete, steel, architectural and piping were provided by local contractors based in Mongolia. Labour rates and equipment usage rates used throughout the estimate were provided by the same source as the commodity prices. The cost estimate, based on the designs presented in this DFS, has a predicted accuracy level of ±15%.

Table 20-7 - Summary by Area

Aker Solutions	Dornod Project	EST6-04/20/2009-14:43:54
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	COST IN USD	Final - March 05, 2009

	DESCRIPTION	HOURS	LABOR	EQP USAGE	MATERIAL	SUBCONTR	EQUIPMENT	TOTAL
000	Site Development	226177	3,280,497	395,604	617,778	0	0	4,293,879
100	Mining	20467	311,315	40,935	276,250	4,187,000	1,747,900	6,563,401
114	Mine Dry and Main Offices	33426	502,045	199,366	1,065,060	0	30,800	1,797,272
120	Underground Development	19136	291,063	38,272	586,060	44,889,000	1,550,100	47,354,497
125	Services (Electrical, Air, Water)	29	447	58	924	5,093,000	0	5,094,431
129	Mobile Mine Equipment	0	0	0	0	12,583,600	0	12,583,600
143	Explosive Storage Facility	53668	790,201	299,967	852,392	0	0	1,942,561
160	Mine Haul Road	45381	639,610	83,780	249,420	0	0	972,811
210	Crushing	142210	2,150,469	978,220	2,519,019	0	1,961,178	7,608,888
301	Building Structure	287133	4,418,615	2,584,593	8,442,591	0	0	15,445,800
310	Grinding	95312	1,447,678	583,230	713,700	0	9,515,337	12,259,947
311	Thickening	38913	602,067	170,072	305,789	0	1,996,425	3,074,353
320	Leaching	164255	2,486,826	1,113,129	1,283,370	0	4,660,448	9,543,775
330	Leach Neutralization Area	39569	603,965	232,567	397,674	0	1,009,769	2,243,976
350	Resin in Pulp and Resin Elution Area	62237	962,648	353,045	731,014	0	8,689,721	10,736,429
355	Impurity Precipitation and Gypsum Film	23030	355,643	111,775	361,764	0	727,893	1,557,077
360	Product Recovery and Loadout	32243	498,075	173,146	351,288	0	1,279,744	2,302,254
370	Effluent Treatment and Reverse Osmosis	34204	531,214	155,531	449,905	0	1,607,027	2,743,679
380	Utilities & Reagents	324414	5,067,361	1,780,970	3,798,687	200,000	9,034,655	19,881,675
490	Central Control System	4742	72,139	9,486	46,066	0	952,231	1,079,922
590	Tailings	56712	895,342	269,572	1,133,711	0	540,702	2,839,328
592	Residue Management Area	0	0	0	0	6,185,574	0	6,185,574
610	Main Sub-Station	33581	508,604	87,183	844,154	0	3,172,500	4,612,441
620	Site Electrical Distribution	0	0	0	0	581,500	0	581,500
635	Sewage System, Treatment and Distribution	13305	199,475	101,988	239,007	0	0	540,470
651	Incineration Facilities	2600	39,552	5,200	39,646	0	11,200	95,600
659	Environmental Monitoring System	0	0	0	0	247,500	0	247,500
720	Boiler Plant	55395	915,697	276,977	1,539,500	0	370,000	3,102,174
730	Facility Building	17950	273,022	35,900	519,336	0	246,700	1,074,959
732	Assay Laboratory Facility	2366	35,998	4,733	45,163	400,000	31,700	517,595
733	Warehouse	17467	262,071	104,884	337,821	0	96,200	800,977

Table 20-7 - Summary by Area

Aker Solutions	Dornod Project	EST6-04/20/2009-14:43:54
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	COST IN USD	Final - March 05, 2009

	DESCRIPTION	HOURS	LABOR	EQP USAGE	MATERIAL	SUBCONTR	EQUIPMENT	TOTAL
734	Security Check Point	3399	50,989	15,689	54,753	0	8,700	130,132
740	Maintenance Facility	12962	193,550	70,491	263,069	0	99,200	626,311
745	Plant Mobile Equipment	0	0	0	0	0	2,892,000	2,892,000
760	Railroad	0	0	0	0	1,811,879	0	1,811,879
780	Communicationn System	1182	17,979	2,364	13,813	0	29,200	63,357
790	Permanent Camp Facilities	0	0	0	0	2,270,000	0	2,270,000
815	Incoming Power	0	0	0	0	0	0	0
890	Railroad Upgrades	0	0	0	0	7,963,452	0	7,963,452
910	Temporary Buildings and Facilities	0	0	0	0	2,500,000	0	2,500,000
911	Temporary Construction Utility Services	0	0	0	0	884,000	0	884,000
912	Winter Work & Lost Productivity	0	0	0	0	387,000	0	387,000
920	Construction Equipment & Small Tools	0	0	0	0	2,972,000	0	2,972,000
950	Construction Camp Facilities	0	0	0	0	6,225,560	0	6,225,560
955	Construction Power & Fuel	0	0	0	0	8,160,000	0	8,160,000
960	Spare Parts	0	0	0	0	2,410,389	0	2,410,389
965	Initial Fills	0	0	0	0	3,816,836	0	3,816,836
970	Freight & Insurance	0	0	0	0	5,431,361	0	5,431,361
975	Vendor Representatives	0	0	0	0	964,000	0	964,000
980	Owner's Cost	0	0	0	0	12,420,000	0	12,420,000
991	EPCM	0	0	0	0	37,747,944	0	37,747,944
992	Commissioning & Start-up	0	0	0	0	4,714,316	0	4,714,316
993	Third Party Engineering	0	0	0	0	1,000,000	0	1,000,000
995	Contingency	0	0	0	0	37,717,106	0	37,717,106
REPORT TOTALS		1863378	28,404,169	10,278,741	28,078,737	213,763,017	52,261,335	332,786,000

Table 20-8 - Summary by Commodity

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	DESCRIPTION	HOURS	LABOR	EQP USAGE	MATERIAL	SUBCONTR	EQUIPMENT	TOTAL
D0	Mining	0	0	0	0	66,752,600	0	66,752,600
D1	Earthworks	327067	4,716,107	579,081	1,462,257	16,208,405	0	22,965,851
D2	Concrete	311860	4,585,069	3,113,608	6,295,400	0	0	13,994,077
D3	Structural Steel	173110	2,850,089	2,685,044	7,056,601	0	0	12,591,735
D4	Architectural	182548	2,667,975	381,531	2,164,376	2,270,000	0	7,483,883
D5	Mechanical	381620	5,888,396	1,908,100	1,250,000	600,000	38,609,684	48,256,180
D6	Electrical	200846	3,054,873	401,692	4,955,850	581,500	8,711,550	17,705,466
D7	Instrumentation	73989	1,125,386	147,979	794,058	0	2,048,101	4,115,525
D8	Piping	212334	3,516,272	1,061,703	4,100,192	0	0	8,678,167
D9	Plant Mobile Equipment	0	0	0	0	0	2,892,000	2,892,000
IA	Temporary Building & Facilities	0	0	0	0	2,500,000	0	2,500,000
IB	Temporary Construction Utility Services	0	0	0	0	884,000	0	884,000
IC	Winter Work and Lost Productivity	0	0	0	0	387,000	0	387,000
ID	Construction Site Support & Operations	0	0	0	0	2,972,000	0	2,972,000
IE	Construction Camp and Catering	0	0	0	0	6,225,560	0	6,225,560
IF	Power During Construction	0	0	0	0	8,160,000	0	8,160,000
IG	Spare Parts	0	0	0	0	2,410,389	0	2,410,389
IH	Initial Fills	0	0	0	0	3,816,836	0	3,816,836
IJ	Freight and Insurance	0	0	0	0	5,431,361	0	5,431,361
IK	Vendor Representative	0	0	0	0	964,000	0	964,000
IL	Owner Costs	0	0	0	0	12,420,000	0	12,420,000
IN	EPCM	0	0	0	0	38,747,944	0	38,747,944
IP	Commisioning and Startup	0	0	0	0	4,714,316	0	4,714,316
IQ	Contingency	0	0	0	0	37,717,106	0	37,717,106
REPORT TOTALS		1863378	28,404,169	10,278,741	28,078,737	213,763,017	52,261,335	332,786,000

20.5.2 Basis of Estimate

The capital cost estimate includes for the following.

- Direct costs of new equipment for the processing facilities
- Construction materials and installation labour
- Temporary buildings and services
- Construction support services
- Project infrastructure
- Spare parts
- Initial fills (inventory)
- Freight
- Vendor Supervision
- Owner's cost
- Engineering, Procurement and Construction Management
- Commissioning and start up
- Contingency.

20.5.3 Direct Cost Elements

The direct costs are all the costs associated with permanent facilities. This includes equipment and material costs, as well as construction and installation costs.

Process Equipment

Equipment pricing is based on the equipment list, specification and process flow diagrams. Budgetary prices were obtained from Vendors of major equipment and in-house data was used from similar project for items not quoted for. Estimated cost based on local data for platework was used to estimate the remaining equipment; tanks, bins and chutes. Costs for installation of equipment are based on unit man-hour requirements.

Other direct costs were priced based on actual takeoffs:

- Earthwork / site work
- Concrete
- Structural steel
- Buildings and architectural
- Electrical
- Instrumentation and controls
- Piping.

It was assumed that rock required for site preparation and the tailings will be provided at no cost during the preproduction stage, only cost for placement has therefore been allowed for in the estimate.

20.5.4 Indirect Cost Estimate

The indirect costs cover all the costs associated with temporary construction facilities and services, construction support, freight, Vendor representatives, spare parts, initial fills and inventory, Owner's costs, EPCM, commissioning and start-up assistance.

20.5.5 Construction Facilities

The costs for construction facilities include all temporary facilities, services and operation, site office operations, security buildings and services, construction warehousing and material management, construction power and utilities, site transportation, medical facilities and services, garbage collection and disposal, and surveying.

20.5.6 Spare Parts

The cost for spare parts is factored based on equipment costs where Vendors did not provide cost for spares needed for the first year of operations.

20.5.7 Initial Fills (Inventory)

The estimated cost for initial fills of reagents is based on 3 months of operating requirements. Budget quotations were obtained for reagent pricing.

20.5.8 Freight

The freight costs were either provided by Vendor or estimated based on weights and typically include for containerised and break-bulk shipping, and each are respectively divided into ocean freight and inland freight. For imported equipment, the cost of freight and export packing, ex-works to a local port, is included with the cost of the equipment.

Freight insurance is included in the Owner's cost.

20.5.9 Vendor Representatives

The requirement for Vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment as deemed necessary for equipment guarantees or warranties has been included in the estimate. Typically, the cost for this item is inclusive of salary and travel.

20.5.10 Taxes and Duties

Taxes and duties have been excluded.

20.5.11 Engineering, Procurement and Construction Management

EPCM has been calculated assuming North American rates and includes estimates for travel allowance and other incidentals. Third party engineering has also been included in the estimate.

20.5.12 Contingency

A contingency allowance of 11.4% of process plant and infrastructure direct and indirect costs has been included in the estimate. P&E, based on their experience, has allowed a 15% contingency on the mining portion. The overall average contingency therefore is 13.3% of total direct and indirect costs, exclusive of Owner's costs.

The 11.4% contingency on plant and infrastructure costs reflects the potential growth in capital costs within the same scope of work. It includes variations in quantities, differences between estimated and actual equipment and material prices, labour cost and site-specific conditions. It also accounts for variation resulting from uncertainties that are clarified during basic and detailed engineering, when designs and specifications are finalised. A capital cost contingency analysis has been performed for the Project using the @Risk program. A Monte Carlo simulation examined the impact of varying within a range of assigned confidence the cost of the total labour, total construction equipment, commodities and each component of the indirect cost. Bell-shaped distributions were generally assumed with a standard deviation of one.

Commodities (i.e., earthworks, concrete, structural steel, architectural, process equipment, electrical, instrumentation and piping) were broken out by area by source code. A distinction between material and plant equipment was also made. A different confidence range was then applied to the each source code for each commodity at the area summary level.

Based upon 10,000 iterations for different simulations, the statistically most probable contingency was 11.4% at the 90% confidence level.

20.5.13 Owner's Cost

Included in the estimate as provided by the Owner.

20.5.14 Capital Cost Qualifications and Exclusions

Qualifications

- All construction work will be executed by local contractors.

Exclusions

- Sunk costs
- Sustaining capital
- Deferred capital
- Working capital
- Financing and interest during construction
- Additional exploration drilling
- Escalation
- Corporate withholding taxes
- Legal costs
- Process royalty fees
- Metallurgical testing costs
- Condemnation testing.

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20.6 Operating Cost Estimate

20.6.1 Summary

Operating costs for the Project reflect fourth quarter 2008 dollars. The exchange rates used to convert other currencies to US dollars are shown in Table 20-9.

Table 20-9
Exchange Rates Used in the Report

1 month average	01Dec08 to 31Dec08	31 days
3 month average	01Oct08 to 31Dec08	92 days
6 month average	01Jul08 to 31Dec08	184 days

		1 month	3 months	6 months	USE
CAD: USD	High	0.84600	0.95930	1.00240	
	Low	0.76870	0.76800	0.76800	**0.81**
	Average	**0.80994**	0.82882	0.89549	
EUR: USD	High	1.47190	1.47190	1.60380	
	Low	1.25490	1.23290	1.23290	**1.35**
	Average	**1.34718**	1.31883	1.41277	
AUD: USD	High	0.71370	0.80960	0.98490	
	Low	0.62890	0.60050	0.60050	**0.67**
	Average	**0.66908**	0.67306	0.78240	
ZAR: USD	High	0.10700	0.12150	0.13910	
	Low	0.09301	0.08410	0.08410	**0.101**
	Average	**0.10054**	0.10139	0.11529	
CNY: USD	High	0.14616	0.14650	0.14660	
	Low	0.14480	0.14390	0.14390	**0.146**
	Average	**0.14616**	0.14646	0.14643	
MNT: USD	High	0.00085	0.00087	0.00087	
	Low	0.00079	0.00079	0.00079	**0.00081**
	Average	**0.00081**	0.00085	0.00086	

The DFS operating cost estimates are prepared by area and component, and consider the mining plan and processing schedule.

Life-of-mine operating costs are presented in Table 20-10. Note that the Years 2008 to 2010 are considered as preproduction and their cost is included in the mine capital.

Table 20-10
Life-of-Mine Operating Costs

Year		Tonne Milled (x '000)	Mining (USD)	Plant (USD)	G&A (USD)	Total (USD)	Cost/Tonne Milled (USD)
2009							
2010							
2011							
2012	1	854	32,976,454	20,443,546	7,040,000	60,460,000	70.83
2013	2	1,225	44,664,514	31,246,486	7,040,000	82,951,000	67.72
2014	3	1,225	43,142,514	31,246,486	7,040,000	81,429,000	66.47
2015	4	1,225	44,169,514	31,246,486	7,040,000	82,456,000	67.31
2016	5	1,225	47,345,714	30,880,286	6,300,000	84,526,000	69.00
2017	6	1,228	46,680,714	30,880,286	6,300,000	83,861,000	68.29
2018	7	1,225	44,334,714	30,880,286	6,160,000	81,375,000	66.43
2019	8	1,225	50,113,714	30,880,286	6,160,000	87,154,000	71.15
2020	9	1,225	52,096,714	30,880,286	6,160,000	89,137,000	72.76
2021	10	1,225	31,863,386	22,334,614	4,977,000	59,175,000	48.31
2022	11	1,225	28,903,738	20,930,262	4,977,000	54,811,000	44.74
2023	12	1,225	29,184,738	20,930,262	4,977,000	55,092,000	44.97
2024	13	1,225	27,133,738	20,930,262	4,977,000	53,041,000	43.30
2025	14	1,225	29,708,738	20,930,262	4,977,000	55,616,000	45.40
2026	15	1,262	20,756,000	14,626,000	4,977,000	40,359,000	31.98
TOTAL		**18,044**	**573,074,904**	**389,266,096**	**89,102,000**	**1,051,443,000**	**58.26**
Cost/lb U_3O_8		45,279,000	12.71	8.60	1.97	23.22	
Cost/Tonne Milled			31.76	21.56	4.94	58.26	

Note that the above amounts do not include VAT or the interest costs associated with the leasing of mining equipment. The interest on the leased equipment is shown in the Project Cash Flow, Table 20-34.

20.6.2 Basis of Estimate

(a) Expatriate Labour Costs

It has been recognised that some of the more technical skills will be difficult to source in Mongolia. Initially, these needs will be satisfied by the employment of expatriate specialists. These operator-trainers will be employed for as long as it takes to train suitable Mongolian replacements. It is estimated that it will take up to 5 years to accomplish this. At the end of the 5-yr period, the number of expatriots will be greatly reduced. A breakdown of the expatriate replacement schedule is given in Table 20-11.

The costs of these expatriate experts are given in Table 20-12.

Table 20-11
Expatriate Replacement Schedule

Position	Years 1 - 4	Years 5 - 6	Years 6 - 8	Years 9	Years 10 - 16
General Manager	1	1	1	1	1
Plant Superintendent	1				
Plant Maintenance Superintendent	1				
Administration Superintendent	1				
Mine Superintendent	1	1		1	1
Chief Geologist	1	1	1	1	1
Chief Engineer	1	1	1	1	1
Mine Engineer	1			2	2
Mine General Foreman	2	2			
Supervisor Load & Haul					2.5
Maintenance General Foreman	1	1	1	1	1
Electrical General Foreman	1	1	1	1	
Lead Mechanic				1	4
U/G Maintenance Trainer	1	1	1	1	
Mine Trainers	2	2	2	1	0.5
Lead Miners/Dev. Mentors	14	12	10	10	
Mine Geologists				2	2
TOTAL	**29**	**23**	**18**	**23**	**16**

Table 20-12
Cost of Expatriate Employees

Position	Payroll Burden (%)	Total Cost (USD/a)	Salary Cost (USD/a)
General Manager	30	259,200	199,385
Mill Superintendent	30	216,000	166,154
Administration Superintendent	30	225,391	173,378
Mine Superintendent	30	210,000	161,538
Chief Geologist	30	168,000	129,231
Chief Engineer	30	168,000	129,231
Mine Engineer	30	140,000	107,692
Mine General Foreman	30	140,000	107,692
Maintenance General Foreman	30	140,000	107,692
Electrical General Foreman	30	140,000	107,692
U/G Maintenance Trainer	30	140,000	107,692
Mine Trainers	30	140,000	107,692
Lead Miners/Dev. Mentors	30	110,000	84,615

The expatriate workers will be sourced in established mining camps. The personnel employed will be hired before start up so that they can train the new workers. An amount of USD 400,000 has been included in the owners cost for training of the personnel.

It is unlikely that suitable people skilled in mining will be readily available in Mongolia. Up to 29 expatriate miners and mine supervision will be used at Dornod in the initial years.

Salary levels used in the operating cost estimate for these expatriates come from Aker Solutions' files and reflect those used in a recent mine in Southeast Asia. It has been assumed that the more senior employees are sourced in North America, while the less experienced workers will come from areas such as China or the Philippines.

(b) Local Labour Costs

All personnel other than the expatriates will be recruited in Mongolia. A total of 933 people will be employed during the peak employment year. A breakdown of the workforce is presented in Table 20-13.

Table 20-13
Total Mine Manpower - Average Year

	Staff	Hourly	Total
Mine	46	665	711
Mill	22	127	149
G & A	27	36	63
Camp Personnel	8	2	10
TOTAL	**103**	**830**	**933**

Local labour rates were as a result of a labour survey conducted in the Choi Balsaan area. The rates used are shown in Table 20-14, and were confirmed by Khan. They reflect current salaries being paid by other mining companies in Mongolia. A 30% burden over the annual salary has been used in the estimate.

Table 20-14
Cost of Local Labour (USD)

	Monthly Salary	Yearly Salary	30% Burden	Annual Cost
Mine Manager	3,250	42,250	11,700	53,950
Superintendents	1,500	19,500	5,400	24,900
Foremen	900	11,700	3,240	14,940
Technical Specialists	1,000	13,000	3,600	16,600
Planner	400	5,200	1,440	6,640
Clerk	400	5,200	1,440	6,640
Loader Operators	737	9,577	2,652	12,229
Plant Operators	650	8,450	2,340	10,790
Plant Helpers	520	6,760	1,872	8,632
Training Operators	585	7,605	2,106	9,711
Mechanis	780	10,140	2,808	12,948
Instrument Techs	858	11,154	3,089	14,243
Electricians	780	10,140	2,808	12,948
Mechanical Helpers	607	7,887	2,184	10,071

During the operation of the mine, the total mining labour complement varies as the mining moves from the No. 7 Deposit to the No. 2 Deposit. This distribution is indicated in Table 12-15.

Table 20-15
Life of Mine, Underground and Open-Pit Manpower

Year	Management	Engineering	Geology	Administration Support	Safety and Training	Mine Supervision	Development Crews	Mining	Mine Services	Maintenance	Total
1	6	7	8	10	3	12	246	213	112	94	711
2	6	7	8	10	3	12	246	213	112	94	711
3	6	7	8	10	3	12	246	213	112	94	711
4	6	7	8	10	3	12	246	213	112	94	711
5	6	7	8	10	3	12	164	213	112	94	629
6	6	7	8	10	3	12	164	213	112	94	629
7	6	7	8	10	3	12	82	213	112	94	547
8	6	7	8	10	3	12	82	213	112	94	547
9	8	13	8	13	4	14	82	239	112	108	601
10	2	11	7	6	2	8		106		44	186
11	2	11	7	6	2	8		106		44	186
12	2	11	7	6	2	8		106		44	186
13	2	11	7	6	2	8		106		44	186
14	2	11	7	6	2	8		106		44	186
15	2	11	7	6	2	8		106		44	186
16	2	11	7	6	2	8		106		44	186

The ratio of expatriates to total labour complement in the average years of the mine life is 2.5%. This is less than the number allowed for enterprises of this type (Gold Mining, Registered Capital 500 million tugriks or more and more than 500 employees).

20.6.3 Mine Operating Costs

Mine operating costs have been calculated by P&E. Details are given in the DFS.

20.6.4 Mine Operating Cost Summary

Table 20-16a and 20-16b summarise the life-of-mine mining operating costs. Note that Years -3 to -1 are preproduction years and that this cost has been accounted for in the mining capital.

Table 20-16a
Life-of-Mine Underground Mining Costs

Year	Tonnes Mined	Ore Silling	Longhole Mining Downholes	Longhole Mining Uppers	Pillar Recovery	Equipment Leasing	Mine services	**TOTAL**	**Cost Per Tonne Mined**
	(Tonnes)	(USD)	(USD)	(USD)	(USD)	(USD)	(USD)	(USD)	**(USD/t)**
1	854,000	8,715,000	6,792,000	3,072,000	320,000	0	13,997,000	32,896,000	**38.52**
2	1,228,000	6,991,000	15,668,000	4,092,000	0	3,597,000	16,205,000	46,553,000	**37.91**
3	1,226,000	6,549,000	15,309,000	2,717,000	2,304,000	2,998,000	16,185,000	46,062,000	**37.57**
4	1,226,000	4,561,000	11,954,000	114,000	11,277,000	2,303,000	16,185,000	46,394,000	**37.84**
5	1,226,000	5,114,000	14,629,000	768,000	5,598,000	1,497,000	16,185,000	43,791,000	**35.72**
6	1,229,000	5,681,000	14,855,000	3,576,000	4,018,000	562,000	16,482,000	45,174,000	**36.76**
7	1,225,000	622,000	12,245,000	759,000	12,177,000	0	16,462,000	42,265,000	**34.50**
8	1,225,000	0	13,643,000	4,932,000	5,892,000	25,629,000	16,739,000	66,835,000	**54.56**
9	1,195,000	0	4,159,000	427,000	20,503,000	0	16,443,000	41,532,000	**34.75**
TOTAL	**10,634,000**	**38,233,000**	**109,254,000**	**20,457,000**	**62,089,000**	**36,586,000**	**144,883,000**	**411,502,000**	**38.70**
USD/t Mined		**3.60**	**10.27**	**1.92**	**5.84**	**3.44**	**13.62**	**38.70**	

Table 20-16b
Life-of-Mine Open-Pit Mining Costs

Year	Tonnes Ore Mined (Tonnes)	Tonnes Waste Mined (Tonnes)	Total Mined (Tonnes)	10m Ore Benching (USD)	10m Waste Benching (USD)	5m Ore Benching (USD)	5m Waste Benching (USD)	Total Ore Cost (USD)	Total Waste Cost (USD)	**Total Cost (USD)**	Per tonne Ore (USD/t)	Per Tonne Mined (USD/t)
8		5,000,000	5,000,000		8,160,000		1,341,000	0	9,501,000	**9,501,000**		1.90
9	422,912	6,237,088	6,660,000	652,900	9,167,900	90,900	3,875,000	743,800	13,042,900	**13,786,700**	32.60	2.07
10	1,225,000	18,025,000	19,250,000	2,004,300	28,162,200	263,400	3,875,000	2,267,700	32,037,200	**34,304,900**	28.00	1.78
11	1,225,000	18,025,000	19,250,000	1,815,700	25,386,600	263,400	3,875,000	2,079,100	29,261,600	**31,340,700**	25.58	1.63
12	1,225,000	18,025,000	19,250,000	1,834,600	25,664,200	263,400	3,875,000	2,098,000	29,539,200	**31,637,200**	25.83	1.64
13	1,225,000	18,025,000	19,250,000	1,703,700	23,739,100	263,400	3,875,000	1,967,100	27,614,100	**29,581,200**	24.15	1.54
14	1,225,000	18,025,000	19,250,000	1,883,200	26,379,200	263,400	3,875,000	2,146,600	30,254,200	**32,400,800**	26.45	1.68
15	858,588	12,589,905	13,448,493	1,319,900	18,425,100	184,600	2,707,000	1,504,500	21,132,100	**22,636,600**	26.36	1.68
TOTAL	**7,406,500**	**113,951,993**	**121,358,493**	**11,214,300**	**165,084,300**	**1,592,500**	**27,298,000**	**12,806,800**	**192,382,300**	**205,189,100**	**27.70**	**1.69**
USD/T Mined				1.68	1.60	2.15	2.15	1.73	1.69	1.69		

Interest costs on leasing of mining equipment has been included in Tables 20-16a and 20-16b.

20.6.5 Mill Operating Costs

Life-of-mine mill operating costs are shown in Table 20-17.

Table 20-17
Life- of-Mine Milling Costs

Year		Tonne Milled (x '000)	Labour (USD)	Consumables Consumables	Power (USD)	Maintenance (USD)	Mill Other (USD)	Total (USD)	Cost/Tonne Milled (USD)
2009									
2010									
2011									
2012	1	854	2,179,637	13,115,601	3,743,996	1,029,311	375,000	20,443,545	23.95
2013	2	1,225	2,179,637	21,859,336	4,991,995	1,715,519	500,000	31,246,487	25.51
2014	3	1,225	2,179,637	21,859,336	4,991,995	1,715,519	500,000	31,246,487	25.51
2015	4	1,225	2,179,637	21,859,336	4,991,995	1,715,519	500,000	31,246,487	25.51
2016	5	1,225	1,813,437	21,859,336	4,991,995	1,715,519	500,000	30,880,287	25.21
2017	6	1,225	1,813,437	21,859,336	4,991,995	1,715,519	500,000	30,880,287	25.21
2018	7	1,225	1,813,437	21,859,336	4,991,995	1,715,519	500,000	30,880,287	25.21
2019	8	1,225	1,813,437	21,859,336	4,991,995	1,715,519	500,000	30,880,287	25.21
2020	9	1,225	1,813,437	21,859,336	4,991,995	1,715,519	500,000	30,880,287	25.21
2021	10	1,225	1,813,437	13,313,663	4,991,995	1,715,519	500,000	22,334,614	18.23
2022	11	1,225	1,813,437	11,909,311	4,991,995	1,715,519	500,000	20,930,262	17.09
2023	12	1,225	1,813,437	11,909,311	4,991,995	1,715,519	500,000	20,930,262	17.09
2024	13	1,225	1,813,437	11,909,311	4,991,995	1,715,519	500,000	20,930,262	17.09
2025	14	1,225	1,813,437	11,909,311	4,991,995	1,715,519	500,000	20,930,262	17.09
2026	15	1,262	1,868,459	12,270,654	5,143,458	1,767,570	515,171	21,565,312	17.09
								-	
TOTAL		**18,041**	**28,721,377**	**261,211,850**	**73,783,389**	**25,098,628**	**7,390,171**	**396,205,415**	**21.96**
Cost/lb U3O8		45,105,211	0.64	5.79	1.64	0.56	0.16	8.78	
Cost/Tonne Milled (USD/Tonne)			1.59	14.48	4.09	1.39	0.41	21.96	

The total operating cost (USD 396,205,415) included the first fills and spare parts costs.

Mill Operating Supplies

Grinding media and crusher-liner consumption rates were either obtained from equipment suppliers or from similar operations elsewhere.

For No. 2, Deposit, industry standard acid and oxidant consumption numbers were used. The more difficult No. 7 Deposit will consume more acid, due to its higher carbonate content. For this material, acid and oxidant consumption rates, as experienced in the recent SGS testwork, were used.

Magnesia and hydrogen peroxide consumption in the uranium precipitate circuit were assumed to be as obtained in commercial plants elsewhere. These numbers were compared with those obtained in the testwork at SGS and were found to be similar.

All other reagent usage was assumed to be at average industry standard levels.

Reagent prices were the result of actual supplier quotes.

Most of the standard industrial chemicals were sourced in China. An allowance for rail transport to the site at USD 0.65/t km was used to calculate a delivered cost.

As some of the speciality chemicals are currently only available in North America, these were sourced from North American suppliers. An allowance for shipping and rail transport was added to these costs. It may be possible to replace these brand name chemicals with equivalent ones manufactured in Southeast Asia. In this case, a saving would result.

Sulphur is used to produce steam and sulphuric acid. This is the most expensive of the reagents used. Aker Solutions was able to get pricing for this commodity FOB the Russian – Mongolian border from a nickel smelter in the former Soviet Union. Rail costs were added to this number to bring the sulphur from the border to the mine site. As the sulphur price during the fourth quarter of 2008 was very volatile, the current Russian price was adjusted to reflect a longer term historical delivered prices.

Note that an import duty of 5% is applied to all imports. This, and a Value Added Tax (VAT) of 10% apply to all of these purchases. These two taxes are recoverable in the form of rebates from the Mongolian government. They have not been included in the operating cost estimate. The timing of the tax rebates will be accounted for in the cash flow model.

These consumptions and costs used are shown in Tables 20-18 and 20-19, respectively. Grinding steel and liner costs reflect current North American prices.

Table 20-18
Estimated Annual Steel Consumptions

	Tonne Consumed (a)	Consumed/t Milled (kg/t)	Price/t Including Shipment (USD/t)	Cost/Year (USD)	Cost/t Milled (USD)
Grinding Balls Ball & SAG Mills	743.6	0.607	1,677	1,247,288	1.02
Mill Liners SAG	399.71	0.326	1,269	507,098	0.41
Mill Liners Ball Mills	167.15	0.136	1,269	212,051	0.17
Crusher Liners	4.02	0.003	1,269	5,102	0.00
				1,971,539	**1.61**

Table 20-19
Estimated Annual Reagent Consumptions

	Consumed	Consumed/t Milled	Price/unit Including Shipment (USD/unit)	Cost/Year (USD)	Cost/t Milled (USD)
Barium Chloride	42	0.03	455.52	19,132	0.02
Ferric Sulphate(45%)	517	0.42	387.98	200,586	0.16
Floculent (t)	128	0.10	4974.59	636,747	0.52
Diesel (t)	714	0.58	909.25	649,201	0.53
Hydrogen Peroxide (t)	2041	1.67	579.66	1,183,085	0.97
Lime (t)	28511	23.27	113.80	3,244,498	2.65
Magnesia (t)	345	0.28	286.37	98,797	0.08
Product Drums (each)	1720	1.40	31.62	54,379	0.04
RIP Resin (m^3)	99	0.08	5516.61	546,144	0.45
Sodium Hydroxide (t)	698	0.57	432.17	301,656	0.25
Sulphur (t)	101205	82.62	125.41	12,692,398	10.36
TOTAL				**19,626,623**	**16.02**

An amount of USD 264,175/yr has been allowed (including transport) to cover miscellaneous consumables.

20.6.6 Mill Power

Power consumption was estimated in a load study presented as part of the equipment list. A factor of 0.85 was used to estimate the operating load from the connected load. A utilisation of 85% and 93% was estimated for the crushing and remaining plant sections, respectively.

The power is assumed to be supplied via a new power line to the Mongolian grid. This line will be built by the Mongolian Government. A delivered cost of power of USD 0.075/kW.h has been used to estimate the operating cost.

20.6.7 Mill Labour

One hundred and forty-nine people will be employed in the mill. This will comprise 22 staff supervisors and 127 hourly employees. Of the staff members, initially two are expatriates. These people will be replaced by local employees by Year 5.

Salaries, as indicated above, were used in the estimate.

20.6.8 Maintenance Costs

These costs were estimated as a percentage of the purchased machinery costs. Separate requirements were estimated for mechanical, electrical and instrumentation equipment as shown in Table 20-20.

Table 20-20
Mill Maintenance Cost

	Equipment Cost (USD)	% Applied	Annual (USD)
Purchase Price of Mechanical Equipment	38,609,684	3.5	1,351,339
Purchase Price of Electrical Equipment	13,667,400	2	273,348
Purchase Price of Instrumentation Equipment	2,842,159	2	56,843
Puchase Price of Mobile Equipment	2,892,000	4	115,680
TOTAL	**66,712,929**		**1,797,210**
Import Duty (5%)			**89,861**
Mill Maintenance (USD)			**1,887,071**
USD/t ore milled			**1.54**

20.6.9 Concentrator "Other" Costs

These costs were entered in the form of allowances. This was based on experience with other similar operations in Asia. An amount of USD 550,000 was allowed for in each year of the mine life.

20.6.10 General and Administration Other Costs

These costs were entered in the form of allowances. This was based on experience with other similar operations in Asia.

Insurance costs were as a result of an estimate provided by Khan's insurance company.

Camp costs were calculated at a rate of USD 20/man/d in camp.

Road maintenance considered the grading and snow removal on 140 km of road between the mine and Choi Balsam.

The G&A other costs are shown in Table 20-21.

Table 20-21 **G&A Annual Costs** (USD)			
	USD/a	**Duty**	**Total**
Training	30,000		30,000
Safety Equipment	20,000	$1,000	21,000
Medical Supplies	15,000	$375	15,375
Security Supplies	10,000	$500	10,500
Radiation Protection and Monitoring	20,000	$1,000	21,000
Environmental	35,000	$875	35,875
Tailings Pond Work	30,000		30,000
Legal and Accounting (auditing)	40,000		40,000
Consultants	80,000		80,000
Insurance Property	400,000		400,000
Insurance Liability	250,000		250,000
Shipping Agent	20,000		20,000
Communications	30,000		30,000
Office Supplies	20,000	$1,000	21,000
Computer Supplies	20,000	$1,000	21,000
Petrol	20,000	$1,000	21,000
Camp Costs	3,615,964	$36,160	3,652,123
Road Maintenance	50,000		50,000
Travel Locals	153,300		153,300
Travel Expats	180,000		180,000
Business Meals and Entertaining	30,000		30,000
Courier	20,000		20,000
Bank Fees	3,000		3,000
Other	80,000		80,000
Land Lease			
Standby Plant Heating	22,166	$1,108	23,275
G & A Power	563,203		563,203
TOTAL Without VAT	**5,757,633**	**44,018**	**5,801,651**
Cost/Tonne Milled	**$ 4.70**	**$ 0.04**	**$ 4.74**

20.6.11 General and Administration Labour

A total of 63 people will be employed in this section, 27 staff members and 36 hourly employees. They will see to the management, administration, personnel, accounting and purchasing needs of the operation. The health and safety, environmental, radiation protection and security requirements will also be located in this section. A breakdown of the G&A labour costs are indicated in Table 20-22.

Table 20-22
G&A Site Labour

Function	Quantity	Annual salary	Extended cost
Mill Manager (expat)	1	$259,200	$259,200
Admin Superintendant (expat)	1	$225,391	$225,391
G & A Staff			
Human Resources/ H & S Superintendent	1	$24,900	$24,900
Environmental/ Radiation Superintendent	1	$24,900	$24,900
Purchasing specialist	1	$24,900	$24,900
Chief Accountant	1	$24,900	$24,900
Medical nurse First Aid room	1	$16,600	$16,600
Office Clerks	3	$6,640	$19,920
Translators	3	$14,940	$44,820
Head of Security (Contract - 15% overhead))	1	$17,181	$17,181
Security Personnel (Contract - 15% overhead)	12	$9,927	$119,122
Mongolian Doctor (Contract)	1	$28,635	$28,635
G & A Hourly			
H & S Trainers	2	$16,600	$33,200
Environ Technicians / Analysts	4	$16,600	$66,400
Warehouse and Tool Crib Staff	8	$6,640	$53,120
Radiation Technichians	2	$16,600	$33,200
Surface Day Gang Janitors	3	$6,640	$19,920
Rail Crew & Road Maintenance	6	$6,640	$39,840
Workshop Foreman	1	$14,940	$14,940
Workshop Artisans	4	$12,948	$51,792
Workshop Helpers	2	$10,071	$20,141
Maintenance planner	0	$9,100	$0
Pool Drivers	4	$10,071	$40,283
Sub-total	**63**		**$1,203,305**
Burden @ 10%% (30% in Salaries)			**$120,330**
Annual Total			**$1,323,635**
Unit cost ($ US per tonne of milled ore)			**$1.05**

Life-of-mine G&A costs are shown in Table 20-23.

Table 20-23
Life-of-Mine G & A Costs

Year		**Tonne Milled** (x '000)	**Labour** (USD)	**Power** (USD)	**Other** (USD)	**Total** (USD)	**Cost/Tonn Milled** (USD)
2009							
2010							
2011							
2012	1	854	1,323,635	563,203	5,152,755	7,039,593	8.25
2013	2	1,225	1,323,635	563,203	5,152,755	7,039,593	5.75
2014	3	1,225	1,323,635	563,203	5,152,755	7,039,593	5.75
2015	4	1,225	1,323,635	563,203	5,152,755	7,039,593	5.75
2016	5	1,225	863,944	563,203	4,873,035	6,300,182	5.14
2017	6	1,225	863,944	563,203	4,873,035	6,300,182	5.14
2018	7	1,225	863,944	563,203	4,733,175	6,160,322	5.03
2019	8	1,225	863,944	563,203	4,733,175	6,160,322	5.03
2020	9	1,225	863,944	563,203	4,733,175	6,160,322	5.03
2021	10	1,225	863,944	563,203	3,550,035	4,977,182	4.06
2022	11	1,225	863,944	563,203	3,550,035	4,977,182	4.06
2023	12	1,225	863,944	563,203	3,550,035	4,977,182	4.06
2024	13	1,225	863,944	563,203	3,550,035	4,977,182	4.06
2025	14	1,225	863,944	563,203	3,550,035	4,977,182	4.06
2026	15	1,262	863,944	563,203	3,550,035	4,977,182	3.94
TOTAL		**18,041**	**14,797,927**	**8,448,049**	**65,856,818**	**89,102,794**	**4.94**
Cost/lb U3O8		45,279,000	0.33	0.19	1.45	1.97	
Cost/Tonne Milled			0.82	0.47	3.65	4.94	

20.7 Overview of the Uranium Industry

20.7.1 Mining and Milling

Uranium ore is recovered by excavation or by in-situ leaching techniques. Excavation may be open-pit or underground mining. In general, open-pit mining is used where deposits are close to the surface and underground mining is used for deep deposits, typically greater than 120-m deep. Underground mines have relatively small surface disturbance and the quantity of material that must be removed to access the ore is considerably less than in the case of an open-pit mine. In-situ leaching involves pumping a liquid into the ground to dissolve the uranium and then pumping that liquid back to the surface. (Source: Khan Resources Inc. (KRI), AIF dated December 12, 2008.)

After the uranium ore has been mined, it is milled. Milling, which is generally carried out close to a uranium mine, extracts the uranium from the ore. At the mill, the ore is crushed and ground to a fine slurry. Sulphuric acid or a strong alkaline solution is used to dissolve the uranium to allow the separation of uranium from the waste rock. It is then recovered from solution and precipitated as uranium oxide (U_3O_8) concentrate. This is sometimes referred to as “yellowcake” and generally contains more than 80% uranium. The original ore may contain as little as 0.1% uranium. After drying and usually heating, it is packed in 200-L drums as a concentrate. The remainder of the ore, containing most of the radioactivity and nearly all the rock material, becomes tailings, which are placed in engineered facilities near the mine (often in mined-out pits).

20.7.2 Conversion and Enrichment

Uranium found in nature consists largely of two isotopes, U-235 and U-238. The production of energy in the form of heat in nuclear reactors is from the “fission” or splitting of the U-235 atoms. Natural uranium contains 0.7% of the U-235 isotope. The remaining 99.3% is mostly the U-238 isotope which does not contribute directly to the fission process. Most nuclear reactors require uranium enriched to 3% to 5% U-235 as their fuel. The Canadian-designed Candu and the British Magnox reactors use natural uranium as their fuel. (Source: KRI)

Uranium enrichment requires the material to be in gaseous form. The product of a uranium mine is not directly usable and the uranium oxide must be converted into uranium hexafluoride (UF6) which is a gas at relatively low temperature. There are conversion plants in Europe, Russia and North America. At a conversion facility, the U_3O_8 is first refined to uranium dioxide, which can be used as the fuel for those types of reactors that do not require enriched uranium. Most is then converted into uranium hexafluoride, ready for the enrichment plant.

Uranium is enriched into U-235 by gaseous diffusion or centrifuge technology. Both of these processes work on the principle of separating the lighter U-235 from the heavier U-238, when in the form of uranium hexafluoride gas. At present, the gaseous diffusion process accounts for about 40% of world enrichment capacity. However, because they are old and energy-inefficient, most gaseous diffusion plants are being phased out over the next 5 years and the focus is on energy-efficient centrifuge enrichment technology which will replace them.

20.7.3 Price

There is no formal exchange for uranium as there is for other commodities such as gold or oil. Uranium price indicators are developed by a small number of private business organisations that independently monitor uranium market activities, including offers, bids and transactions. Such price indicators are owned by and proprietary to the business that has developed them.

The uranium spot price (USD/lb U_3O_8) steadily increased from USD 7/lb in December 2000 to a peak of USD 135/lb in June 2007. Since that time, the uranium spot price has ranged from USD 45 to USD 123 and was USD 54 at December 4, 2008.

A survey of forecast contract uranium price (USD/lb U_3O_8) from various financial institutions was obtained for the period 2008 to 2014. The average price ranged from a low of USD 66.54/lb in 2009 to a high of USD 82.50/lb in 2011. The uranium price of USD 65/lb was used in the Definitive Feasibility Study.

20.7.4 Demand

About 435 reactors with combined capacity of 370 GW net require 78 500 t of uranium oxide concentrate containing 66 500 t of uranium from mines (or the equivalent from stockpiles or secondary sources) each year. Capacity is growing slowly, and at the same time, the reactors are being run more productively, with higher capacity factors and reactor power levels. However, these factors increasingly fuel demand are offset by a trend for increased efficiencies, so demand is dampened – over the 20 years from 1970, there was a 25% reduction in uranium demand per kilowatt-hour output in Europe, due to such improvements, which continue. Each gigawatt net of increased capacity will require about 195 t U/a of extra mine production routinely, and three times this for the first fuel load. Fuel burn up is measured in megawatt days per tonne U, and many utilities are increasing the initial enrichment of their fuel (e.g., from 3.3% to more than 4.0% U-235) and then burning it longer or harder to leave only 0% U-235 in it.

The demand on uranium fuel is much more predictable than with most if not all other mineral commodities, because of the cost structure of nuclear power generation, with high capital and low fuel costs. Once reactors are built, it is very cost effective to keep them running at high capacity and for utilities to make any adjustments to load trends by cutting back on fossil fuel use. Demand forecasts for uranium thus depend largely on installed and operable capacity, regardless of economic fluctuations. For instance, when South Korea's overall energy use decreased in 1997, nuclear energy output actually rose, to replace import fossil fuels.

Looking 10 years ahead, the market is expected to grow slightly. Demand thereafter will depend on new plants being built and the rate at which older plants are retired. Licensing of plant lifetime extensions and the economic attractiveness of continued operation of older reactors are critical factors in the medium-term uranium market. However, with electricity demand by 2030 expected (by the OECD's International Energy Agency) to double from the demand in 2004, there is plenty of scope for growth in nuclear capacity in a greenhouse-conscious world.

20.7.5 Supply

Mines in 2005 supplied some 49 000 t of U_3O_8 containing 41 600 t of U, about 64% of existing utilities' annual requirements. The balance was supplied from secondary sources or stockpiled uranium held by utilities, which stockpiles are now largely depleted. As well as existing and new mines, nuclear fuel supply may be from secondary sources, including recycled uranium and plutonium from spent fuel, as mixed oxide fuel, re-enriched depleted uranium tails, ex-military weapons-grade uranium, civil stockpiles, and ex-military weapons-grade plutonium.

20.7.6 Uranium Producers

The uranium industry is concentrated with a small number of companies controlling a majority of the production. In 2007, seven companies marketed 85% of the world's uranium mine production (see Table 20-24). Also, in 2007, the top seven uranium producing countries accounted for 89% of the world's total production, led by Canada at 23% (see Table 20-25).

Table 20-24
Major Uranium Producers – Companies

Company	**Production** (t U)	**World Share** (%)
Cameco	7 770	19
Rio Tinto	7 172	17
Areva	6 046	15
KazAtomProm	4 795	12
ARMA	3 413	8
BHP Billiton	3 388	8
Navoi	2 320	6
Uranium One	784	2
GA / Heathgate	673	2
Others	4 918	11
TOTAL	**41 279**	**100**

(Source: KRI)

Table 20-25
Major Uranium Producers – Countries

Company	Production (t U)	World Share (%)
Canada	9 476	23
Australia	8 611	21
Kazakhstan	6 637	16
Russia (est.)	3 413	8
Nigeria	3 153	8
Nambia	2 879	7
Uzbekistan	2 320	6
U.S.A.	1 654	4
Ukraine (est.)	846	2
Others	2 290	5
TOTAL	**41 279**	**100**

(Source: KRI)

20.7.7 Mongolia

Introduction

Mongolia is landlocked country, located in northeast Asia between Russia and China. The country has a total of 1 565 600 km^2 and shares a 4673-km-long border with China on its eastern, western and southern sides, and a 3485-km-long border with Russia to the north. The population of Mongolia is estimated at 2.7 million people with approximately 1 million people living in Ulaanbaatar, the capital and largest city. Some 40% of the population lives in the countryside, primarily subsisting as nomadic.

20.8 Economic Analysis

20.8.1 General Parameters

The financial analysis model covers the time span from Year -3 through Year +15. The pre-production years are Years -3, -2 and -1. Production years are from +1 to +15. Underground mining is from Years +1 to +9, whilst open-pit mining will commence from Years +10 to +15. Year 16 is allowed for Project closure.

The mill feed rate from the mine is 1 225 000 t/a, with first year of production at 854 000 t, thus allowing the mill to ramp up to full production. The total ore mined over the life of mine is 10 634 000 t. The average head grade over the life of mine is 0.133% U_3O_8. The average head grade for underground mining is 0.174% and for the open pit 0.074%.

The process recovery for uranium (U_3O_8) is 84.5% for the underground and 89.28% for the open pit. Over the life of mine, the total production of U_3O_8 is 20 538 t (45 279 000 lb).

Product pricing is based on the recommendation of Khan and is assumed to be on an FOB mine site of USD 65/lb U_3O_8.

20.8.2 Financial Data

Tables 20-26 and 20-27 summarise the financial analysis model. NPV is calculated on a mid-year and end-year basis.

Table 20-26
Financial Data
(USD '000)

	TOTAL
Revenue	2,943,111
Operating Costs, Mine Site	1,051,443
Other Operating Costs including Royalties	158,109
Total Operating Costs	1,175,028
Total Initial Capital Investment Costs	371,174[1]
Nett Initial Capital Investment Costs	332,786
Sustaining Capital Investment Costs	154,706
Pretax Cumulative Cash flow	1,242,203
Taxes, Income	317,273
After Tax Cumulative Cash flow	924,929

[1]Initial capital investment plus VAT.

Table 20-27
IRR and NPV Values
(USD '000)

	End of Year	
	Pre-tax	After Tax
IRR	36.4%	29.1%
NPV @ 0%	1,242,203	924,929
NPV @ 10%	406,827	275,993
Payback Period, Years	1.9	2.3

The operating cost distribution is summarised in Chart 20.1.

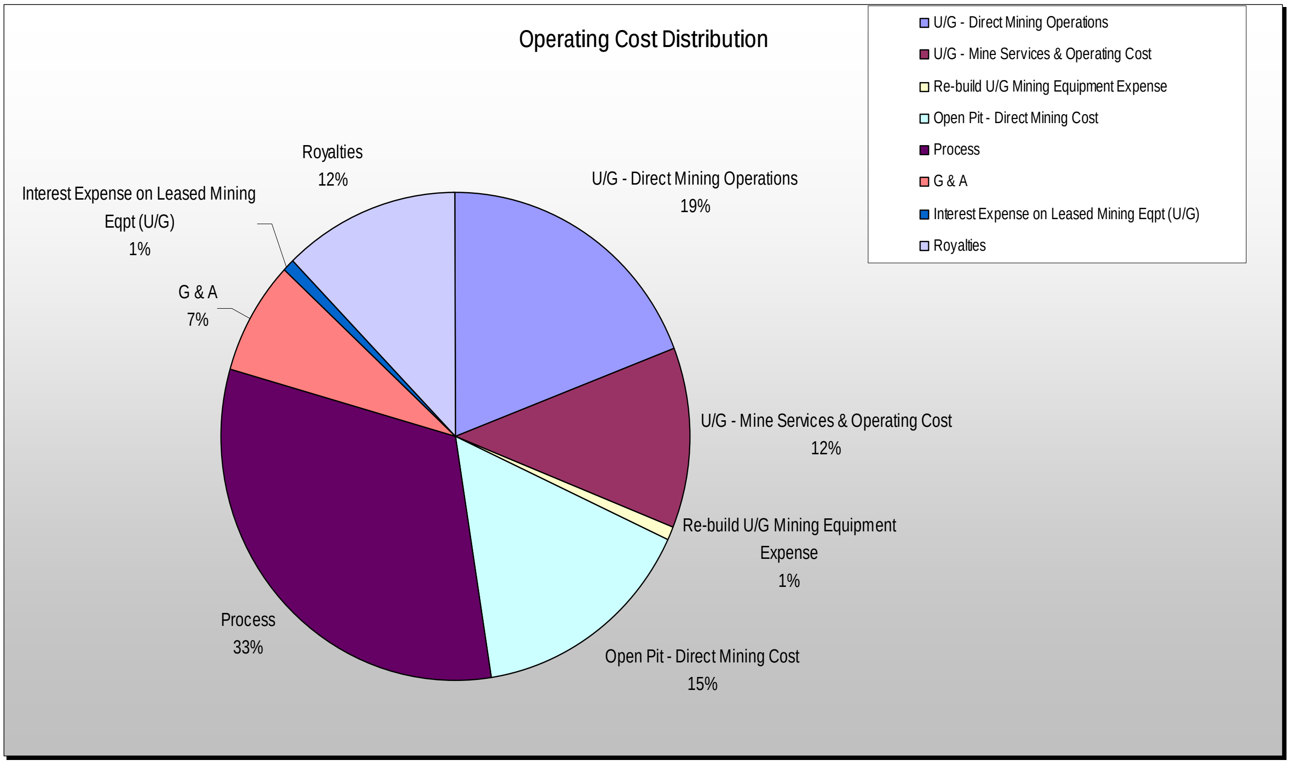


Chart 20.1
Operating Cost Distribution

The cumulative cost values for revenue, operating costs, capital investment costs and taxes and royalties are presented in Chart 20.2.

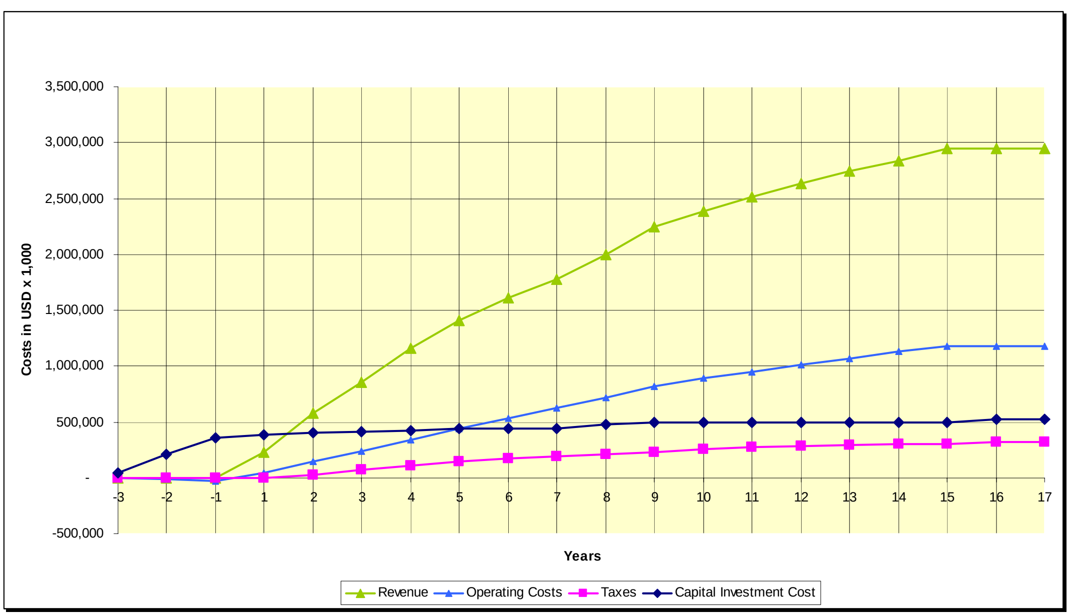


Chart 20.2
Financial Costs

20.8.3 Taxation

The project is subject to graduated levels of taxation and flat rate royalty based on gross revenue. Income Tax is payable at a rate of 10% for initial income of 3,000,000,000 togorgs (USD 1.94 million) and below and at a rate of 25% for income over the 3,000,000,000 togorgs threshold. Royalty is payable at 5% of gross revenue.

20.8.4 Closeout

The closeout cost estimated at USD 37.4 million. USD 1.4 million is for closeout engineering and is applied in year +15 whilst the closeout cost is applied in year +16

20.8.5 Sensitivities

The following Chart 20.3 Sensitivity Graph revolves around the after tax NPV @ 10% of USD 275,993,000 calculated on end of year basis.

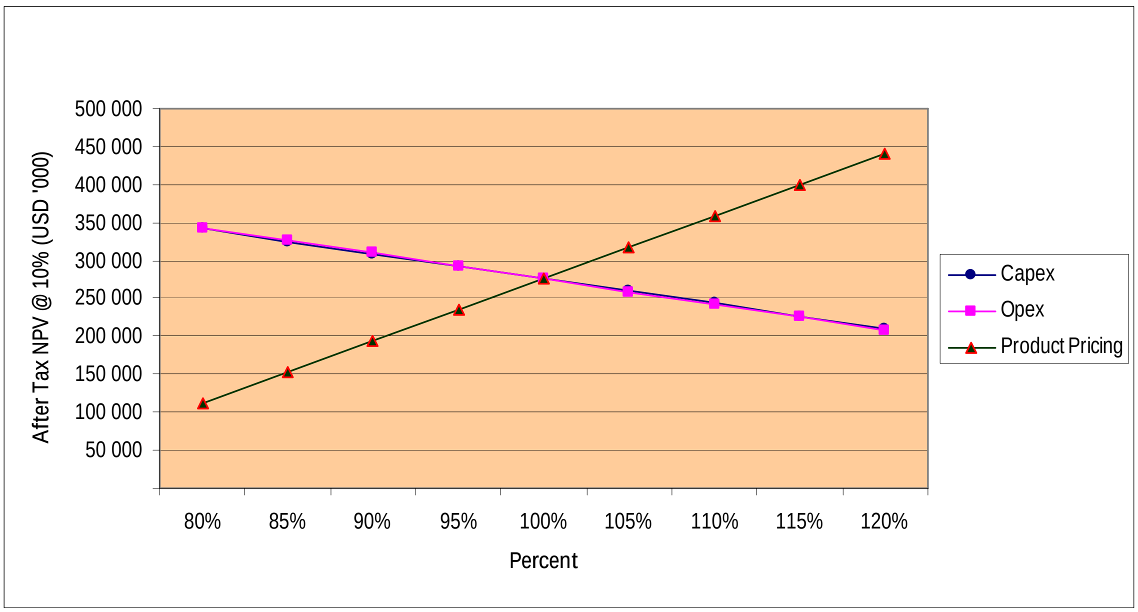


Chart 20.3
Sensitivity Graph

The following sensitivity tables present various scenarios. The After Tax NPV at 10% is the base for the tables 20-28 to 20-33

Table 20-28 Operating Cost Versus Revenue Sensitivities on After Tax IRR - compares the effect of assigning different percentages to the revenue and operating cost will have on the After Tax IRR.

Table 20-29 Change in Initial Capital Cost Versus Revenue Sensitivities - compares the effect assigning different percentages to the revenue and Initial Capital Cost will have on the After Tax NPV.

Table 20-30 Operating Cost Versus Revenue Sensitivities - compares the effect of assigning different percentages to the revenue and operating cost will have on the After Tax NPV.

Table 20-31 Operating Cost Versus Initial Mining Capital Cost Sensitivities - compares the effect assigning different percentages to the operating cost and Capital Costs excluding Sustaining Capital and Closure will have on the After Tax NPV.

Table 20-32 Operating Cost Versus Total Capital Cost Sensitivities - compares the effect assigning different percentages to the operating cost and Total capital Costs (initial plus sustaining capital costs) will have on the After Tax NPV.

Table 20-33 Grade Versus Revenue - compares the effect assigning different percentages to the grade (diluted U3O8) and revenue will have on the After Tax NPV.

Table 20-28 - Operating Cost Versus Revenue Sensitivities on After Tax IRR

		Opex - Including Royalties				
After-Tax IRR	**29.05%**	90%	95%	**100%**	105%	110%
	110%	35.2%	34.3%	33.5%	32.7%	31.8%
	105%	33.0%	32.2%	31.3%	30.5%	29.6%
Revenue, Commodity Pricing	**100%**	30.8%	29.9%	**29.1%**	28.1%	27.2%
	95%	28.5%	27.6%	26.7%	25.7%	24.8%
	90%	26.1%	25.2%	24.20%	23.2%	22.2%

Table 20-29 - Change in Initial Capital Cost Versus Revenue Sensitivities

		Change in Capex				
After-Tax NPV @ 10% (USD x '000)	**275 993**	90%	95%	**100%**	105%	110%
	110%	385 330	371 827	358 324	344 821	331 318
	105%	344 165	330 662	317 159	303 656	290 153
Revenue, Commodity Pricing	**100%**	302 999	289 496	**275 993**	262 490	248 987
	95%	261 834	248 331	234 828	221 325	207 822
	90%	220 668	207 165	193 662	180 159	166 657

Table 20-30 - Operating Cost Versus Revenue Sensitivities

		Opex - Including Royalties				
After-Tax NPV @ 10% (USD x '000)	**275 993**	90%	95%	**100%**	105%	110%
	110%	392 550	375 437	358 324	341 211	324 098
	105%	351 163	334 161	317 159	300 156	283 154
Revenue, Commodity Pricing	**100%**	309 777	292 885	**275 993**	259 101	242 210
	95%	268 390	251 609	234 828	218 047	201 265
	90%	227 004	210 333	193 662	176 992	160 321

Table 20-31 - Operating Cost Versus Initial Mining Capital Cost Sensitivities

		Opex - Including Royalties				
After-Tax NPV @ 10% (USD x '000)	**275 993**	90%	95%	**100%**	105%	110%
	110%	282 771	265 879	248 987	232 096	215 204
	105%	296 274	279 382	262 490	245 599	228 707
Capex excluding Sustaining Capital & Closure	**100%**	309 777	292 885	**275 993**	259 101	242 210
	95%	323 280	306 388	289 496	272 604	255 713
	90%	336 783	319 891	302 999	286 107	269 216

Table 20-32 - Operating Cost Versus Total Capital Cost Sensitivities

		Opex - Including Royalties				
After-Tax NPV @ 10% (USD x '000)	**275 993**	90%	95%	**100%**	105%	110%
	110%	277 076	260 152	243 228	226 303	209 379
	105%	293 427	276 519	259 611	242 702	225 794
Capex - Including Sustaining & Closure	**100%**	309 777	292 885	**275 993**	259 101	242 210
	95%	326 127	309 252	292 376	275 501	258 625
	90%	342 478	325 618	308 759	291 900	275 040

Table 20-33 - Grade Versus Revenue

		Grade				
After-Tax NPV @ 10% (USD x '000)	**275 993**	90%	95%	**100%**	105%	110%
	110%	274 676	316 500	358 324	400 148	441 973
	105%	237 313	277 236	317 159	357 082	397 005
Revenue	**100%**	199 949	237 971	**275 993**	314 015	352 037
	95%	162 586	198 707	234 828	270 949	307 070
	90%	125 223	159 443	193 662	227 882	262 102

Table 20-34 Project Cash flow covers the period from the approval to proceed with the project until post production closure. The cash flow includes revenue, operating costs, capital investment costs (pre-production and sustaining), working capital and taxation. Financial statistics calculated on mid year and end of year basis for NPV, IRR and payback period for both pre-tax and after tax are included in the cash flow model.

Table 20-34 – Project Cash Flow
(USD x '000)

Only U/G Mining Equipment Leased				Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Mining																							
Underground - Ore (No. 7 Ore)	000 mt			10,634				854	1,228	1,225	1,225	1,225	1,228	1,225	1,225	1,199	-						
Open Pit - Ore (No. 2 Ore)	000 mt			7,413												26	1,225	1,225	1,225	1,225	1,225	1,262	-
Total Available to Stockpile	000 mt			-																			
Ore Feed to Crusher, Annual								854	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,262	-
Concentrate Production																							
Ore, Annual Production	000 mt			18,041				854	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,262	-
Diluted Grade (%U3O8)								0.224	0.234	0.183	0.208	0.166	0.136	0.115	0.149	0.164	0.093	0.082	0.075	0.070	0.058	0.066	
Contained U3O8	000 mt			23.999				1.912	2.872	2.244	2.542	2.037	1.664	1.410	1.831	2.015	1.141	0.999	0.921	0.856	0.713	0.844	
Recovered U3O8	000 mt			20.538				1.615	2.426	1.896	2.148	1.721	1.406	1.191	1.547	1.703	1.018	0.892	0.822	0.764	0.636	0.753	
Recovered U3O8	000 lbs			45,279		-	-	3,561	5,349	4,180	4,735	3,793	3,099	2,626	3,410	3,754	2,245	1,967	1,812	1,685	1,403	1,660	-
Revenue	USD x '000			2,943,111		-	-	231,493	347,658	271,692	307,781	246,565	201,439	170,669	221,624	244,012	145,926	127,865	117,802	109,503	91,163	107,921	-
Total Revenue	**USD x '000**			**2,943,111**	**0**	**0**	**0**	**231,493**	**347,658**	**271,692**	**307,781**	**246,565**	**201,439**	**170,669**	**221,624**	**244,012**	**145,926**	**127,865**	**117,802**	**109,503**	**91,163**	**107,921**	**0**
Operating Costs		USD/t feed																					
U/G - Direct Mining Operations	USD x '000	21.78		231,661				18,899	28,379	26,879	27,906	26,109	28,130	25,803	24,467	25,089	-						
U/G - Mine Services & Operating Cost	USD x '000	13.69		145,600				14,078	16,286	16,264	16,264	16,264	16,561	16,542	16,819	16,522	-		-	-			
Re-build U/G Mining Equipment Expense	USD x '000	0.94		9,946								4,973	1,989	1,989	995								
Open Pit - Direct Mining Cost	USD x '000	25.07		185,867											7,832	10,486	31,863	28,903	29,184	27,133	29,709	20,756	
Process	USD x '000	21.57		389,266				20,444	31,246	31,246	31,246	30,880	30,880	30,880	30,880	30,880	22,335	20,930	20,930	20,930	20,930	14,626	-
G & A	USD x '000	4.94		89,103				7,040	7,040	7,040	7,040	6,300	6,300	6,160	6,160	6,160	4,977	4,977	4,977	4,977	4,977	4,977	-
Subtotal Operating Plant Site	**USD x '000**	**58.26**		**1,051,443**	**-**	**-**	**-**	**60,460**	**82,951**	**81,429**	**82,456**	**84,526**	**83,861**	**81,375**	**87,154**	**89,137**	**59,175**	**54,811**	**55,092**	**53,041**	**55,616**	**40,359**	**-**
Shipping & Transportation	USD x '000	-		-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest Expense on Leased Mining Eqpt (U/G)	USD x '000	1.03		10,953				3,595	2,997	2,303	1,497	561											
Royalties	USD x '000	8.15		147,156		-	-	11,575	17,383	13,585	15,389	12,328	10,072	8,533	11,081	12,201	7,296	6,393	5,890	5,475	4,558	5,396	-
VAT (on Process Op. + G&A)	USD x '000	3.23		58,240				2,492	3,573	3,573	3,573	3,852	3,852	3,992	3,992	3,992	4,321	4,180	4,180	4,180	4,180	4,307	-
Refund of VAT (Process Op. + G&A)	USD x '000	(3.23)		(58,240)				(2,492)	(3,573)	(3,573)	(3,573)	(3,852)	(3,852)	(3,992)	(3,992)	(3,992)	(4,321)	(4,180)	(4,180)	(4,180)	(4,180)	(4,307)	-
Refund of Duty & VAT on Initial Capex	USD x '000	(1.91)		(34,524)	(2,980)	(11,921)	(11,921)	(7,702)															
	USD x '000			-																			
Total Operating Cost	**USD x '000**	**65.11**		**1,175,028**	**(2,980)**	**(11,921)**	**(11,921)**	**67,928**	**103,331**	**97,316**	**99,342**	**97,416**	**93,933**	**89,908**	**98,235**	**101,338**	**66,472**	**61,204**	**60,982**	**58,516**	**60,174**	**45,755**	**-**
Investment Costs																							
Initial Capital -U/G Mining	USD x '000			66,753	4,494	20,467	41,792	-		-													
Capitalized U/G Mining Operations	USD x '000			3,864		-	3,864																
Initial Capital - Process & Infrastructure	USD x '000			266,033	32,390	136,396	86,299	10,949															
Deferred Taxes on Capital	USD x '000			-																			
Duty & VAT on Initial Capex				34,524	2,980	14,282	14,282	2,980															
Total Initial Investment Costs				**371,174**	39,864	171,145	146,236	13,929															
Purchase Open of Pit Equipment	USD x '000			51,512											25,038	21,024	5,450						
Principal Repayment on U/G Mining Equipment	USD x '000			25,628				3,721	4,319	5,014	5,820	6,755											
Sustaining Capital -U/G Mine Services.	USD x '000			1,354				75	333	121	332	121	121	121	86	44							
Sustaining Capital -U/G Mine Infrstructure	USD x '000			-				-	-	-	-	-	-	-	-	-							
Sustaining -U/G Waste Development	USD x '000			11,651				1,388	2,823	1,128	4,560	1,300	290	162	-	-							
Sustaining Capital -U/G Trackless & Track Eqpt.	USD x '000			3,300				-	-	-	-	3,300	-	-	-	-							
Sustaining Capital - Open Pit	USD x '000			238														60	60	60	58	-	
Expand Truck Shop & Upgrade Haul Road	USD x '000			4,486											4,486								
Sustaining Capital Tailings	USD x '000			18,638					4,080			5,860		5,448			3,250						
Sustaining Capital - Fresh Water Source	USD x '000			500										500									
Working Capital	USD x '000			0	-	-	-	10,980	5,723	(972)	327	(311)	(563)	(651)	1,346	502	(5,636)	(851)	(36)	(399)	268	(2,331)	(7,396)
Closure	USD x '000			37,400																		1400	36,000
				-																			
Total Sustainig Capital				**154,706**				16,164	17,278	5,290	11,039	17,025	(152)	5,580	30,955	21,570	3,064	(791)	24	(339)	326	(931)	28,604
Total Investment Costs	**USD x '000**			**525,881**	**39,864**	**171,145**	**146,236**	**30,093**	**17,278**	**5,290**	**11,039**	**17,025**	**(152)**	**5,580**	**30,955**	**21,570**	**3,064**	**(791)**	**24**	**(339)**	**326**	**(931)**	**28,604**
Pre-Tax Financial																							
Cashflow, Annual	**USD x '000**			**1,242,203**	**(36,884)**	**(159,224)**	**(134,316)**	**133,472**	**227,049**	**169,085**	**197,400**	**132,124**	**107,658**	**75,180**	**92,435**	**121,104**	**76,391**	**67,452**	**56,796**	**51,325**	**30,663**	**63,096**	**(28,604)**
Cashflow, Cumulated	USD x '000			1,242,203	(36,884)	(196,107)	(330,423)	(196,951)	30,098	199,183	396,583	528,707	636,365	711,545	803,980	925,084	1,001,475	1,068,927	1,125,723	1,177,048	1,207,711	1,270,807	1,242,203
Pay-back Period	Years			1.9	-	-	-	-	1.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Internal Rate of Return (IRR)				**36.4%**																			
Net Present Value		Rate	Mid Point	End of year																			
	USD x '000	5.0%	720,937	703,562																			
	USD x '000	7.5%	553,880	534,209																			
	USD x '000	10.0%	426,684	406,827																			
	USD x '000	15.0%	252,668	235,614																			
Taxation																							
Depreciation of Initial Capital Asset	USD x '000			447,018				46,763	47,667	47,916	48,895	50,230	21,312	21,134	26,208	29,443	29,806	18,805	18,760	13,748	9,546	16,784	-
Depreciation of Sunk Cost Capital Asset	USD x '000			-																			
Total Depreciation				447,018	-	-	-	46,763	47,667	47,916	48,895	50,230	21,312	21,134	26,208	29,443	29,806	18,805	18,760	13,748	9,546	16,784	-
Loss for the Year from Operations	USD x '000			-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Loss Available from Previous & Current Year/s	USD x '000			-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Utilized in current Year				-					-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
				-																			
Total Loss Carried forward	USD x '000			-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Current																							
Income subject to Tax	USD x '000			1,286,542				109,100	196,660	126,459	159,544	98,919	86,194	59,626	97,181	113,231	49,649	47,856	38,059	37,239	21,443	45,382	-
Income Tax	USD x '000			317,273				26,984	48,874	31,324	39,595	24,439	21,258	14,616	24,005	28,017	12,121	11,673	9,224	9,019	5,070	11,055	-
Windfall Tax	USD x '000			-																			
Total Taxes	**USD x '000**			**317,273**	**-**	**-**	**-**	**26,984**	**48,874**	**31,324**	**39,595**	**24,439**	**21,258**	**14,616**	**24,005**	**28,017**	**12,121**	**11,673**	**9,224**	**9,019**	**5,070**	**11,055**	**-**
After Tax Financial																							
Cashflow, Annual	**USD x '000**			**924,929**	**(36,884)**	**(159,224)**	**(134,316)**	**106,488**	**178,175**	**137,761**	**157,805**	**107,685**	**86,401**	**60,564**	**68,430**	**93,088**	**64,269**	**55,779**	**47,572**	**42,306**	**25,593**	**52,042**	**- 28,604**
Cashflow, Cumulated	USD x '000			924,929	(36,884)	(196,107)	(330,423)	(223,935)	(45,760)	92,001	249,805	357,490	443,891	504,455	572,885	665,973	730,242	786,021	833,593	875,899	901,492	953,533	924,929
Pay-back Period	Years			2.3	-	-	-	-	-	2.3	-	-	-	-	-	-	-	-	-	-	-	-	-
Internal Rate of Return (IRR)				**29.1%**																			
Net Present Value		Rate	Mid Point	End of year																			
	USD x '000	5.0%	517,404	504,935																			
	USD x '000	7.5%	387,689	373,921																			
	USD x '000	10.0%	289,464	275,993																			
	USD x '000	15.0%	156,361	145,807																			

20.9 Project Implementation

The Project Execution Plan, Figure 20.17, outlines the summary of major activities leading to successful completion of the Project. The major activities are grouped into major categories: Agreements, Environmental Assessment, Engineering, Mining, Construction, Commissioning and Ramp up.

The scheduled start of the EPCM activities is October 09, 2009, dependent on receiving Government of Mongolia approval for the Project. The schedule identifies activities occurring during the first half of 2010 necessary to maintain the planned completion date.

The overall duration from March, 2010 to achieving full production is 33 months. From the start of detail engineering to completion of pre-commissioning is 28 months. The construction duration of the surface facilities is 18 months. A 3-month duration for production ramp-up is planned.

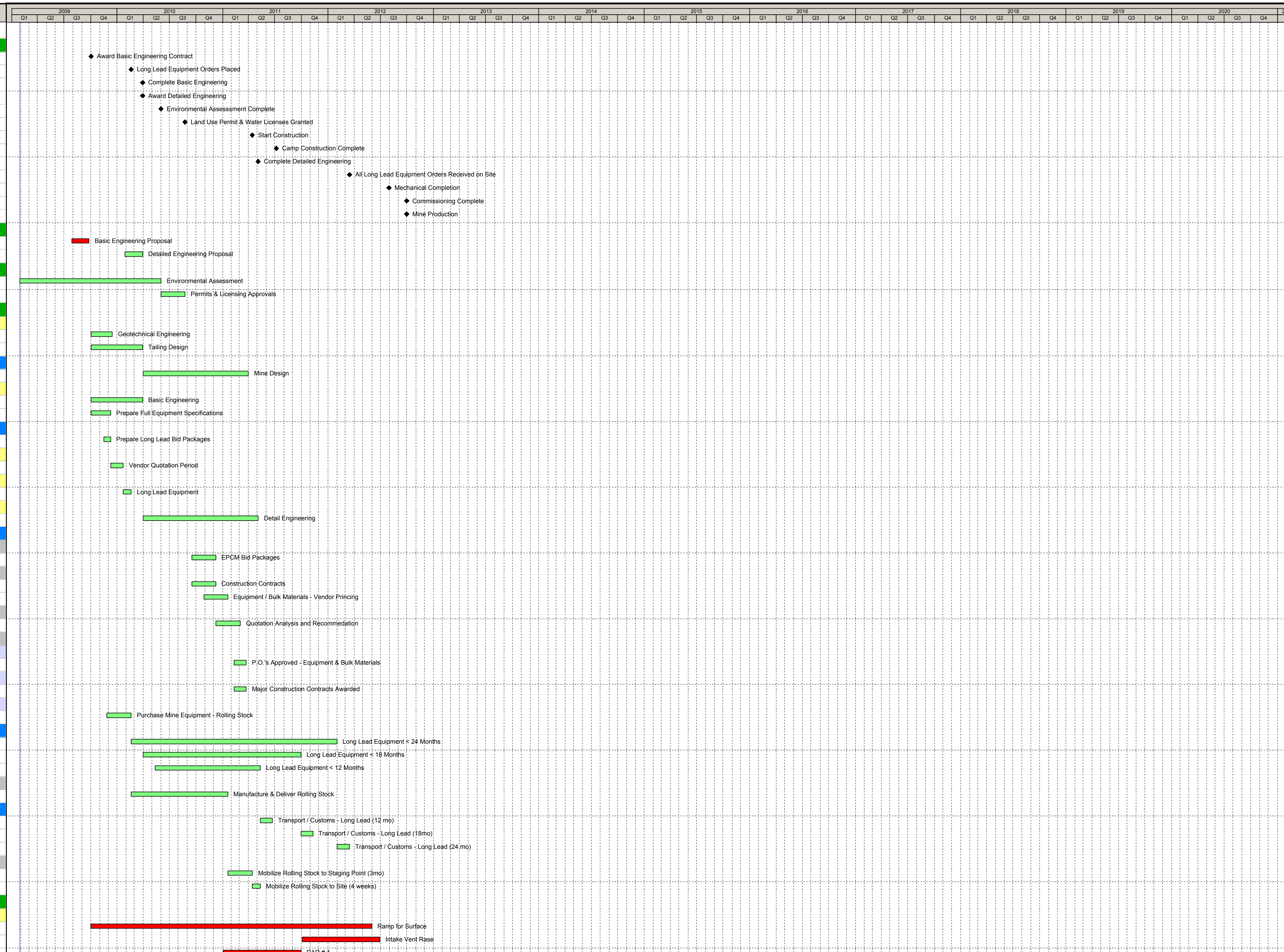
20.9.1 Key Project Dates

The following activity key dates are identified.

- October 2, 2009 Award Basic Engineering Contract
- February 18, 2010 Long Lead Equipment Orders Placed
- March 30, 2010 Complete Basic Engineering and Award Detailed Engineering
- August 24, 2010 Land Use Permit & Water Licenses Granted
- April 14, 2011 Start of Construction
- May 5, 2011 Complete Detailed Engineering
- July 7, 2011 Construction Camp Completed
- March 15, 2012 All long lead equipment orders received on site
- July 31, 2012 Mechanical Completion
- September 30, 2012 Commissioning Complete

Khan Resources Inc

Activity ID	Activity Name	Original Duration	Start	Finish
Dornod Uranium Project		3045	29-Jan-09	01-Oct-20
MILESTONES		781	02-Oct-09	30-Sep-12
M0020	Award Basic Engineering Contract	0		02-Oct-09
M0030	Long Lead Equipment Orders Placed	0		18-Feb-10
M0040	Complete Basic Engineering	0		31-Mar-10
M0050	Award Detailed Engineering	0		30-Mar-10
M0060	Environmental Assesssment Complete	0		01-Jun-10
M0070	Land Use Permit & Water Licenses Granted	0		24-Aug-10
M0080	Start Construction	0	14-Apr-11	
M0090	Camp Construction Complete	0		07-Jul-11
M0100	Complete Detailed Engineering	0		05-May-11
M0110	All Long Lead Equipment Orders Received on Site	0		15-Mar-12
M0120	Mechanical Completion	0		31-Jul-12
M0130	Commissioning Complete	0		30-Sep-12
M0140	Mine Production	0	30-Sep-12	
AGREEMENTS		176	27-Jul-09	30-Mar-10
M0210	Basic Engineering Proposal	44	27-Jul-09	25-Sep-09*
M0220	Detailed Engineering Proposal	44	27-Jan-10	30-Mar-10
ENVIRONMETAL ASSESSMENT		409	29-Jan-09	24-Aug-10
M0310	Environmental Assessment	349	29-Jan-09	01-Jun-10
M0320	Permits & Licensing Approvals	60	02-Jun-10	24-Aug-10
ENGINEERING		639	02-Oct-09	15-Mar-12
Engineering by Others		389	02-Oct-09	31-Mar-11
E0010	Geotechnical Engineering	52	02-Oct-09	15-Dec-09
E0020	Tailing Design	128	02-Oct-09	31-Mar-10
Mine Engineering		260	01-Apr-10	31-Mar-11
E0110	Mine Design	260	01-Apr-10	31-Mar-11
Basic Engineering		128	02-Oct-09	31-Mar-10
E0310	Basic Engineering	128	02-Oct-09	31-Mar-10
E0320	Prepare Full Equipment Specifications	49	02-Oct-09	10-Dec-09
Bid Packages - Long Lead Equipment		18	16-Nov-09	10-Dec-09
E0410	Prepare Long Lead Bid Packages	18	16-Nov-09	10-Dec-09
Vendor Quotation Period		30	10-Dec-09	21-Jan-10
E0510	Vendor Quotation Period	30	10-Dec-09	21-Jan-10
Long Lead Equipment P.O's (Khan)		20	21-Jan-10	18-Feb-10
E0610	Long Lead Equipment	20	21-Jan-10	18-Feb-10
Detailed Engineering		600	26-Nov-09	15-Mar-12
E1010	Detail Engineering	285	01-Apr-10	05-May-11
Procurement		345	26-Nov-09	24-Mar-11
Develop Bid Pacakages (Procurement)		60	16-Sep-10	09-Dec-10
E2010	EPCM Bid Packages	60	16-Sep-10	09-Dec-10
Vendor Quotation Period		90	16-Sep-10	20-Jan-11
E2110	Construction Contracts	60	16-Sep-10	09-Dec-10
E2120	Equipment / Bulk Materials - Vendor Princing	60	28-Oct-10	20-Jan-11
Bid Analysis		60	09-Dec-10	03-Mar-11
E2210	Quotation Analysis and Recommedation	60	09-Dec-10	03-Mar-11
P.O Commitment Approved by Khan		345	26-Nov-09	24-Mar-11
Equipment & Materials		30	10-Feb-11	24-Mar-11
E2220	P.O.'s Approved - Equipment & Bulk Materials	30	10-Feb-11	24-Mar-11
Major Contracts		30	10-Feb-11	24-Mar-11
E2230	Major Construction Contracts Awarded	30	10-Feb-11	24-Mar-11
Mine Equipment (Khan Resources)		60	26-Nov-09	18-Feb-10
E2240	Purchase Mine Equipment - Rolling Stock	60	26-Nov-09	18-Feb-10
Manufacture		510	18-Feb-10	02-Feb-12
E3010	Long Lead Equipment < 24 Months	510	18-Feb-10	02-Feb-12
E3020	Long Lead Equipment < 18 Months	390	01-Apr-10	29-Sep-11
E3030	Long Lead Equipment < 12 Months	260	13-May-10	12-May-11
Mine Equipment (Khan Resources)		240	18-Feb-10	20-Jan-11
E3110	Manufacture & Deliver Rolling Stock	240	18-Feb-10	20-Jan-11
Transportation & Logistic		300	20-Jan-11	15-Mar-12
E3210	Transport / Customs - Long Lead (12 mo)	30	12-May-11	23-Jun-11
E3220	Transport / Customs - Long Lead (18mo)	30	29-Sep-11	10-Nov-11
E3230	Transport / Customs - Long Lead (24 mo)	30	02-Feb-12	15-Mar-12
Mine Equipment (Khan Resources)		80	20-Jan-11	12-May-11
E3310	Mobilize Rolling Stock to Staging Point (3mo)	60	20-Jan-11	14-Apr-11
E3320	Mobilize Rolling Stock to Site (4 weeks)	20	14-Apr-11	12-May-11
MINING		2870	01-Oct-09	01-Oct-20
Mining Underground		782	01-Oct-09	28-Sep-12
N1010	Ramp for Surface	696	01-Oct-09*	31-May-12
N1020	Intake Vent Rase	195	03-Oct-11*	29-Jun-12
N1030	RAR # 1	194	04-Jan-11*	30-Sep-11



Actual Work
Remaining Work
Critical Remaining Work
Milestone

Figure 20.17

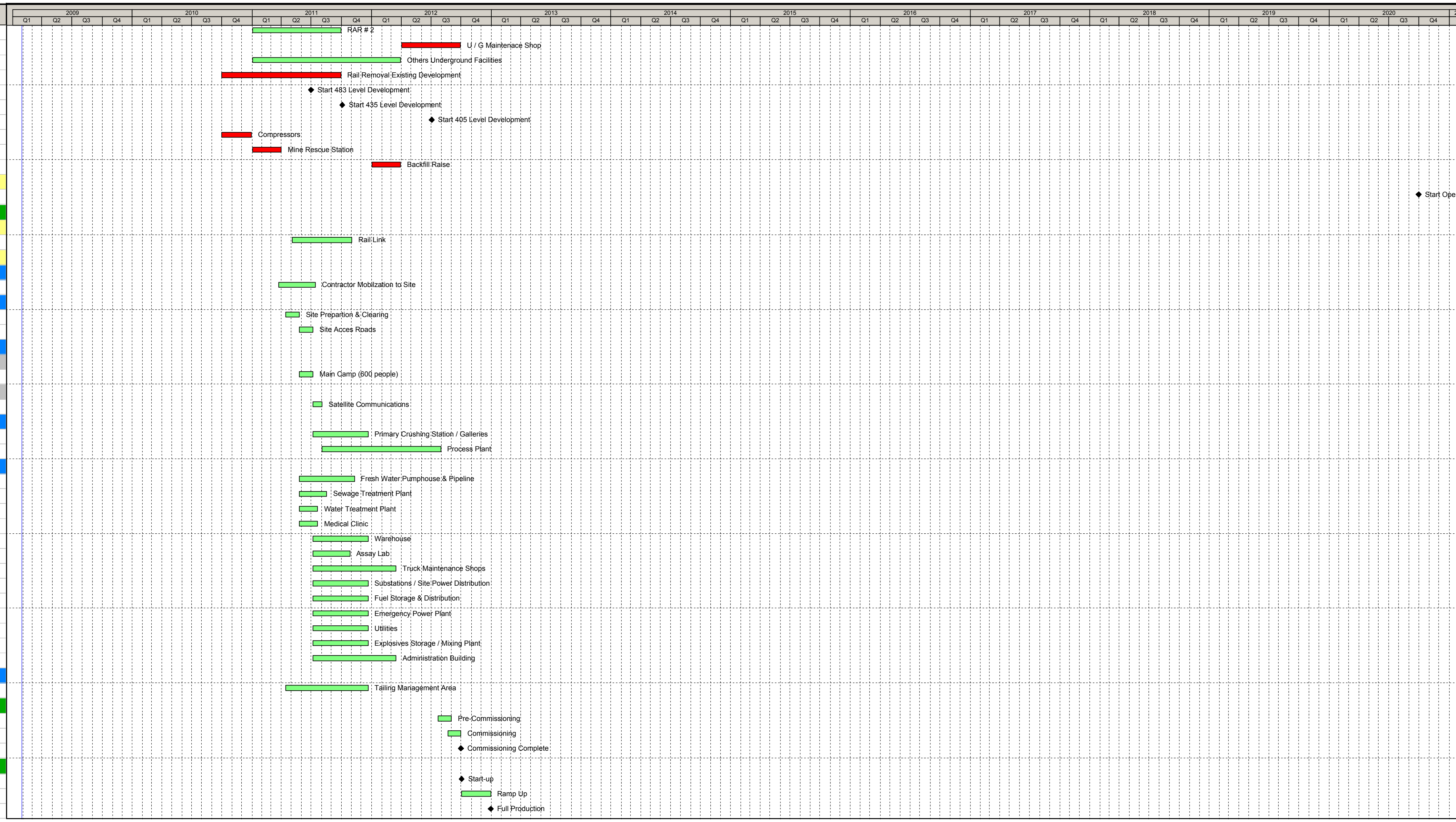
AkerSolutions

Dornod Uranium Project

Project Execution Schedule

Khan Resources Inc

Activity ID	Activity Name	Original Duration	Start	Finish
N1040	RAR # 2	194	04-Jan-11	30-Sep-11
N1050	U / G Maintenace Shop	130	02-Apr-12*	28-Sep-12
N1060	Others Underground Facilities	324	04-Jan-11	30-Mar-12
N1070	Rail Removal Existing Development	261	01-Oct-10*	30-Sep-11
N1080	Start 483 Level Development	0	01-Jul-11*	
N1090	Start 435 Level Development	0	03-Oct-11*	
N1100	Start 405 Level Development	0	02-Jul-12*	
N1110	Compressors	65	01-Oct-10*	30-Dec-10
N1120	Mine Rescue Station	65	03-Jan-11*	01-Apr-11
N1130	Backfill Raise	65	02-Jan-12*	30-Mar-12
Open Pit Year 8		0	01-Oct-20	01-Oct-20
N2010	Start Open Pit	0	01-Oct-20*	
CONSTRUCTION		353	24-Mar-11	31-Jul-12
Construction Off Site (by Khan)		130	05-May-11	03-Nov-11
C0010	Rail Link	130	05-May-11	03-Nov-11
Construction On Site		353	24-Mar-11	31-Jul-12
Mobilization		80	24-Mar-11	14-Jul-11
C0110	Contractor Mobilzation to Site	80	24-Mar-11	14-Jul-11
Earthwork		60	14-Apr-11	07-Jul-11
C0210	Site Prepartion & Clearing	30	14-Apr-11	26-May-11
C0220	Site Acces Roads	30	26-May-11	07-Jul-11
Camp		50	26-May-11	04-Aug-11
Main Camp		30	26-May-11	07-Jul-11
C0310	Main Camp (600 people)	30	26-May-11	07-Jul-11
Satellite Communications (Khan)		20	07-Jul-11	04-Aug-11
C0410	Satellite Communications	20	07-Jul-11	04-Aug-11
Ore Process Facilities		278	07-Jul-11	31-Jul-12
C0510	Primary Crushing Station / Galleries	120	07-Jul-11	22-Dec-11
C0520	Process Plant	258	04-Aug-11	31-Jul-12
Infrastructure		210	26-May-11	15-Mar-12
C1010	Fresh Water Pumphouse & Pipeline	120	26-May-11	10-Nov-11
C1020	Sewage Treatment Plant	60	26-May-11	18-Aug-11
C1030	Water Treatment Plant	40	26-May-11	21-Jul-11
C1040	Medical Clinic	40	26-May-11	21-Jul-11
C1050	Warehouse	120	07-Jul-11	22-Dec-11
C1060	Assay Lab	80	07-Jul-11	27-Oct-11
C1070	Truck Maintenance Shops	180	07-Jul-11	15-Mar-12
C1080	Substations / Site Power Distribution	120	07-Jul-11	22-Dec-11
C1090	Fuel Storage & Distribution	120	07-Jul-11	22-Dec-11
C1100	Emergency Power Plant	120	07-Jul-11	22-Dec-11
C1110	Utilities	120	07-Jul-11	22-Dec-11
C1120	Explosives Storage / Mixing Plant	120	07-Jul-11	22-Dec-11
C1130	Administration Building	180	07-Jul-11	15-Mar-12
Tailing Handling Facilities		180	14-Apr-11	22-Dec-11
C2010	Tailing Management Area	180	14-Apr-11	22-Dec-11
COMMISSIONING		50	22-Jul-12	30-Sep-12
C3010	Pre-Commissioning	40	22-Jul-12	31-Aug-12
C3020	Commissioning	40	21-Aug-12	30-Sep-12
C3040	Commissioning Complete	0		30-Sep-12
RAMP UP		65	01-Oct-12	31-Dec-12
C3030	Start-up	0	01-Oct-12	
C3050	Ramp Up	91	01-Oct-12	31-Dec-12
C3060	Full Production	0	31-Dec-12	



Actual Work
Remaining Work
Critical Remaining Work
Milestone

Figure 20.17

AkerSolutions

Dornod Uranium Project

Project Execution Schedule

21 Interpretation and Conclusions

The Definitive Feasibility Study commissioned by Khan for the Dornod Project shows a positive economic outcome, including the following key results.

(a) Mineral Reserves

(i) The proven and probable reserve estimate for the No. 2 Deposit open-pit mine, at 0.028% U_3O_8 cutoff grade, is 7 407 000 t grading 0.074% U_3O_8. Mining dilution of 15% at a 0.018% U_3O_8 grade is included.

(ii) The proven and probable reserve estimate for the No. 7 Deposit at a 0.061% U_3O_8 cutoff is 10 634 000 t grading 0.174% U_3O_8. Underground mining recovery of 88% and dilution of 10% at 0% U_3O_8 grade is forecast.

(b) Mining

(i) Underground and open-pit mines are planned to produce a total of approximately 1 225 000 t of ore per year, at a rate of 3500 t/d.

(ii) A total of 18.04 Mt of ore at an average grade of 0.133% U_3O_8 will be mined from the Nos. 7 and 2 Deposits over a period of slightly more than 15 years.

(c) Environmental and Social Impacts

Many adverse effects that could occur from the Project will be eliminated or minimised by proper design, maintenance, management, and mitigation measures. The net social and environmental analysis assumes that the environmental and social management, monitoring, and reclamation measures will be implemented as discussed in both the ESIA and ESMP.

Table 21-1 summarises the potential net environmental and social impacts of the Project. Net impacts were calculated based on worst-case impact scenarios (i.e., gross impacts), minus the effects of all proposed prevention and mitigation measures. This provides an estimate of the net impacts, both short- and long-term, that can be anticipated as a result of the Project's construction activities, operational activities and closure.

This analysis indicates that implementation of the environmental and social management, mitigation, monitoring, and reclamation measures adopted by Khan will eliminate or minimise the potential negative environmental and social impacts of the Project; and, will provide economic and social benefits to the region.

Table 21-1
Summary of Net Environmental and Social Impacts

Environmental Parameter	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
Topography	Construction of new land features such as the water management facilities, RMA, overburden placement areas, roads, sediment and water runoff controls, mining support facilities; and expansion of the pre-existing open pit.	Recontour new land features at closure; Cap and revegetate RMA at closure; Revegetate disturbed areas as practical; Decommission and / or demolish mining facilities at closure; Backfill water and sediment ponds at closure; Progressively backfill underground mine and air shafts during operations; and Allow the open pit to flood naturally after closure.	Short-term: Changes can be significant with newly constructed land features and mine expansion. Long-term: Except for the expanded open pit, topographic changes will be minimal due to recontouring and revegetation.
Air	Gaseous emissions from stationary and mobile sources; Fugitive dust emissions; and Increased noise levels and blasting vibrations.	Minimise land disturbance; Cover or revegetate exposed soils or erodible materials to reduce dust generation; Suppress dust with water or surfactants; Utilise low-sulphur fuel; Construct appropriate stack heights for emissions; Install pollution control features; Maintain equipment; Implement enclosure and cladding of processing facilities; Install proper noise barriers and / or noise containments at / near source equipment and at facility boundaries; and Optimise traffic routes and speed limits.	Short-term: Slight increases in dust. Increased noise levels and vibrations. No significant impact from gaseous emissions. Long-term: No significant impacts from dust after closure. Gaseous emissions, noise and vibrations will cease after closure.
Geology and Mineral Resources	Alteration of the geologic configuration; and removal of uranium ore.	Inherent to open-pit and underground mining; no mitigation measures exist.	Alteration of the geologic configuration; and, Removal of uranium ore.

Table 21-1
Summary of Net Environmental and Social Impacts

Environmental Parameter	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
(cont)			
Soils and Sediment	Removal of topsoils; Alteration of the soil profile; Increased erosion; Decreased soil productivity; Possible contamination; and Increased sediment transport or production.	Stockpile and preserve topsoils for use in reclamation; Control surface-water runoff and maintain the zero-discharge facility as designed; Minimise traffic and enforce low speed limits; Control erosion and sedimentation with sediment fences, vegetative strips, ponds, and ditches; Properly treat contaminated soil; and Monitoring.	Short-term: Significant direct impact from soil displacement. Long-term: No significant impact due to reclamation. Net sediment impacts are not anticipated.
Surface Water	Accidental release of contaminants.	Follow standard operating procedures; Control and treat surface-water runoff as planned; Maintain the zero-discharge facility as designed; Control erosion and sedimentation; Routinely inspect and maintain equipment and water management facilities; Implement a Spill Control Plan; Properly store and handle chemicals; Immediately clean up accidental spills or release of contaminants; and Monitoring.	No short- or long-term impacts to surface water are anticipated. The Project is designed as a zero-discharge facility; and, there are no natural surface-water bodies within and near the Project area.
Groundwater	Alteration of subsurface recharge-discharge relationships; Alteration of groundwater flow pattern and conditions; Reduced potentiometric surface elevation; Reduced groundwater availability; and Groundwater quality alteration from accidental releases and spills, from mining and geologic exploration / monitoring / well drilling, and from flooding.	Recycle mine process water; Collect and store water resources; Decommission and reclaim on-site water management facilities at closure; Regularly monitor groundwater quantity and quality, including the cone of depression from dewatering activities; Properly store and handle chemicals; Routinely inspect and maintain equipment and water management facilities; Promptly clean up accidental spills; Line the RMA and other water management facilities; and Monitoring.	Short-term: Alteration of subsurface recharge-discharge relationships and groundwater flow conditions; and, reduced groundwater availability and groundwater resources. Long-term: No significant impacts to groundwater are anticipated.

Table 21-1
Summary of Net Environmental and Social Impacts

Environmental Parameter	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
(cont)			
Biological	Direct vegetation removal; Species population reduction or changes in species diversity; Chronic and self-perpetuating erosion-prone areas; Fragmentation and loss of habitat; Introduction of new or invasive species; Increased human presence and activity; Changes in vegetation structure / composition; Decline in wildlife health; and Displaced wildlife.	Update baseline environmental data; Stockpile non-commercial vegetation and slash for reclamation; Support refuges in the vicinity; Implement an effective reclamation and revegetation plan; Select native seed mixtures; Actively control invasive species as necessary; Protect sensitive species; Support a periodic environmental monitoring program; and Collaborate ecosystem management with outside stakeholders.	Short-term: Significant impact to vegetation from removal. Possible reduction in some wildlife populations in areas of disturbance. Long-term: No significant impacts; positive net gain in grassland vegetation due to reclamation and revegetation of previous industrial barrens.
Social	Increased employment opportunities; Purchase and / or utilisation of Mongolian supplies and services; Increased tax base; Land improvements; Community development programs; Decreased grazing area; Decreased water resources; Relocation of herders; Short-term increased land disturbance; Increased demand on infrastructure and services; and Short-term increased risk to human health and safety.	Continue to pay land-use and water-use fees to the Soum Government; Reclaim / revegetate disturbed land; Consult regional government and local services to plan the expansion and/or necessary maintenance of infrastructure and services; Restrict access to the Project area for health and safety reasons; Provide employee safety training as well as immediate medical attention in case of an accident; and Implement the PCDP.	Short-term: Increased employment opportunities; purchase and / or utilisation of Mongolian supplies and services; increased tax base; land improvements; and implementation of community development programs; minor impact to herders; increased land disturbance; increased risk to human health and safety. Long-term: Improved roads and public infrastructure; improved general economy in the region.

Table 21-1
Summary of Net Environmental and Social Impacts

Environmental Parameter	Potential Gross Impacts	Mitigation Measures	Potential Net Impacts
(cont) Radiological	Radon emanation; Release of radioactive particulates; Accidental spills; Mobilization of materials (e.g., uranium) into groundwater; Vegetation uptake of radionuclides; and Human and wildlife exposure to radionuclides.	Restrict access to the Project area; Line water facilities with appropriate materials to prevent seepage to groundwater; Monitor water management facilities; Suppress dust; Protect stockpiled topsoil for use in reclamation; Construct effective water management features; Implement the RPP; Reduce and monitor employee exposure; Provide training, guidance and PPE to employees; Plan comprehensive transport, shipping containers, and accident response; Utilise up-to-date mining and reclamation techniques; Upon closure, reduce radiation exposure from the Project activities and from the previous un-reclaimed mining activities by others through soil cover and reclamation measures; and Monitor environmental parameters through the Project life.	Short-term: Restricted access; increased human radiological exposure; increased potential wildlife exposure to radionuclides; increased potential vegetation uptake of radionuclides; radon emanation; increased radioactive particulate release. Long-term: Restricted access; land improvement due to reclamation of previous mining impacts; reduced radiation exposure to wildlife due to soil cover from Khan's reclamation of previous mining activities by others at the site (although natural levels of radiation will remain in this area).

(d) Residue Disposal

(i) It is feasible to develop a residue management area and a polishing pond within the Project licence boundary. However, if Khan could get access to land outside the current license area, other off-site locations should also be considered to minimise containment dam requirements.

(ii) The RMA upstream dam will be lined with a composite liner that will extend down through the colluvium layer and tie into the saprolite layer. The low permeable saprolite layer beneath the permeable colluvium layer is considered to be sufficient seepage barrier at the RMA basin. Further investigation, however, is recommended during the next stage of Project engineering to confirm the reliability and extent of the layer as a seepage barrier.

(e) Water Management

(i) Flow modelling indicates that prior to commencement of open-pit mining, the existing Open Pit Lake can provide adequate water for the Project operations under mean annual precipitation conditions. After Year 7, additional water will be required from other sources such as groundwater to keep the system in balance after the Open Pit Lake has been depleted. The water required from other sources to run the operations varies between 7 m^3/hr in Year 8 and 25 m^3/hr in Year 15 for mean annual precipitation conditions.

(ii) Water collected in the RMA pond is pumped directly to the mill from a pump barge to meet makeup water requirements. Additional water needed for the process is pumped from the existing Open Pit Lake for the first 7 years when the mine is an underground operation, and then from the Water Collection Pond after Year 7.

(f) Closure Plan

(i) The Closure Plan proposed for the Dornod Uranium Project is developed to best international practices for both existing site infrastructure and any new proposed infrastructure.

22 Recommendations

The Project economics are robust and show a favorable rate of return. The authors recommend that Khan proceed to the next phase of basic engineering and construction. In addition to the work typical of that level of engineering, the authors recommend the following specific items be included.

22.1 Mining (P&E Mining Consultants Inc.)

(a) Place a critical priority on designing radiation mitigation measures for equipment and work rotations for workers.

(b) Immediately finalise the underground preproduction develop contractor tender package and expedite the tendering and awarding process, with careful consideration of work standards and performance guarantees.

(c) Design the training programme for workers in conjunction with the mining contractor.

(d) Design and implement radiation monitoring regimes.

(e) Identify potentially long-lead mining equipment items and expedite purchase.

(f) Investigate equipment leasing opportunities.

(g) Early in the mine design stage, validate the detailed stope design and sequencing to ensure short- and long-term mine plans can be achieved.

(h) Identify other potential Chinese suppliers, critically assess capabilities and select and negotiate detailed delivery orders (with comprehensive specifications lists), where applicable and appropriate.

(i) Undertake full geotechnical review for No. 2 Deposit 2 to 3 years before underground mining ceases.

(j) Investigate possibility of waste rock storage on adjacent properties.

(k) Investigate possibility of expanding pit onto adjacent properties.

22.2 Processing (Aker Solutions)

(a) The ore samples tested to date from the No. 7 Deposit have shown considerable variability in terms of their metallurgical response to acid leaching, their ability to be thickened and in the energy consumption required in the milling process. Additional representative samples need to be taken from the ore deposit to confirm the basic parameters used in the DFS. After dewatering, the underground mine additional bulk sample can be taken. These samples should be selected so that they represent various rock units within the deposit, but so that preference is given to material to be mined during the earlier years. These samples need to undergo leaching under standard conditions to confirm liberation size, leach time, acid and oxidant requirements, and the leach recovery.

(b) Additional SAG mill work index testing is required to confirm mill power requirements, mill size and the necessity for a pebble crusher in the milling circuit. This will require about 50 samples from various locations within the deposit.

(c) Testwork has previously been carried out by the Russians on the No. 2 Deposit which has to be confirmed. This needs to be done before basic engineering to indicate the correct design parameters.

(d) Additional testing is required to confirm the performance of the RIP and elution circuits. This work should be done under the direction of Kemix and Cleanteq, respectively. To date, only one resin has been tested. Resins from competitors should also be screened for use in the plant.

(e) Khan should produce some yellowcake under the conditions specified in the DFS and send this to potential purchasers to confirm that trace elements conform to their requirements.

(f) Additional work should be done with Pasteteck to see if thickened tailings or paste backfill can be produced with the Dornod tailings. This will improve the water balance and reduce the volume of the tailings dam. This work should be done to confirm the need for the tailings thickener and, if required, its affect on the water balance.

(g) Optimisation of the types and dosage rates for the various flocculent additions needs to be conducted.

(h) The design criteria specified that the mine had to be situated on land currently owned by the joint venture. This has resulted in a very cramped layout. The tailings dam has had to incorporate additional structures to fit it into the available space. The possibility of acquiring additional land holdings should be considered to improve the efficiency of the design and operations.

(i) The confined space has resulted in the man camp being situated down wind of the tailings area. This has resulted in less than ideal living conditions due to noise from the mine and the emanation of radon progeny and potentially radioactive dust. The camp should be relocated to an area up wind and away from the mine.

(j) A tradeoff study should be done to evaluate the inclusion of a second (standby) disk filter ahead of the leach circuit.

(k) The control philosophy has assumed a limited amount of instrumentation for the Project. This assumption needs to be reassessed and, if thought beneficial, additional control circuitry should be included. Priority should be given to acid and steam addition in the leach circuit and pH control in the various precipitation steps.

(l) A heat exchanger has been included to preheat neutral thickener pulp with leach discharge material. This heat exchanger needs to be able to withstand both abrasion from the solids in the pulp and also from the residual acid. Special attention needs to be given to the materials of construction of this exchanger.

(m) The mixers in the leach circuit and in the RIP circuit need to be able to startup after a power outage and the settling of the solid particles in the tanks. Vendor input, with respect to the design horsepower for this, is required.

(n) It may be possible to replace the Ground Ore Repulp Tank with a shaft type repulper in the Disk filter discharge. This cost saving should be investigated.

(o) An in-line screen to remove trash should be included ahead of the RIP circuit. This would remove wood and plastic which will plug the pump cell screens.

(p) Consideration should be given to the automation of the RIP carousel tank advancing system. It is anticipated that the rotation be changed every 5 to 6 hours. This is a complicated process and is likely to be mishandled from time to time. The additional cost would likely be easily paid for in lost production.

(q) Proprietary elution packages are currently being developed by various Vendors. Some of these claim to be able to produce higher grade and purer elluants. The use of such a system will reduce the size of the downstream equipment. This opportunity should be evaluated.

(r) It may be possible to reduce the cost of the RIP equipment by specifying less expensive materials of construction. The current specification assumes relatively high chloride contents in the solutions. Attempts should be made to quantify the chloride loadings.

(s) Additional efforts should be made to more accurately estimate the extent of annual resin loss. A replacement of 30% of the total inventory per year has been assumed in the operating costs. The value of using silica resistant resin should also be quantified.

(t) Some recently designed plants have incorporated a repulp tank ahead of the yellowcake centrifuge. This is done to reduce the impurities in the product. Purchasers of the uranium oxide should quantify this requirement.

22.3 Geotechnical (Golder)

(a) Piezometers should be installed in the current proposed plant site area as part of detail design to check actual groundwater conditions near and around the plant site, and to define the seasonal (and yearly) variations of the groundwater table.

(b) It is recommended that the colluvium be stripped from the plan limits of the proposed plant structures prior to engineered fill placement and / or foundation construction.

(c) Heavy proof rolling of the rough grade level should be included as part of earthworks to expose more competent portions of the saprolite and / or foundation construction.

(d) For raft foundations and spread footings founded on the very stiff to hard, native saprolite soils, a maximum allowable bearing pressure of about 200 kPa is recommended for preliminary design of the structures in the plant site area, assuming that all rafts / footings are founded at least 1.5-m below the final adjacent ground surface.

(e) For raft foundations and spread footings on silty clay engineered fill, a maximum allowable bearing pressure of 150 kPa is recommended to design. Where foundations are constructed directly on the properly prepared bedrock, an allowable bearing pressure of 2500 kPa may be assumed for preliminary design assuming that all loose, shattered and / or fractured rock within the footprint of the foundations is removed and replaced with mass concrete prior to construction.

(f) Static slope stability analyses were carried out to assess the stability of the proposed ore stockpile at the south end of the plant. The stockpiles were assumed to have a diameter of 20 m and a height of about 7 m (based on an angle of repose of 35º for the crushed ore). For this geometry, a factor of safety of about 1.2 to 1.3 is estimated to the global stability of the piles, based on the limited soils information currently available. A more detailed assessment of the stability of the ore stockpile(s) will be required as part of detail design.

(g) Information regarding the location(s) and design (i.e., height) of the proposed WRSF has not been provided to Golder. Based on the GIS information collected by Golder, the existing waste dump piles are considered to be between about 15- to 20-m high, and constructed with a side slope, angle of approximately 45º (i.e., the estimated angle of repose of the waste rock). New waste rock storage piles should be constructed with the same side slopes to no more than 1.5 m in height. If heights in excess of 15 m are required, a site-specific geotechnical investigation is recommended to assess the stability of the new waste rock storage piles.

(h) The elevations of the bedrock surface should be defined across the plant site, and the proposed rough grade level of the site adjusted (or stepped) accordingly to minimise the amount of overburden cutting, bedrock ripping (or blasting), and filling required.

(i) The thickness and geotechnical properties of the saprolite in the plant site area will also need to be better defined. By carrying out additional in-situ testing and laboratory testing on samples obtained from within the plant site area, the shear strength and stiffness can be better assessed, which could lead to an increase the allowable bearing pressure (and therefore decrease in the cost of foundations) for footings founded directly on the native saprolite soils.

22.4 Water Management Plan (Golder)

(a) Perform surface water quality and groundwater quality modelling to observe seasonal variations during all of the stages of the Project.

(b) Monitor the seepage and water quality into the open pit during operations, in order to refine availability of water and water treatment alternatives, if required.

(c) Perform proper hydrogeological and water quality investigations during dewatering of underground workings and operations, in order to have a great chance of getting better information and refine the water balance.

(d) Perform field investigations along different watershed around the Project site, In order to identify possible sources of water.

(e) Investigate void ratio of the deposited residue.

22.5 Residue Management Area (Golder)

(a) Additional geotechnical and thickening testing on the residue.

(b) Detailed geotechnical investigations and in-site testing on the RMA basin to define the extent and the reliability of the saprolite layer as a seepage barrier.

(c) Geotechnical testing on the waste rock that will be used for the construction of the containment dam and perimeter dikes.

(d) Development of a seepage model for the RMA that incorporates the hydrogeology of the site and geochemistry of the residue.

(e) Assessment of risk associated with Shaft No. 2 on the planned underground operations.

(f) Evaluation of the option of commingling the residue with the waste rock.

(g) Investigation of the alternative negotiating land adjacent to the property boundary to minimise capital cost required to construct the perimeter dikes.

22.6 Environmental and Social Programs (AATA)

(a) Continue existing baseline sampling programs on air quality, surface water and groundwater.

(b) Perform air quality modelling with more detailed Project information.

(c) Start hi-vol sampling at the future plant site, downwind and upwind along Project boundaries.

(d) Evaluate other locations for the mine camp (e.g., to the NW corner of the land use permit area).

(e) Line water management facilities to prevent seepage to groundwater, especially for the RMA (recommend double liner on the basin floor as well).

(f) Allow at least 2 years before capping the RMA to dry the residues (revise current Project closure plan accordingly).

(g) Construct a landfill separately (from the RMA).

(h) Locate licenced hazardous waste disposal facility.

(i) Conduct a survey on regional industrial activities (e.g., closest industrial sites, type of business, amount of emission / effluent discharge; potential regional impacts etc.).

(j) Select native seed mixtures for revegetation and control invasive species.

(k) Restrict access to the Project area for reasons of health and safety.

(l) Perform public consultation and disclosure at least annually.

(m) Evaluate and implement the ESMP.

(n) Expand the Radiation Protection Plan.

22.7 Conceptual Closure Plan (Golder)

(a) Further geochemical characterisation of the open-pit walls, waste rock and residue during operations, in order to better gauge their potential for metal leaching.

(b) The construction of trial cover plots, for both the RMA and WRSF, of varying materials and thicknesses, in order to optimise the design to best promote vegetation, while inhibiting infiltration, and diffusion of radon gas and gamma radiation.

(c) Waste rock seepage quality should be monitored during operations, in order to better refine the post-closure water quality monitoring program (a potential cost-saving opportunity upon closure).

(d) Water quality of the RMA pond should be periodically monitored during operations, in order to gauge whether lime treatment upon closure will be necessary before construction of the overflow spillway.

(e) Further geotechnical testing to gain a better understanding of the subsurface extent of the saprolite layer in the RMA.

(f) Performance of the saprolite in the RMA as impermeable barrier should be monitored, with respect to seepage.

(g) The backfilling of the ramp / portal and vent raises, along with the capping of Shaft No. 3 upon completion of underground mining activities will reduce the overall closure liability at the end of mine life.

23 References

AATA International Inc., 1995, Environmental Due Diligence Report, Dornod Uranium Mine, Mongolia: Report Prepared for WM Mining, Denver, Colorado, August 1995.

Aker Kvaerner, 2007, Dornod Uranium Project – Prefeasibility Study Report: prepared for Khan Resources Inc., August 2007.

Agnerian, H., Heymann, L.; (2006) Technical Report on the Preliminary Assessment of the Dornod Uranium Project Mongolia, for Khan Resources Inc. October 25, 2006.

Agnerian, H., 2006, Report on the Dornod Uranium Property, Mongolia: Scott Wilson Technical Report for Khan Resources Inc., June 30, 2006.

Agnerian, H., 2004, Report on the Dornod Uranium Property, Mongolia: Scott Wilson Technical Report for Khan Resources Inc., December 29, 2004.

Blake, H.J., 2004, Report to Khan Resources Inc. Re Acquisition of Uranium Property: McLean & Kerr LLP, Toronto, October 10, 2004.

Brel, A.I., Melnik, N.A., Panin, V.F. and Zizdo, G.K., 1988?, Certificate, Technological Sample No. 150: Internal Report by Scientific Institute Research Laboratories and Sosnovsk Production and Mining Association.

Bruce, Don, 1998, Dornod Mine Report No. 7 – Underground Mine, Mining Plan and Costs.

Diatchkov, S.A., 1994, Principles of Classification of Reserves and Resources in the CIS Countries: Mining Engineering, pp. 214-217.

Eco-Trade, 2006. Environmental Baseline Study, Dornod Uranium Mine, Mongolia. Ulaanbaatar, 2006.

Galsan, J., 2004, Personal Communication.

Golder, 2009a, Dornod Uranium Project Geotechnical Field Investigations. Draft Report 07-1118-0023 (4000) by Golder Associates Ltd. prepared for Khan Resources Inc. (in progress).

Golder, 2009b, Preliminary Foundation Recommendations – Plant Site and Waste Dump, Technical Memorandum 07-1118-0023 (8000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated January 27, 2009.

Golder, 2009c, Preliminary Site Wide Water Balance Results – Rev C, Dornod Feasibility Study, Technical Memorandum 07-1118-0023 (7003) by Golder Associates Ltd. prepared for Khan Resources Inc., dated January 22, 2009.

Golder, 2009d, Dewatering Test Work on the Un-neutralized Dornod Tailings (Acid leach solution and leach slurry), Dornod Uranium Project, Draft Report 07-1118-0023 (11000) by Golder Paste Tec prepared for Golder Associates Ltd., dated January 19, 2009.

Golder, 2009e, Dornod Uranium Project Residue Management Area, Final Report – Version 4.0, 07-1118-0023 (6000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated March 13, 2009.

Golder, 2009f, Dornod Uranium Project Water Management Plan, Final Report – Version 4.0 07-1118-0023 (7000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated March 13, 2009.

Golder, 2009g, Dornod Uranium Project Conceptual Closure Plan, Final Report, Version 4.0, 07-1118-0023 (9000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated March 13, 2009.

Golder, 2008a, Dornod Project: Final Baseline Report 07-1118-0023 (2000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated August 15, 2008.

Golder, 2008b, Dornod Uranium Project: Development of deterministic peak ground accelerations for preliminary analysis design of tailings dam. Technical Memorandum 07-1118-0023 (2160) by Golder Associates Inc. prepared for Golder Associates Ltd., dated November 26, 2008.

Golder, 2008c, Dornod Uranium Project: Summary of Dewatering Testing Results Including Path Forward for Future Testing. Technical Memorandum 07-1118-0023 (RCN1) by Golder Paste Technology prepared for Golder Associates Ltd., dated June 5, 2008.

Golder, 2008d, Residue Management Area Siting Study – Dornod Uranium Project, Technical Memorandum 07-1118-0023 (5003) by Golder Associates Ltd., prepared for Khan Resources Inc., dated April 18, 2008.

Golder, 2008e, Dornod Project: Alternative Residue Management Area Configurations, Technical Memorandum 07-1118-0023 (6000) by Golder Associates Ltd., prepared for Khan Resources Inc., dated December 5, 2008.

Golder Memo 07-1118-0023, Impact of Modifications to Scoping Study Recommendations – March 2008.

Golder, 2006a. Review of the Dornod Environmental Baseline Study Report. July 2006. Ref. No. 05-1118-041.

Golder Report 05-1118-041, Mine Geotechnical Underground Design for Dornod Project, Mongolia, Golder 2006.

Golder, 2006b. Dornod Uranium Project Screening Report. December 18, 2006. Ref. No. 05-1118-041A.

Golder 2006c. Mine Geotechnical Underground Design for Dornod Project, Mongolia. September 2006. Ref. No. 05-1118-041.

Golder, 2005a. Radioactivity Data Review - Dornod Uranium Deposit. December 5, 2005. Ref. Nr. 05-1118-041.

Harper, G., 2005, Saddle Hills area uranium prospects, Exploration Licences 3367X, 4846X, 4969X, 4970X, 4974X, 6995X, 7023X, 7405X, 7685X and 9283X, Dornod Aimag, eastern Mongolia: Report for Western Prospector Group Ltd., May4, 2005.

Intergen Safety and Environment Solutions Inc. titled “Dornod Underground Ventilation Modeling” March, 2009.

Khan Resources Inc., September 11, 2007, August 23, 2007, August 17, 2007. Company Press Releases regarding Special Exploration Licence 9282X.

Khan Resources Inc., 2004, Miscellaneous Technical Information.

Kiselev, V.Y., Samoilenko, Y.A., Pelmenev, M.D., Gubkin, G.N., Trinikov, I.P., Rubtsov, G.V., Avtsyev, B.V. and Shleyder, V.A., 1982, Protocol No. 8975, Dornod Uranium Deposit, Mongolia (in Russian): USSR Ministry of Geology, Irkutsk, March 29, 1982.

Kita, J., 2006, Personal Communication.

Kryukov, V.G. and Stolpovitsky, V.E., 1997, Dornod Project, Site Plan and Infrastructure (List 2): Priargunsky Kombinat, 1997.

Lynch, M.M., 2004, Personal Communication with Lynch & Mahoney, Law Offices, October 2004.

MacCormack, 1998a, An Addendum to the Report on the Geology and Mineral Resource Estimates of the Dornod Mine Project, Mardai, Northeastern Mongolia: Scott Wilson RPA Report prepared for World Wide Minerals Limited, February 19, 1998.

MacCormack, 1998b, Report on the Geology and Mineral Resource Estimates of the Dornod Mine Project, Mardai, Northeastern Mongolia: Scott Wilson RPA Report prepared for World Wide Minerals Limited, January 9, 1998.

Mariano, A.N., 2007, Report on a Search for Uranium Mineralization in the Dornod Uranium Deposit of Mongolia.

Mining Journal, 1998, Uranium in Mongolia: Mining Journal, July 10, 1998.

Ministry of Nature and Environment of Mongolia. 1997. Mongolian Red Book. Ulaanbaatar. 388 pp.

Miselev, R.Y., Trikilov, I.M., Bilomitsov, Y.A., Samovlenko, R.A. and Rogov, Y.R., 1985, Report on the Mineral Resources of Zones 2б and 2B, Dornod Uranium Deposit: USSR Ministry of Geology (in Russian), January 2, 1985.

MNS 900:2005 (Mongolian National Standard), Environment, Health Protection, Safety: “Drinking Water: Hygienic Requirements and Quality Control, Ulaanbaatar, Mongolia”, 2005.

National Atlas of Mongolia. 1990. National Atlas of Mongolia. Moscow, Academy Press.

Pecherkin, Y. and Solovyev, N.S., 1983, Regional Geological Map of Dornod Aimag: Ministry of Geology, U.S.S.R, 1983.

Rogov, I.G. and Yurchenko, I.A., 1988, Report on Mineral Reserves of Dornod Uranium Deposit (in Russian): Russian-Mongolian Geological Expedition, USSR Ministry of Geology, Irkutsk, 1988.

Rogov, I.G. and Yurchenko, I.A., 1987, Gurvanbulakskoye and Dornotskoye Uranium Deposits, Report on Results of Mineral Reconnaissance and Reserve Estimate of January 7, 1987, 6 Volumes (in Russian): Russian-Mongolian Geological Expedition, USSR Ministry of Geology, Irkutsk, 1987.

Scientific Research Institute VIMS Laboratories, 1998?, Report No. 209 on Vein No. 7 of the Dornod deposit, Mongolia.

SGS, An investigation into THE RECOVERY OF URANIUM FROM THE DORNOD DEPOSIT prepared for AKER SOLUTIONS / KHAN RESOURCES Project 11516-004 – Report No. 1 January 14, 2009

Shacklette, H. and Boerngen, J. 1984. Element Concentration in Soils and Other Surficial Material of the Conterminous United States. Washington (DC): United States Geological Survey (USGS) Professional Paper #1270. 105 pp.

Tatarnikov, A.A., Hikovich, I.M. and Komarova, T.M., 1988, Instructions Required to Determine the Specific Gravity in Ore by Gamma Method, Ministry of Geology, U.S.S.R. (in Russian).

Thompson, J.T., 2004, Personal Correspondence.

Thompson, J.T., 2002, Dornod Uranium Mine, Eastern Mongolia, Open Pit Mine #2, Mining Study and Economic Analysis; Summary and Update to Report Entitled “Dornod Uranium Mine, Eastern Mongolia, Mining Study and Economic Analysis” by World Wide Uranium Holdings Inc.: John Thompson and Associates, July 8, 2002.

Thompson, J.T., 2000, Dornod Uranium Mine, Eastern Mongolia, Scoping Study and Economic Analysis: Report for World Wide Uranium Holdings Inc. by John Thompson and Associates, April 30, 2000.

Tserenmur, D., 2004, Personal Communication.

UNSC, 1993. United Nations Scientific Committee on the Effects of Atomic Radiation- Report to the General Assembly, with scientific annexes, New York (http://en.wikipedia.org/wiki/Uranium_in_the_environment).

Western Prospector Group Ltd., 2006a, Western Prospector Buys out Option Agreement on Saddle Hills Project Licence: Western Prospector Group Ltd. Press Release of June 21, 2006.

Western Prospector Group Ltd., 2006b, Moving Forward with the Saddle Hills Uranium Project: Western Prospector Group Ltd. Press Release of June 26, 2006.

World Wide Uranium Holdings Inc., 2002, Dornod Uranium Mine, Eastern Mongolia, Mining Study and Economic Analysis: Internal Company Report, (Re-compiled and Re-formatted June 2002.

World Wide Uranium Holdings Inc., 2002, Dornod Uranium Mine, Eastern Mongolia, Mining Study and Economic Analysis: Internal Company Report, April 20, 2002.

World Wide Uranium Holdings Inc., 1998, Dornod Uranium Mine, Eastern Mongolia, Summary of Mining Study and Economic Analysis: Internal Company Report, April 16, 1998.

World Wide Minerals Ltd., 1995, Feasibility Study, Dornod Open Pit Uranium Mine, Erdene/Mardai, Mongolia: Internal Company Report.

24 Date and Signature Page

This report titled “Technical Report (NI 43-101) on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia” prepared for Khan Resources Inc., effective April 22, 2009, was prepared and signed by the authors.

Dated at Toronto, Ontario

April 22, 2009 (SIGNED)

Hrayr Agnerian, M.Sc. (Applied), P.Geo.
Scott Wilson Roscoe Postle Associates Inc.

Dated at Brampton, Ontario

April 22, 2009 (SIGNED)

Eugene Puritch, P.Eng.
P&E Mining Consultants Inc.

Dated at Brampton, Ontario

April 22, 2009 (SIGNED)

Malcolm Buck, P.Eng.
P&E Mining Consultants Inc.

Dated at Penetanguishene, Ontario

April 22, 2009 (SIGNED)

Leslie H. Heymann, P.Eng.
Senior Process Engineer
Aker Solutions

CERTIFICATE of AUTHOR

I, Hrayr Agnerian, P.Geo., Suite 501, 55 University Avenue, Toronto, ON, M5J 2H7**,** as an author of this Report entitled “Technical Report on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia”, prepared for Khan Resources Inc., and dated April 22, 2009, do hereby certify that:

1. I am an Associate and a Consulting Geologist with Scott Wilson Roscoe Postle Associates Inc. of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7.

2. I am a graduate of the American University of Beirut, Lebanon in 1966 with a Bachelor of Science degree in Geology, of the International Centre for Aerial Surveys and Earth Sciences, Delft, the Netherlands, in 1967 with a diploma in Mineral Exploration, and of McGill University, Montreal, Quebec, Canada, in 1972 with a Masters of Science degree in Geological Engineering.

3. I am registered as a Professional Geoscientist in the Province of Ontario (Reg. #0757) and Saskatchewan (Reg. #4305) and as a Professional Geologist in the Province of Quebec (Reg. #302). I have worked as a geologist for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Review and report as a consultant on more than 75 mining operations and projects around the world for due diligence and regulatory requirements, including:

     – Technical review of two uranium mining areas, Kazakhstan, for World Wide Minerals Limited

     – Monitoring and management of numerous uranium joint venture exploration projects in Saskatchewan (1977 – 1987)

     – Preparation of a Technical Report on the Berezitovoye Gold and Silver Deposit and Mineral Resource Estimate, Amur Oblast, Russia (Siberia), for High River Gold Mines Ltd.

     – Estimate of the Mineral Resources of the Anoki deposit and Anoki South Zone in the Kirkland Lake area, for Queenston Mining Inc.

     – Preparation of a Technical Report on the Salave Gold Project, Spain, for Rio Narcea Gold Mines Ltd.

     – Preparation of a Technical Report on the Volta Grande Gold Project for Verena Minerals Corporation

     – Audit of the Mineral Resources of the Boston, Doris and Madrid areas of the Hope Bay Gold Project, Nunavut Territory, for BHP Minerals Canada

     – Audit of the Mineral Resources of the Joe Mann Mine, Québec, for Campbell Resources Inc.

     – Mineral Resource estimate of the Holloway Zone, Kirkland Lake area, for Freewest Resources Ltd.

- District Geologist with a major Canadian mining company, responsible for project management and monitoring of several uranium and rare earth projects in the Athabasca basin
- Project Geologist in charge of base metal exploration programs in northern Québec and Newfoundland
- Exploration Geologist with a number of Canadian mining companies in charge of base metal exploration in Québec, Alberta and NWT.

4. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

5. I visited the Dornod Property October 4, 2004, and from December 19 to 20, 2005.

6. I am responsible for preparation of Items 7 to 17, and parts of Items 6 and 19 (covering property location, leases and Mineral Resources) of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of National Instrument 43-101.

8. My prior involvement with the property consists of three Technical Reports for the issuer dated December 29, 2004, June 30, 2006, and October 25, 2006.

9. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

10. To be best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective date: September 27, 2007

Signing Date: April 22, 2009

{SIGNED AND SEALED}

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Hrayr Agnerian, M.Sc. (Applied), P.Geo.

CERTIFICATE of AUTHOR

I, Eugene J. Puritch, P. Eng., residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am President of P&E Mining Consultants Inc.

2. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for Bachelor's Degree in Engineering Equivalency. I have practiced my profession continuously since 1978. My summarised career experience is as follows:

   - Mining Technologist - H.B.M.&S. and Inco Ltd. 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine 1984-1986
   - Self-Employed Mining Consultant – Timmins Area 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator 1995-2004
   - President – P & E Mining Consultants Inc. 2004 - Present

3. I am a mining consultant currently licensed by the Professional Engineers of Ontario (License No. 100014010) and registered with the Ontario Association of Certified Engineering Technicians and Technologists as a Senior Engineering Technologist. I am also a member of the National and Toronto CIM.

4. I authored or co-authored portions of Items 3, 4, 5, 19, 20, 21 and 22, of the Technical Report entitled "Technical Report (NI 43-101) on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia" and dated April 22, 2009.

5 I visited the Dornod site on November 14, 2006 and January 16 to 17, 2008.

6. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

7 I am an independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

8. I have had prior no prior involvement with the Dornod property that forms part of the subject of this Technical Report.

9. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

10. I am a “Qualified Person” for the purposes of NI 43-101 due to my experience and current affiliation with a professional organization (Professional Engineers of Ontario) as defined in NI 43-101.

Effective date: March 11, 2009

Signing Date: April 22, 2009

{SIGNED AND SEALED}

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Eugene Puritch, P.Eng.

CERTIFICATE of AUTHOR

I, Malcolm Buck, M.Eng., P. Eng., residing at 164 Castle Crescent, Oakville, Ontario, Canada do hereby certify that:

1. I am a Senior Associate Mining Engineer at P&E Mining Consultants Inc..

2. I am a graduate of The Technical University of Nova Scotia, with a Bachelor of Engineering in Mining Engineering (1983). I have also obtained a Masters of Engineering, in Mining Engineering (Mineral Economics) from McGill University (1986). My summarised career experience is:

   - Practiced my profession continuously since 1983.

   - Extensive and progressively more senior operational and engineering duties at base metals, gold and uranium mining operations and development projects.

   - 15 years experience performing all types of feasibility studies and due diligence and strategic planning studies for mines and mining companies.

3. I am licensed by the Professional Engineers Ontario (License No. 5881503). In addition, I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I authored or co-authored portions of Items 3, 4, 5, 19, 20, 21 and 22, of the Technical Report entitled "Technical Report (NI 43-101) on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia" and dated April 22, 2009.

5. I visited the Dornod site on January 30, 2005.

6. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

7. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Effective date: March 11, 2009

Signing Date: April 22, 2009

{SIGNED AND SEALED}

____
Malcolm Buck, P.Eng.

CERTIFICATE of AUTHOR

I, Leslie H. Heymann, Suite 400, Davisville Centre, 1920 Yonge Street, Toronto, Ontario M4S 3E2, do hereby certify that as author of the processing section contained in the report entitled “Technical Report NI 43-101 on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia” for Khan Resources Inc. hereby make the following statements

1. My name is Leslie Harold Heymann and I am a Senior Process Consultant employed by Aker Solutions. My office address is Suite 400, Davisville Centre, 1920 Yonge Street, Toronto, Ontario, M4S 3E2.

2. I am a graduate in Chemical Engineering of the University of the Witwatersrand, Johannesburg, South Africa [B.Sc. (Chem Engl) 1969].

3. I am registered as a Professional Engineer in the Province of Ontario. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

4. I am a Qualified Person for the purposes of National Instrument 43-101.

5. This Report is based on my personal review of Technical Reports and other data supplied by the Issuer, on discussions with the Issuer and its representatives, and on information available from technical publications.

6. My relevant experience for the purpose of this Report is:

    - I have worked in the production end of the uranium industry between 1970 and 1981; this work included the metallurgical management of four uranium plants and the management of one underground uranium mine

    - I have published several documents in my field.

7. I have been practicing as a Professional Engineer in both Canada and South Africa for over 25 years.

8. I authored portions of Items 3,4 5, 18, 21 and 22 of the “Technical Report (NI 43-101) on the Definitive Feasibility Study for the Dornod Uranium Project, Mongolia”, dated April 22, 2009.

9. I am not aware of any material fact or material change with respect to the subject matter of this Report, which is not reflected in the Report the omission to disclose which makes this Report misleading.

10. I am independent of the Issuer.

11. I have read NI 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1, Technical Report and Companion Policy 43-101CP, and this Report has been prepared in compliance with NI 43-101.

Signing Date: April 22, 2009

{SIGNED AND SEALED}

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Leslie H. Heymann, P.Eng.

25 Additional Requirements for Technical Reports on Development and Production Properties

26 Illustrations