APPENDIX IX.3

PREDICTED QUANTITY OF WATER DISCHARGED FROM THE SNAP LAKE DIAMOND PROJECT

PREDICTED QUANTITY
OF WATER DISCHARGED FROM THE
SNAP LAKE DIAMOND PROJECT

Sections for inclusion to

GEOTECHNICAL AND HYDROLOGICAL INVESTIGATION
FOR
OPTIMISATION OF SNAP LAKE MINE DESIGN

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TABLE OF CONTENTS

Page

LIST OF FIGURES

LIST OF TABLES
(in text)

LIST OF APPENDICES

1.0 HYDROGEOLOGY

1.1 INTRODUCTION

This section summarises the methodology and findings of the hydrogeologic portion of the investigation conducted by Hydrologic Consultants, Inc. of Colorado (HCI) to predict the quantity of water discharge from the Snap Lake Diamond Project. To predict the quantity of discharge, a fully three-dimensional, finite-element, ground-water flow model of the proposed Snap Lake Diamond Project and surrounding area was developed. The model was used to define the potential amount and distribution of ground-water inflow to the various mine workings over time under the currently proposed mine plan (Base Case).

1.2 EXISTING HYDROGEOLOGIC DATA

Data and information provided by De Beers Mining Canada, Inc. (De Beers), AMEC Simons Mining and Metals (AMEC), and other members of the Snap Lake Underground Group, most notably Golder Associates Ltd. (Golder), were used by HCI to develop a conceptual hydrogeologic model of the mine area and to construct the ground-water flow model. The most relevant information and data included:

- the location and topographic setting of the proposed mine (shown in Figure 1);
- the areal geology (Stubley, 2000; provided by De Beers);
- a select number of geologic cross-sections from the *GEMCOM* geologic block model of the Snap Lake Diamond Project area (provided by De Beers);
- proposed mine plans (provided by AMEC);
- the observed ground-water discharge to the initial exploration drift in 2000 (Golder, 2001); and,
- discharge rates and hydraulic conductivity data from hydraulic testing (under the direction of Golder) in the exploration drift as part of the 2001 Advanced Exploration Program (AEP).

The conceptual hydrogeologic model of the Snap Lake Diamond Project area is shown in cross-section in Figure 2. The primary hydrogeologic units of interest, from shallowest to deepest, are:

- Metavolcanics – part of the Yellowknife Supergroup that comprises a relatively small portion of the country rock in the area to be mined;
- Granite – a series of leucogabbros to anorthosites that comprise the vast majority of the country rock in the area to be mined;
- Kimberlite – the diamondiferous host rock occurring as a dyke approximately 1.5 to 2.5 m thick and dipping to the east-northeast at about 14°; and,
- Lakebed sediments – a thin veneer of till and organic materials on the bottom of Snap Lake.

Figure 2 also indicates schematically the assumed depth of relaxation (or exfoliation) due to post-glacial unloading, the depth of permafrost beneath land, and the zone of subsidence that will develop above the mined area (to be discussed in more detail in Section 2.4).

Hydraulic testing of eight drillholes was conducted during the 2001 AEP to estimate the hydraulic conductivity of the granite and metavolcanics rock units (Figure 3). Two types of hydraulic tests were conducted in the testholes: 1) discharge (or flow recession) and 2) shut-in recovery. The estimates of hydraulic conductivity derived from these field data are summarised in Appendix A.

The hydraulic properties of all of the hydrogeologic units and leakance factors used in the ground-water flow model, based either on the results of field tests or assumed, are summarised in Table 1. A discussion on the estimation of the leakance factor is provided in Appendix C.

Figure 4 indicates how the values for hydraulic conductivity incorporated into the model honour the limited field data and compare to previous estimates by Golder (1999, 2000, and 2001) and typical values from the literature (the green bars in Figure 4).

Table 1. Hydraulic Properties of Hydrogeologic Units and Leakance Factors Used in the Ground-Water Model

Hydraulic Properties								
Unit	**Material**	**Hydraulic Conductivity (m/day)**			**Specific Yield**	**Specific Storage**	**Uncertainty Factor**	**Notes**
		K_x	K_y	K_z	S_y ( )	S_s (m^{-1})		
1	Granite – exfoliated	2×10^0	2×10^0	2×10^{-2}	0.01	1×10^{-5}		1,2
2	Granite - hanging wall	2×10^{-1}	2×10^{-1}	1×10^{-2}	0.005	1×10^{-6}	10, 0.1	3
3	Granite – footwall	4×10^{-2}	4×10^{-2}	1×10^{-2}	0.005	1×10^{-6}	10, 0.1	
4	Metavolcanics – exfoliated	2×10^{-1}	2×10^{-1}	3×10^{-1}	0.01	1×10^{-5}		2
5	Metavolcanics - hanging wall	2×10^{-2}	2×10^{-2}	5×10^{-3}	0.005	1×10^{-6}		3
6	Metavolcanics – footwall	4×10^{-3}	4×10^{-3}	1×10^{-3}	0.005	1×10^{-6}		
7	Kimberlite Dyke	5×10^{-4}	5×10^{-4}	1×10^{-4}	0.005	1×10^{-6}		
8	Backfill Material	1×10^0	1×10^0	2×10^{-1}	NA	NA	10, 0.1	4
9	Layer Above Backfill	2×10^0	2×10^0	2×10^0	0.005	1×10^{-4}	10, 0.1	5
NA	Lakebed Sediments	1×10^{-3}	1×10^{-3}	1×10^{-3}	NA	NA	10, 0.1	6

Leakance Factors			
Material / Location	**Leakance Factor (m^2/day)**	**Uncertainty Factor**	**Notes**
Mine Areas	0.3	10, 0.1	7
Drifts in footwall rock	0.1	10, 0.1	7
Drifts in ore rock	0.15		7
Lakebed Sediments	0.1 to ~1800 (average ~ 14)	see above	6, 8

Notes:

1. Based on shut-in recovery tests in testhole UG-45 and UG-176.
2. Zone of exfoliation (due to glacial rebound) is most likely very gradational, but in the model it is assumed to occur abruptly at a depth of 50 m.
3. Will increase K_z of rock above mine blocks and below neutral axis by a factor varying from 3.3 to 10 after block is extracted.
4. The nominal specific yield of the backfill is very low because the paste backfill will be emplaced as a slurry very near saturation.
5. Arbitrarily assumed to be equivalent to exfoliated granite (i.e., highest value of rock hydraulic conductivity), but isotropic.
6. The lakebed sediments are not explicitly represented in the model. They are simulated in the model by the leakance factor applied to the constant head nodes at the bottom of the lake.
7. Estimated and assigned globally to all applicable drain nodes.
8. Explicitly calculated (dependent on discretization, among other factors) and assigned per node.
9. The depth of permafrost beneath land is assumed to be 50 m. It is assumed that permafrost is essentially impermeable and the top of the ground-water model is simply 50 m deeper in those areas.

1.3 NUMERICAL GROUNDWATER FLOW CODE USED IN STUDY

The groundwater flow model constructed for this investigation utilised the numerical code *MINEDW* developed by HCI that solves three-dimensional ground-water flow problems with an unconfined (or phreatic) surface using the finite-element method. *MINEDW* has several special attributes that were specifically developed to address conditions unique either is or is not unique (kind of like half-pregnant) to mine dewatering (Azrag et al., 1998). For example, a new feature was added to *MINEDW* to handle the special technique of mining in Snap Lake Diamond Project and the development of a relaxation zone above the mining panel areas.

2.0 GROUNDWATER FLOW MODELING

The following section describes the construction and limited calibration of the numerical groundwater flow model of the Snap Lake Diamond Project area.

2.1 MODEL GRID AND DISCRETISATION

Utilising the finite-element method, the three-dimensional model domain for the Snap Lake Diamond Project groundwater flow model has been subdivided into many smaller sub-volumes or elements. The elements are triangular prisms; and the corners of the prisms are points referred to as nodes. The hydraulic properties are assigned to the elements representing the various hydrogeologic units, and the model-calculated water levels and flows are associated with the nodes. Therefore, every element in the numerical model is associated with a hydrogeologic unit that has specified values for the hydraulic properties of horizontal (K_x and K_y) and vertical (K_z) hydraulic conductivity, specific storage (S_s), and specific yield (S_y). These values are summarised in Table 1.

The finite-element model for the Snap Lake Diamond Project encompasses approximately 89 km^2 in the vicinity of Snap Lake. The grid for the model is composed of 134,652 elements and 71,792 nodes (Figure 5). The finest discretisation is within the proposed mine footprint, where the average size of the elements is about 600 m^2.

Figure 5 also indicates the hydrogeologic units simulated in the uppermost layer of the model. It is assumed that the hydraulic properties of any specified hydrogeologic unit are consistent and invariant within the model domain. The vertical discretization of the Snap Lake Diamond Project model is shown in the cross-section on Figure 6.

2.2 MODEL BOUNDARIES

The model boundaries of the Snap Lake Diamond Project model were designated as variable-flux boundaries. This type of boundary condition, as it is incorporated in *MINEDW*, simulates infinite hydrogeologic units that have the same hydraulic properties as the units at the boundary. Using an analytical solution, the variable-flux boundary condition calculates the flow across the boundary as the result of calculated changes in ground-water levels at the boundary.

The bottom boundary of the model is located about 125 m below the kimberlite and is assigned a no-flow boundary condition. The upper boundary of the model beneath Snap Lake is a constant head boundary set at 444 mamsl (meters above mean sea level). The upper boundary for the remaining area of the model domain beneath land is a no-flow boundary, simulating the bottom of the permafrost layer, at an elevation of 394 mamsl. The initial water levels for the model area were defined by lake levels in the surrounding area, as described by Golder (D. Chorley, written communication, 2001).

2.3 SIMULATION OF HYDROGEOLOGY

The major hydrogeologic units previously described are represented by individual layers and zones within the model grid (Figures 5 and 6). The model incorporates fifteen layers east of the kimberlite outcrop and two layers west of the outcrop. These layers, from top to bottom, are:

- Layers 1 through 4 - exfoliated granite or metavolcanics;
- Layers 5 through 11 - hanging wall granite or metavolcanics;
- Layer 12 - a 1-m thick layer to represent highly conductive zone between backfill material and back rock after mining has occurred (hydraulic conductivity value changes after the area is mined and backfilled);
- Layer 13 - kimberlite dyke; and,
- Layers 14 and 15 - footwall granite or metavolcanics.

2.3.1 Major Geologic Units

2.3.1.1 Granite and Metavolcanic Units

In areas beneath land, permafrost (which is assumed for all practical purposes to be impermeable) is assumed to occur to a depth of 50 m. In these areas, therefore, the top of the model is 50 m lower than natural ground surface.

It was also assumed that the top approximately 50 m of granite or metavolcanics has experienced relaxation or exfoliation due to glacial rebound effects. Thus, hydraulic conductivity values that are one order-of-magnitude greater than the underlying hanging wall geologic units (Table 1) were assigned to this layer.

2.3.1.2 Lakebed Sediments

The sediments at the bottom of Snap Lake are believed to be composed mainly of organic material with an estimated average thickness of approximately 0.5 m. In the ground-water flow model, the low hydraulic conductivity sediments are represented by a leakance factor calculated using an assumed hydraulic conductivity of 1×10^{-3} m/day. It is also implicitly assumed that the material is sufficiently plastic to flow into any cracks that might propagate to the floor of the lake due to mining-induced subsidence.

2.3.1.3 Kimberlite Dyke

The kimberlite dyke is simulated as a single geologic layer 2.5 m thick. As shown on the cross-section on Figure 8, the dyke crops out to the west. The hydraulic conductivities of the kimberlite are assumed to be 5×10^{-4} m/day (horizontal) and 1×10^{-4} m/day (vertical).

2.3.2 Geologic Structures

The two main geologic structures in the Snap Lake Diamond Project area are the Snap and Crackle Faults (Figure 1). It is assumed that when these faults are encountered during excavation of drifts across them, the fault zones will be effectively grouted. Thus, HCI did not consider it necessary to explicitly represent the Snap and Crackle Faults in the model. We are not aware of any other major structures at this time.

2.3.3 Backfill Material

Mine panels will be backfilled immediately after mining during the development stage of the current mine plan. The nominal backfill will include paste, concrete pillars, and various “cracks” (between the concrete pillars and un-mined rock and possibly between the paste and un-mined rock on the sides and back). We have assigned a horizontal hydraulic conductivity value for the nominal backfill of 1 m/day (Table 1 and Figure 4). The highly conductive zone above the backfill material created by the separation of the fill from the back and/or tensile relaxation of the back is represented by a layer 1 m thick having a hydraulic conductivity of 2 m/day.

2.4 SIMULATION OF MINE PLAN

Figure 7 indicates the general mine areas for the currently planned Base Case scenario that are simulated in the ground-water model. Year-by-year excavations of mine blocks are shown in Figure 8.

Currently, the mine plan includes 20 years of mining beginning in mid-2005. The total area of the mine plan is approximately 505 acres (2,044,300 m^2), with the southern approximately 60% of the footprint under Snap Lake. The mine panels align in a northwest-southeast direction. Mining will begin approximately 300 m down-dip from the western edge of the mine footprint and will continue down-dip for about 16 years. During the last 7 years of the mine life, mining will be occurring both down- and up- dip of the initial mine panel.

The model is first run under steady state conditions to generate a reasonable representation of the regional ground water gradient. Hydraulic stresses in the model are initiated with the beginning of the conveyor drift, north and south ramps, and the ore and haulage drifts 4.5 years before extraction of the first mine block starts. Advance of the various drifts is then simulated along the specified routes to reach the various mine blocks, as designated by AMEC.

A mine block to be extracted in a specified year is sub-divided into twelve “sub-blocks”. In the model, these sub-blocks are mined in sequence such that six sub-blocks are active every month. Six months after mining of a specified sub-block is simulated to begin, it is assumed that the void is backfilled and that subsidence occurs above the area. Both backfilling and subsidence are assumed to occur instantaneously. This is simulated by “turning on” so-called drain nodes within the specified sub-blocks at the specified times. These drain nodes simulate flow from the ground-water system into the mine workings. They are assigned hydraulic heads corresponding to the elevations (assuming pressure is equal to zero) that they represent. A drain node simulates inflow only if the calculated hydraulic heads in the surrounding nodes are higher than the elevation of the drain node. If this is not the case, then the drain node is deactivated. Each drain node is also assigned a leakance factor representing the local resistance to inflow (Appendix C). Drain nodes representing the conveyor, ramps, and haulage drifts in the footwall rock remain active throughout the entire model simulation once they are activated.

Four simultaneous steps occur in the model to simulate the backfilling and subsidence, as described above and shown schematically in Figure 9:

1) The drain nodes representing the sub-block are “turned off”.

2) The hydraulic conductivity of the kimberlite layer is replaced by the hydraulic conductivity of backfill (an increase by a factor of 2,000).

3) The hydraulic conductivity of the zone above the backfill (1 m thick layer representing the increased hydraulic conductivity of the backfill material created by the separation of the fill from the back).

4) In order to simulate the potential effect of subsidence, the hydraulic conductivity of the elements in the hanging wall rock above the sub-block up to the assumed "neutral axis" of deflection is increased starting with a factor of 10 just above the backfill to a factor of 3.3 at the neutral axis. As shown in Figure 6, the neutral axis is currently assumed to be about the mid-point of the hanging wall rock between the kimberlite and the bottom of the exfoliation zone.

2.5 MODEL CALIBRATION

There are very little inflow data and no water level data with which to calibrate the ground-water model. Thus, the only calibration was to:

- The reported inflow in the exploration drift (Figure 3) between July and December 2000 (Golder, 2001) and the inflows to the 2001 AEP development, and

- Estimated outflow to ground water from Snap Lake under pre-mining conditions (D. Chorley of Golder, written communication, 2001).

The sole purpose of this calibration was to determine if appropriate magnitudes of leakance factors were being utilised in the model. This very limited calibration indicated the leakance factors were reasonable. Without water level data, the hydraulic conductivity values could not be refined during the calibration process.

3.0 PREDICTED INFLOWS

Using the ground-water flow model and mine plan as described, the predicted inflows to the mine from the various mine workings over time are presented in Figure 10. The predicted peak inflow is approximately 25,000 m^3/day and occurs in 2018, about 12 years after mining begins. Inflow to the various mine workings over time was determined and the majority of the water inflow to the mine area is from the footwall and ore level ramps and drifts. During a majority of the mine life, this component of inflow comprises approximately 70% of the total inflow.

Because of the very limited data on which the ground-water flow model for the proposed Snap Lake mine is based, HCI conducted a relatively comprehensive uncertainty analysis to quantify the potential variations in predicted inflows due to variations in key input parameters. A first-order approximation to the variance of the inflow, which is a function of several hydraulic parameters, is given by (Benjamin and Cornell, 1970):

$$Var\left[Q_t\right] \approx \sum_{i=1}^{n}\left\{\left(\frac{\Delta Q_t}{\Delta x_i}\right)^2 Var\left[x_i\right]\right\} \tag{1}$$

where:

Q_t = inflow rate at time t,
x_i = a specific hydraulic parameter (e.g., K_z of the granite),
$\Delta Q_t/\Delta x_i$ = rate of change of Q_t with respect to change in value of parameter x_i, and
n = total number of hydraulic parameters considered in the analysis.

Equation 1 indicates that each of the hydraulic parameters contributes to the overall variance of the predicted inflow in proportion to its own variance and the cumulative rates of change of the inflow with respect to each of the varied parameters.

Using Equation 1, HCI evaluated the effects of different input values for seven parameters:

- the hydraulic conductivity of the hanging wall granite, footwall granite, backfill material, lakebed sediments, and the "crack"/relaxation zone in the back; and,
- the leakance factors for the mine area and drifts.

As indicated in Table 1, the values of each of these parameters was varied by a factor of 10 and 0.1. It was also assumed that the range of values for each of these parameters is log normally distributed and that one order of magnitude on each side of an "expected" value (i.e., the value put into the model) comprises two standard deviations (2σ) or the 95 percent confidence interval of a set of reasonable values. A more complete description of this method of uncertainty analysis is given in Appendix B.

For the seven parameters identified above, a total of 14 simulations were completed. The results of the uncertainty analysis indicate that the model predictions are most sensitive to changes in the leakance values used for the haulage drifts and mine panels. The ranges of predicted inflows over time derived from Equation 1 are shown in Figure 11. We have indicated the expected inflows and the inflows within the 68 percent (1σ) and 95 percent (2σ) confidence intervals. For reference, we have also indicated the range of inflows previously predicted by Golder (2000) for approximately the same mine plan.

The numerical ground-water flow model was also used to estimate the proportion of Snap Lake water inflow to the mine workings and other surrounding lakes versus water from storage (i.e., lateral flow). The results indicated that the proportion of the lake water in early time was approximately 50%, was essentially constant at about 70% from year 2013 to 2020, and then increased to approximately 90% by the end of the mine life.

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FIGURES

APPENDICES

TABLE A-1

Summary of Hydraulic Conductivity Values from Field Data from Golder

Location	Type of Hydraulic Test	Geologic Unit	Calculated K (m/day)	Notes
BH-24	Falling Head	Exfoliated Metavolcanics	1×10^{-2}	Golder (1999)
Initial Exploration Drift	Inflow	Metavolcanics	1×10^{-2}	Calculated by Golder (2001) using Goodman et al. (1965) method

Appendix B

Analysis of Sensitivity and Uncertainty

For the purpose of the predictive ground-water flow modeling of inflow to the Snap Lake Mine, variances in the key input parameters can be estimated by assuming a) physically realistic ranges for the values of these parameters, b) that the values are log-normally distributed, and c) that ± one log cycle of the values encompasses 95 percent (two standard deviations or 2σ) of the values in the realistic range.

The standard deviation, σ_e, of each parameter can be calculated from:

$$\boldsymbol{\sigma_e = C_v \cdot x} \qquad \textbf{(B-1)}$$

where

C_v = coefficient of variation, and
x = expected value.

For log-normally distributed values, the coefficient of variation can be calculated from (Benjamin and Cornell, 1970):

$$\boldsymbol{Var[\,] = \sigma_e^2 = \ln\left(C_v^2 + 1\right)} \qquad \textbf{(B-2)}$$

where σ_e^2 is the variance (or square of the standard deviation) of the natural logarithms of the values of the given parameter. Under the assumption stated above of two standard deviations of values ranging log-normally over one log cycle, $\sigma_e = 0.576$ and

$$\boldsymbol{(0.576)^2 = \ln\left(C_v^2 + 1\right)} \; . \qquad \textbf{(B-3)}$$

Taking the exponential of both sides of Equation B-3,

$$\boldsymbol{e^{(0.576)^2} = C_v^2 + 1} \qquad \textbf{(B-4)}$$

and solving for C_v

$$\boldsymbol{C_v = 0.63} \; . \qquad \textbf{(B-5)}$$

Thus, from Equations B-1 and B-5, the variance of each parameter is

$$\boldsymbol{Var\,[x_i] = \sigma_e^2 = (0.63\,x)^2} \qquad \textbf{(B-6)}$$

A first-order approximation to the variance of the passive inflow, which is a function of several hydraulic parameters, can then be calculated by (Benjamin and Cornell, 1970):

$$\boldsymbol{Var\left[Q_t\right] \approx \sum_{i=1}^{n}\left\{\left(\frac{\Delta Q_t}{\Delta x_i}\right)^2 Var\left[x_i\right]\right\}} \qquad \textbf{(B-7)}$$

where:

Q_t = inflow rate at time t,
x_i = a specific hydraulic parameter (e.g., K_z of the granite), and
$\Delta Q_t/\Delta x_i$ = rate of change of Q_t with respect to change in value of parameter x_i.

Equation B-7 indicates that each of the hydraulic parameters contributes to the overall variance of the predicted total inflow rate in proportion to its own variance and the cumulative rates of change of the total inflow rate with respect to each of the varied parameters.

Table B-1 summarises the sensitivity ($\Delta Q_t/\Delta x_i$) and the results of the uncertainty analysis for the Base Case. The predicted inflows are most sensitive to the leakance factor of the mine panels and, to a slightly lessor degree, of the haulage drifts. Most interestingly, the predicted inflows are virtually insensitive to the hydraulic conductivity of the backfill.

It should be noted that HCI has somewhat "extended" the method of multi-variant analysis of Benjamin and Cornell (1970) by applying it to cases where the values of some of the parameters change with time (e.g., the hydraulic conductivity of the hanging wall rock below the neutral axis) and space (e.g., the area of the mine panels and, hence, the number of drain nodes with leakance factors). As a consequence, the resulting values of σ/Q_t where Q_t is the expected value of inflow at a specified time varies with time over the mining sequence. For three somewhat arbitrarily selected times, σ/Q_t ranged from 29 to 39 percent for the Base Case (Table B-1). To generate the range of inflow shown in Figure 11, the simple average of the three calculated σ/Q_t values shown in Table B-1 was used (33%).

TABLE B-1

Summary of Uncertainty Analysis of Model Predictions - Base Case

Uncertainty Analysis Model Simulation	Parameter Values		Var[x_i]	May 2011			March 2018			April 2020		
	Expected	Varied		Q_t	n	n*Var[x_i]	Q_t	n	n*Var[x_i]	Q_t	n	n*Var[x_i]
"Expected"				23933			25661			23155		
Kz of hanging wall granite	0.01	0.1	3.97E-05	24451	3.32E+07	1.32E+03	26438	7.46E+07	2.96E+03	23674	3.32E+07	1.32E+03
	0.01	0.001	3.97E-05	23760	3.69E+08	1.46E+04	25315	1.47E+09	5.85E+04	22896	8.29E+08	3.29E+04
Kz of footwall granite	0.01	0.1	3.97E-05	24624	5.90E+07	2.34E+03	26266	4.52E+07	1.79E+03	23587	2.30E+07	9.14E+02
	0.01	0.001	3.97E-05	23501	2.30E+09	9.14E+04	25229	2.30E+09	9.14E+04	22723	2.30E+09	9.14E+04
Factor for backfill	2000	20000	1.59E+06	24624	1.47E-03	2.34E+03	25834	9.22E-05	1.46E+02	23328	9.22E-05	1.46E+02
	2000	200	1.59E+06	23846	2.30E-03	3.66E+03	21686	4.88E+00	7.74E+06	19008	5.31E+00	8.43E+06
K of lakebed in leakance factor	0.001	0.01	3.97E-07	24106	3.69E+08	1.46E+02	25747	9.22E+07	3.66E+01	23242	9.22E+07	3.66E+01
	0.001	0.0001	3.97E-07	23069	9.22E+11	3.66E+05	24970	5.90E+11	2.34E+05	22118	1.33E+12	5.27E+05
Leakance factor of mine panels	0.3	3	3.57E-02	37238	2.43E+07	8.67E+05	49680	7.91E+07	2.83E+06	41818	4.78E+07	1.71E+06
	0.3	0.03	3.57E-02	15984	8.67E+08	3.10E+07	18662	6.72E+08	2.40E+07	15638	7.75E+08	2.77E+07
Leakance factor of haulage drifts	0.1	1	3.97E-03	38707	2.69E+08	1.07E+06	52186	8.69E+08	3.45E+06	52790	1.08E+09	4.30E+06
	0.1	0.01	3.97E-03	18749	3.32E+09	1.32E+07	17453	8.32E+09	3.30E+07	14256	9.78E+09	3.88E+07
Factor for zone above backfill	200	2000	1.59E+04	24797	2.30E-01	3.66E+03	26006	3.69E-02	5.85E+02	23328	9.22E-03	1.46E+02
	200	20	1.59E+04	23760	9.22E-01	1.46E+04	25661	0.00E+00	0.00E+00	23155	0.00E+00	0.00E+00
					Σ =	4.65E+07		**Σ =**	7.14E+07		**Σ =**	8.16E+07
					σ =	6823		**σ =**	8451		**σ =**	9032
					σ ÷ expected =	29%		**σ ÷ expected =**	33%		**σ ÷ expected =**	39%

Notes:
$n = (\Delta Q_t/\Delta x_i)^2$
Cv = 0.63

Appendix C

Estimation of Leakance Factor

The “leakance factor” concept is developed to account for the hydraulic resistance to seepage from or to saturated porous media. In the case of a lake, such as Snap Lake, the interface between the body of water and the porous media is well defined; it’s thickness and hydraulic properties can be estimated from direct measurements. However, in the case of the mine drifts, the leakance factor can only be estimated from the calibration of the model-computed seepage to the observed flows into the drifts. In the finite element method, fluxes are associated with nodes, in the case ground-water leaving the porous media and entering the drifts these nodes are called “drain nodes”. The drain nodes is calculated in *MINEDW* in a similar manner to most other codes from the Darcy relationship as follows:

$$\boldsymbol{Q = KiA} \tag{1}$$

$$\boldsymbol{Q = K \frac{\Delta h}{\Delta L} A} \tag{2}$$

where:

Q = ground-water seepage discharge [L^3/T],
K = hydraulic conductivity [L/T],
i = hydraulic gradient [ ],
A = seepage area [L^2] in *MINEDW* the area is estimated as the area associated with a drain node.

A and ΔL are simply generic dimensions related to the individual nodes to which the seepage flux is associated.

Equation 2 is often simply written as,

$$\boldsymbol{Q = C_L \Delta h} \ . \tag{3}$$

where C_L is the leakance factor.

For “complete” connection where the resistance to flow is that of the material surrounding the node, the leakance factor will be set to a large value. However, if there is some additional resistance to flow (or “skin” attributable to non-Darcian flow, etc.), then Equation 2 can be expressed as:

$$Q = f_s K \frac{D_2 D_3}{D_1} \Delta h = K' \frac{D_2 D_3}{D_1} \Delta h \qquad (4)$$

where f_s is the "skin factor" and where Equation 4 can again be simplified to

$$Q = C_L^{'} \Delta h \ . \qquad (5)$$

It should be noted that as indicated in both Equations 2 and 4, the leakance value is grid-dependent and, consequently, in a model with variable discretisation, highly variable. *MINEDW* calculates a node-by-node value for the leakance factor with an input value for *K'* and the area associated with a node in that specific area of the model that represents the mine drifts. In either case (i.e., using Equation 2 or 4), the selection of a value for *K'* and ΔL (almost never assumed simply *K* in flow to underground mines) are rather arbitrary and the can only be accurately determined from calibration to actual inflow data when available.

The following Table presents the average leakance factors, average nodal areas and K'/ΔL values used in the Snap Lake model:

Leakance Factors			
Location	**Leakance Factor (m^2/day)**	**Average Node Area (m^2)**	**Average K'/ΔL (1/day)**
Mine Areas	0.3	1758	.00018
Drifts in footwall rock	0.1	1758	.00006
Lakebed Sediments	0.2 to 9.9 (average = 1.2)	5304	.00023