Integrated Mine Evaluation — Implications for Mine Management

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ABSTRACT

Mine management is often expected to make rapid evaluation decisions at different stages of projects based on limited and uncertain data. The challenge is exacerbated by having to distil technical complexity into a financial model that is usually designed to produce only one or two key indicators, eg net present value (NPV), internal rate of return (IRR). Mining is a complex environment with many sources of uncertainty ranging from sampling to economics. In order to optimise investment decision-making, an appropriately structured evaluation framework must be utilised.

An evaluation framework should be designed to encapsulate and integrate the complexity across the evaluation cycle, ie sampling, resource estimation, mine planning and treatment, and financial and economic modelling. This complexity is diverse and ranges from sampling support, scale effects to understanding the impact of variability, uncertainty and flexibility on operational efficiency and economic viability. These complexities, combined with time and capital constraints, usually do not allow all facets of evaluation to be integrated into the model. The model must strike a balance between simplified estimation techniques and sufficient incorporation of aspects of the project that will make a material difference to the investment decision.

This paper demonstrates the impact of the scale of measurement on the valuation of a mineral project. Comparisons are made between global estimation averages using a top-down approach and local estimates using a bottom-up approach. Three sampling campaigns were conducted on a virtual orebody to compare the relative NPV accuracies. Stochastic forward models were run on foreign exchange rates and are compared with the results from a fixed foreign exchange rate model.

INTRODUCTION

This paper explores the impact of the measurement scale on the estimate of a mineral project's NPV. Scale of measurement refers to dimensions in both space and time that are related to the key variables of the project, such as ore volume (thickness), grade, density, costs, revenue, foreign exchange rates, etc. Why is this important? Given a complex geological deposit and volatile price environment, it is suggested that the valuation of a mineral project may be materially affected by the use of large scale, annual average estimates for major variables. An integrated mine evaluation approach should be adopted using short-term, operational scale numerics that are accumulated into annual estimates to derive more realistic NPVs.

Many of the well-established resource and reserve classification codes refer to a mineral resource as having some 'reasonable and realistic prospects for eventual economic extraction' (JORC (2004), SAMREC (2000), NI43-101 (2001)). These codes offer guidelines for assessing the criteria required to define mineral reserves but do not stipulate any quantitative confidence limits associated with tonnages, grade and revenue estimates. The selection of measurement scales is ultimately based on the judgement of a competent person. In order to quantify the impact of the selected scale on valuation, it is recommended that the process incorporate a quantitative assessment of the impact of these effects. This assessment should include both the modelling of unsystematic (specific) risks for resources and reserves, and systematic (market) risks, such as foreign exchange variability and costs of commodities such as oil, steel, concrete, etc. This would facilitate the setting of confidence limits around project valuation.

It is unrealistic to create predictions of resource and reserve estimates on a small block scale when sample data are limited and spread out over a large area. Thus, in many cases production estimates of tonnages and grades are computed on an annual basis rather than a shorter-term scale (eg daily, weekly, etc). The sum of the local reserve depletions in a year is not equal to the total expected production derived from the average global reserve depletions. This is true for mineral projects that have a high degree of short-scale geological and mineralisation variability but only limited sampling data. The effect is amplified when resource variability has a substantial impact on mining rate and treatment efficiency. The problem is further exacerbated for marginal projects which usually cannot afford the cost and potential time delays of spending additional evaluation capital on attaining close-spaced sampling data.

As the scale of data acquisition changes (ie more or less data are acquired), the mean and dispersion of the data will change. The impact of scale on a single variable is largely dependant on the distribution of the underlying phenomenon, eg for grade or density. If many sample data were acquired, the shape of the distribution (specifically, the means and variances) for each variable would be well defined. In most cases of evaluation, however, only limited sampling data are acquired and as a result, changes in the means and variances of individual resource variables could have a material impact on the project value. As variances are additive, the cumulative impact could result in over- or under-estimation of the NPV.

Two different evaluation approaches are selected in this paper to demonstrate the impact of measurement scale, viz. top-down versus bottom-up techniques. The former refers to annual forecasts that are calculated from depleting resource estimates through a global mine plan. Average expected values per annum are used as inputs into the mine plan to produce a NPV estimate. An alternative approach utilises a bottom-up evaluation technique whereby additional sampling data allow finer resolution resource models to be created. These finer scale models provide a way to carry out a quantitative assessment of the impact that resource variability has on daily mine output. Annual cash flow forecasts are derived from accumulations of daily depletions based on localised resource estimates.

While it may appear that these two methods would produce similar NPV results, there are cases where they do not. A case-study of an underground mine in Canada is presented where diamonds are contained in an irregular dyke that intruded into a fractured granitic host rock. Two sources of uncertainty were modelled. Firstly, geology was evaluated as a form of unsystematic (specific) risks due to the uncertain thickness of a mineralised dyke and its undulating top surface. Secondly, economic uncertainty, in the form of foreign exchange rate volatility between the US dollar and the Canadian dollar, was integrated into the evaluation model as a systematic (market) risk.

A virtual orebody (v-bod) was created using a non-conditional geostatistical simulation based on actual sampling data to provide a method of comparing the top-down and bottom-up approaches with 'reality' in the form of a v-bod. Comparisons were made between the two techniques and the v-bod. Three sampling campaigns were conducted on the v-bod and resource and reserves estimates were recalculated each time using the additional information to assess the impacts on differences between the top-down and bottom-up approaches.

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EVALUATION PRACTICES

Project evaluation comprises three main components: project uncertainty, project structure, and value numerics, Samis and Davis (2005). The authors of this paper focused solely on project uncertainty. Firstly, because in their experience technical complexities and correlations between variables cannot be captured easily in the typical evaluation of mineral projects; and secondly, because the impacts of technical uncertainty and variability are not clearly communicated to management.

Geostatistical techniques are routinely used to estimate grade, geology and density resource models for most mineral commodities (Matheron, 1973; Krige, 1951). Since geostatistical simulations were developed (Matheron, 1973; Journel, 1974), they have been used to model the inherent variability and compare the impact of different mining methods or support sizes on resources and reserves. Early work (Dowd, 1976; Dumay, 1981; Chica-Olmo, 1983; de Fouquet, 1985) focused on understanding the influence of technical aspects related to complex mining constraints and on quality control during production. As computer power increased, more simulations could be run and different types of simulation methods were developed that allowed more complex types of geology to be modelled.

Since the 1990s, the impact of uncertainty on project economics became increasingly important as more marginal projects were discovered. Ravenscroft (1992); Berckmans and Armstrong (1997); Dimitrakopoulos *et al* (2002) and Dowd and Dare-Bryan (2004) have used a combination of objective functions and geostatistical techniques to evaluate the impact of resource risks on the mine plan and determine their probabilistic impacts on NPV. These techniques incorporate resource uncertainty in the scheduling optimisation algorithm compared to traditional mine planning methods which could result in sub-optimal reserves.

Over the past 15 to 20 years the techniques used in financial valuation of mineral projects have also evolved. Standard discounted cash flow (DCF) is used as the baseline for decision-making, but most mining companies now understand its limitations (Davis, 1995; Smith, 2000). Firstly, the technical and financial parameters used as input in NPV calculations are subject to uncertainty; secondly, mine management can and do react to changing circumstances (eg rising or falling commodity prices) by adapting the mine plan. Monte Carlo simulations coupled with geostatistical orebody simulations overcome the first limitation; real options were developed to overcome the second one.

According to Brealey and Meyers (2003) the first person to have recognised the value of flexibility was Kester (1984) in an article in the *Harvard Business Review.* The following year, Mason and Merton (1985) reviewed a range of applications to corporate finance and in their seminal paper, Brennan and Schwartz (1985) applied option pricing techniques first developed in finance to the evaluation of irreversible natural resource investments using Chilean copper mines to illustrate the procedure. To simplify the mathematics, they assumed that the reserves were perfectly homogeneous and that the grades were perfectly known. From a mining point of view, these assumptions are unrealistic. Galli and Armstrong (1997), Carvalho *et al* (2000) and Goria (2004) have overcome this by combining geostatistics with option pricing.

In their paper, Brennan and Schwartz (1985) used a geometric Brownian motion based on Black and Scholes (1973) method with a convenience yield proportional to price in order to model the copper price. This was necessary to try and reproduce the natural variability of commodity prices over time. In contrast to many other commodities, diamond prices are not as volatile. Factors like the oil price and the exchange rate are more volatile and have a material impact on project value; the oil price affects costs and the exchange rate influences the company's revenue. The authors have chosen to focus on the exchange rate for this study.

Many models have been developed for interest rate and foreign exchange rates, ranging from simple extensions of Black and Scholes (1973) through Vasicek (1997) and on to the latest models with stochastic volatility. The book edited by Hughston (1996) provides a good overview of the subject. The authors chose to use the Garman and Kohlhagen (1983) which is a simple extension of Black and Scholes. In this model the drift term is replaced by the difference between the domestic and foreign interest rates. If S_t denotes the spot exchange rate at time t and r_d and r_f are the domestic and foreign interest rates, then:

$$dS_t = (r_d - r_f)\, S_t dt + \sigma_S S_t dW_t$$

where:

σ_S is the volatility of the exchange rate

dW_t is a Brownian element

Two advantages of this model are that the exchange rates generated are lognormally distributed and hence positive, and that the parameters are easy to estimate.

The evaluation of a mineral project is a complex and technically challenging process, further complicated by numerous estimates of variables, covariance relationships and associated uncertainties. This paper captures a few crucial aspects of the evaluation pipeline, summarised in the following four sections:

- sampling and resource modelling,
- estimation of reserves (mine-planning and treatment),
- financial evaluation and economic modelling, and
- analysis and interpretation.

SAMPLING AND RESOURCE MODELLING

Sampling data in any evaluation model are fundamental in producing estimates that reflect reality. Although including more samples reduces uncertainty associated with both the mean and variance of resource estimates, it does not alter the natural variability within the deposit. The limitations of designing a sampling campaign for multiple variables have been discussed before Kleingeld and Nicholas (2004). Three variables were considered in this evaluation model:

- the geometrical variability of the top surface of the dyke (v1),
- thickness related to the volume of the dyke, and
- grade (in carats per hundred tonnes).

Core drilling was used to delineate geological variability on three different grid densities; 75 by 75 m, 50 m by 50 m and 25 m by 25 m, creating scenarios one, two and three, respectively. A 50 m by 50 m drilling grid was used to sample for grade, using large diameter drilling (LDD). Grade was not deemed to have any significant variability between scenarios and therefore, a single sampling campaign sufficed. The same grade estimates were applied to each scenario.

A v-bod was created using a non-conditional geostatistical simulation based on data gathered from a combination of drilling information, bulk-samples and face mapping from an exposed part of the dyke. It is assumed to be the 'reality' on which the various sampling campaigns were conducted to generate sample data. Table 1 describes the design of the simulated sampling campaigns on a virtual orebody (v-bod); sampling occurred at point support and simulation grid nodes were 4 m by 4 m in dimension. The limitation of this approach is that only a single v-bod was created due to the time constraints and all conclusions are directly a function of both the data used to seed the v-bod and the design of the subsequent sampling campaigns. Sample data

TABLE 1

Sampling campaign design – summarising the three sampling campaigns and the v-bod.

Description	V-bod	Scenario 1	Scenario 2	Scenario 3
	Reality	Wide-spaced	Moderate	Detailed
Grid dimensions	4 m × 4 m	75 m × 75 m	50 m × 50 m	25 m × 25 m
No of samples/ nodes	399 360	1136	2556	10 224
Sample per cent of V-bod	-	0.28%	0.64%	2.56%

were used as input to generate kriged estimates and spatial simulations of grade, dyke thickness and geometric surface undulations of the dyke. A single mine plan was created based on the kriged estimates and overlain onto each estimate and simulation to determine the reserves. All output was fed into the financial model.

Base maps of the v-bod and each sampling campaign are shown in Figure 1 (warmer colours represent higher values while darker colours are low values).

RESERVES

The degree of resource complexity will have little impact on an operation's financial outcome for models that are generally unconstrained in terms of mining and treatment thresholds (assuming that the resource estimates have been accurately estimated). This applies to scenarios where abundant flexibility is included in the mining plan so that no bottlenecks occur in the extraction or treatment processes. The rate and scale of mining would deviate very little from plan as a result of resource variabilities.

In contrast, mining operations (such as the underground example in this paper) that operate under strict geotechnical and geohydrological constraints in environmentally sensitive areas do not have the luxury of unlimited mining and treatment flexibility. These mines cannot easily respond to changes in tonnages or grades as a function of resource variability. In the case of marginal operations with limited capital expenditure, the impact of this limited responsiveness is further exacerbated by the presumption of 'smoothed' ore horizons due to kriging with limited sampling data. The impact of this 'smoothing' will be demonstrated in this paper.

There are multiple factors to consider at this stage, ranging from resource uncertainties, mining and treatment constraints, financial cost per tonne data and economic volatilities with respect to commodity pricing and consumable costs. Identifying those factors that have the biggest impact on project value is essential but can be a very complex and time-consuming process. This is largely driven by the number of variables that have to be considered and the complex interaction between variables, which are associated with different uncertainties and variabilities.

While legal, social, political and environmental factors may influence managerial decision-making, the authors have elected to concentrate on the mining and treatment components of this model as discussed below.

Mine – planning and design

In this example a conventional room and pillar underground method is considered with an option of slashing and drifting, depending on whether the dyke thickness was less than a specified mining threshold. An average extraction rate of 75 per cent was used. Each mining block of size 250 m by 250 m was depleted based on a combination of rim tunnels, stope tunnels and stope slashing. An average daily call of 3150 treatment tons was imposed on the project by management. The mine plan and treatment plant were designed to meet this production requirement on average per year.

The tabular nature of this deposit and mining, geohydrological and geotechnical restrictions severely limit the sequencing and optimisation of extraction. Simplistic assumptions were made regarding the selection sequence of blocks based on the highest value blocks being extracted first to maximise the time value of money. While the authors recognise the work done by Dimitrakopoulos and Ramazan (2004); and Dowd and Dare-Bryan (2004) involving the optimisation of blocks given resource and reserve uncertainties, there was insufficient time to include this in the study. The mine plan provided an opportunity to understand the interaction of the spatial nature of the reserves with the temporal realisation of its value.

Mine blocks were depleted at a smallest mining unit (SMU) scale of 4 m by 4 m with a minimum mining height requirement of 2.0 m for equipment access into stope tunnels. Maximum mining heights of stopes were constrained to 2.2 m while rim tunnels were 3.5 m.

- rim tunnels were 4 m x 4 m x 3.5 m (height),
- stope tunnels were 4 m x 4 m x minimum 2.0 m (height), and
- stope blocks were 4 m x 4 m x minimum 1.0 m (height).

Pillar dimensions varied depending on the support required but no span greater than 8 m was created.

Recovery modelling

The estimation of the mean recovery factor and its variance is critical in determining the quantity of recovered material at a predetermined throughput treatment rate. The recovery factor depends largely on three key considerations:

- characteristics of the ore type,
- its liberation and separation properties, and
- the design and interaction of the treatment process in relation to this ore type.

The challenge of achieving efficient recoveries is to understand these complex three-way interactions. As the evaluation model required simplistic assumptions to be made, the authors assumed a linear relationship between the proportion of kimberlite ore and the waste.

The impact of the recovery factor on the recovered carats can be very marked especially if there are constraints on the system. For example, if the cut off grade is close to the statistical mean, subtle variations in the mean cut-off grade could significantly impact the project NPV. If the cut-off grade is raised, the average grade above cut-off increases which may require mining that is too selective using the current mine design and equipment.

Plant design, by its nature, requires a best fit for the 'average expected feed' and hence cannot incorporate the daily feed variation that may occur over the project's LOM. Conventional approaches to plant optimisation Parker (1997) usually entail:

- adapting the plant to accept the variability,
- installing a stockpile blending system, and
- adapting the mining method to increase the number of faces or draw points and use smaller equipment to improve selectivity.

The example in this study is fairly fixed in terms of its mine design and equipment selection. In addition, environmental policies limit the creation of large stockpiles. A total stockpile capacity of 3000 tons was created, which included capacity from an underground storage bin. While some degree of flexibility was available to adapt the plant settings to the ore variability, this was more suited to weekly and monthly fluctuations but would not cater for daily variations in the system.

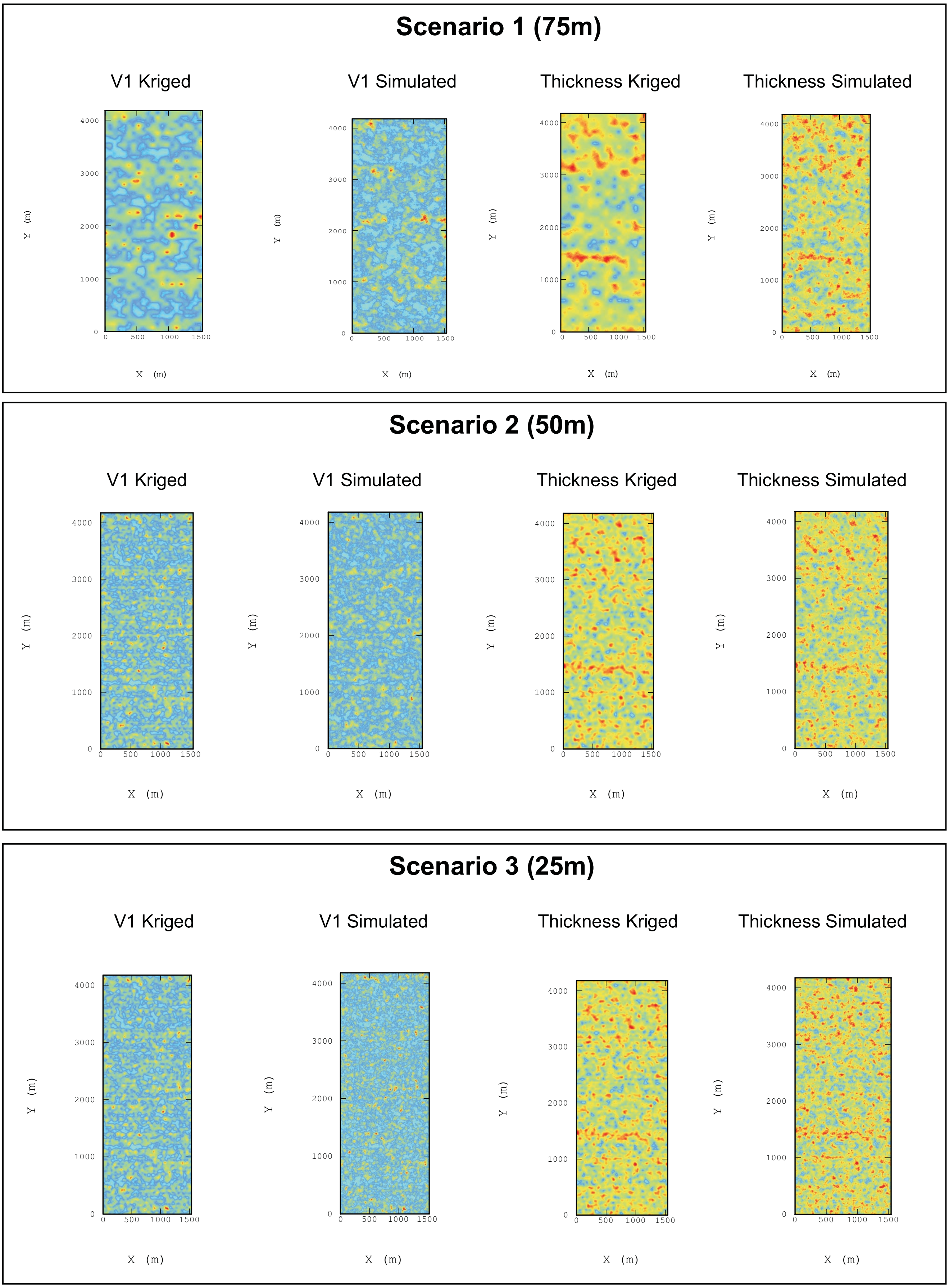

FIG 1 - Comparison of the thickness and v1 base maps for the kriged and simulated outputs of each scenario with that of the v-bod. Grade was held constant between scenarios.

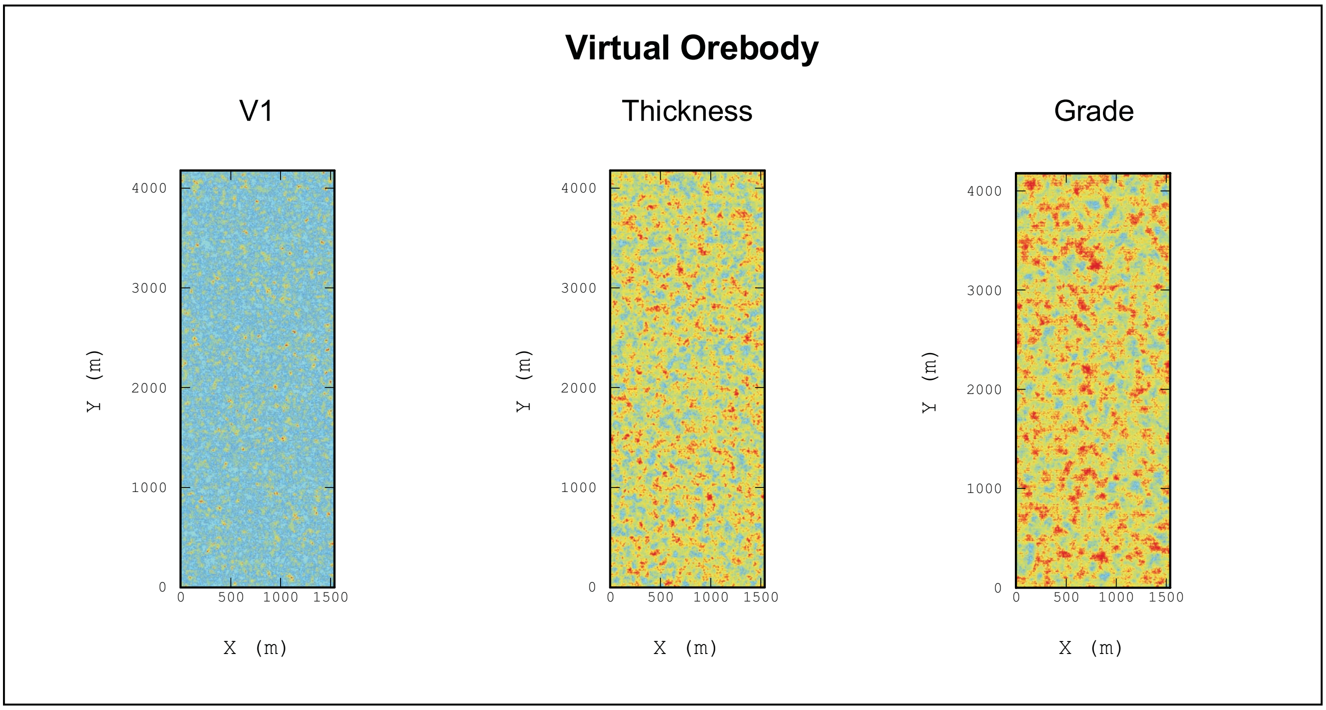

FIG 1 (continued) - Comparison of the thickness and v1 base maps for the kriged and simulated outputs of each scenario with that of the v-bod. Grade was held constant between scenarios.

TABLE 2

Resource simulation output showing the statistical differences between the v-bod and each scenario for grade, dyke thickness and the geometrical variability of the dyke surface (v1).

	V-bod	Scenario 1 (75 m)		Scenario 2 (50 m)		Scenario 3 (25 m)	
		Kriged	Sim 1	Kriged	Sim 1	Kriged	Sim 1
Mean thickness	1.70	1.70	1.66	1.70	1.71	1.70	1.69
Variance thickness	0.23	0.11	0.18	0.11	0.15	0.13	0.20
Mean v1	1.88	1.90	1.90	1.90	1.91	1.90	1.91
Variance v1	0.18	0.09	0.17	0.09	0.10	0.09	0.17
Mean grade	191	195	187	195	195	195	192
Variance grade	1985	1062	2860	1062	1523	1062	2004

While dynamic simulations are considered as a possible means to estimate the short-scale variability in the recovery efficiency, a simpler, pragmatic approach was sought to ascertain the impact in the case study.

In this model, depletions of the simulated 4 m by 4 m SMUs provided the ore-waste proportion information. A simplistic, linear relationship was imposed on treatment recoveries in relation to the proportion of kimberlite and waste; recovery efficiency improved as the proportion of kimberlite increased. A plant surge capacity constraint was included to assess the impact of varying dyke thickness (on a 4 m by 4 m SMU scale) on the feed rate variability using an 'event-based' simulation.

The principle strategic levers that were considered in this mining and treatment sections were:

- annual mining rate in order to produce 3150 tons per day,
- bin storage capacity of 3000 tons,
- SMU selection (4 m × 4 m × height),
- the maximum mining ramp angle (17 degrees), and
- a threshold imposed on the waste/kimberlite proportion (70/30); if any blasted block had more than 70 per cent waste, it was not sent to the treatment plant.

Mine plan and treatment output

The daily production variations for scenario three are shown in Figure 2 together with resource variability in relation to mining and treatment constraints. The recovered carats after deducting all losses due to the waste threshold vary considerably on a daily basis.

Output from the mining and treatment phase on an annual basis is tabulated in Table 3 for the v-bod and each of the three scenarios.

FINANCIAL MODELLING

Given the uncertainties associated with each component of the model, the conventional practice of quoting a single NPV output is deemed idealistic and often, misleading. Conversely, running hundreds (or thousands) of stochastic realisations to quantify uncertainty in each component may be excessively time-consuming and expensive and could result in superfluous data that have little material impact on the NPV. A balance must be struck. The financial model must be sufficiently flexible to accommodate multiple input scenarios for both global and local estimates yet quick when generating NPV outputs.

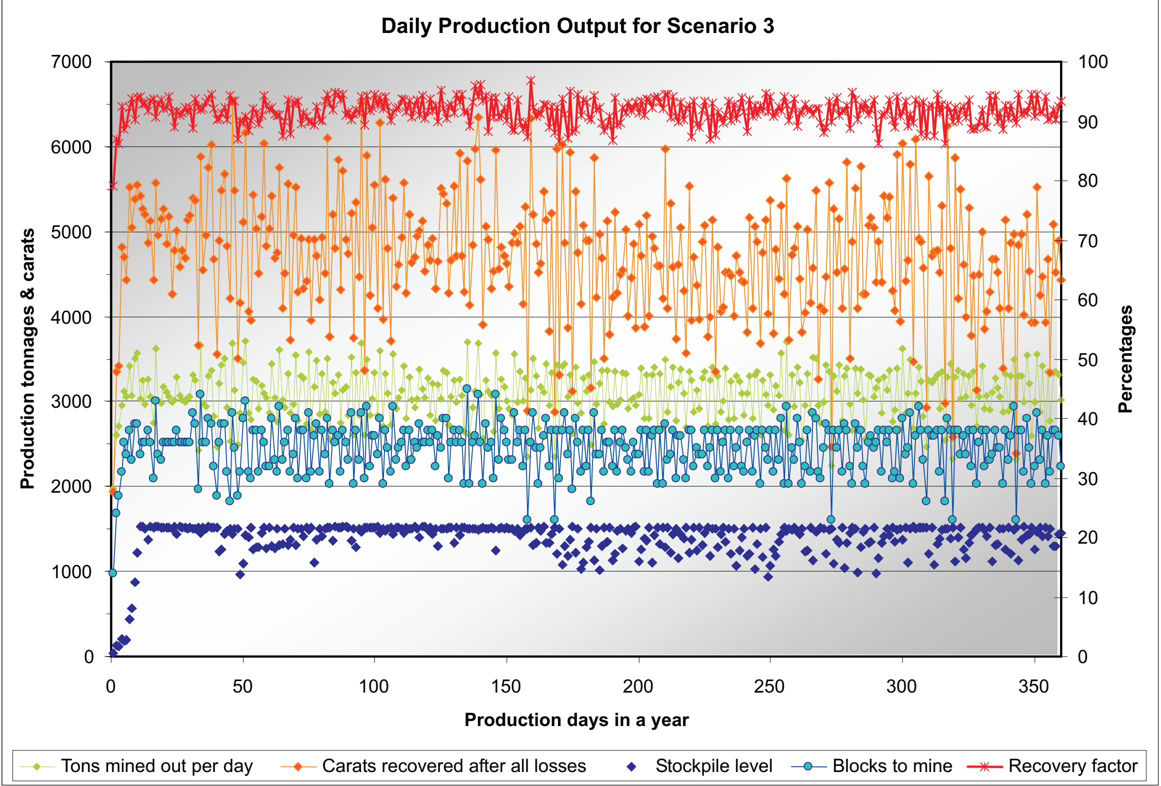

FIG 2 - An example of the daily production output in one year for scenario 3.

TABLE 3

LOM production output showing the annual production output for the v-bod and the three scenarios (recovered carats and grade are shown after all deductions).

	V-bod	Kriged results			Simulated results		
		Scenario 1	Scenario 2	Scenario 3	Scenario 1	Scenario 2	Scenario 3
Total tons (milion)	10.8	10.8	10.8	10.8	10.8	10.8	10.8
Recovery factor	92.5	93.6	93.3	93.1	92.1	92.5	93.0
Recovered carats (million)	16.2	16.6	16.6	16.5	15.7	16.2	16.4
Recovered grade	149.6	153.8	153.7	153.2	145.7	149.8	151.9

Financial models are often designed as disparate systems, usually in a spreadsheet form, to compute the financial value of a project based on hard-coded production output from mine plans. So they have difficulty in capturing dynamic, technical linkages between resource, mining, treatment and economic models. The evaluation framework of conventional models allows limited risk and sensitivity analyses to be conducted as they do not assess the impact of correlated variables across the evaluation pipeline.

In conventional sensitivity analyses, all parameters, except the one in question, are held constant in the evaluation model. While this helps to identify which variable has the highest influence on the NPV, it cannot capture the range and probability of realistic scenarios when parameters vary simultaneously. Monte Carlo simulations (MCS) are a useful tool but should be used in conjunction with other geostatistical and economic modelling tools to model the spatial and time-based variabilities. Examples of economic stochastic variables are foreign exchange rate prices, commodity prices, oil and diesel prices etc.

A few of the key financial concepts are discussed below forming the building blocks of an integrated mine evaluation approach. Financial values have been adjusted to maintain confidentiality of the Canadian diamond mine.

Bottom-up versus top-down evaluation

Temporal scale is one of the most important aspects to be considered in the design of a financial model. The time interval in which cash flows are estimated must correspond with the time interval in which mining and treatment production data are measured and accumulated. These reserves in turn, depend upon the mine plan's ability to react to resource variability at the appropriate operational short scale. In addition to the unsystematic (project specific) risks, the financial model should also take due cognisance of systematic (market, economic related) risks by incorporating these stochastic variables at the appropriate time scale (support size).

This section of the paper demonstrates that cash flow constituents derived from annual estimates in a top-down approach will not correctly reflect the asymmetries due to operational variability on a local, daily basis. A more accurate way of deriving annual cash flow estimates needed to make decisions on projects would be to accumulate the appropriate values from a bottom-up approach, ie daily, monthly, quarterly then derive annual estimates for NPV forecasts.

The bottom-up approach entailed estimation (via geostatistical kriging techniques) of the main resource variables into a fine resolution grid (SMUs of 4 m by 4 m) based on sampling data from each campaign. Each SMU was analogous to a mining blast that was assessed to ascertain if it met the necessary mining and plant criteria, before either contributing to the daily plant call of 3150 tons per day or dumped to the waste bin if it comprised more than 70 per cent waste. These daily accumulations were added together to form monthly, quarterly and annual production totals forming inputs into the cash flow models to derive NPVs for each scenario.

For the top-down approach, it was assumed that the mine plan only incorporated sufficient detail to deplete large-scale mine blocks of dimensions 250 m by 250 m. This implied that local mine plans (within each large-scale mine block) were not available to allow sequential depletion of the SMUs to accumulate tonnages and carats in a given year. Although the resource was modelled on a finer resolution (SMUs of 4 m by 4 m), these values were averaged into larger 250 m by 250 m mine blocks. The mine plan was designed to deplete on average 3.3 large-scale mine blocks per annum.

The average resource values for each year were run through this mine plan, assuming a fixed daily plant call of 3150 tons per day could be attained. Total recovered carats were calculated as a function of depleting the average estimated tonnages (per large-scale mine block) at a fixed throughput rate of 3150 tons per day, then multiplying the depleted carats with an average recovery factor per large-scale mine block. The carats per large-scale mine block were accumulated into annual cash flow models to produce global NPV estimates for each of the three kriged scenarios.

Table 4 presents local versus global NPVs calculated for each scenario using bottom-up and top-down approaches, respectively.

Technical discount rate

Many approaches have been developed to include technical risks in projects. Davis (1995) and Samis *et al* (2006) have argued against using a single discount factor to the aggregate net cash flows; they favour discounting each component as a function of its specific risk level. The authors, however, elected to use a single discount rate for the following reasons:

- It is still used in practice today as a baseline metric for financial comparisons.
- It allowed uncomplicated calculations of the NPV and the principles of this study are applicable to any other approach used.
- This study assessed the evaluation of a single project rather than a portfolio of projects. Project (technical) risks could be diversifiable if a large portfolio of projects were considered. The overall variance of the portfolio would reduce as a function of the number of projects in the portfolio, Markowitz (1952).

In this study, a ten per cent discount rate has been used. The standard NPV formula is well known where *CF* refers to the cash flow in each period *i* and *r* is the discount rate (see Equation 1). This equation can be rewritten as a weighted sum to illustrate the impact of the discount rate on the variance of the DCF (see Equation 2).

$$NPV = \frac{\sum CF_i}{(1+r)^i} - I_0 \tag{1}$$

$$DCF = \sum CF_i * \left(\frac{1}{(1+r)^i}\right) \quad \text{or} \quad DCF = \sum CF_i * w_i \tag{2}$$

When risk analyses are conducted to ascertain the impact of the uncertain cash flows on a project's NPV, the mean net cash flow in each period, *i*, will be reduced by the weighting factor, *w*. This penalises cash flows in later years. The variance of the cash flows also reduces but by the square of the weighting factor, *w*, so this has an even greater effect on the variance than on the mean. In this example there are two opposing effects; on the one hand the variance reduces over time but as the sampling information is sparser in later years, less knowledge exists about the continuity of the dyke or the grade variability in those years.

DCF analysis and time windows

NPV is a metric to assess whether the project makes a profit after all debts, invested capital and interests have been repaid. Once the NPV estimate has been determined, the second step is to plot the annual DCFs as this shows when the major proportion of cash flows fall and whether there are any irregularities over the LOM. The annual, locally-derived NPVs using the kriged estimates for scenario one and three are CAD$ 32.9 million and CAD$ 28.3 million, respectively. Figure 3 compares the annual cash flows and DCF values for these two scenarios.

Figure 3 shows that the period between (2008 and 2012) accounts for more than 60 per cent of the project's positive annual cash flow and 70 per cent of the DCF value. As cash flows generated after 2012 are discounted at values of 50 per cent and higher, management would have to make significant operational changes in order to increase net cash flows beyond 2012. Money would be better spent on attempting to improve the net cash flows earlier on to maximise the NPV. Risk mitigating controls could be implemented such as mining or treatment modifications or by reducing the technical risk proportion in the discount rate through further sampling.

TABLE 4

Financial output (in CAD$) showing the differences between the global NPV, using a top-down approach versus that of the NPV annual based on a bottom-up approach (all values were calculated using a flat forex rate).

	V-bod	Kriged		
		Scenario 1	Scenario 2	Scenario 3
Global annual NPV	-	91.6	80.1	73.9
Local annual NPV	2.1	32.9	31.4	28.3
Differences	-	58.8	48.7	45.6
	V-bod	**Simulated**		
		Scenario 1	**Scenario 2**	**Scenario 3**
Global annual NPV	-	12.0	39.7	58.1
Local annual NPV	2.1	(26.1)	3.6	18.1
Differences	-	38.0	36.1	40.0

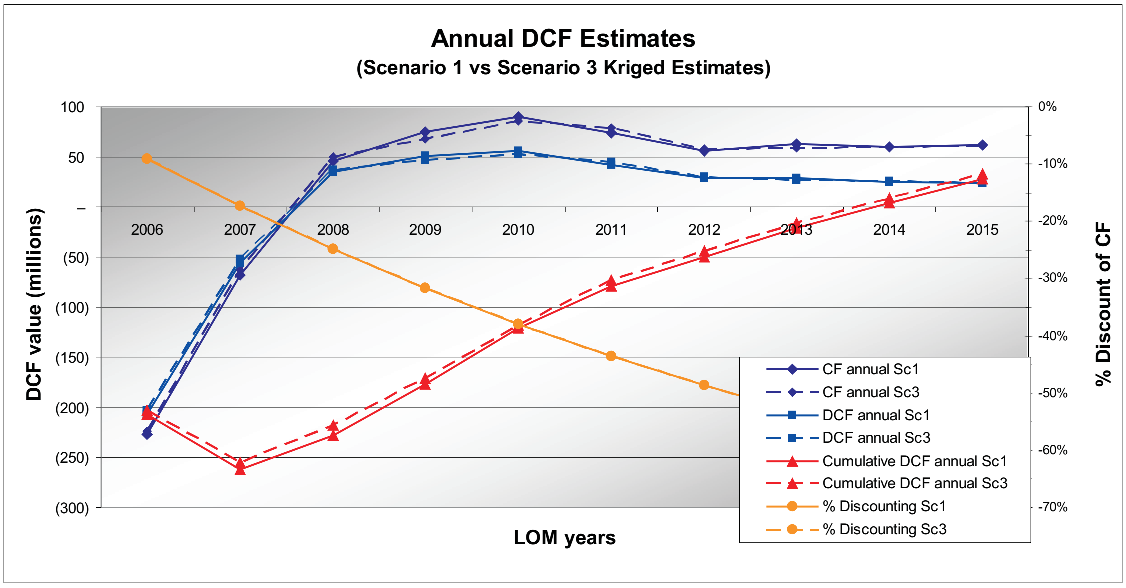

FIG 3 - Comparison of net cash flows (CF), discounted cash flows (DCF), cumulative discounted cash flows and the percentage discounting applied to the net cash flows for scenario 1 and scenario 3.

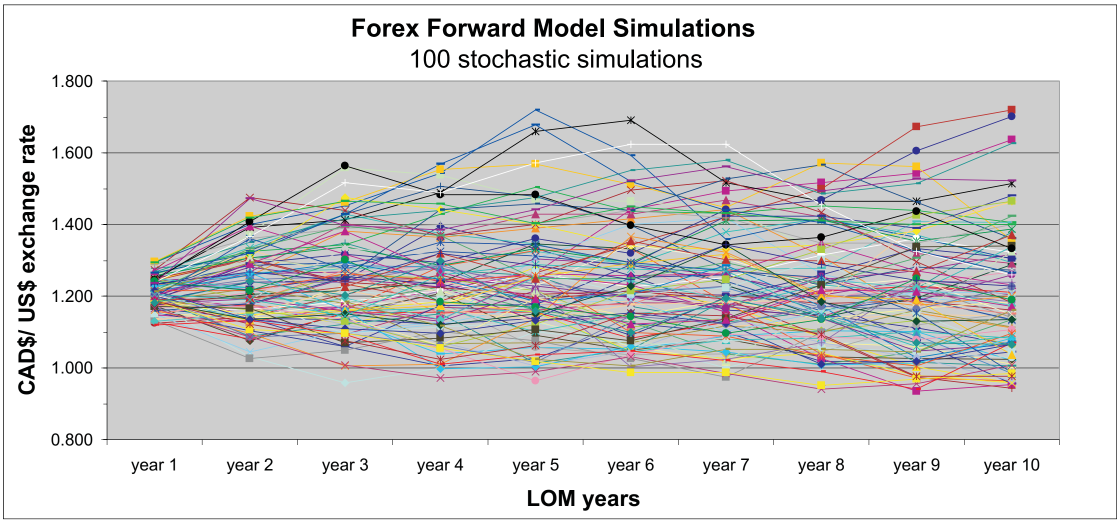

FIG 4 - The forex rate stochastic output per year from 100 simulations.

Economic (forex) uncertainty

Two scenarios considering forex uncertainty were integrated into the evaluation model. In both cases, the forex rate was applied only to the revenue component as sales from diamonds were in notional US$ whereas all costs were assumed to be sourced locally. The first scenario assumed a flat rate of 1.21 CAD$ to a US$. This corresponds to a forward forex price. Transaction costs were ignored. The NPV results of the three scenarios relative to the v-bod using the flat rate were shown in Table 4.

The second scenario assumed that the project management team would expose the project to the forex rate volatility. Forex stochasticity was modelled using a Garman and Kohlhagen (1983) to incorporate mean reversion and volatility parameters. A total of 100 simulations were run over a ten-year period emulating the forex uncertainty (Figure 4).

Each of the 100 simulations was incorporated into the financial model to produce NPV estimates for each scenario and for the v-bod. NPV histograms and cumulative probability plots for the v-bod are shown in Figures 5 and 6.

NPV comparisons incorporating the forex rate simulations are tabulated in Table 5 for each scenario and for the v-bod. All values shown were calculated using the local estimation technique (bottom-up approach).

ANALYSIS AND INTERPRETATION

The case study demonstrated the impact of resource and economic stochasticities on a project's NPV as a function of both sampling and temporal uncertainties. A v-bod was constructed from actual sampling data, derived from a Canadian mine, to provide a method of comparing scenarios against a simulated version of reality. Three sampling campaigns at grids of 75 m, 50 m and 25 m were conducted on the v-bod to produce scenarios one, two and three. It was shown that global annual NPV estimates derived in a top-down fashion, markedly over-estimated the v-bod NPV. Comparisons between scenarios showed material differences in the NPV estimates.

Global NPVs derived from kriged estimates for the three scenarios (75 m, 50 m and 25 m) were CAD$ 91.6 million, CAD$ 80.1 million and CAD$ 73.9 million, respectively.

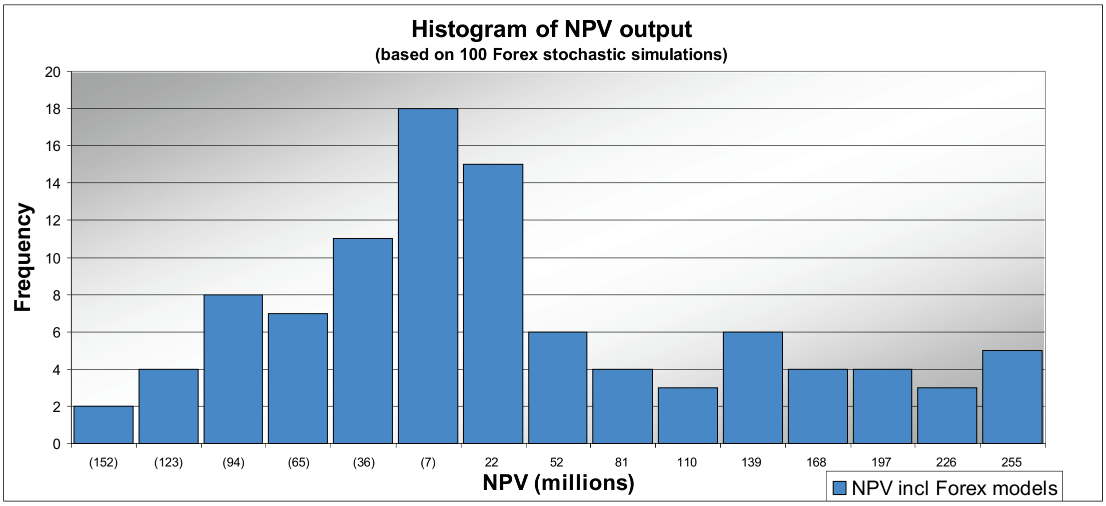

FIG 5 - An NPV histogram for v-bod after including 100 forex simulations.

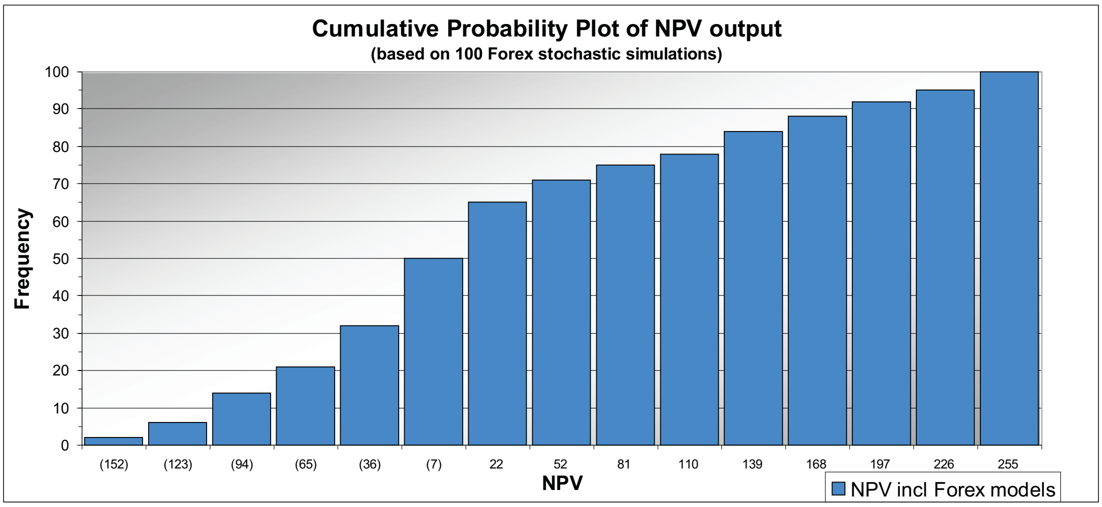

FIG 6 - The cumulative probability plot of the NPV for v-bod after including 100 forex simulations.

TABLE 5

Economic forex output (in CAD$) showing the maximum, minimum and 50th percentile NPVs of the three scenarios relative to the v-bod after including forex rate modelling (per cent differences are relative to the v-bod P50 value).

	V-bod	Scenario 1	Scenario 2	Scenario 3
Maximum NPV (annual)	255. 2	292. 5	291. 5	287. 6
Minimum NPV (annual)	(177.4)	(150.3)	(152.8)	(155.2)
NPV P50 (annual)	(6.7)	24. 5	22. 7	19. 6
P50 difference (%)	-	468%	440%	394%

As drilling grid densities increased from 75 m to 50 m and 25 m intervals, the uncertainty of v1 and dyke thickness reduced and the estimates improved relative to the actual v-bod NPV (CAD$ 2.1 million). Nonetheless, all global estimates over-estimated the v-bod NPV estimate by a magnitude of 43 to 35 times (75 m to 25 m scenarios).

Local NPVs derived from kriged estimates for the three scenarios (75 m, 50 m and 25 m) were CAD$ 32.9 million, CAD$ 31.4 million and CAD$ 28.3 million, respectively. Similarly, the NPV estimates improved as more samples were taken. Local estimates over-estimated the v-bod NPV estimate by a magnitude of 15 to 13 times (75 m to 25 m scenarios).

Note that the number of samples are significantly large (1136 samples for the 75 m scenario, 2556 samples for the 50 m scenario and 10 224 samples for the 25 m scenario). The more complex a deposit is (in terms of geological structures and mineralisation dispersion), the more sample holes will be required to reduce uncertainty and produce more accurate estimates of the statistical means and variances of relevant variables. Greater NPV differences between sampling scenarios would be expected if fewer samples were taken.

While kriging exercises produced the best unbiased estimates for key variables, they tend to provide 'smoothed' resource estimates based on limited data. It is this 'smoothing effect' and its interaction with mining constraints that result in over-estimation of the grades, thickness and v1 variables. NPV estimates would be over-estimated relative to the actual deposit. Contrary to kriging, spatial simulations provide a better indication of the range of variabilities to be expected. Insufficient time was available to generate a range of simulated realisations for comparison purposes. Thus, only a single simulated realisation was selected as an example of the expected differences in mean values.

Local NPVs based on conditional simulated estimates for the three scenarios (75 m, 50 m and 25 m) were negative CAD$ 26.1 million, CAD$ 3.6 million and CAD$ 18.1 million, respectively. These simulated outcomes are significantly lower than the kriged estimates and closer to the actual v-bod NPV. This may give the impression that conditional simulations provided more accurate estimates than kriging, but these simulations represent only one extraction from a range of simulations. This could represent the tenth or 90th percentiles (P10 or P90) of the simulated distribution outputs. Further work is necessary to generate the e-type estimate from a complete range of conditional simulations and compare it with the kriged result.

The use of a flat forex rate was compared with a stochastic forward model that considered forex rate volatility. A fixed forex rate of 1.21 was used (February 2006 CAD$:US$ rates) to derive a v-bod NPV of CAD$ 2.1 million. Table 5 shows the probable range in NPVs for the V-bod and three kriged scenarios when each of the 100 forex models were run through the financial model. The medians (ie 50th percentile or P50) for scenarios one, two and three were CAD$ 24.5 million, CAD$ 22.7 million and CAD$ 19.6 million, respectively.

Using the variable forex rates, the P50 of the v-bod NPV reduced from CAD$ 2.1 million to negative CAD$ 6.7 million (four times less). This would imply that the project is susceptible to forex rate volatility. However, as shown in Figures 5 and 6, there is upside opportunity when the 50th to 90th percentiles are considered. Projects that are particularly revenue or cost sensitive may benefit by conducting forward modelling of the forex rate as it allows management to gain an improved understanding of the range of probable NPVs. The costs of hedging against downside risks of forex rate fluctuations should be weighed against the negative impact that it may have on project value.

The estimation of resources strives to create a view of the quantity of *in situ* material that can reasonably be mined. It is this 'reasonable expectation' of 'mineability' that implies it is impossible to estimate resources totally independently of all external factors. These factors include the economic and technological limits that have to be imposed, and the scale and rate of mining.

As noted from this study, the optimal operational strategy of a mine is related to a number of key factors that need to be defined at an appropriate spatial and temporal scale:

- resource complexity, in terms of the continuity of mineralisation within geological structures; and thickness of the ore zone;
- design of sampling campaign(s) to detect the means and variances of selected variables; specifically considering sample support size and quantity of samples;
- resource modelling; kriged estimates to determine the means of grade, thickness, etc and geostatistical simulations to assess the probabilistic impact of variabilities on the evaluation model;
- design of the mine plan in response to resource complexities;
- mining and treatment logistical, environmental and financial constraints;
- financial cash flow model with respect to revenues, costs and other aspects, such as taxes and royalties that emphasise asymmetries in cash flows;
- economic stochasticity of foreign exchange rates and commodity prices for steel, diesel, concrete costs, etc; and
- encapsulation of different sources of technical risks in the evaluation model.

While this study focused exclusively on a diamond mine example, it is believed that the key aspects mentioned above are true for most mineral projects that have complicated resource models but only limited sampling data, and restrictive mining and treatment constraints.

CONCLUSIONS

Mine evaluation requires an integrated, holistic approach as the valuation of intangible resource and reserve assets are based on uncertain data that are linked to several components of the valuation pipeline. The complexity of valuing mineral projects lies in evaluating a number of spatial and time-dependent variables, within an appropriate time scale. These variables may or may not be correlated with each other. There is usually a high degree of uncertainty about the true means and variances of these variables which complicates the design of an optimal integrated evaluation system.

As noted from this study, selection of the appropriate time measurement scale in which to evaluate a number of diverse variables in a mineral resource project is critical in attaining realistic NPV estimates. Further analysis demonstrated the knock on effects that both uncertainty and variability have on the evaluation pipeline. For this reason, the evaluation model components cannot be optimised individually; the seamless integration of resource, mining and treatment, and financial components is required in order to achieve an optimal balance of the system. There are three main effects that have been investigated in this model.

The first is the impact that a complex and uncertain resource has on the design of an optimal mine plan. The effectiveness of the design is usually determined by a combination of the inherent, stochastic variability of the deposit and the uncertainty of predictions of this complexity that arises from limited sampling data. If restrictive mining and treatment constraints are imposed onto a complex resource model, the adaptability of the mine will decrease. Where possible, the flexibility in the mine plan should be matched to both the estimated degree of resource complexity and the uncertainty that the mine design team has about that complexity.

Secondly, synchronisation between the treatment plant and the mining extraction process has a huge impact on the asymmetries in the cash flow model. If mining constraints, such as mining rate of advance, development tonnes, dilution, etc are not aligned with the treatment constraints in terms of storage bin capacity or plant throughput, the mine could produce more tonnes at a time when the plant cannot treat it, or conversely, the mine will produce less tonnes at a time when the plant's capacity exceeds that of the mine. Imbalances in these constraints result in time wastage and inevitably, lost profit opportunities.

Lastly, the process of integrating this model revealed which resource, mining and treatment parameters have the biggest impact on project value. This process would assist the competent person in identifying those areas which are uncertain and could lead to a material difference in valuation. Once an integrated development system has been developed, it will be possible to explore the upside potential of optimally matching economic forecasts with mining and treatment parameters. Forward models of cost and revenue data should be at an appropriate time scale that is aligned with the estimates of cash flow forecasts and reserve calculations. Changes in revenue as a function of commodity price or exchange rates, or costs related to oil and diesel, steel and concrete prices could have a material impact on a project's value. The derivation of reserves will also be influenced by these economic stochasticities.

While a balance was sought between a pragmatic yet sufficiently detailed evaluation model, inclusion of realistic mining and treatment constraints necessitated the construction of a more complex evaluation model to reflect the value of additional sampling data. Mining and treatment constraints in response to resource variability defined the key relationships within the evaluation model that resulted in different NPVs between scenarios.

Further work is pending in the following areas:

- modelling spatial correlations between thickness, geometrical surfaces of the dyke and grade using the latest sampling data;
- mine plan sequencing and optimisation in response to resource uncertainty;
- dynamic recovery modelling with particular emphasis to liberation, separation and their interaction with the ore properties;
- economic stochastic modelling related to oil, steel and concrete prices;
- response of the evaluation model (feed-back and feed-forward loops) to different sources of uncertainty;
- real options valuation to ascertain the impact of flexibility in the model;
- development of an integrated software platform to rapidly evaluate projects.

> *As a last remark, beware that uncertainty … arises from our imperfect knowledge of that phenomenon, it is data-dependent and most importantly model-dependent, that model specifying our prior concept (decisions) about the phenomenon. No model, hence no uncertainty measure, can ever be objective* (Goovaerts, 1997).

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