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Reconciliation of the McKinnons Gold Deposit, Cobar, New South Wales

By S M Elliott[1], D V Snowden[2], A Bywater[3], C A Standing[4] and A Ryba[5]

ABSTRACT

McKinnons open pit gold mine, located within the Cobar Basin in New South Wales, is characterised by complex mineralisation controlled by the intersection of steep faults in the hinge zone of an anticline Resource estimation using indicator variography and indicator kriging was based on RC drilling on 25 m sections together with grade control RC drilling on 12 m by 12 m and 6 m by 6 m patterns. Rigorous sampling and checking procedures were used to verify the reliability of the drill hole data. Blast hole data was used to assist grade control in defining ore block boundaries.

Production shows there is excellent reconciliation of high-grade tonnage and grade with geostatistical model expectations. However, the mined low-grade tonnage, which has been stockpiled and awaits processing, appears to have been substantially reduced when compared to the model.

A retrospective study of the spatial and statistical relationships between blast hole assays, exploration RC assays, block model grades and production has highlighted a number of classical issues related to the following aspects of resource estimation:

- *inability of geostatistical estimates to respond to geologically distinct boundaries when gradational behaviour has been assumed;*
- *the difference between setting cut-offs on assays versus smoothed block grades;*
- *the sensitivity of the estimate to the nugget effect;*
- *the potential for more low-grade tonnage according to the geostatistical block estimate compared with polygonal grade control estimates;*
- *the problem of deciding whether a block is ore or waste in a high nugget effect environment; and*
- *the optimum resource drill density pattern.*

The operation has committed to a trial campaign processing material classified as low-grade (0.7 - 1.3 g/t material based on assays) which is equivalent to an apparent block cut-off of 0.85 g/t. If the model is correct the low-grade material should be 0.15 g/t to 0.2 g/t higher grade than shown in the production inventory and this will verify the volume variance relationship in a high nugget environment. If not, the case study will illustrate the dangers of allowing geostatistical model to wander beyond the limits of geological reality.

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INTRODUCTION

McKinnons open pit gold mine commenced mining in January 1995, based on a final pit design proved ore reserve of 2.39 Mt at a grade of 1.75 g/t gold with 134 200 contained ounces. This was defined by a close spaced RC drilling pattern ranging from 12 m by 12 m to 6 m by 6 m. An erratic gold distribution required further grade control drilling (blast hole sampling) during mining for ore boundary definition. Mining was completed in December 1996, with stockpiled run of mine (ROM) ore (>1.3 g/t) milled to August 1997. The low-grade stockpiled ore (0.7 – 1.3 g/t) is currently being milled. Detailed mapping and ore monitoring occurred during the mining and milling cycle, to validate parameters used by the Ore Reserve estimate. This paper analyses the similarities and discrepancies between the final pit ore reserve estimation with final ROM milled tonnes and grade along with grade control estimates of low-grade stockpile material.

McKINNONS GOLD DEPOSIT

The McKinnons gold deposit is described by Elliott (1995), Rugless and Elliott (1995), Bywater *et al* (1996) and Elliott *et al* (1998) so only a brief outline of the regional and local geology will be provided in this paper. A summary geology map of the McKinnons gold deposit is shown in Figure 1.

The Cobar Basin comprises Early Devonian marine sediment and minor volcanic rocks deposited in a deep sedimentary basin with flanking shallow-water sediments along its margins (Glen, 1987). The first deformation folding event occurred shortly after deposition at 390 – 400 Ma followed by a second deformation event in the mid-Devonian.

The McKinnons gold deposit outcropped on a low lying ridge of silicified weakly metamorphosed monotonous laminated to massive siltstone, mudstone and fine grained quartz sandstone, known as the Amphitheatre Group, near the western margin of the basin. The deposit is up to 350 m long, 120 m wide and 80 m deep. Mineralisation occurs from the surface (202 mRL) to the 122.5 mRL, with highest grades centred around 190 m to 170 mRL. Textures show that gold has formed by primary and secondary (supergene) processes. In the oxide zone, gold is enriched forming economic mineralisation associated with iron oxide and quartz. In the primary zone gold is largely uneconomic averaging around 0.3 g/t. It occurs in quartz veins, silica breccia and pyrite veins. Alteration associated with mineralisation includes an extensive outer carbonate, intermediate pyrite and inner silica zones.

Mineralisation is structurally controlled forming near the intersection of steep faults in the hinge zone of an anticline resulting in complex mineralisation controls. Both infill and replacement silica textures are associated with mineralisation. The ore deposit trends NW with internal controls on individual pods oriented north and north-east parallel to cleavage and fault directions. Bedding has variable dips between 15° - 35° towards the north and north-west, with local disruptions common in fault wedges.

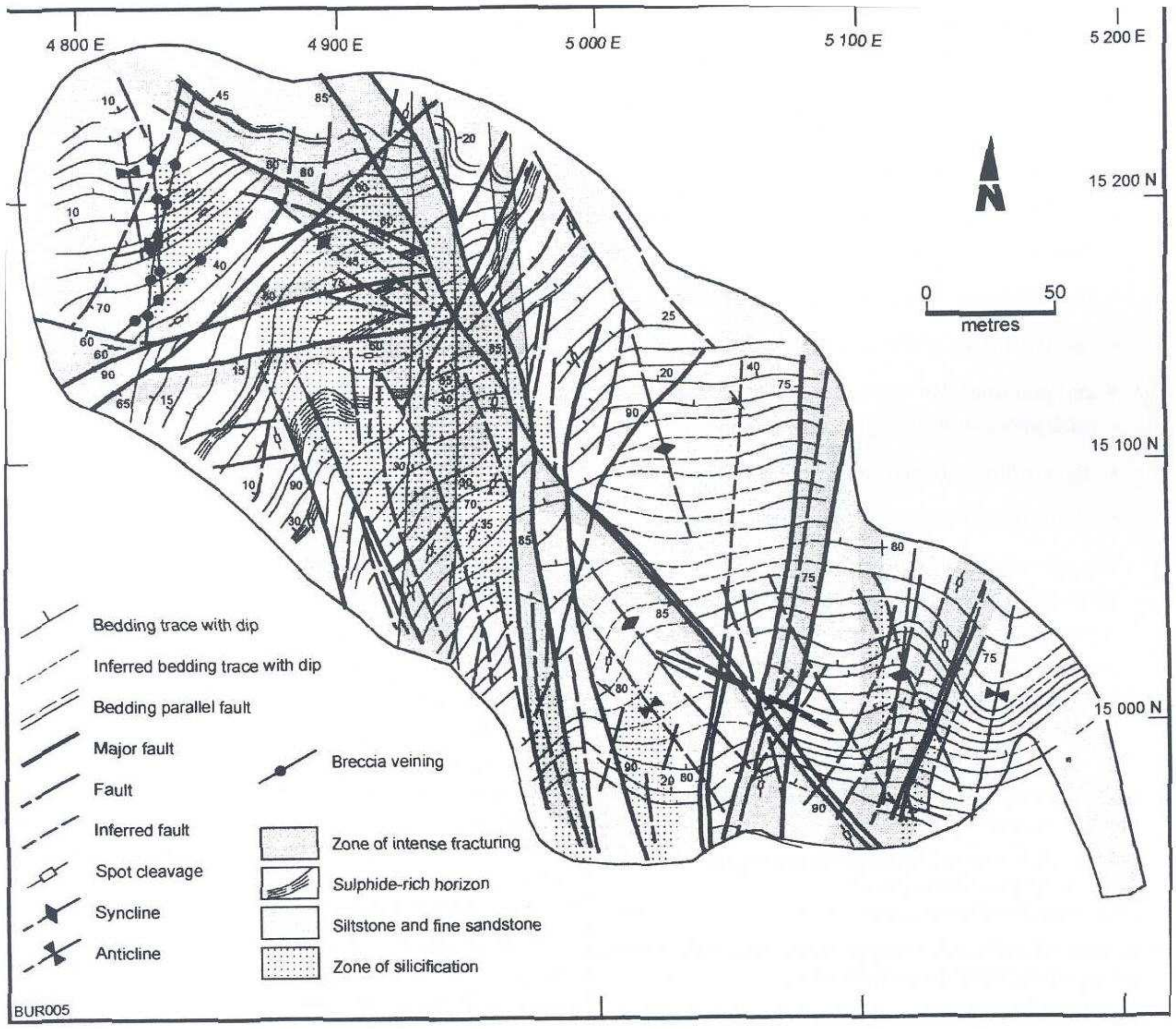


Figure 1 – Geology of the McKinnons gold deposit 180 mRL (after Elliott et al, in press)

The bulk of McKinnons mineralisation is situated within one main ore block in the central portion of the pit (Figure 2). The boundary of this pod forms an irregular polygon that required detailed drilling (3.5 m x 3.5 m pattern) for accurate boundary definition. A major 040° trending breccia zone bounds the western side of this block, and it is bounded to the NE by a 325° trending fault. Ore controls to the SE and SW boundaries are less readily definable. Figure 2 shows that other smaller pods occur to the SE of the main pod. Low-grade mineralisation has been more difficult to define visually, with irregular quartz veining and minor to absent silicification.

Although ore feeder fault structures assist in controlling the location of the ore pods, geological features cannot be used alone to define ore boundaries, which are largely dependent on assay grades.

RESOURCE DEFINITION AND DATA QUALITY

A range of drilling methods has been used to outline the resource at McKinnons. Geological and sampling data are essentially dependent on RC drilling methods.

Geopeko completed diamond, cross-over RC percussion and open hole percussion drilling at McKinnons between 1990 and 1992 .A total of 65 holes were drilled, totally 4920 m percussion and 1070 m diamond. Between December 1993 and June 1994, Burdekin completed 98 face sampling RC percussion holes totalling 5743 m and five diamond holes totalling 389 m. Most drilling of the deposit has been carried out on 25 m spaced east-west at approximately 20m intervals. Some holes were drilled oblique to the east-west grid to check continuity of mineralisation. The average hole diameter was 130 mm. Geopeko and Burdekin surveyed all drill collars and carried out down hole surveys.

Three to four kilogram samples were taken for assay every metre and reduced to 90 per cent passing 106 mm. Routine 50 g fire assay incorporating as AAS aqua regia finish was used with a lower detection limit of 0.01 g/t gold.

Burdekin adopted rigorous sampling and checking procedures during all its drilling programs to verify reliability of data (Elliott, 1995). The procedures included using independent control samples, inter-laboratory checks, replicate samples, duplicate samples, and large 3-4 kg samples reduced to a fine fraction before a subsample was taken for analysis. Accuracy and precision tests checked the reliability of the assays for mineral resource estimation. Accuracy is estimated to be

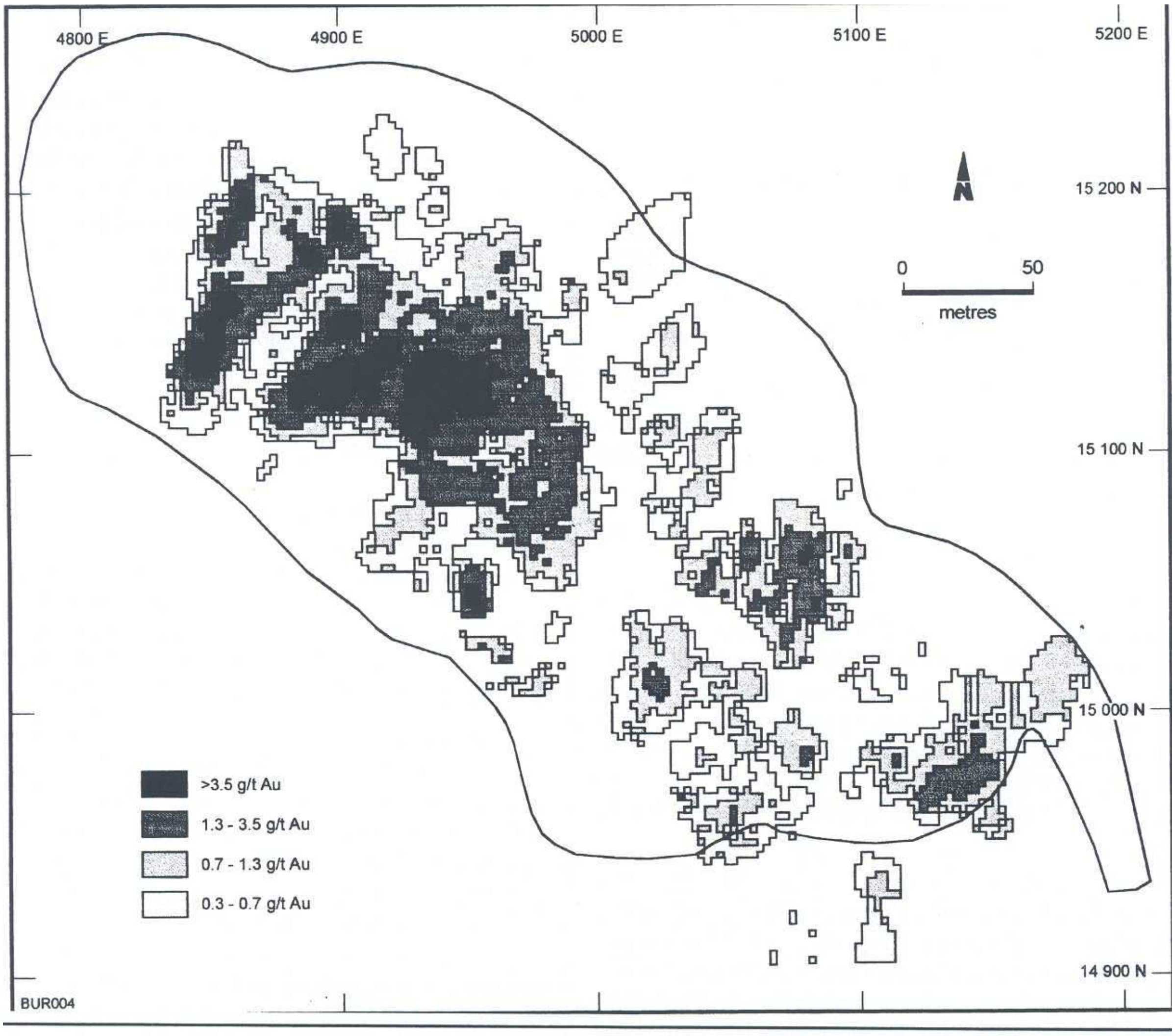


Figure 2 – Pit plan showing ore blocks at 180 mRL (after Elliott et al, in press)

between three per cent to seven per cent from control sample results and precision ranged from $\pm$9 per cent to $\pm$13 per cent at two standard deviations from duplicate assay results.

Twinning RC versus diamond and RC versus RC holes checked reliability of the drilling method. This check on drilling methods investigates assay value variability and smearing characteristics. Although differences were noted between twinned holes, they were interpreted to represent natural variability in the deposit's gold distribution rather than being the result of poor quality drilling, sampling or analytical methods. No smearing of gold grades was evident and all drilling in the deposit was carried out above the water-table. Sample recoveries were commonly constant and considered acceptable.

Another area for resource estimation error is bulk density determinations. They were carried out using drill core on ore and waste material throughout the deposit. A total of 233 determinations were made in the oxide zone and a bulk density of 2.5 t/m³ is used for waste and ore. In the oxide zone there is no apparent change in the average bulk density value with depth. Approximately ten per cent of the Ore Reserve occurs in the primary zone. Twenty-one ore and waste samples from the primary zone gave an average bulk density of 2.6 t/m³. The risk of a significant error in the bulk density affecting the resource estimate is considered low.

The reliability of the data used in the pre-development resource estimation was sufficient for it to be placed in a Measured Mineral Resource category according to the July 1996 JORC Code.

PRE-DEVELOPMENT RESOURCE DRILLING

A pre-development close spaced drilling program was completed over the entire deposit prior to the commencement of mining. The perceived main benefits for this strategy were:

1. to reduce project risk;
2. to enable mining rates to exceed treatment rates in a small pit to reduce mining costs;
3. to improve Lerchs-Grossmann (LG) optimisations and pit design; and
4. to reduce grade control costs and sample turnaround time during mining.

This program consisted of drilling 528, 120 mm face sampling RC holes, totalling 34 390 m between August and mid-October 1994 using two UDR650 rigs on double shifts. The deposit was drilled out on a 12 m by 12 m to 6 m by 6 m pattern. The same rigorous procedures for checks on quality control used in the ore delineation program were also adopted in this program. The main difference was that a 20 g aqua regia digest with an AAS finish was adopted instead of the 50 g fire assay.

The Ore Reserve estimated at June 1994 (2.2 Mt at 1.91 g/tAu; 135 200 oz) and the January 1995 final pit estimate (2.39 Mt at 1.75 g/tAu; 134 200 oz) was similar. Mineralisation would have been sterilised in the ramp by the original pit design but the new pit design was able to include it as ore. The benefits of this pre-development drilling that cost $1.2 million and added four months to the timetable can be reviewed now that mining has been completed.

It is evident that detailed quality data collected from the McKinnons deposit for Ore Reserve estimation does not stop errors introduced by poor grade control/mining practice or inappropriate resource modelling parameters or techniques. McKinnons may have benefited more by not undertaking the detailed pre-development drill out program and undergoing a more traditional detailed grade control program during mining. The planned reconciliation of the low-grade material will clarify this.

MINING

Open pit mining at McKinnons involved excavation, drill and blast by a contractor with grade control, ore delineation, extraction and milling by Burdekin personnel (Discombe and Engelhardt, 1996). To optimise profit, the mining occurred on double shifts with the mining rate higher than the milling rate. Mining commenced in February 1995 and was completed in December 1996. Five metre benches were each mined in two 2.5 m flitches. Treatment by the 500 000 tpa CIP plant is planned to continue until the end of 1998. Treating the ROM ore (>1.3 g/tAu) was completed in August 1997 allowing mill tonnage and grade reconciliations to be carried out to check original assumptions and parameters used in the ore reserve estimation.

RESOURCE MODELLING

A geostatistical modelling technique was adopted using indicator kriging. The resource was modelled on 2.5 by 2.5 m blocks on 2.5 m flitches based on uncut 1 m assay data. The gold grades approach a lognormal distribution in all seven domains, high-grade outliers are present and there is evidence of mixed populations. The global statistics for 1 m composites (n = 48 455) above a mineralisation indicator grade of 0.1 g/t reflect an average grade of 1.60 g/t with a variance of 22.2 (log variance 1.55). Individual domains range in average grade from 0.9 g/t to 1.9 g/t (above 0.1 g/t).

Full indicator variography was undertaken for each of nine indicator grades, based on the deciles of the data above the mineralisation indicator grade. The subtle anisotropy interpreted from the variography confirmed the changes in strike and dip interpreted geologically. Short range structures of 4 m to 12 m, up to a maximum of 22 m were nested within structures of the order of 20 m to 60 m. The nugget effect was notably high for all but one domain, comprising 45 per cent to 70 per cent of the total variability. This reflects the poddy nature of the mineralisation.

Full indicator kriging was used to interpolate block grades and indicator cut-offs were set at the data deciles (10^{th} to 90^{th} percentile) plus the 95^{th} and 97.5^{th} percentiles of each data set. This method of interpolation is appropriate for skewed data and mixed populations such as exist at McKinnons. It was not possible to define precise geological boundaries or to accept that a robust mineralisation envelope (0.1 g/t) existed at the time. Therefore soft boundaries were applied between domains. Concern about excessive edge dilution was addressed by re-running the model constrained within all blocks exceeding 0.25 g/t, using only the data from within this constraint. Hard boundaries were used at the edge of the mineralisation envelope and soft boundaries within the envelope, between the domains. Owing to time constraints median indicator kriging was used for the second model and the data was cut to the 99th percentile. There was no significant change to the resource estimate.

GRADE CONTROL METHOD

Grade control was undertaken using a manual polygonal methodology incorporating the data from the original RC holes plus blast hole data to refine the edges of the mining block. Ore boundaries were digitised in plan for each 2.5 m flitch, with grades calculated by arithmetic average. No smoothing was applied at this stage. A top cut of 15 g/t was applied to raw RC and blast-hole data, as determined by statistical analysis and reconciled trial ore parcels. This technique resulted in the strict application of a 1.3 g/t assay cut-off for a high-grade material and a 0.7 g/t cut-off for low-grade material, with 0.3 - 0.7 g/t material being delivered to the mineralised waste stockpile.

Dilution in the pit was reduced by drilling 5 m deep holes and inserting poly-pipe, then surveying the pre and post-blast positions to monitor movement. Ore boundaries were then adjusted for heave to reduce dilution.

A detailed reserve drilling had been undertaken prior to mining, grade control drilling during mining was restricted to strategic areas for defining boundary margins. This would involve very little sampling in the high-grade central cores of large pods. This caused the grade control estimated grade of the ROM ore to be under-estimated (2.36 g/t) compared to milled grade (2.65 g/t).

Approximately 30 samples were taken of the low-grade stockpile each month during mining as a check on pit grade control values. Although stockpile sampling is considered a rough method it serves as a guide. The 0.75 g/t average of 314 stockpile samples is lower than the grade control value of 0.91 g/t gold.

GEOSTATISTICAL CHARACTERISTICS

A retrospective study of the statistical relationships between blast hole assays, exploration RC assays, block model grades and production has highlighted a number of classical issues related to the following aspects of resource estimation:

- inability of geostatistical estimates to respond to geologically distinct boundaries when gradational behaviour has been assumed;
- the difference between setting cut-offs on assays versus smoothed block grades;
- the sensitivity of the estimate to the nugget effect;
- the potential for more low-grade tonnage according to the block estimate compared with polygonal grade control estimates;
- the problem of deciding whether a block is ore or waste in a high nugget effect environment; and
- the optimum resource drill density pattern.

Nugget effect

The nugget effect describes how well sampling results can be reproduced by repeated sampling at the same location. It incorporates both the natural inherent variability of the deposit plus variability because of sample size, sample preparation and analysis. Heterogeneous mineralisation is sensitive to the method of sampling and could give variable results from a single location. Precision checks previously given indicate sampling and analytical errors are at acceptable levels.

Recognition of the nugget effect is vital to resource estimation. The higher the nugget effect, the higher the degree of smoothing required in the estimation where the estimate is a weighted average of samples within the range of influence of the block being evaluated. Kriging is an estimation technique which includes the nugget effect in the derivation of sample weights. The higher the nugget effect the higher the degree of smoothing; that is samples are more evenly weighted. If the nugget effect is low enough the block derives its average grade from the closest sample grade. Otherwise the block grade is a weighted average of the samples within the range of influence.

Volume variance effect

The volume variance effect, which reflects the change of support and hence the decrease in variability between sample sized volumes and blocks representing selective mining units, causes a regression relationship between sample estimates and actual block grades. Samples overestimate block grades in high-grade areas and underestimate them in low-grade areas. Cut-offs should be applied to block estimates in order to optimise ore/waste decision and avoid misclassifying ore as waste.

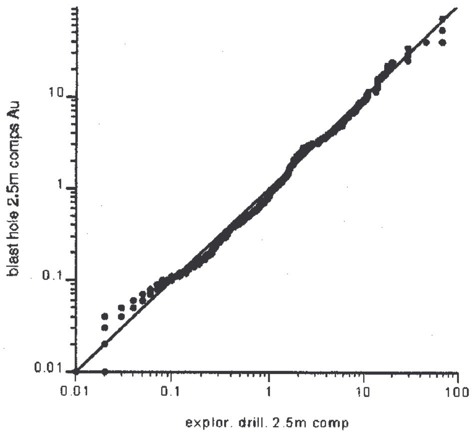
Statistical reconciliation within test area

A comparison between 2.5 m blast hole samples, exploration RC composites and block model estimates in a test area representing the extent of all blast hole sampling at McKinnons is summarised in Table 1 and the Q-Q plots in Figures 3 to 5.

TABLE 1

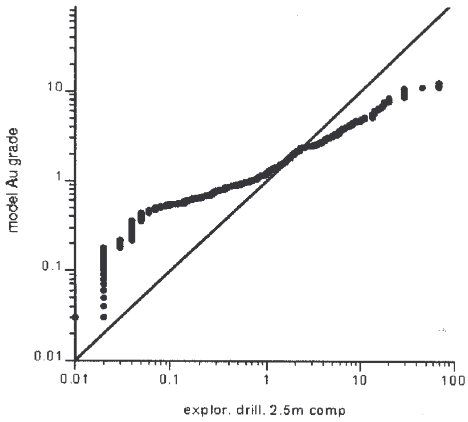
Comparative 2.5m statistics in test area

	Model blocks 2.5 x 2.5 x 2.5m	Blast hole samples 2.5m bench	Exploration RC 2.5m composition
Number		3860	
Minimum	0.03	0.01	0.01
Maximum	12.55	108.0	69.03
Mean	1.08	1.20	1.24
Median	0.77	0.26	0.31
Log Variance	1.26	2.97	3.22



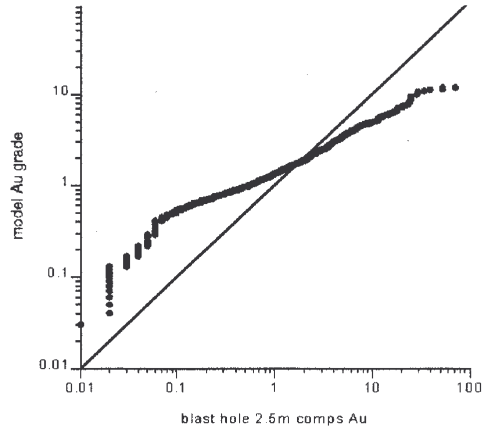
McKinnons B4 and A8 Benches

Figure 3 – Q-Q plot comparing 2.5 m composites from RC and blast hole sampling



McKinnons B4 and A8 Benches

Figure 4 – Q-Q plot comparing model blocks with 2.5 m RC composites



McKinnons B4 and A8 Benches

Figure 5 – Q-Q plot comparing model blocks with 2.5 m blast hole composites

There is no bias between the exploration RC and blast hole grades as evidenced by their comparable statistics and the 1:1 relationship in Figure 3. Comparisons between block grades and either raw exploration RC sample grades or blast hole sample grades (Figures 4 and 5 respectively) both illustrate a marked regression caused by the lower variability of the model block grades. In this case the block grades are higher than the sample grades in the low-grade areas and lower than the sample grades in the high-grade areas. The overall block average is similar to the average sample grades. The log histograms and log probability plots in Figures 6, 7 and 8 illustrate the smoothing in the model block grades compared with the raw data. Delineation of mining blocks at cut-offs below 1 g/t would outline more tonnes at higher grades than polygons defined on raw assay grades (either blast hole or RC).

Conditional bias and kriging efficiency

The relationship between input assays and actual block grades can be predicted from the variography using the method recom-

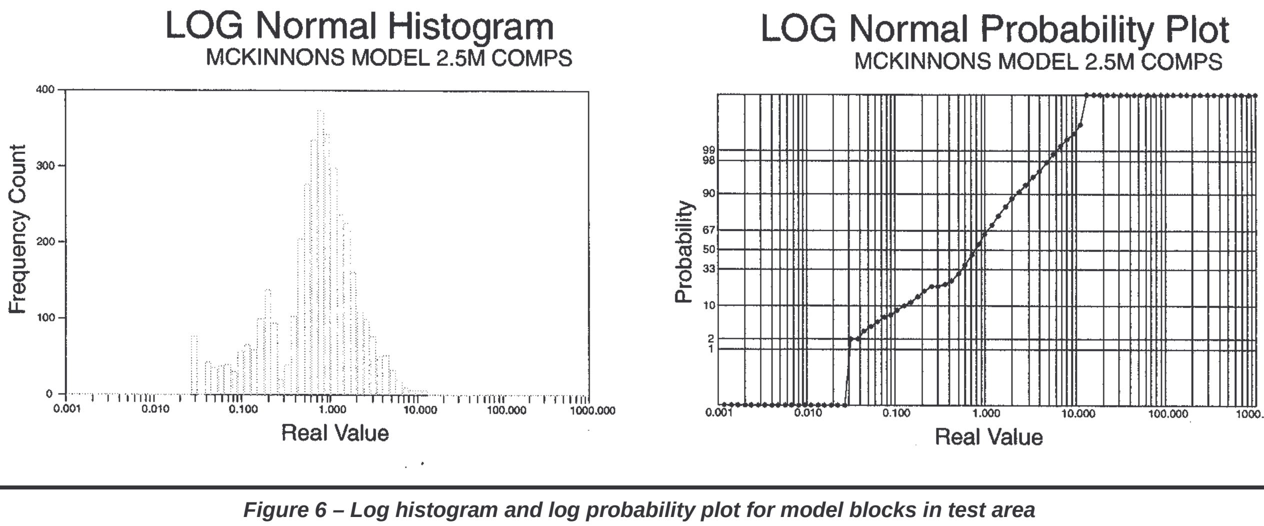


Figure 6 – Log histogram and log probability plot for model blocks in test area

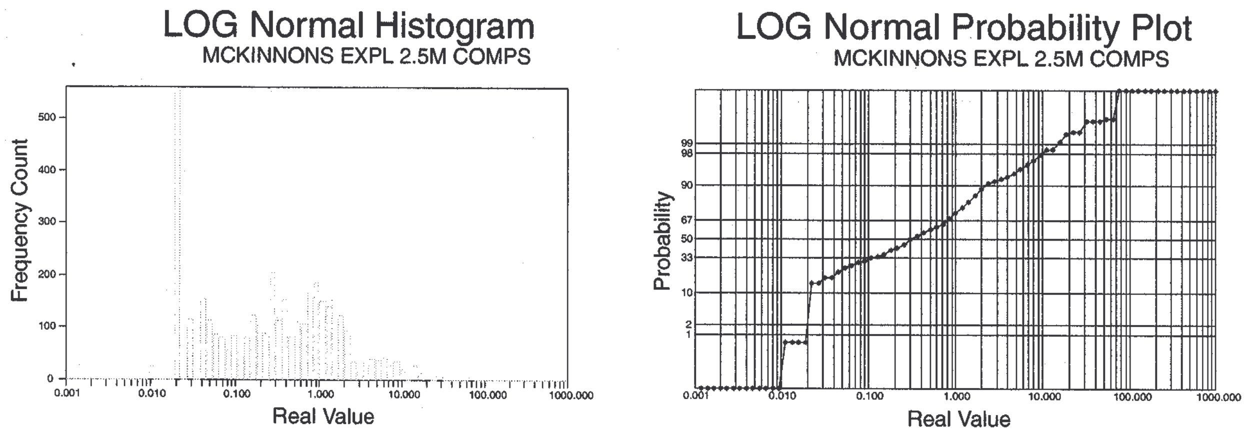


Figure 7 – Log histogram and log probability plot for 2.5m RC composites in test area

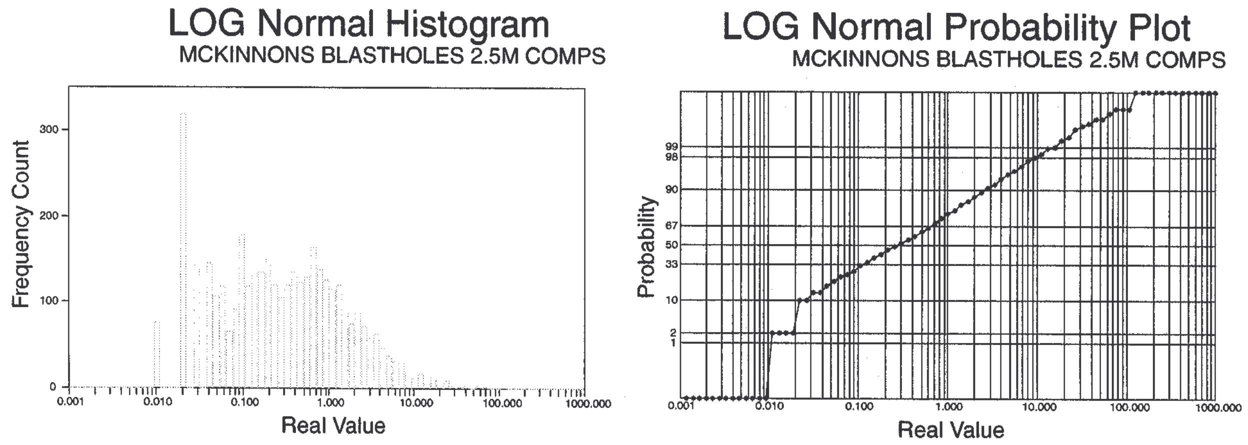


Figure 8 – Log histogram and log probability plot for 2.5m blast hole composites in test area

mended by Krige (1986) to review whether the model has the appropriate amount of smoothing according to geostatistical theory.

Table 2 shows the regression co-efficient, which is the measure of conditional bias, and kriging efficiency which is the measure of reliability for a given block size and grid spacing using a typical variogram from one of the domains at McKinnons.

TABLE 2

Regression coefficient and kriging efficiency

Sampling grid	Regression co-efficient	Kriging efficiency
8 x 8m grid, 1m comps	0.89	48%
8 x 8m grid, 2.5m comps	0.93	49%

These parameters suggest that the kriged blocks would not have much conditional bias (a regression co-efficient of 1 is perfect) but kriging efficiency is low (above 50 per cent is considered advisable by Krige). The kriging efficiency would be improved by increasing the block size.

The apparent discrepancy between sample and block grades is, according to this test, an expected outcome and it suggests that the block grades would eventuate if the cut-off was set on smoothed estimates. This is an important relationship to understand and, if block grades are indeed realistic, there is potential to retrieve more ore that may have been misclassified as waste using a manual polygonal grade control method. However, although there may be no bias as such, the reliability of individual block estimates could be low. Hence highly selective mining would be likely to fail. Ore/waste decisions would most optimally be made on larger selective mining units.

MODEL VERSUS PRODUCTION COMPARISONS

At the onset of mining, it became evident that there was reasonable reconciliation of high-grade material with model expectations but that low-grade tonnage was substantially reduced (see Table 3).

TABLE 3

Reconciliation and feasibility, grade control and mill Ore Reserves.

Feasibility reserves final mine pit design (Jan 1995)			Grade control and mill reserves		Difference		
Grade	Tonnes x 1000 t	Grade g/t Au	Tonnes x 1000 t	Grade g/t Au	Tones %	Grade %	Contained Au %
ROM	1151	2.62	11751	2.651	+2	+1	+3
Low	1239	0.94	5632	0.912	-55	-4	056
Total	2390	1.75	1738	2.09	-27	+19	-13

Notes:

1. **Mill reconciled Ore Reserve for ROM at August 1997.**
2. **Grade control Ore Reserve of low-grade, survey adjusted in January 1997.**
3. **Grade control Ore Reserve of ROM is 1 215 000 t grading 2.36 g/t Au.**

Reconciliation figures for ROM (>1.3 g/t) between grade control and milled tonnes and grade indicates a contained gold variance of +3 per cent. The overall loss in contained gold of -13 per cent is largely attributed to the dramatic drop in mined low-grade tonnes (-55 per cent) when compared to the Ore Reserve model.

Detailed grade control drilling and bench mapping in conjunction with face channel sampling showed that ore boundaries are often well defined as opposed to being gradational boundaries as assumed in the ore reserve model by using essentially unconstrained boundaries.

Figure 9 shows the grade tonnage curve for the model reported within the pit outline as well as within the limits of the mining blocks on 12 consecutive benches mined at McKinnons (175 – 205 mRL). Also plotted is the production estimate for cut-offs of 0.3 g/t (mineralised waste), 0.7 g/t (low-grade) and 1.3 g/t (high-grade).

Within the mining outlines, it appears that the 0.7 g/t assay cut-off represents an effective block cut-off of 0.85 g/t. Mining to a block cut-off of 0.7 g/t would have presented 11 per cent more tonnes at nine per cent lower grade than was actually mined as low-grade. The resource model defines further low-grade material external to the mining outlines accounting

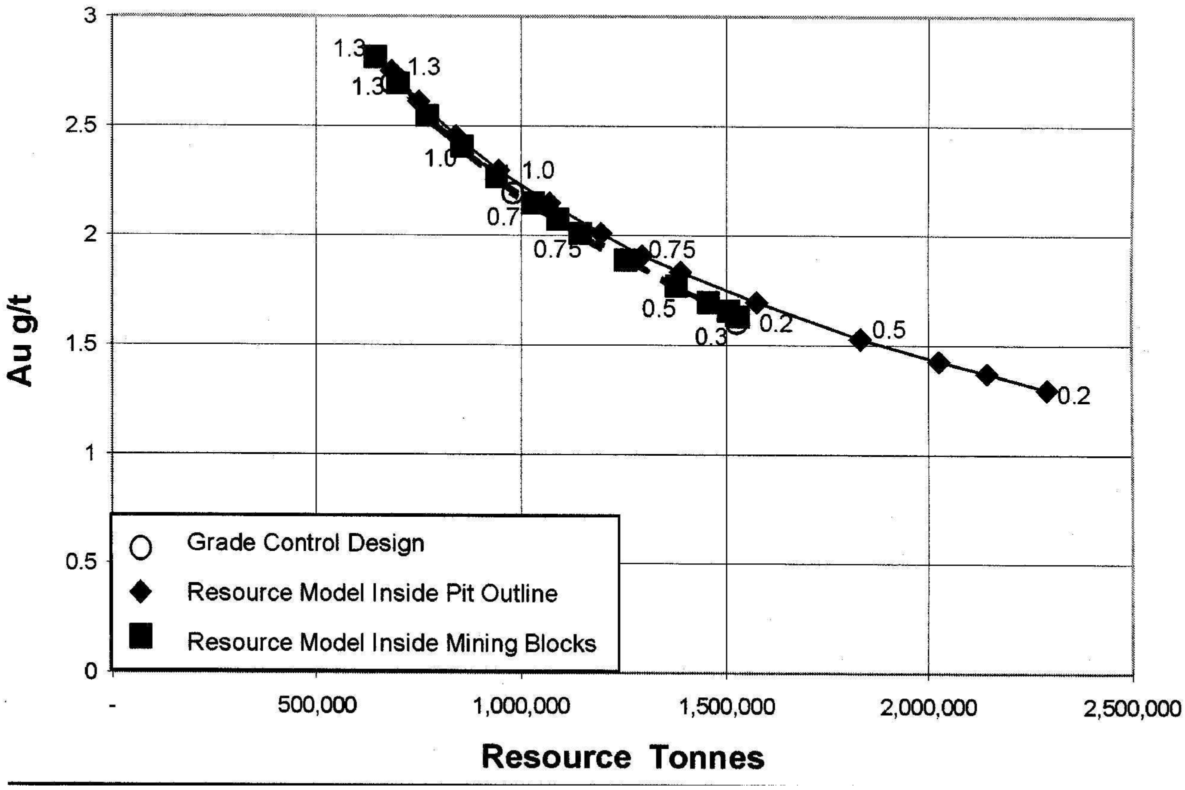


Figure 9 – Grade tonnage reconciliation between model and production estimates (174 – 205 m RL)

for an additional 11 per cent in tonnes, with an overall decrease of nine per cent in grade. The significance of this discrepancy is even more marked at the 0.3 g/t cut-off for which the model defines a resource of 2.1 Mt at 1.37 g/t compared with 1.5 Mt actually mined at 1.6 g/t.

However, besides examining data statistically it is important to also examine it spatially to assist in understanding the problem. The block model plans indicate a smearing effect or zonation from high-grade to low-grade and through to mineralised waste. This is not observed in the actual spatial distribution when manual ore boundaries were applied associated with raw drilling data. A typical comparison between raw RC data with the overlying block grade for a low-grade section of the pit is given in Table 4. Note that there are many examples where significant differences exist between the raw grades and corresponding block grade. At the mine a manual cut-off grade was used because there appeared to be a sharp cut-off between 0.7 g/t material and waste.

This is demonstrated by Figure 10 which shows the 0.7 g/t block model boundary being significantly larger than the raw grade control 0.7 g/t boundary. This relationship is observed on each bench and may account for the significant overcall in low-grade tonnage (or extra 'ore', if the model is correct) suggested by the geostatistical ore reserve modelling technique.

DISCUSSION

Whether the extra 'ore' is real or not is yet to be decided. At the mine site it is firmly believed that because the assay data is lower than the block grades in these low-grade areas – in fact many samples are totally barren – the block model must be in doubt for the low-grade categories. This is explained, as being because of the existence of sharp mineralisation contacts which the unconstrained model did not consider. These contacts are difficult to map in the pit in relation to iron staining; silicification and faults so in practise the ore boundaries are largely based on assay grades. Statistically, if there are no visible geological boundaries the combination of high nugget effect and the volume variance effect lends credence to the possibility that there could be extra 'ore' on the mineralised waste stockpiles and mineralised waste on the waste stockpiles.

In an attempt to understand the relationships observed, the operation has committed to processing material classified in the pit as low-grade (0.7 – 1.3 g/t material based on assays) which is equivalent to an apparent block cut-off of 0.85 g/t. The results of this trial should be definitive as to the merits of using block grades compared with assay grades at McKinnons. If the model is correct, the low-grade material should be about 0.15 to 0.2 g/t higher than shown in the production inventory. The high nugget effect and the volume variance relationship could have hidden some or all of the missing low-grade material. If not, this case study will illustrate the dangers of allowing geostatistical models to wander beyond the limits of geological reality. In the case of McKinnons, this was a difficult limit to set on close spaced exploration RC drilling and remained difficult even using blast hole data and pit mapping. The difficulties experienced both during resource modelling and mining highlight the complexity of the McKinnons deposit and this reconciliation exercise reaffirms the importance of taking cognisance of the full spectrum of technical parameters from feasibility through to mining in an attempt to ensure optimisation at all stages of the project.

TABLE 4

Comparison of raw RC grades with block model grades

(182.5m – 180 mRL)

No of samples	RC grade g/t Au	Classification	Bloc grade g/t Au	Classification
4	0.02	W	0.42	MW
5	0.04	W	1.13	LG
3	0.25	W	0.45	MW
4	0.04	W	0.29	MW
4	0.02	W	0.44	MW
4	0.15	W	0.69	LG
3	0.03	W	0.27	MW
4	0.24	W	1.89	ROM
4	0.02	W	0.57	MW
2	0.43	MW	0.81	LG
4	0.30	MW	0.72	LG
4	0.63	MW	0.75	LG
4	0.49	MW	0.78	LG
4	0.08	W	0.87	LG
4	0.09	W	0.35	MW
4	0.49	MW	1.38	ROM
4	0.26	W	0.96	LG
4	0.15	W	0.85	LG
4	0.37	MW	1.36	ROM
4	0.10	W	0.63	MW
4	0.42	MW	0.76	LG
4	0.15	W	0.90	LG

Notes:

W – Waste (<.3 g/t)

MW – mineralised waste (0.3 – 0.7 g/t)

LG – low-grade (0.7 – 1.3 g/t)

ROM – Run of Mine grade (>1.3 g/t)

CONCLUSIONS

The accuracy of the feasibility ore reserve tonnes at McKinnons appeared to vary substantially between high-grade (>1.3 g/t) and low-grade (0.7 – 1.3 g/t) as defined by later grade control and mill reconciliations. The high-grade material milled is within one per cent to two per cent of the feasibility model's tonnes and grade which is an excellent result. However, the low-grade material determined by grade control

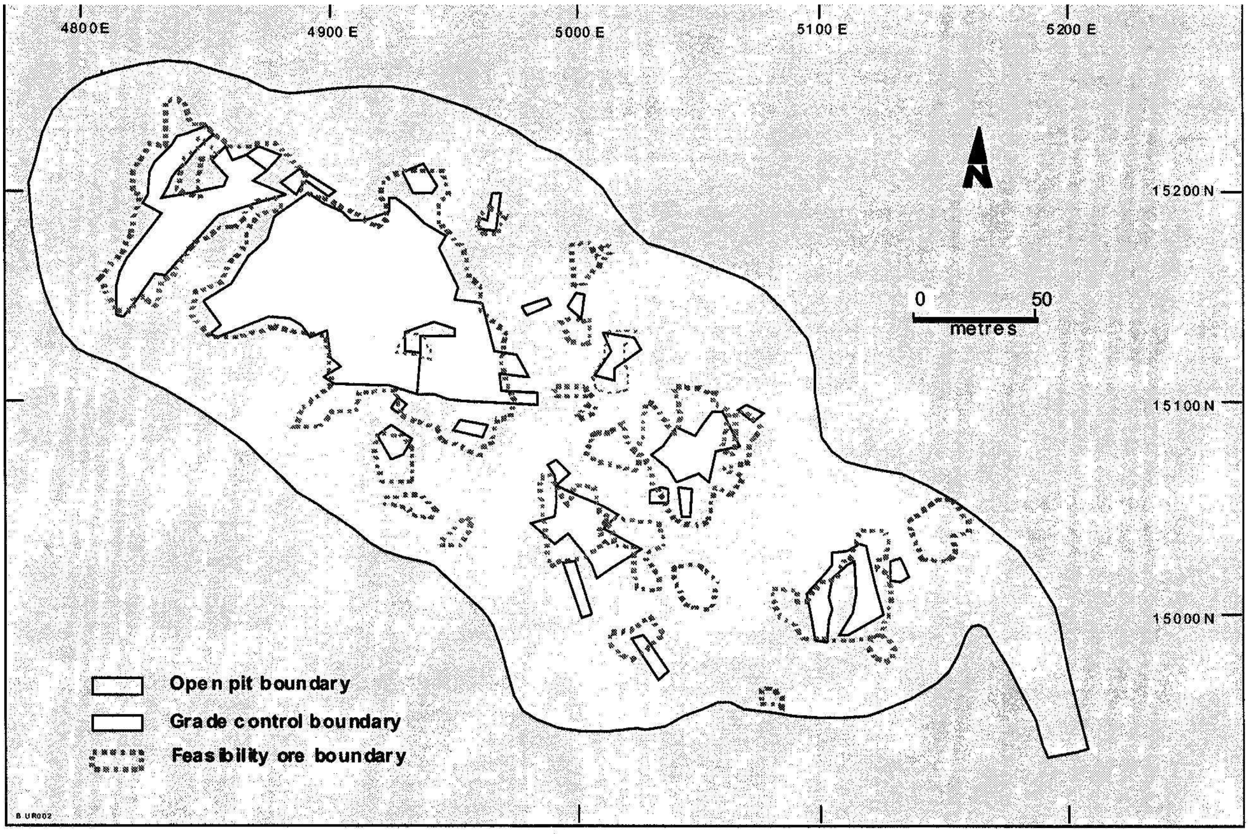


Figure 10 – Pit plan showing grade control and block model 0.7 g/t Au boundaries.

shows a 55 per cent shortfall using the same modelling parameters. This has led to an estimated 13 per cent shortfall of contained gold.

If the geostatistical modelling is correct, it would suggest misclassification of low-grade tonnage during mining caused by the high nugget effect and the volume variance relationship. A higher block grade would be expected compared with the arithmetic average of raw drill assays for the mined low-grade pods. Site personnel believe the unconstrained modelling method used may have created non-existent low-grade blocks, citing as evidence stockpile sampling, sharp mineralisation boundaries based on close spaced grade control patterns, bench mapping and face channel sampling.

This issue should be resolved after the low-grade material has been processed to check if the grade is higher than estimated by grade control methods (1.1 g/t instead of 0.9 g/t).

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ADDENDUM

Low-grade material has now been processed at McKinnons permitting the following addendum to this article. Mining was completed in December 1996 and treatment of the ROM ore finalised in November 1997. Treatment of low-grade ore commenced in August 1997 and was finalised in October 1998 and treatment of the mineralised waste commenced in May 1998 and is still in progress.

As mentioned, site expectations based on grade control were that the low-grade ore would average 0.9 g/t whereas the kriged block model indicated that the average grade of this ore would be higher (1.1 g/t). During the months of December 1997 to April 1998 only material from the low-grade stockpile was being treated. Based on these five months the average grade of the low-grade ore is 1.08 g/t, which is in line with the block model prediction.

Unfortunately the figures provided from the stockpiles differ from those reported as having been crushed and milled. In order to undertake a full reconciliation of the ROM and low-grade ore, tonnages have been obtained from back-calculation of the proportion of each ore type recorded from the stockpiles and the reported total milled tonnes. The monthly reported grade, calculated tonnages and the average grade of the low-grade ore (based on the assumption that the average grade from December 1997 to April 1998 is representative of the low-grade ore stockpile) permitted calculation of the overall milled tonnes and grade of the ROM and low-grade ore. These figures are listed in Table 5.

The kriged block model was reported within the final pit outline and the predicted resource is presented as a grade-tonnage curve in Figure 11. The milled low-grade ore lies on the grade-tonnage curve predicted by the block model, but is at a block cutoff grade of 1.0 g/t. The tonnage and grade of the low-grade ore was accurately predicted by the kriged block model, but grade control practices have outlined ore based on an effective block cut-off grade of 1.0 g/t rather than the 0.7 g/t planned.

TABLE 5

Milled ore to October 1998 (based on back-calculation of monthly production figures).

	Tonnes	Grade g/t Au	Contained Au
Total ROM (>1.3 g/t)	1 204 000	2.64	3 177 000
Total LG (0.7 – 1.3 g/t)	399 000	1.08	430 000
Total MW (0.3 – 0.7 g/t)	251 840	0.74	186 050
Total (>0.3 g/t)	1 854 624	2.05	3 793 787

The kriged block model thus accurately predicted the tonnes and grade of the ROM ore and illustrates that the low-grade ore was mined at a higher block cut-off grade than intended, resulting in the misclassification of low-grade ore as mineralised waste. A higher than anticipated cut-off grade has also been applied during delineation of the mineralised waste. The back-calculated average grade of the mineralised waste is 0.74 g/t Au which is 0.23 g/t higher than grade control expectations. Some mineralised waste may have been classified as waste, which may have resulted in the overall loss of gold.

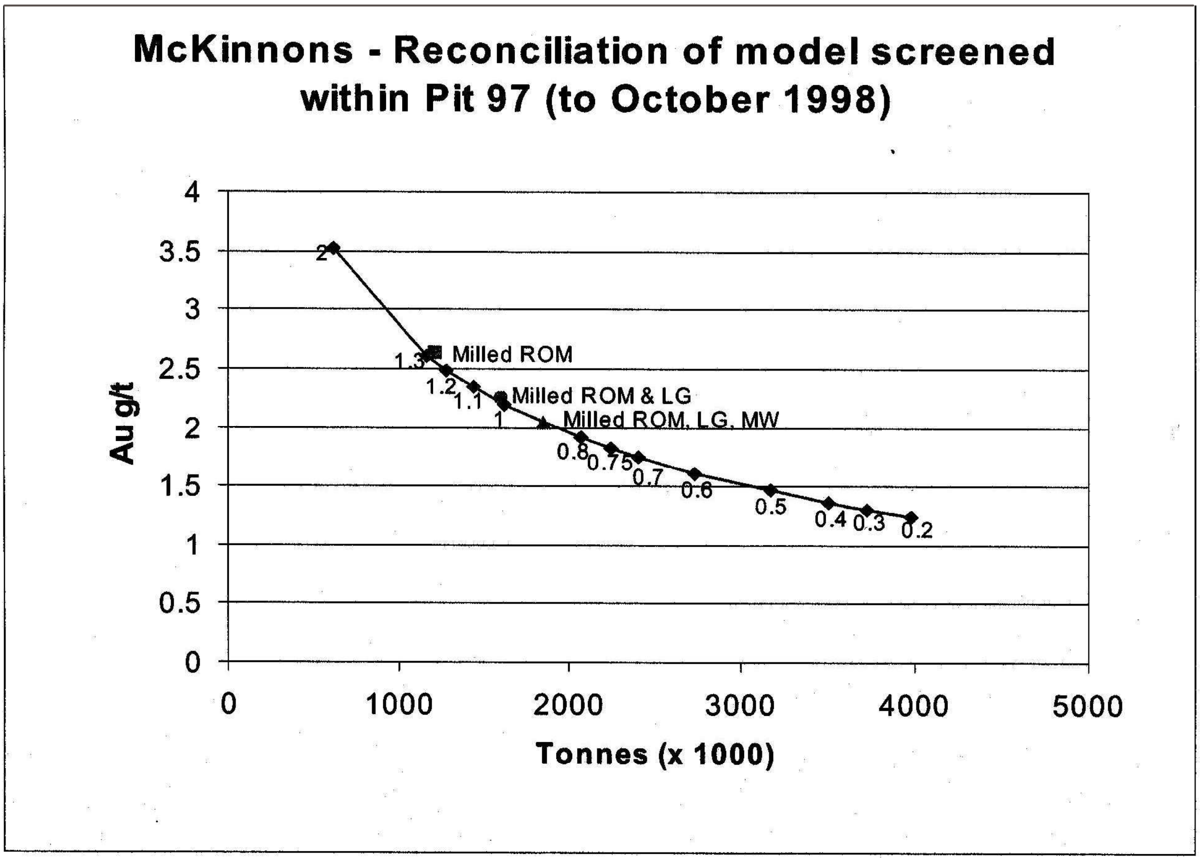

Figure 11 – Grade tonnage reconciliation between model and production figures. (ROM: 1.3 g/t assay cut-off, LG: 0.7 g/t assay cut off, MW: 0.3 g/t assay cut-off).

The regression effect and volume – variance relationship are illustrated in Figure 12. Where the cut-off grade is less than the mean grade of the sample data, then the blocks (as mined and predicted by the block model) will define more tonnes and less grade than the samples, whereas if the cut-off grade is greater than the mean grade then the blocks will define less tonnes and more grade than samples. At McKinnons the mean grade of the samples used to create the kriged block model is 1.3 g//t, which is the same as the ROM cut-off grade. Hence the samples and the block model both accurately predicted the tonnes and grade of ROM ore. The low-grade ore and mineralised waste cut-off grades, of 0.7 g/t and 0.3 g/t respectively, are both lower than the mean grade of the ample data. Definition of the low-grade ore and mineralised waste, based on the sample data, has defined less tonnes and more grade than predicted by the kriged model.

It is possible that some of the extra tonnage estimated at very low grades could be due to the lack of geological constraints applied to the limits of the model, particularly if structures are observed to be discrete rather than gradational as assumed during modelling. It is thus difficult to assert the location of the optimal cut off at the low-grade end of the grade-tonnage curve. There are many combinations of hard, soft or transitional boundary styles that may be inherent even within a single deposit.

This case study illustrates the pit-falls of ore block delineation based on individual assay results in a high nugget environment, where individual assay results are not representative of block grades and where ore delineation must be based on blocks. The recognition of the volume-variance effect could have prevented the misclassification of ore during mining. As an industry practice, grade control procedures should not place undue importance on individual sample results and should be cognisant of the fact that blocks are mined, not samples. Grade control personnel should be aware of the statistical and geostatistical sensitivities as well as the geological and mining constraints at an operation. A team approach to grade control is recommended to ensure the optimal production result is achieved by combining knowledge from all the relevant disciplines involved.

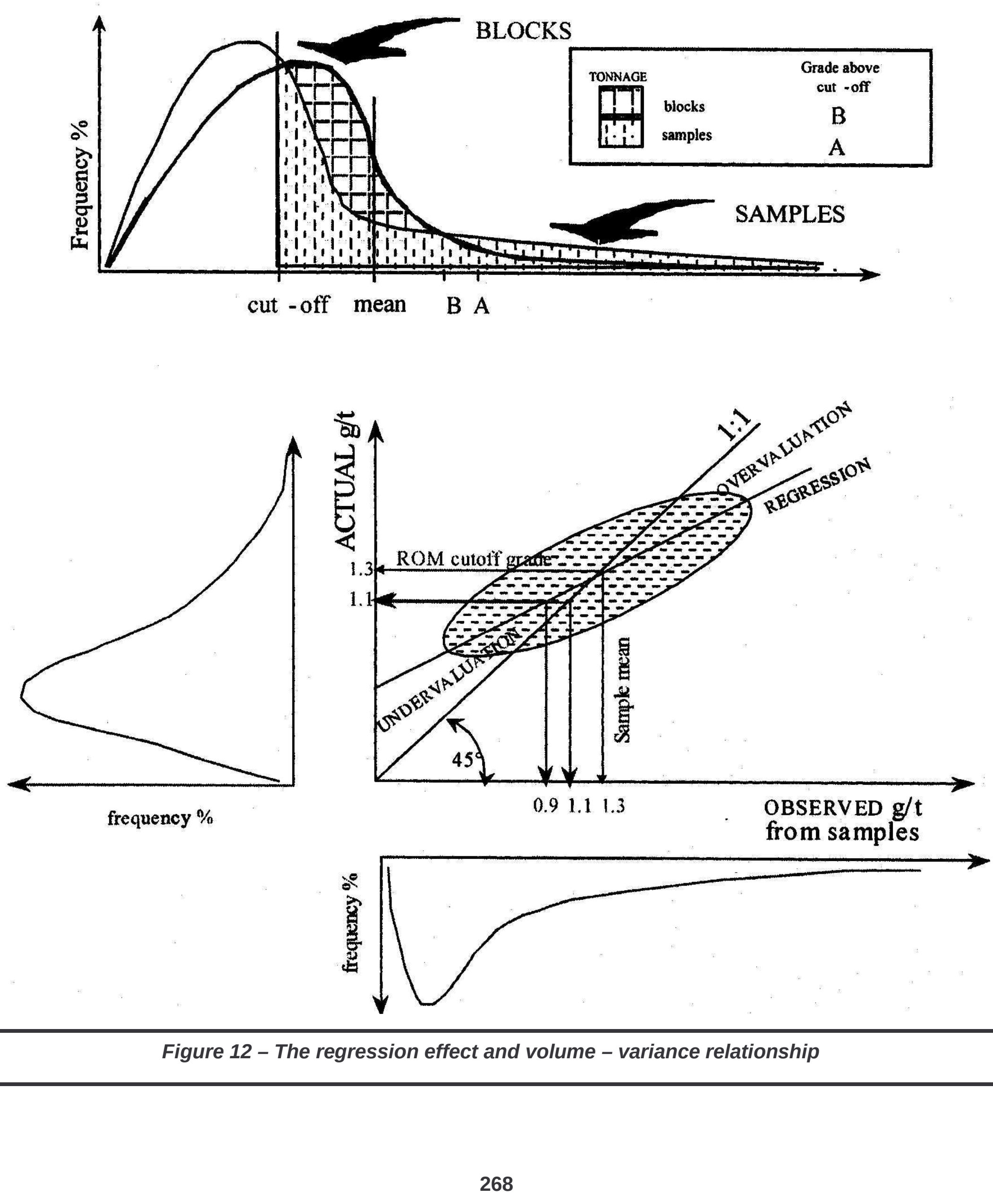

Figure 12 – The regression effect and volume – variance relationship