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CURRAGH RESOURCES INC

INTER-OFFICE MEMORANDUM

FARO OFFICE

DATE: December 7, 1990

TO: W. DUNN
CHIEF ENGINEER

FROM: DAVE TENNEY
CHIEF GEOLOGIST

SUBJECT: GEOSTATISTICAL REPORT

Attached are the following:

1) Letter to Dr. Dagbert dated October 22, 1990 with attached questions concerning the preliminary evaluation and the final Phase II report.
2) Letter from Dr. Dagbert to me with reply dated November 16, 1990.
3) Attached to above letter - Pages 1-4 of clarifications
4) Memorandum "Comparison of Mill Head Grades and Blasthole Predicted Crusher Feed Grade."
5) Also attached pages ii-iv of "conclusions and recommendations"
6) Also attached pages 2-12 of Phase I report with minor corrections.
7) Also attached pages 19,24,and 55 of Phase II report with corrections.

1,2,3,4, and 5 above should be added to Geostat's correspondence file. The new pages 2-12 (Phase I Report) and pages 19,24, 25 (of the Phase II Report) should be used to replace those pages in the reports already in your possession.

Any comments you may have on the final report would be much appreciated. When the new data has been digested we might wish to consider having another meeting with Dr. Dagbert to discuss in more detail the prediction of mill head feed grades from diamond drilling and blashole data.

D. Tenney.

Dave Tenney
Chief Geologist

DT:cc

cc: E. Beaumont
D. Basso
G. Jilson
J. Hendry
L. Pigage
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October 22, 1990

Dr. Michel Dagbert
Geostat System International
4385 rue St. Hubert, Suite 1,
Montreal, Quebec

Dear Michel:

Thank you for the final version of the Phase II report. Attached is a list of questions which also cover the "Preliminary Evaluation". Many of them result from my incomplete geostatistical education, and some concern what are obviously typos. I very much enjoyed reading your conclusions and recommendations, and am very interested in implementing Kriging as a means of predicting mill feed grades (and tonnages) from blasthole data. We will also have to review our method for calculating computer generated ore reserves.

It looks as though you were able to do a good job of answering nearly all our questions, and we are now in a potentially much better position to estimate grade and tonnage from the Faro Pit than we were previously. The question of evaluating the impact of incorrectly interpretated geology remains a problem which, as you pointed out, will not likely be solved by block modelling.

I am recommending we continue this geostatistical work with a study of the Grum Deposit.

If you like you can send me any updated pages and I will replace those now in the original.

Yours truly,
CURRAGH RESOURCES INC.

Dave.

Dave Tenney
Chief Geologist

DT:cc

attmts.

LIST OF QUESTIONS

1 - 9 - PRELIMINARY EVALUATION

10 - 22 - FINAL PHASE II REPORT

1) Figs: 4,5,17,18,21,25 - show three parallel straight lines. What do they represent. (regression line ± one S.D.?)

Preliminary Evaluation:

2) Page 2 - (Bottom) - correction -

- 2EC - pyritic semi-massive sulphide ore (40)
- 2E/F - massive pyritic ore (50)

3) Page 3 - correction - 2nd line paragraph # 2

- ....multiple regression ......(not linear)

4) Page 4 - correction - Bottom 2nd paragraph

- " Latest model (F 9003) uses power 1 (1/D) and 2 "

- Could you explain the ... and 2?

5) Page 5 - Paragraph # 4 -

" ....1/D weighting seems to give better results than $1/D^2$

Results in Table # 1 suggests both 1/D and $1/D^2$ are close to blasthole results.

6) Page 7 - Bottom Paragraph

"...... we don't compare block estimates with block true values: we rather compare the distribution of block estimates with the distribution of bench intercept values in the same area."

Are you saying we should do the latter rather than the former? What is the practical value of this statement?

7) Page 8 - Bottom line / Top Page 9

Are you recommending a block model with "Geological" block estimates? Does the complexity of geology at Faro really compare well to iron ore deposits in Labrador and Western Australia?

8) Page 10 - Paragraph # 1

".... linear regression...."

Should read "multiple regression".

9) Page 11 - 2nd Paragraph from bottom should read:

"…. but they are the only ones with digitized coordinates"

Phase II Report:

10) Page 19 - Top Paragraph

"Enveloppes" should read "envelopes" on both line 3 and 4.

11) Page 19 -

Could I interpret results in Table # 5 to get an inverse distance power of 1.2 or 1.3 instead of 1.4 in rock types 20 - 30?

12) Page 24 - Bottom Paragraph

Does "connectivity" mean "continuity" in a geological sense or in a statistic sense?

13) Page 24 - Bottom Paragraph

When you do indicator Kriging, is the variogram constructed from the indicator values "0" or "1" (zero or one)? I assume that the indicators ("0" and "1") are assigned to model blocks based upon the Kriged grade of those blocks.

14) Page 26 - Bottom Paragraph

I assume relative standard errors of monthly and yearly estimates referred to are grade estimates not metal content estimates. Am I correct?
Also is there any significance to the use of "relative"? (Note that Table 9 refers to tonnage and grade separately).

15) Fax transmission dated 09/10/90 - Page 1 of 1

You state grade is underestimated by up to 17% in 20/30 rock types and up to 7% in 50/70 rock types. This suggests (as currently blasthole grades are in excess of mill heads) that dilution is greater than we had supposed.

16) Same fax sheet as above - lines 2/3 of message -

"correction of block estimates for proportion of blast holes in block"

Does this mean the correction of grade and tonnage in ore reserve blocks derived from the computer block model using the statistically determined distribution of blasthole tonnages and grades within the reserve block?.

17) (Page 38 2nd line from bottom) -

"tonnages are to high and grade too low" - compared with what? (BM S?). On a later page 55 you show Kriged BH assays give better results than polygonal (nearest neighbour) estimates of block grades, which seems in apparent contradiction with above statement - my understanding may be at fault here.

18) Is "lognormal short cut" (Page 39) an empirical method or is there a logical mathematical basis for using it which would apply to our situation at Faro?

Could you easily prepare comparison for several blocks showing polygonal (nearest neighbour), Kriged and Kriged + Lognormal short cut estimates of block grades?

Are you suggesting (based on P. 55) that we should Krige our blasthole grades to obtain a block grade or should a lognormal short cut correction be applied to Kriged grade as you did to diamond drill hole composite block grades (P. 39/40)?

19) Page 40 - Last Paragraph

"the correction is achieving its goal: reduce tonnage and increase grade."

Why is this a goal. If this correction in fact gives a better indication of "in situ" grade and tonnage then we must reevaluate dilution. (See question 15)

20) Page 55 - 2nd paragraph from bottom - 10th line

Formula should be: 3.2 / (1.0 -0.56)

21) Page 55 - Bottom Paragraph Line 5

By "true" do you mean "real" as in Line 2 of same paragraph.

22) Page 55 - Bottom Line -

If our selectivity has improved this must imply our dilution is higher than we thought.

23) P. ii Conclusions 4 - Line 4 -

Uncertainty about lead and zinc grade is <u>more</u> in rock types 20 - 30?

SYSTÈMES GEOSTAT INTERNATIONAL INC.

Montreal, November 16th 1990

Mr. Dave Tenney
Curragh Resources Inc.
P.O. Box 1000
Faro, Yukon
Y0B 1K0

Dear Dave,

Thank you for your letter of October 22. Please find herewith the corrected pages for the first and second reports as well as a few pages of answers to your list of questions.

Don't hesitate to communicate with me if you need additional clarifications.

Yours sincerely,

GEOSTAT SYSTEMS INTERNATIONAL INC.

Michel Dagbert, Manager

Encl.

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Geostatistical Analysis of Faro Drill hole and Blast hole Data

CORRECTIONS AND CLARIFICATIONS

1- You are correct. The three parallel lines on the correlation plots of figures 4, 5, 17, 18, 21 and 25 are the regression line of y (vertical axis) on x (horizontal axis) in the middle and the ± one standard deviation lines on both sides. Assuming normal distributions for x and y values (lognormal distributions if we use log scale), the regression line gives an estimate of y given x and the two surrounding lines give the limits of 68% confidence interval for that estimate. For example, if we take the scattergram of % Zn vs % Pb for the composites in ore type 50 on Figure 4, given a composite with a 4% Pb, the best value of % Zn that we can predict for that composite is 6.2% but the 68% confidence for this predicted value is quite large from 4.8% to 7.6%.

2- Corrected page 2 attached

3- Both terminologies are correct. The density formula is a multiple regression since it involves several variables (% Pb, % Zn and % Fe) but it is also linear since it is a linear function of these variables. The exact terminology should then be: "multiple linear regression". Corrected page 3 attached.

4- "and 2" means the inverse of the power 2 (square) of distances. Corrected page 4 attached.

5- I agree that both weighting (1/D and 1/D^2) give results very similar to BH values specially for metal quantities. However if we look at grade and tonnage estimates, 1/D is marginally better than 1/D^2. Corrected page 5 attached.

6- The statement is misleading and I decided to suppress it completely. The rest of the text explains more clearly the problem of comparing block values above cut-offs and blast hole values above the same cut-offs in the same area. Basically, we mean that, even if we had exact 25' x 35' x 20' block values and blast holes in the same zone, the proportion and grade of those blocks above a cut-off, say 4% Pb + Zn, is not the same as the proportion and grade of blast holes above the same cut-off. If the overall grade in the zone is more then 4%, we have more tonnes in the blocks above the cut-off and at a lower grade. This comes from the lower variability of block values compared to sample values. Corrected page 7 attached.

7- Yes, I would suggest that, at least on a few sections, a purely geological block reserve estimation using the standard section method be attempted and results compared to 1) the inverse-distance regular block model on the same sections. 2) blast hole data in the same area. Of course the geological structure of Faro is not the same as that of the sedimentary iron ore deposits of Labrador or Western Australia specially the magnitude and layering. However, like in those deposits, we have at Faro:

1- a strong original statigraphic control.

2- intensive deformation of the original sedimentary structure.

8- Multiple linear regression. Corrected page 11 attached.

9- Corrected page 12 attached.

10- Corrected page 19 attached.

11- Yes. Table 15 indicates that in fact any power of inverse distance between 1.0 and 1.5 might give good results in ore type 20-30. In ore type 40-70, it would be any power between 1.5 and 2.0. Corrected on page 19 and conclusions.

12- Connectivity is used here in a geometrical sense. If ore lenses above a given cut-off are well connected and not scattered throughout the deposit in small patches, there is a good "connectivity" of ore above the cut-off. Additional explanations on page 24.

13- The indicator variograms are not used here to do indicator kriging (i.e. estimate the percentage of bench intercepts above a cut-off in a block) but rather to derive the estimation variance for the average value of the indicator (hence the proportion of ore above cut-off) over a large zone. Anyway it is right that it is computed after assigning a 1 to composites above the cut-off and 0 to composites below the cut-off.

14- Relative standard errors of 5% on monthly estimates and 2% on yearly estimate refer to tonnage estimates above cut-off. Error is slightly less for grade estimates (see table 9). Error on metal content estimates is even lower because of the negative correlation between tonnage and grade (if tonnage is overestimated, grade would tend to be underestimated).

Relative error means that error is not expressed in absolute terms (i.e. % Pb + Zn) but in relative terms: the error is divided by the estimate. Sometimes this is called a "precision".

15- If we take 413 blocks of 20/30 and the 568 blast holes in those blocks, we find that 59% of the blocks have an estimated grade (derived from D.H. bench composites of 20/30 around) above the 6% cut-off at an average grade of 6.86% whereas 53% of the blast holes are above the same cut-off with an average grade of 8.33% (Table 13) hence a difference of 17% on grade. If blast hole grades are in excess of mill heads, then mill heads are closer to the block estimates.

Now we are not too much surprised to hear that mill heads are less than blast hole grades. In fact, on page 55 of our report, when we analyze the continuity of blast hole grades, we come to the conclusion that, with the current processing of blast hole values (nearest neighbour), we overestimate high grade. We give an example for 6% cut-off: estimated grade is 9.37% but real grade is 8.57%, a 9% difference. There is a dilution when going from blast holes to block estimates but it is somewhat compensated by a dilution when going from blast holes to mill heads.

16- Yes this is exactly it: the original block grade estimate is taken as the mean of the distribution of blast hole grades in the block and the cut-off is applied to that distribution (not to the mean).

17- This is what we explain in #15. Block estimates derived from D.H. do not compare well to blast holes when you apply a cut-off because of the natural dilution when going from blast hole to blocks. This dilution is somewhat compensated by the fact that you don't mine individual blast holes but blocks around them.

18- The "lognormal short cut" is an empirical method that just consists in assuming that the distribution of blast hole grades in a block is lognormal with a mean equal to the block grade estimate and a variance derived from the variogram. In the case of Faro, it would be more appropriate to use a "normal shortcut" but the only thing that we had programmed at the time of the study was the lognormal model (now when variances are small, normal and lognormal are not that different).

The lognormal shortcut is designed to provide block reserves above cut-off directly comparable to individual blast holes in the same blocks. Now since block mined are not individual blast holes and there is some dilution here, it is a good idea to compare block estimates without lognormal correction, block estimates with lognormal correction, blast holes and kriged blast holes in the same zone.

Anyway, at the time of mining, when all decisions are taken from blast holes, it is definitely better to do the delineation of ore zones above a cut-off with kriged blast holes rather then individual blast holes. At this time, there is no need for a lognormal correction of the kriged blast holes grades (since you actually mine blocks and not individual blast holes).

19- Again the goal is to have block grades derived from DHs closer to those of blast holes in the same zone.

20- Corrected page 55 attached.

21- "True grade" is in fact "real grade". On the corrected page 55.

22- In the test zone, if we apply a 6% cut-off on the kriged blast holes with no segregation between ore types, we get 1560 B.H.s with an expected grade of 8.46% and a real grade of 8.52%. If we krige blast holes in separate ore types, we get 1519 BHs above 6% with an expected grade of 8.65% and a real grade of 8.64%. Selectivity has improved because 1) we get less discrepancy between expected grade and real grade 2) real grade above cut-off is higher. By all means, this is much better than applying the cut-off on individual blast holes: then we get 1367 BHs with an expected grade of 9.37% but a real grade of 8.57%.

23- No, as indicated by table 8, the uncertainty about lead and zinc grade is less in type 20-30 than in type 40-70. Although continuity is better in 40-70, variability of grade above cut-offs is less in 20-30 (standard deviations of composites grades above 3% and 6% cut-offs are 2.45% and 1.96% respectively in 20/30 and 3.09% and 2.46% respectively in 40/70).

GEOSTAT SYSTEMS INTERNATIONAL INC.

Michel Dagbert, Manager
11/16/90

CONCLUSIONS AND RECOMMENDATIONS

1- In the calculation of tonnes and grade above cut-off, we have identified 3 major sources of differences between estimates derived from D.H. data and values derived from B.H. data in the same area: spatial interpolation errors, oversmoothing of distance weighted estimates and uncertainty of the geological model.

2- **The spatial interpolation errors** can be quantified by means of variograms calculated on drill hole bench composite values in the various ore types. If we stick to the current segregation of the quartzite ores (20, 30) and sulfide ores (40 - 70), then the average precision of 25' x 35' x 20' block % Pb + Zn grade estimates at a 95% confidence level are: ± 67% in 20-30 and ± 50% in 40-70 from drill holes on a 140' grid, ± 47% in 20-30 and ± 37% in 40-70 from drill holes on a 70' grid.

3- The above confidence intervals are quite large but they apply to rather small blocks. If we now consider groups of blocks which may represent a month, a quarter or a year production, we have the following average precision (still at a 95% confidence level) for tonnage and %Pb + Zn grade estimates above cut-offs : for a month and with the 140' grid: from ± 44% for tonnage above 6% cut-off to ± 14% for grade above 6% - for a month but with the 70' grid: from ± 22% to ± 7% - for a year with the 140' grid: from ± 13% to ± 4% - for a year with the 70' grid: from ± 6% to ± 2%. Previous figures assume a production made 50% from 20-30 and 50% from 40-70.

4- Precision of estimates for large volumes depends of drilling grid, ore type and cut-off. Uncertainty on tonnage above cut-off increases with cut-off and, at the 6% cut-off, is higher in 20-30 than in 40-70. Uncertainty on % Pb + Zn grade above cut-off is not as bad. It now decreases with cut-off and is less in 20-30 than in 40-70. Also, to get a ± 5% precision on all monthly estimates, we need a drilling grid between 35' and 40' at 3% cut-off and between 15' and 20' at 6% cut-off. The ± 2% yearly precision is easier to achieve: we can get it with just a 50' grid at the 3% cut-off but we need a 25' grid at the 6% cut-off.

5- The above quantification of spatial interpolation errors rests on variograms of D.H. bench composite grades in the two groups of ore types. With close to a thousand data points in each group, those variograms are well defined : relative nugget effects range from 35% (40-70) to 50% (20-30) of total variation; in 40-70, horizontal variations are isotropic with a maximum range of 600'; in 20-30, range along N-S (400') is much longer than along E-W (150'); in both groups, there is a strong vertical anisotropy with vertical range not exceeding 60' in 20-30 and 180' in 40-70. Variogram features and the resulting assessment of the magnitude of spatial interpolation errors are confirmed by a D.H. bench composite reuse study. For example, in 20-30, out of 719 reestimated composites, 480 (67%) have an estimation error less than the predicted standard error and 683 (95%) have an estimation error less than twice the predicted standard error: this is an almost perfect fit of estimated and actual errors.

6- Variograms of D.H. bench composites also indicate only minor differences between kriged estimates and inverse distance estimates for the same blocks. The power of inverse distance which gives results closer to kriging is between 1.0 and 1.5 and between 1.5 and 2.

7- The oversmoothing of distance weighted block estimates is quite noticeable in the Faro long term block model: if you apply a cut-off on block estimates derived from D.H. bench composites, you always find more tonnes and a lower grade than what is indicated by the distribution of blast holes in the same area. In 20-30, tonnage differences are as much as 30% (5% cut-off) and grade differences reach 17%. In 50-70, maximum tonnage and grade differences are respectively 14% and 7%.

8- A way to overcome the oversmoothing of straight block estimates is to infer the likely proportion and grade of blast holes above the cut-offs in each block. A simple correction of block estimates dubbed "lognormal" short-cut reduces maximum tonnage differences to 8% in 20-30 and 3% in 50-70. With that correction, maximum grade differences are only 5% in 20-30 and 2% in 50-70.

9- All the above conclusions assume that the "geological model" is correct e.g. when we assign a code 20 to a block (and thus, we infer its grade from just 20-30 D.H. bench composites around), the type of ore in that block is actually 20 (or 30). To validate that assumption, we have compared the ore type of a D.H. bench composite to the ore type of the nearest B.H. (in the same bench and at a distance of less than 20'). We found that the classification of the predominantly high grade types 50 + 60 + 70 is consistent from drill holes to blast holes since nearly 70% of the D.H. composites of that type have a B.H. of the same type next to them. However, we have more problems with types 20 and 30: in that case, more than 50% of the D.H. composites of those types have a B.H. of type 50-70 nearby. We suggest that the criteria used to classify D.H. samples and B.H. of type 20-30 be revised so that we have more consistency in the classification of the two types of samples.

10- Another suggestion related to the geological model is to separate type 40 from types 50, 60 and 70 in the estimation of blocks even if they all refer to sulfide ore. Statistics of type 40 D.H. composite grade values indicate that they are much lower than for the other categories of sulfide ores. As a result, there is a danger to underestimate a type 50-70 block with a type 40 composite next to it and overestimate a type 40 block with a type 50-70 composite nearby. In fact, in the comparison of block estimates with B.H. values in those blocks, better results were achieved after isolating type 40 composites (and type 55 blast holes).

11- This geostatistical analysis of D.H. bench composites data can be extended to blast holes. The idea here is to see whether there is a better procedure to delineate ore limits in blasts than just contour individual blast hole grades. From the test data set of about 2000 blast holes in the south part of benches 3450, 3470 and 3490, we found that:

+ if we krige the grade of blast blocks from blast holes and then apply the cut-off on those blast block estimates, the actual grade of the material that we recover is the estimated grade.

+ if we contour individual blast hole values, the actual grade is less than the expected grade (in the example that we ran we noticed a 8.5% shortfall at the 6% cut-off). Moreover, we get less tonnes than if we do the selection on kriged estimates (12% in our example).

+ recovery based on kriged blast blocks estimates is even better if we process blast holes of type 20-30, 50-60-70 and 40-55 separately.

+ blast block kriging can easily be implemented with blast hole variogram models. Horizontal variograms of groups of a few hundred of blast holes around a blast are quite interpretable. They generally show the same features as the corresponding variograms of D.H. bench composites. In particular, their nugget effect is the same hence it looks like the sampling precision of blast holes is as good as that of drill holes.

FOREWORD

This study has been initiated by a letter of April 25 , 1990 , from Dave Tenney , chief geologist of the Faro operations for Curragh Resources Inc. to Geostat . This letter stated the three main objectives of the contemplated geostatistical analysis i.e. 1) estimate the likely error (global and local) of current computer-generated inverse distance (ID) grade and tonnage estimates , 2) determine the necessary spacing between holes to achieve a given confidence on the same estimates (in later discussions , it was mentioned that management objectives are a maximum +/-5% deviation for monthly grade and tonnage estimates and +/-2% deviation for annual estimates) ,3) using blast hole data , compare the ID estimates with truly geostatistical estimates (kriging) for the same blocks .

In its answer letter of April 30 , Geostat proposed to conduct a two phases study : Phase 1 would include a visit of Faro operations by a Geostat's representative (M. Dagbert) for gathering the necessary information and preparing a detailed scope of work . Phase 2 would be the analytical work itself conducted in Montreal .

Phase 1 was accepted by Curragh Resources on June 12 . The site visit was conducted by M. Dagbert in the week of July 16 and this is the report for that phase of the study .

INTRODUCTION

Curragh Resources Inc. is currently mining lead-zinc-silver sulphide ore from the Faro open pit northeast of Whitehorse in the Yukon . The mine is mostly a standard truck and shovel operation with some underground mining from a ramp at the bottom of the pit . Shovels are 12 yd3 and trucks 150-170 tons . Bench height is 20 ft . Ore production is about 450,000 tonnes/month or 15,000 tonnes/day . Mining in the pit is now restricted to the S (south) and E (east) phases . The central (A phase) is now completed . An estimated 6 M2 tonnes above 5 % Pb+Zn is left to be mined .

Initial exploration of the deposit was done on NE-SW sections at 140 ft intervals . Holes were mostly vertical and spacing between holes on the sections was also about 140 ft . In the recent years , in-fill drilling on about a 70x70 ft grid has been conducted in the remaining E and S parts of the pit. These new holes are generally vertical but some of them are dipping to the NE or SW . Drill hole assay intervals are generally 2.5 to 5 ft long . Each interval is assigned a lithological code (see below). Assay values are %Pb,%Zn,g/t Ag and g/t Au . A pulp specific gravity is also available for most intervals .

The Faro deposit is considered as a polydeformed stratiform deposit .Several different rock and ore types can be identified . Main ore types are from the outside into the core of the deposit :

2A	Graphitic quartzite ore (code 20)
2BCD	Pyritic quartzite ore (code 30)
2EC	Pyritic semi-massive sulphide ore (code 40)
2E/F	Pyritic massive sulphide ore (code 50)
2G	Barytic massive sulphide ore (code 60)
2H	Pyrrhotitic massive sulphide ore (code 70)

The quartzite ore types (20 and 30) form the outside "shell" of the deposit . The massive and semi-massive sulphide ore types (40 to 70) form the core of the deposit . Most of the ore left to be mined in the S and E phase of the pit is mostly of types 20,30,40 and 50

Main waste rocks are the FW and HW phyllites (codes 100-120) , a calc-silicate breccia cap to the east (code 170) and dike material (codes 180-190) .

The stratiform deposit has been subjected to intense folding and faulting . Isoclinal folds subparallel to schistosity can be recognized . Low angle thrust faults as well as post-metamorphic extensional faults with displacement of up to several hundred feet have been interpreted on sections and plans .

Grade control uses blast hole assay data (%Pb,%Zn and %Fe) . A specific gravity is derived from grade values using a multiple linear regression formula derived from pulp SG values in drill core samples . Lithological codes of material found in each blast hole are also recorded . Blast holes are on a square grid with side from 20 to 25 ft . Blast holes are classified according to %Pb+%Zn grade limits : 3-5% is low grade (3-4 is LLG and 4-5 is LGA or LGC),5-6% is medium grade and above 6% is high grade . A first calculation of ore in the blast is done after classifying each blast hole and assigning a tonnage from the volume of influence of the blast hole (generally the blast hole grid cell) and the inferred density of the blast hole material . A second classification is done after grouping neighbor blast holes with similar grades and lithology . Limits of that second classification are used to flag ore in the blast .

LONG TERM PREDICTION FROM DRILL HOLE DATA

Current modelling technique :

Current estimation of ore reserves from drill hole data is done through a regular block model and the PC-EXPLOR/PC-MINE software package . The block grid is parallel to the NW strike of the deposit (N of grid) as well as the NE direction of drill hole sections (E of grid) . Block size is 25 ft E-W x 35.35 ft N-S x 20 ft vertical (from 1300 to 2400 tonnes depending of density) . Of course , the block reserve grid is using the same benches as the mining operation . Maximum number of columns and rows in the grid is 128 . Maximum number of benches is 50 .

In a first step , a "geological" block model is defined . This means that a single lithological code is assigned to the block . This is done by digitizing the interpreted limits of lithological units on the cross-sections . Traces of section lithological intercepts are then drawn on bench plans (mid-bench elevation) and a new interpretation of lithological limits is then done on each bench plan . It is from this interpretation that the block geology assignment is made : lithological code of the block is that of its center on the interpreted bench plan .

The next step is the interpolation of average grades (%Pb, %Zn , g/t Ag and g/t Au) and specific gravity (SG) of each block from surrounding drill hole samples . Original drill hole assay intervals are grouped into 20 ft bench composites . Composite grades are SG weighted averages of portion of drill hole assay intervals in the composites . In the latest models (F8910 and F9003) , the highest assay values (above 95th percentile of the distribution) are clipped before compositing . Important too is the lithological assignment of the composite from the lithology of intervals in the composite . It looks like the rule used is majority one i.e. the code of composite is the code of intervals with the longest cumulated length in the composite (although the internal company report describing the comparison between drill hole predictions and blast hole values mentions a different rule : code of composite is code of its mid-point i.e. a rule similar to that used for blocks) .

The selection of composites used to interpolate a block is done according to lithological code and maximum distance to block center . In the latest block models (F8908 , F8910 and F9003) blocks of quartzite ore (codes 20 and 30) are interpolated from composites of the same type and blocks of semi-massive and massive sulphide ore (40 to 70) are interpolated from composites of the same type . Hence , there are only two categories of ore for block grade interpolation . Maximum distance from block center is defined with a search ellipsoid centered on the block : long axis of the ellipsoid is always along strike (NW-SE) with a search distance between 120 and 300 ft ; intermediate axis is across strike (NE-SW) , horizontal or dipping 12 deg. or 20 deg. to the SW depending of the part of the deposit where the block is located - search distance along that axis varies from 70 to 200 ft ; short axis is along vertical (or nearly vertical) with a search distance from 14 to 37.5 ft . Block interpolation is done in two or three passes with an increasing search ellipsoid in each pass . Hence , in the first pass , only blocks close to drill holes are interpolated and the block values are very much dependent of those drill hole values . This similarity is also achieved in the latest model (F9003) by restricting the interpolation to the six closest composites (of the right lithological code) in the search ellipsoid .

Finally , once the neighbor composites are selected , the interpolation method is an inverse distance one : the final block estimate is a weighted average of selected composite values with weights inversely proportional to some power of the distance between block and composite centers . Early models used a power 2 ($1/D^2$) . Latest model (F9003) uses power 1 (1/D) and 2 ($1/D^2$) i.e. inverse distance and inverse squared distance.

Validation of modelling technique

Verification of the long term reserve model is done periodically by comparing blast hole values with estimated block values within the same limit (one year production or any particular bench in a one year production) . In Table 1 , we are reproducing the results of that comparison for the latest model F9003 which uses the latest in-fill drill hole data on the 70 x 70 ft grid in the S and E parts of the pit and both 1/D and $1/D^2$ weightings . Other parameters of that most recent model are :

-assay data and SG clipped to 95th percentile before bench compositing
-2% reduction of SG (to allow for porosity) before compositing
-blocks of 20/30 estimated by composites of 20/30 . Blocks of 40-70 estimated with composites of 40-70 .
-search radii for first pass : 120 x 70 x 19 ft
-search radii for second pass : 240 x 140 x 19 ft
-minimum number of composites : 2
-maximum number of composites : 6

The model appears to do a very good job in 1989 but it is too much conservative in the three previous years where both grade and tonnage are underestimated . This discrepancy is more pronounced at 5% (15% underestimation of metal) than 4% cut-off (12% underestimation of metal) . It can be argued that the better performance of the model in 1989 comes from the simple fact that the additional in-fill drilling used by the model was precisely in the area mined during that year .

If we compare the 1/D and 1/D^2 weighting , as a general rule the metal quantities come very close but we find less tonnes and an higher grade with the 1/D^2: this is to be expected since the higher the power of inverse distance , the less smooth is the distribution of grade estimates .

In 1989 , with both weighting methods and at the two cut-offs , there is still an underestimation of metal by about 2% . This might come from the clipping of high grade sample values : cutting or clipping high grade is fine for local (block) estimates close to those high grade samples but overall it generates a reduction of the estimated metal quantity .

Alltogether , even if metal quantities estimates are very close , the 1/D weighting seems to give slightly better results than the 1/D^2 (tonnage and grade estimates are closer to those indicated by blast holes)

If we do a similar comparison on a bench-by-bench basis for that year 1989 (Table 2 - 1/D weighting only) , we see much more variation between predicted reserves and reserves indicated by blast holes . Benches have been ordered according to tonnage above 4% cut-off indicated by blast holes with only those benches with more than 100 Ktonnes retained . We can see two benches with major discrepancies : 3430 and 3630 . In the first case there is an overestimation of tonnes and grade at the same time . In the second case , this is an underestimation . In both cases the difference on metal is around 12,000 tonnes . In most of the other benches , there is a compensation of errors : grade is overestimated and tonnage is underestimated or vice-versa . Hence metal is generally well predicted . There are also benches like 3410 , 3610 or 3550 where the prediction of both grade and tonnage are right .

Discussion :

When we estimate the average grade of a block with a weighted average of grades of bench composites nearby , there is an inference error the magnitude of which is a function of the natural difference between the grade of those composites and the grade of bench intercepts inside the block . Obviously this difference increases with the distance between the samples and the block . A way to characterize this increase of the difference between bench intercepts as a function of the distance between the intercepts is by calculating variograms of bench composites grade values . To stick to the rules of the current modelling method , we would compute and model variograms of bench composites of type 20-30 and of type 40-70 separately . Then we would use those variograms to infer the likely error for block grade estimates , in each of the two lithological groups , and using a inverse distance weighting method from composites in drill holes on various grid : 140 x 140 ft , 140 x 70 ft , 70 x 70 ft ...... The "likely error" is measured by a standard deviation of the possible errors . This "standard error" can be converted into confidence intervals for grade estimates after assuming a normal distribution for the possible errors : for example , the 95% confidence interval is +/- 2 standard errors . With the variograms of 20/30 and 40/70 bench composites , it is possible to get the standard error with any distance weighted method and obviously find out , in the two lithological groups , which power of the inverse distance gives the least error (this "optimal" weighting would give results very close to true kriging of block grades from composite grades) .

Also from the variograms of bench composite grades in the two lithological groups , it will be possible to get the standard error for the grade estimate of groups of blocks (which is less than for a single block and not a straight combination of standard errors for blocks in the group) . We can look at groups representing a month , 3 months and a year production with various proportions of 20-30 / 40-70 blocks and various drilling grids . As a result of that study , we could derive the drilling grids necessary to get less than 5% relative deviation on monthly estimates and 2% on yearly estimates (at a 95% confidence level) .

Results from the variogram study are fairly theoretical and it would be good to check them against real data . We have two sets of real data that we can use : the drill hole bench composites themselves and the blast hole data . In both cases , we estimate the grade of bench composites or blast holes from bench composites around and in the same lithological type with various distance weighting methods (including kriging) . We then check that the distribution of experimental error has a standard deviation similar to that predicted from the variogram .

The variogram study described above is fine to get the precision of block grade estimates from various interpolation methods using the same composite data . However , the ultimate comparison of those block values is with the grade distribution of blast holes inside the same blocks . Now we know by experience that blocks do not have the same variability as samples , hence , even if we had perfect block values , we would not get the same proportion and average grade of blocks and samples above a given cut-off : at a low cut-off compared to the mean of all values , we expect more tonnes and less grade for blocks ; at a high cut-off , we expect less tonnes and less grade for blocks . In fact , what we should estimate in each block so that we have a fair comparison in the end , is the likely proportion and average grade of bench intercepts (= blast holes) with a grade above the cut-off . Several methods can give this answer . We can examine two of them : 1-(log)normal shortcut : we assume that the distribution of BH grades in a block has a simple shape (normal or lognormal) , the mean of the distribution is the regular block estimate and its variance is a combination of the estimation variance (takes into account oversmoothing from uncertainty) and the expected variability of BH grades in a block 2) indicator estimation : we directly estimate the quantiles of the BH grade distribution in blocks from the surrounding bench composites . Like before , we have to validate those different estimation methods : best is to use blast hole data available . In the area where we have those blast holes we will 1) estimate blocks from bench composites with the regular distance weighting (1/D) procedure 2) do a (log)normal correction on top of those block estimates 3) run a direct indicator interpolation of the blast hole grade distribution in the blocks . We will then compare the results of the 3 methods at different cut-offs with what we get when we apply the same cut-offs directly on the blast hole data .

The two exercices that we just described do not question the "geological" framework of the long term prediction model . Basically we say that a block is 50 (massive sulfide) because its center is interpreted to be in a 50 lens and even if 30 or 40 % of the block might be in a 20 or 30 zone . Also , we say that a composite is 50 because it is mostly made of 50 intervals and even if 30 or 40% of the composite is made of intervals with a 20 or 30 rock type . So we may end up estimating blocks with a significant fraction of low grade ore with composites entirely in high grade . On the other hand , blocks entirely in high grade ore might be interpolated from composites with a significant fraction in low grade . These oddities may happen a few times after considering that most contacts between ore types are subhorizontal and they do not coincide with bench limits . To get around this approximation of the existing geological interpretation in the long term prediction model , it will be necessary to estimate partial blocks i.e. find which different ore types there are in each block and infer the grade of each ore type fraction in the block from the surrounding samples (no longer bench composites) in the same ore type . Practically , this involves the estimation of smaller blocks (e.g. 5 ft cubes) from smaller composites (e.g. 5ft) in the drill holes and the recombining of all the small block estimates in the 20 ft high block . We can test that method on the drill hole bench composites themselves : first we reestimate the grade of those composites using composites of the same type in neighbor holes ; second , we estimate the grade of each different ore type fraction of composites using values of intervals of the same type in the neighbor holes and we recombine these composite fraction estimates to get a full composite grade estimate . We then look at which estimates are closer to the real composite values .

Another approximation of the geological interpretation is the "loose" matching of ore types . With the current model , a massive sulfide block (50) might be interpolated with some semi-massive (40) composites and vice-versa . A strict matching might give better results . Again this can be checked on real data : in that case , we could use blast hole data as the reference values . We reestimate blast hole grades from bench composites with 1) loose matching 2) strict matching and determine which prediction is best . We can also compare the statistics of composite grade values in lithological types grouped together in the estimation of blocks .

A last approximation of the geological interpretation is that it forgets about faults between a block and a composite in the same ore type i.e. two composites 50 ft from a block will have the same influence on the block estimate even if one of them is separated from the block by a fault and the other is not . However taking individual faults into account in the estimation of blocks might just be the limit of block reserve models with distance weighted grade interpolation method . The alternative is a sectional model with a large number of small "geological" blocks interpreted on sections . There is no internal variation of grade in each block : grade estimate is the plain
average of drill hole intervals in the block . "Geological" block estimates can be recombined into any kind of 20 ft high "mining" blocks . This method is the one commonly used in highly folded and faulted stratiform deposits with strong geological controls like the iron ore deposits of Labrador or Western Australia . However building and testing such a detailed sectional model is beyond the scope of this study but it would be worth to, internally, test it on a few sections.

Tasks to be accomplished :

A-Grade interpolation of blocks (or groups of blocks):errors with various methods and drilling grids:

- conduct a statistical study of %Pb , %Zn and %Pb+Zn values of bench composites in units 20-30 and 40-70 (histograms,statistics and correlation)
- calculate and model a 3D variogram of %Pb , %Zn and %Pb+Zn values of bench composites in units 20-30 and 40-70 .
- use variogram models to get standard errors for block estimates of %Pb,%Zn and %Pb+Zn in the two lithological units and with various drilling grids and distance weighting methods (including kriging).
- use variogram models to get standard errors for estimates of average %Pb,%Zn and %Pb+Zn of groups of blocks representing a month , 3 months and a year production .
- validate standard error prediction by estimating bench composites and blast holes from nearby bench composites of the same lithological type . Compare standard deviation of experimental errors with predicted standard errors from variograms .

B-Correction of block estimates for proportion and grade of blast holes in blocks :

- recognize all blocks in the area where we have blast holes with coordinates (benches 3530 to 3570) . Determine the lithological type of those blocks (from bench map or BH lithology)
- perform a regular 1/D grade interpolation of those blocks using bench composites around (20-30 and 40-70 estimated separately) . Get tonnes and grades above various cut-offs (3%,4%,5% and 6%) directly from block estimates .
- perform a (log)normal correction on top of block estimates . Recalculate tonnes and grades at various cut-offs now using the block corrected values (fraction and grade of blocks with BH above cut-offs)
- do an indicator estimation of the BH grade distribution in each block . Calculate a third set of tonnes and grades from those estimated distributions .
- finally , compare the 3 sets of estimates with results obtained after applying cut-offs directly to blast hole data .

C-Testing the "geological framework" of the long term model :

- reestimate bench composite grades from composites in neighbor holes using 1) full composites 2) separate ore type fractions in composites . Determine if composite grade estimates are significantly better in the second case .
- compare statistics, histograms and correlation features of bench composites grades in ore types which are merged together in the estimation (20 and 30 , 40 and 50 and 60 and 70) .
- estimate blast hole grades from nearby bench composites with 1) loose matching 2) strict matching of ore types .

SHORT TERM PREDICTION FROM BLAST HOLE DATA

Current modelling technique :

As explained above , the processing of blast hole data to define the limits of the different ore types in a blast is fairly straightforward : blast hole assays (%Pb , % Zn and % Fe) as well as lithology are put on bench maps . Each blast hole is assigned a volume of influence corresponding to the blast hole grid (from 20 x 20 ft to 25 x 25 ft) . Volume is transformed into tonnage using a specific gravity derived from blast hole grades from a multiple linear regession formula . Then the grade control geologist manually groups neighbor blast holes with similar grades and lithology . Tonnage and grades of material in each ore category (low , medium and high grade) in the blast are then derived from those groupings .

Validation of modelling technique :

The way to validate the prediction of the short term model is to compare them with mill results over the same period of time . Unfortunately , in the case of Faro , this comparison is difficult because of the large stockpiles of the different ore types . One can never be sure of which time the ore currently milled has been mined . Hence when grade control says that , in that blast , there is 60,000 tonnes of high grade at 8.45% Pb+Zn , tonnage is fine because it is exactly what is sent to the high grade pile but grade might less than 8% or more than 9% .

Discussion :

Formally , the current short term prediction model is also a block model with a distance weighting grade interpolation method : blocks are the 20 x 20 ft or 25 x 25 ft volumes of influence of a blast hole and the interpolation method is nearest-neighbor (or polygon) i.e. the grade estimate for a block is the grade of the single sample in the middle . Like for the long term prediction model from drill holes , it is possible to assess the standard error of single blast-block grade estimate or a group of them . We just need compute and model variograms of blast hole grade . Such variograms have already been computed and they seem to be quite interpretable .

The variograms may also indicate that a better blast-block interpolation method than polygon could be used . Those calculated so far tend to show a significant nugget effect (1/3 of total variation is random , probably blast hole sampling error) . In that case , any distance weighting method <u>using several neighbor blast holes</u> around the blast-block to estimate would do better than nearest-neighbor . Obviously , this blast-block grade interpolation method must take into account the various lithologies of the blast holes (BH in dike should not be used to interpolate nearby blast-block around BH in massive sulphide) . Like for long term prediction model from drill hole , the foreseen advantages of this alternative short term prediction model must be checked on the blast holes themselves .

Tasks to be accomplished :

- conduct a statistical study of %Pb , %Zn and %Pb+Zn of the available blast holes in units 20-30 and 40-70 (histograms , statistics , correlation)
- calculate and model 2D (3D?) variograms for those blast holes . Try and relate nugget effect to available check sampling results conducted on blast holes .
- from the variograms , determine the standard error for the grade estimate of a blast-block derived from the blast hole in its center . Also determine the standard error for groups of blast holes .
- define a new blast-block grade interpolation method more consistent with the variograms Illustrate the results of the two methods on a bench .
- cross-validate the two grade interpolation methods on the blast holes themselves

DATA GATHERED

During our visit to Faro , we have gathered all the data necessary to conduct the geostatistical analysis study described here . Data files have been checked on our system and they are usable . We basically have 3 data files :

- drill hole data file : we have lithology and assay data (%Pb , % Zn) for 429 holes from the early 66 ones up to 90-F-93 . We have 3287 lithology intervals (codes 20,30,...) and 10899 assay intervals .
- drill hole bench composite file : we have mid-point coordinates , assigned lithology and calculated %Pb and %Zn as well as SG of a maximum of 7333 20ft D.H. bench composites (5733 with calculated assays)
- blast hole data file : we have coordinates , lithology and assay data (%Pb , %Zn and %Fe) of 1607 blast holes in benches 3530,3550 and 3570 . This is of course just a fraction of all blast holes but they are the only ones with digitized coordinates .

We also have interpreted lithological limits on 3 cross-sections (with drill hole traces) and 4 benches (with location of D.H. bench composites and their values) ; 3 of those benches are those where we have the blast holes (3530-3570) .

In order to illustrate the distribution of standard errors in blocks , we have kriged the %Pb+Zn grade of blocks in two test zones of bench 3450 . First , we plotted all the composites 20-30 and 40-70 in the bench and we defined approximate envelopes around each type of composite (Figure 10) . We filled those envelopes with the 25' x 35.35' blocks and we kriged the average %Pb+Zn grade of each block using composites of the same type around . The search ellipsoid used in each case is defined from the anisotropy of variograms : in 20-30 , the long radius is 400' along N-15-E , the intermediate radius is 150' along E-15-S and the short radius is 60' along vertical . In 40-70 , the long radius is 300' along any horizontal direction and 90' along vertical . Figure 11 is a map of block estimates . Figure 12 is a map of block standard errors . The average standard error is 1.59 %Pb+Zn in 20-30 and 1.73 %Pb+Zn in 40-70 . Those numbers confirm the precision values given above .

We can also use the same test zones to compare I.D. (inverse-distance) and kriged estimates . We have reestimated the %Pb+Zn of block in each ore category by inverse distance methods using various powers of the inverse distance composite-block . Each time we compared our I.D. estimates with kriged estimates for the same blocks and we can compute the average absolute difference for all blocks (Table 5) . We see that the average difference is minimal for a power between 1 and 1.5 in type 20-30 (average difference = 0.37%) and for a power between 1.5 and 2 in type 40-70 (average difference = 0.48 %) . We can also compare reserves above various cut-offs with the two methods (Table 6) .

1-4 Standard errors for monthly , quarterly and yearly production estimates

From variograms , it is also possible to get the magnitude of errors for large block estimates i.e. volumes corresponding to one month , three months or one year production .

The technique used to derive those standard errors is a little bit different than the one used before to get errors of small blocks estimates . We can't combine estimation errors of small blocks in a big block to get the estimation error of the big block since those estimation errors are not independent . We can however combine extension errors i.e. errors produced when the small blocks correspond to the drilling grid cell and the estimation of those small blocks is done with only the sample in their center (polygonal estimation) . In that case the estimation variance for the big block is :

$$V^2=v^2/n$$

where v^2 is the extension variance of one composite to its cell of influence (derived from variograms) and n is the number of drilling grid cells in the big block

At Faro , the big blocks that we have investigated correspond to the average monthly , quarterly and yearly production . To get those volumes , we have considered the production figures for the first four months of 1990 . Depending of cut-off , the flagged-in-pit volume of ore varies from 671,000 bcys at 3% Pb+Zn cut-off to 468,000 bcys at 6% Pb+Zn cut-off. Hence , the "ore bench surface" mined in a month is 226,460 ft2 at 3% and 157,950 ft2 at 6% . For a quarter it is 3 times as much and for a year , 12 times .

To determine the number of drilling grid cells of ore mined in a month , a quarter and a year , we have considered two basic drilling grids : 140' and 70' . Hence , at a 3% cut-off , there are 226,460/(140x140)=12 ore composites on a 140 ft grid in a month volume and 46 composites on a 70 ft grid . At 6% cut-off , number of composites are respectively 8 and 32 .

First uncertainty is on the estimated surface (or volume) of ore above cut-off . This uncertainty depends of the degree of "connectivity" (amount of connection between ore lenses above the cut-off) of the ore above the cut-off and the overall proportion of ore above that cut-off in the ore type being mined . Obviously , we can be more confident on the estimate of volume of ore above cut-off if 80% of the samples are above that cut-off instead of just 20% . More precisely , the "relative" error (or precision) is likely to be higher in the second case . Then we can anticipate that the errors on volumes at 3% are less than at 6% and that we should do better in 40-70 than 20-30 because there is a higher proportion of composites above the usual cut-offs in this ore type . The connectivity of sample values above a cut-off can be measured through a variogram of indicator at that cut-off . The indicator value of a composite is 1 if the grade of the composite is above the cut-off and 0 otherwise . Figure 13 shows horizontal variograms of indicator at 3% Pb+Zn cut-off in 20-30 . Figure 14 has the same type of variograms but for cut-off 6% in 40-70 . As a general rule , indicator variograms have about the same features as the corresponding grade variograms . Once indicator variograms are modelled , we can determine v , the extension standard error

+ low grades 40 and 55 have been pooled together for variogram computation. The variograms show a strong trend at long distances due to the presence of isolated high grade blast holes categorized in those ore types. The model used does not take that trend into account.

To check the performance of various blast hole grade interpolation methods, we can use the same sample reuse method that we had for D.H. bench composites. The idea is to reestimate each of the 2083 B.H. % Pb + Zn grades from blast holes around using various interpolation methods and segregation of blast holes by ore type. Results are in Tables 18 and 19.

For example, if we do the reestimation with no segregation at all, we get a pretty good correlation between original and reestimated B.H. grade if we use kriging (r = 0.70 - Figures 28). It is not so good if we just use the nearest neighbor method (r = 0.56 - Figure 29) even if the dispersion of estimates perfectly matches that of the original data (see table 28). In fact, even if kriged estimates are smoother than real values, we can see one advantage of kriging over nearest-neighbor: the regression line of real on estimate is almost an y = x line. This means that if we mine all blast holes with an estimate of 6% the true average of these blast holes is 6% : we mine what we anticipate. There is no shortfall of grade above cut-off. On the other hand, if we use the nearest-neighbor approach, the regression line is like y = 0.56 x + 3.2. If we enter estimates (x) of more than 3.2/(1.0 - 0.56) = 7.27% Pb + Zn in this equation, the corresponding mean true value (y) is less than the estimate. Hence we overestimate high grade. For example, if we apply a cut-off of 6% Pb + Zn to the nearest-neighbor estimates, we get 1367 blast holes (66%) with an average estimated grade of 9.37% Pb + Zn but an average true grade of 8.57%. If we apply the same cut-off to the kriged estimates, we get 1560 blast holes (75%) with an average estimated grade of 8.46% Pb + Zn and an average true grade of 8.52%. Hence with kriging, we recover more tonnes (12%) at the expected grade. With nearest neighbor we have a grade shortfall of 8.5%.

We get even better results if we do the kriging of blast holes independently in 20-30, 50-60-70 and all the low grades 55-40-00-10 together. Correlation coefficient of real data and estimates is now 0.74 (Figure 30), regression line of real on estimate is virtually y = x. As a result, if we apply the 6% Pb + Zn cut-off on the estimates, we recover 1519 data (73%) with an average estimated value of 8.65% Pb + Zn and an average real value of 8.64%. We get a little less tonnage than in the case when we use all blast holes together but with a higher recovered grade. Our selectivity has improved.

GEOSTAT

MEMORANDUM

To: Dave Tenney, Chief Geologist, Faro Operations, Curragh Resources Inc.

From: Michel Dagbert, Geostat Systems International Inc., Montréal

Subject: Comparison of mill head grades and blast hole predicted crusher feed grade.

Date: November 22nd, 1990

Introduction

We have read your memo of October 10th showing the comparison of daily averages of mill head grades and blast hole grades for the period from April 10 to July 6. First you find that the best correlations are achieved with a zero (0.464) and one day (0.413) lag time between mine and mill and second that the regression of mill head % Pb + Zn grade on BH % Pb + Zn grade is of the form: $y = 0.451 x + 4.369$

This suggests that high blast hole averages above 7.95% should be decreased (but at the same time, blast hole values below that same limit should be increased). To illustrate that correction:

Actual BH average (% Pb + Zn)	Corrected BH average (% Pb + Zn)
6	7.08
7	7.53
8	7.98
9	8.43
10	8.88

Note that this correction 1) relates to average blast hole values over a day (and not individual blast holes) 2) relates to BH values above a certain cut-off (say 4%) and not all BH values.

You relate that correction to the regression of real BH % Pb + Zn grades on their nearest neighbor estimate in our phase 2 geostatistical report of October 3rd: $y = 0.56 x + 3.2$

This regression formula can be interpreted as the average relationship between the grade of a blast hole and the grade of the block (25 x 25) surrounding that blast hole. To illustrate that relationship:

BH grade	Expected grade of block around
(% Pb + Zn)	(% Pb + Zn)
2	4.32
4	5.44
6	6.56
8	7.68
10	8.80
12	9.92
14	11.04
16	12.16

Formally, it looks very similar to the first correction formula (corrected values in the range 6-10% are about the same - values above 7.3% are decreased - values below 7.3% are increased). However we must keep in mind that the second correction function 1) deals with individual blast hole values 2) at no cut-off.

If we assume that mill head grades are averages of blocks grades around blast holes, then the second relationship could represent the relationship between mill heads and blast holes provided that 1) it is limited to blast holes above the cut-off 2) it is based on the comparison of mean blast hole values and mean block values around those blast holes.

Comparison of blast hole estimates at a cut-off.

In fact, if we repeat our comparison of nearest neighbor BH estimate and real BH % Pb + Zn grade but now only for estimates above 4% Pb + Zn, we expect an average grade of 8.48% Pb + Zn (out of 1728 blast holes) but the real average grade of those blast hole is only 8.07% Pb + Zn. The regression formula for that restricted set of blast holes estimated above the cut-off of 4% is now: $y = 0.47\, x + 4.10$ with a correlation coefficient of 0.42.

This is now very close to the type of differences that we can experimentally observe between BH predicted and mill head grades.

It is interesting to note that if we do a comparison between real blast hole grade and kriged blast hole grade from neighbors at the same cut-off of 4%, we get the same good results as we had at no cut-off: 1875 blast holes have an estimate above cut-off (a 8% gain in tonnage compared to nearest-neighbor). The expected average grade is 7.90% % Pb + Zn and the real average grade is also 7.90%. The regression of real on kriged grade is : y = 1.04 x - 0.3 (correlation coefficient = 0.60) i.e. almost the perfect y = x relationship.

Analysis of daily mill head and BH values

We have received a fax copy of the daily values used to derive the first regression equation. For each day, from April 1 to July 31 1990, we have tonnes, % Pb, % Zn and % Pb + Zn as mined that day from BH data and tonnes, % Pb, % Zn and % Pb + Zn as milled that day.

If we sum all the tonnages for the 122 days, we get 1,595,605 tonnes mined and 1,601,957 tonnes milled, i.e. only a difference of 6,352 tonnes or less than 0.4%.

If we calculate the tonnes-weighted average grades for % Pb, % Zn and % Pb + Zn we get:

- from mine (BH) : 3.21% Pb, 5.14% Zn, 8.36% Pb + Zn
- from mill : 3.05% Pb, 5.02% Zn, 8.08% Pb + Zn

There is one extremely high value of % Zn mined for July 3: 14% (next highest is only 6%) which of course generates another high value for % Pb + Zn: 17.12% (next highest is only 9.73%). If we eliminate that suspicious data, we have average mined grades of : 3.21% Pb, 5.06% Zn and 8.28% Pb + Zn. Hence without that addity, the difference between the two lead values stays high but there is now almost no difference for zinc (5.06% vs 5.03% hence less than 1% relative difference).

Assuming a zero lag time between mine and mill, if we look at the correlation plots of % Pb and % Zn (Figure 1 without zero values and odd zinc value) we can see that:

x there is some kind of systematic difference between the two series of lead values: for that metal, we have only 38 days (out of 118) when mill is above mine. A simple non-parametric sign test indicates that this discrepancy is highly significant. The regression equation of lead of mill on lead of mine is: y = 0.538 x + 1.32 with a 0.39 correlation coefficient.

x there is no real systematic difference between the tow series of zinc values: we now have 57 days (out of 117) when zinc of mill is more than zinc of mine. This is not a significant difference. Of course, like in any comparison where the correlation is passable, low mine values correspond to higher mill values and high mine values correspond to lower mill values. Thus the regression of mill on mine is: y = 0.422 x + 2.88 with a 0.47 correlation coefficient.

x for % Pb + Zn, we find almost the same regression as yours: y = 0.347 x + 5.20 with a 0.36 correlation. We have 50 days (out of 117) when mill is higher than mine, a non significant difference at a 90% confidence level. Of course the slight mine-mill unbalance that we observe for % Pb + Zn is mostly generated by the unbalance of lead values.

To illustrate the different behaviour of lead and zinc, we have plotted the weekly fluctuations of mine and mill values (Figure 2). For each day, a weekly value is calculated by averaging (using daily tonnages as weights) the values of that day, the 3 days before and the 3 days after.

As we average grades into more directly comparable tonnages, we clearly see the systematic lead difference and the more random difference for zinc.

Discussion

First we have seen that a comparison of BH predicted and mill head % Pb + Zn daily grades points to a correlation which is very much similar to the correlation between estimated BH % Pb + Zn grade by the nearest neighbor method and real BH % Pb + Zn grade above a cut-off of 4%. Hence it looked at first like the differences that were observed between mill and mine could be explained by the estimation errors of a blast-block grade form the single blast hole in its centre.

After looking at the mine-mill differences in more details, we found that we tend to have a systematic overestimation of lead mill head grades whereas zinc differences appear more random.

Now it may be that, because of a more erratic behaviour, the estimation of the lead grade of blast blocks by just the blast hole in the middle has more serious impact on the recovery of lead than zinc. To test that hypothesis, we have redone the comparison of blast hole estimates and blast hole real grades but now separately for lead and zinc. Then we have looked at the differences, for each element, and only for blast hole estimates above 4% Pb + Zn. Like before we have 1728 blast holes in this case = their average estimated lead grade is 3.22% and the average real grade is 3.09% (very close to the mine-mill statistics of lead). For zinc, the average estimated grade is 5.26% and the average real grade is 4.98% i.e. a similar type of difference that we don't observe in the mine-mill reconciliation.

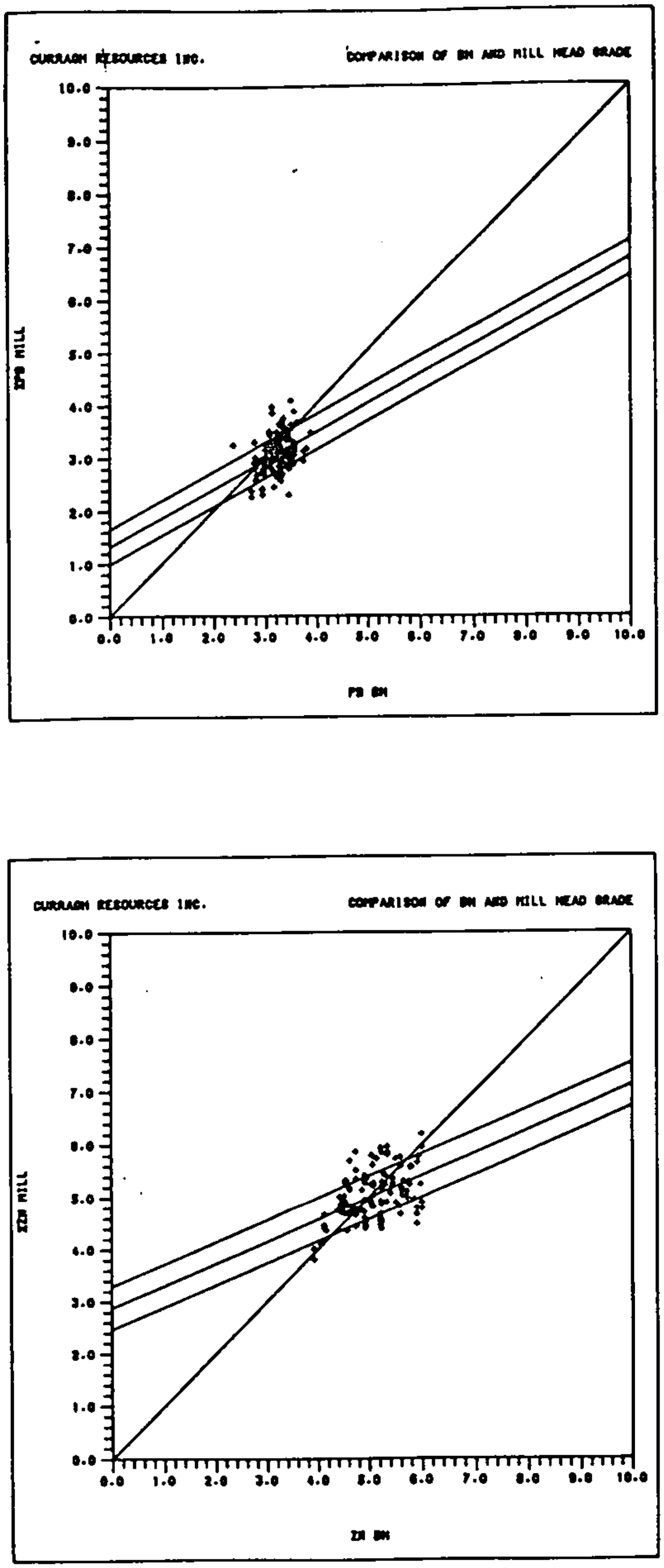
Altogether, it looks like the mine-mill difference that we see for lead can be explained by the way BH data are handled in the grade control procedure whereas we don't have any readily explanation for the good coincidence between zinc BH predicted and mill head grade.

At any rate we think that a true blast hole kriging is preferable to an overall correction of individual blast hole values based on a regression of mill head grades on BH predicted grades before a cut-off is applied to those individual blast hole values. The overall correction does not take into account the local configuration of data (i.e. zones with blast holes of systematic high or low grade where obviously no correction is needed vs zones with blast holes of rapidly changing grades) whereas blast hole kriging does it. Also, the overall correction is not applicable to blast hole data below economic cut-offs (say 4%): in fact, according to that formula, any blast hole would have a corrected value above the cut-off (x = 0, y = 4.369% Pb + Zn).

GEOSTAT SYSTEMS INTERNATIONAL INC.

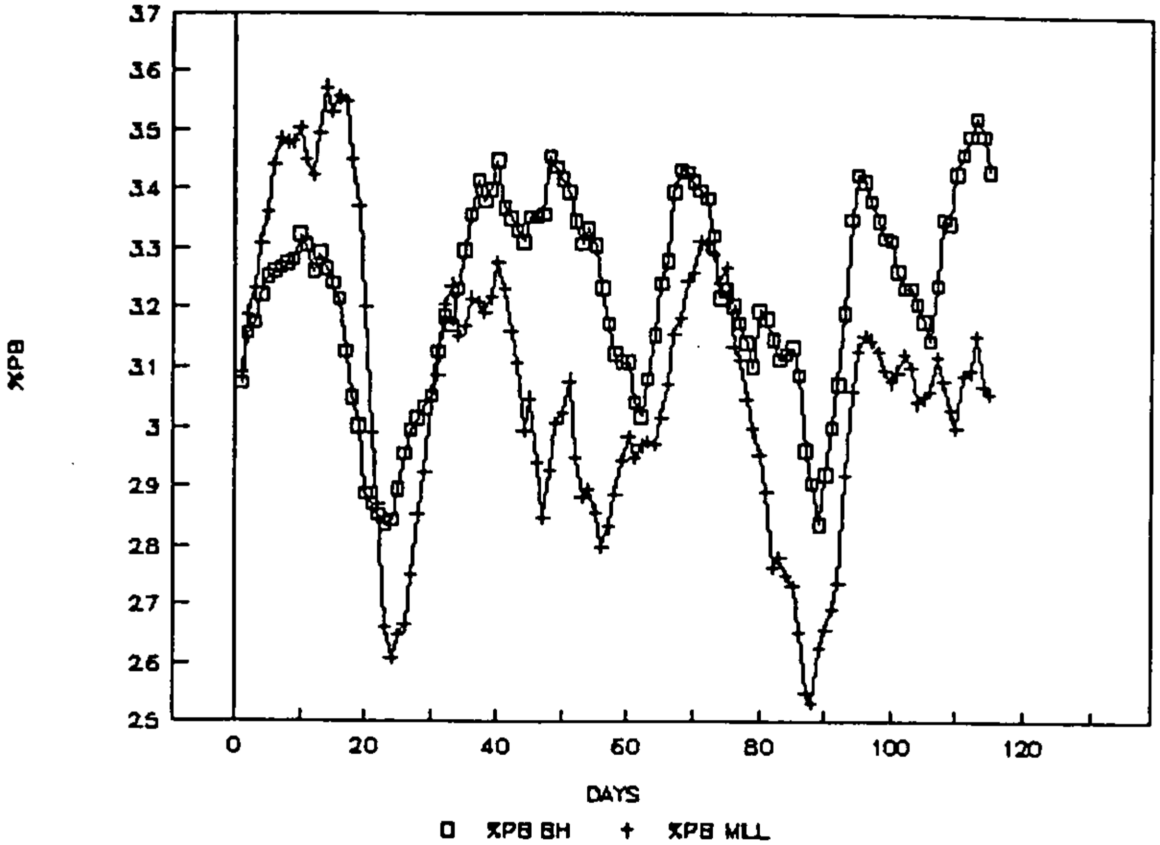
Michel Dagbert, Manager

Figure: 1


CURRAGH RESOURCES INC.
COMPARISON OF BM AND MILL HEAD GRADE
%PB MILL
PB BM
0.0
1.0
2.0
3.0
4.0
5.0
6.0
7.0
8.0
9.0
10.0
CURRAGH RESOURCES INC.
COMPARISON OF BM AND MILL HEAD GRADE
%ZN MILL
ZN BM
0.0
1.0
2.0
3.0
4.0
5.0
6.0
7.0
8.0
9.0
10.0

7 DAYS MOVING AVERAGES FOR %PB

Figure: 2



7 DAYS MOVING AVERAGES FOR %ZN