SPATIAL TRENDS IN ROCK STRENGTH – CAN THEY BE DETERMINED FROM COREHOLES?

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ABSTRACT

Mine planning for a new reserve is based on information obtained from exploratory coreholes. A critical component of an exploration program is the geotechnical evaluation. Poor assumptions about roof conditions greatly add to the risks of mining.

Rock mechanics testing is central to a geotechnical exploration program. Typically, 3 to 5 uniaxial compressive strength (UCS) tests are made to characterize a particular roof unit at a given corehole. The average (mean) of these tests is taken as "The UCS" for that location. Isopach contour maps are then used to show spatial trends in roof strength.

Two issues are raised by this traditional approach. The first is due to the large variability in UCS values that is typical even within a single unit from a single hole. The average UCS might be higher at Corehole A than it is at Corehole B, but the difference may not be statistically significant.

The second issue is whether widely spaced coreholes can identify valid spatial trends in rock strength. The answer depends upon whether rock strength changes over distances that are longer or shorter than the corehole spacing. This is a classical geostatistical problem. While geostatics have been used to investigate many coal quality parameters, they have seldom been used to evaluate rock strength.

This paper describes an extensive investigation of these issues conducted by the National Institute for Occupational Safety and Health (NIOSH) in collaboration with Peabody Energy. The study employed the Peabody Rock Mechanics Data Base which contains more than 10,000 individual test results. Data from four important roof units were subjected to statistical analysis:

- Brereton Limestone above the Herrin 6 seam (Illinois)
- Turner Mine Shale above the No. 9 Seam (Kentucky)
- Sandstone above the Eagle Coal (West Virginia)
- Shale above the Eagle Coal (West Virginia)

The study did not find significant spatial trends in rock strength in any of the cases. Perhaps there are none, or perhaps the exploration coreholes were just too far apart to see them. These results have valuable implications for the design of geotechical exploration programs.

INTRODUCTION

Modern mine planning requires a thorough knowledge of the geotechnical conditions that will be encountered underground. Roof conditions can determine the bolting pattern, longwall pillar size, entry width, and the feasibility of extended cuts. Inaccurate, overly optimistic assumptions can mean unexpected hazards, higher costs, and lower productivity. Unfortunately, there are many examples of large investments that were lost because ground conditions were worse than anticipated.

The Westray coal mine in Canada provided a particularly tragic illustration of the potential consequences of inadequate geotechnical characterization. Westray was designed around large, thick seam, room-and-pillar mining equipment that had been successful in western Canada. The mine's planners did not foresee that the thinly laminated roof shale roof at Westray could not support the wide intersection spans the equipment required. Major roof falls began to occur on a regular basis, and it was obvious that the very existence of the mine was in question. Senior managers were preoccupied with finding the solution to the ground control problems. The diversion of their attention from other major issues and hazards made a significant contribution to the deadly methane explosion that killed 26 miners (Richard, 1999; Comish, 1993).

Most of the data used in coal mine geotechnical site characterization come from exploration coreholes. However, the number and location of coreholes is usually determined by the need to define coal thickness and quality, not rock mechanics. The result is that mine planners must often make due with widely spaced data points.

Therefore, it is important to try to make the most of the data that are available. There are two key issues:

- How to interpret the data from an individual hole, and;
- How to interpolate conditions between holes.

The purpose of this paper is to explore these two issues, and to begin to develop strategies for conducting geotechnical evaluations.

UNCONFINED COMPRESSIVE STRENGTH (UCS)

The two most important geotechnical parameters that can be obtained from coreholes are:

- The thickness of the roof units (lithography), and;
- The structural competence of each unit.

Both are required for the Coal Mine Roof Rating (CMRR), which provides perhaps the most complete picture of the overall roof stability.

The UCS is probably the most widely quoted measure of the structural competence. UCS tests (or UCS index tests, like the Point Load Test) are also one of the three main elements in the CMRR.[1] In the U.S., 3 to 5 samples are typically tested to determine the UCS of a rock unit.

One issue with the UCS is the high variability associated with it. One recent study (Rusnak and Mark, 2000) found that the standard deviation for a typical suite of UCS is 20-30% of the mean (figure 1). To illustrate what this means in plain English, lets suppose 5 tests of a sandstone in Corehole "A" had an average UCS=10,000 psi with a standard deviation of 3,000 psi. A 95% confidence interval around the mean has an upper limit of 13,450 psi and a lower limit of 6,555 psi. That is a pretty big range!

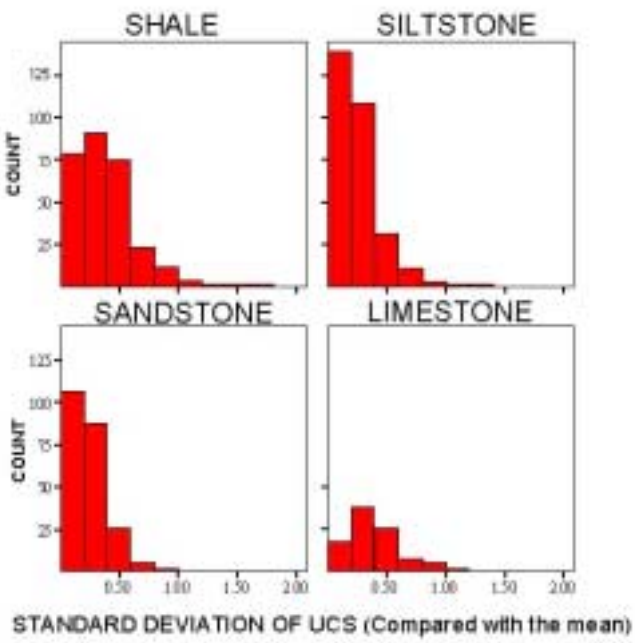

Figure 1. Histograms showing standard deviation of UCS for various rock types as a fraction of the mean (after Rusnak and Mark, 2000).

Now let us further suppose that the average UCS in nearby corehole "B" was 12,500 psi. Would we be right to think that the rock was actually stronger at Corehole B? Probably not.

An even more significant question is how much do we really know about the strength of the sandstone between our two coreholes. Let's suppose for a moment that the tests did tell us the "true" UCS. Would we expect that the sandstone's strength gradually increased from 10,000 psi at Corehole A until it reach 12,500 psi at Corehole B? This is what a contouring package would assume. But how do we know that the strength doesn't actually fluctuate a lot between the two coreholes (see figure 2)?

[1]The other two elements of the CMRRR are (1) the diametral point load strength (a measure of bedding plane strength) and (2) the RQD or fracture spacing (Mark et al., 2002).

Spatial statistics—commonly called Geostatistics—can help us with the answer.

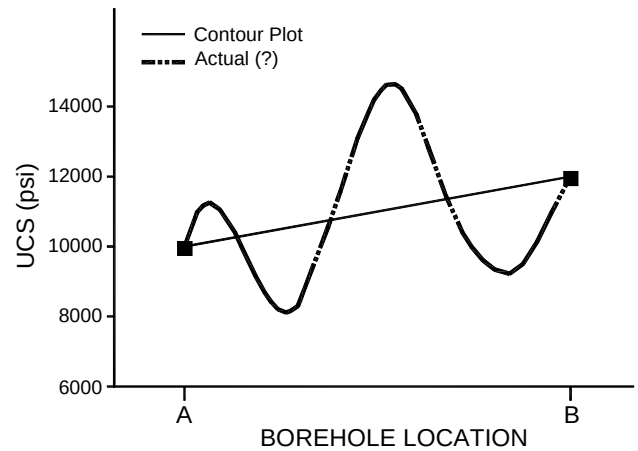

Figure 2. Conceptualization of UCS variation between two coreholes.

GEOSTATISTICS

Geostatistics were first developed to provide better estimates of ore reserves. In coal mining, they have been used to evaluate energy content, sulfur, ash, and other quality attributes of deposits. In fact, geostatistics can be used to study many properties that vary in space, but are measured at distinct locations. Research in fields as diverse as hydrology, forestry, air pollution, and global warming have all made extensive use of geostatistics (Ledvina et al., 1994; Armstrong, 1998).

The basic idea of geostatistics is this. Suppose the goal is to determine the sulfur content in a coal seam. If two coreholes are drilled just 1 ft apart from each other, one would expect that their sulfur values would be very similar. If a third hole is drilled 100 ft away, the sulfur content might be expected to change a little, but still be close the original value. As more holes are drilled further and further away, a distance is eventually reached where the first holes no longer help predict the sulfur content.

Geostatistics helps to *quantify* the spatial relationships that are observed between coreholes. The *variogram* is the central tool. The variogram is constructed by comparing all the possible pairs of data points in a set of drillhole data, and then calculating for each pair:

- The distance (h) between the two holes, and;
- The difference between the values of the parameter being measured (sulfur, in the example above) in the two holes.

An idealized variogram is shown in figure 3. The x-axis is the distance h between the drillholes. The y-axis is the "variance" (the square root of the variance is the standard deviation) for all the data pairs with approximately the same distance h between them. Some valuable information that can be obtained from the variogram are:

- **Range**: the distance h at which there is no *spatial* relationship between the holes (the holes are still related in the sense that they come from the same data set);
- **Sill**: Typically, the value of the variance for the entire data set (i.e., without considering any spatial relationships);

- **Nugget**: The short-range variability (i.e., the difference you might expect to find even with two coreholes drilled right next to each other).

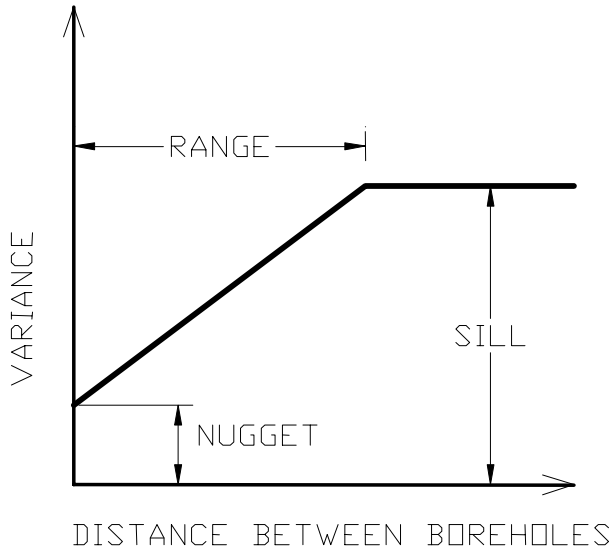

Figure 3. Idealized variogram.

Once the model variogram has been constructed, it can be used to obtain best statistical estimates of values throughout the entire reserve area using the mathematical techniques of "kriging." One big advantage of kriging is that the "kriging error" can also be plotted. The kriging error essentially tells us how much confidence we can have in our estimates of what's going on between the holes.

Of course, this has been a greatly simplified (and non-mathematical) explanation of geostatistics. Before we move on, three other points are worth mentioning:

- **Drift:** Geostatistics only work when it can be assumed that the entire data set has a single mean and standard deviation. If there is a regional trend (for instance, if the sulfur values decrease from north to south), then that "drift" must be removed mathematically before the variogram is constructed.
- **Outliers:** Because the variance is a squared function, even a single outlier can have a huge effect on the variogram. Often, the best thing to do is just remove them (Armstrong, 1998). Outliers can also be truncated (so that they still represent high or low values), or their effect can be minimized by working with the logarithms of the data values.
- **Anisotropy:** The simplest variograms look just at the distance between points, and do not consider the orientation of that distance. However, many geologic phenomena do have preferred orientations, and it is possible to construct variograms in different orientations (as long as enough data are available). Often, a look at a contour plot of the data is enough to determine if anisotropy might be an issue.

AN EXAMPLE OF GEOSTATISTICS APPLIED TO GROUND CONTROL

There are very few reported examples of the application of geostatistics to ground control (Reifenberg, 1994, 1996; Kim et al., 1989). In fact, of the nearly 1,000 papers included in the 22 Conferences in this series, only three have ever mentioned geostatistics.

One notable exception is the case history presented by Ledvina et al. (1994). Ledvina investigated the thickness of the Brereton limestone in the roof above the Herrin No. 6 seam in Illinois. At this mine, the thickness of the limestone was the controlling factor behind roof stability.

Ledvina had three data sets to work with:

- A large area with widely-space coreholes;
- A smaller area with more closely-spaced surface coreholes, and;
- A much smaller underground area, with very closely-spaced roof bolt test holes.

Figure 4a shows the contour plot for the limestone thickness from the widely-spaced coreholes. Note the large number of "bulls-eyes" (closely spaced contours around a single corehole) and large areas of seeming uniformity where data are sparse. The variogram (figure 4b) confirms that this drilling pattern is useless to predict limestone thickness between the coreholes. If there is some spatial relationship, it is clearly operating over distances that are less than the typical spacing between the holes.

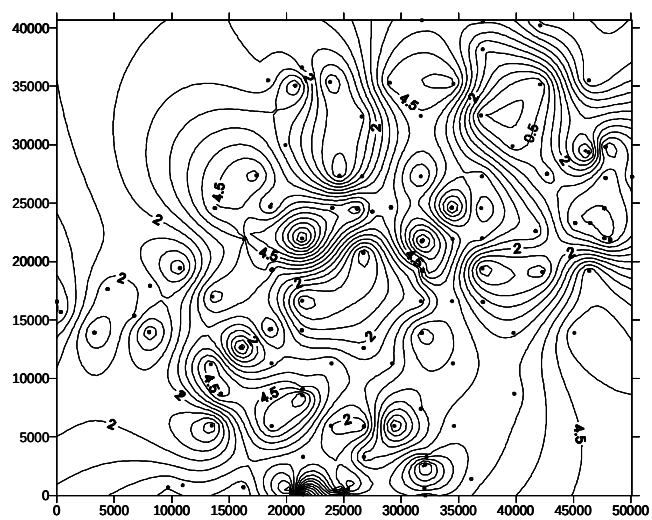

Figure 4a. Contour plot of Brereton Limestone thickness from surface coreholes. Limestone thickness contour intervals are 0.5 ft (after Ledvina et al., 1994).

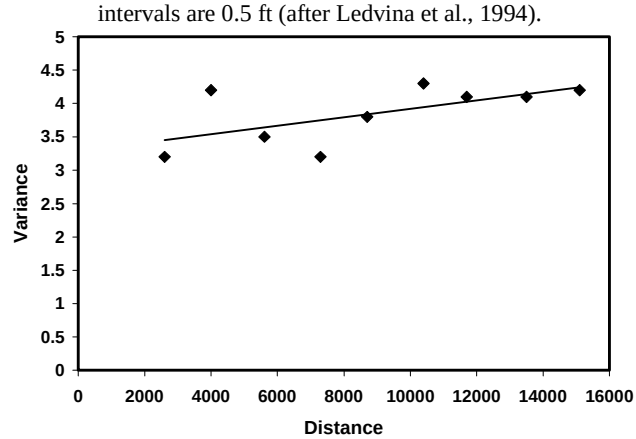

Figure 4b. Variogram constructed from data illustrated in figure 4a (after Ledvina et al., 1994).

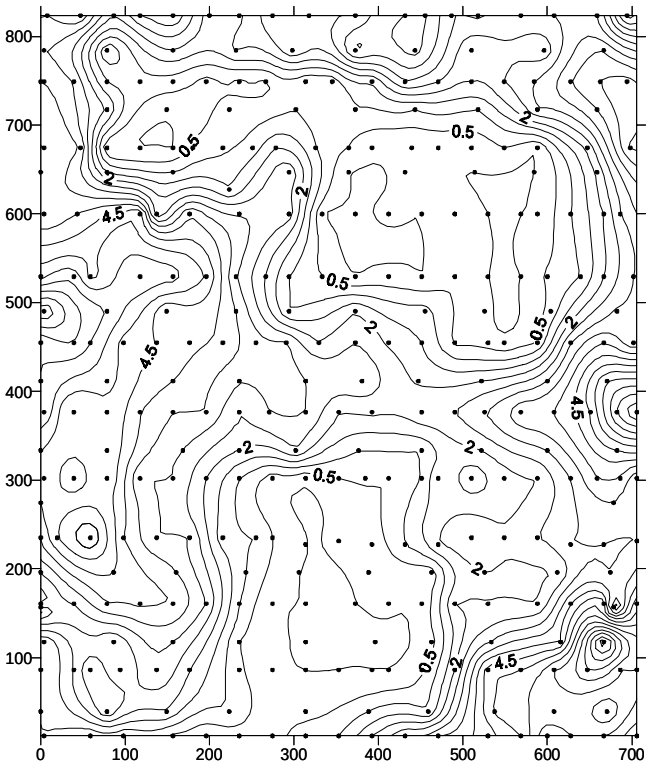

Figure 5a. Contour plot of Brereton Limestone thickness from closely-spaced underground test holes. Limestone thickness contour intervals are 0.5 ft (after Ledvina et al., 1994).

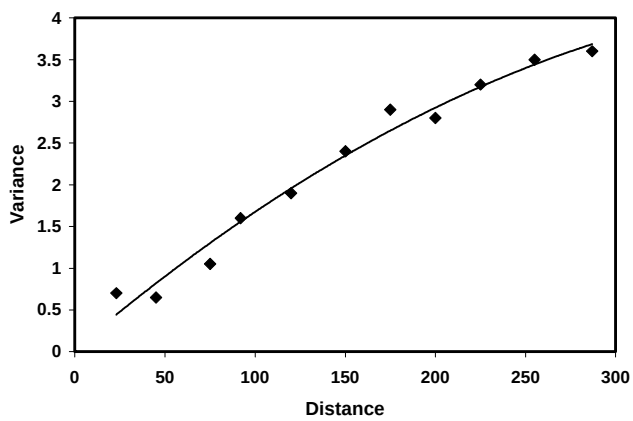

Figure 5b. Variogram constructed from data illustrated in figure 5a (after Ledvina et al., 1994).

In contrast, clear spatial trends can easily be seen in the contour map from the underground drilling area (figure 5a). The contour lines are much more regular, and each is generally defined by several data points. The variogram (figure 5b) has an almost ideal shape, with a nugget of zero and a range of about 300 ft. The implication is that if the mine really needed to know the thickness of the limestone, the holes would have to be drilled less than 300 ft apart.[2]

[2]Ledvina (1991) also provides details on a second case history from a mine working in the Springfield No. 5 seam in Illinois. This case studied the thickness of the Turner Mine Shale, the unit which lay directly above the coal. Again, Ledvina had three data sets with different hole spacings, and again only the underground data set displayed any meaningful spatial correlation, with the range of the variogram being about 300 ft.

It is interesting to note that for both of these data sets, the average limestone thickness was about 2.5 ft, and the standard deviation was about 2 ft. This is important because it tells us that even if our holes are too far apart to be spatially related, it does not mean that we know "nothing" about the rock in between the holes. We still know the *approximate range* of values for that rock (from the mean, the standard deviation, etc), we just don't have any idea *where* the high and low values might be.

ANALYSIS OF UCS DATA

The data used in this study were obtained from the Peabody Energy Rock Strength Data Base. Peabody Energy initiated full-scale rock mechanics testing in the fall of 1986. Testing has been done primarily on core samples obtained from exploration drilling to provide data for mine planning and design. A full range of testing is conducted including uniaxial compressive strength, indirect tensile strength, point load index, triaxial compressive strength, flexural strength, direct shear strength, long term creep, roof bolt anchorage capacity, slake durability, ultrasonic velocity, swelling strain and Atterberg limits. ASTM and ISRM procedures are followed for all rock mechanics testing. Currently, the data base contains rock mechanics test results from more than 1,000 drill holes from the states of WV, IL, KY, IN, CO, and OH.

Four data sets were analyzed for this study:

- Brereton Limestone above the Herrin 6 seam (Illinois)
- Turner Mine Shale above the No. 9 Seam (Kentucky)
- Sandstone above the Eagle Coal (West Virginia)
- Shale above the Eagle Coal (West Virginia)

Table 1 contains details on each data set. The two midwestern data sets have more holes each, but an average of just 2 UCS tests per hole. In the data sets from southern West Virginia, average number of tests per hole was about 5. In the case of the Brereton limestone, the holes were distributed over an area measuring approximately 40 miles by 40 miles. The areas measured about 12 miles by 12 miles in the other three cases.

Table 1. Overall rock unit summary.

State	No. holes	No. UCS tests	UCS average (psi)	UCS standard deviation	UCS variance (10^6)	Size of area (mile^2)
Rock Unit–Brereton Limestone						
IL	108	216	15,251	7,188	51.7	1,744
Rock Unit–Turner Mine Shale						
KY	73	135	6,991	4,040	16.3	109
Rock Unit–Shale above Eagle Coalbed						
WV	45	207	11,457	4,757	22.6	157
Rock Unit–Sandstone above Eagle Coal bed						
WV	50	295	15,266	3,850	14.2	165

The first step was to evaluate the variability of the UCS tests themselves. The goal is to apportion the total variability between:

- *Within-hole variability* due to the range of UCS values encountered in each individual hole, and;

- *Between-hole variability* which is determined by comparing the average UCS values in the different holes to one another.

If the between-hole variability is not significantly larger than the within-hole variability, it is unlikely that any meaningful spatial relationship will be present.

Figure 6 illustrates the concept, using the Eagle Sandstone data set. Only holes where at least 3 UCS tests were conducted are shown. The variability within each hole is signified by the range of UCS (the high-low bar) plotted for each hole. The variability between the holes is indicated by the range of the mean UCS values for the entire data set. The figure shows that the within-hole range of UCS values encountered in many individual holes was similar to range of mean UCS values for the entire data set. Clearly, the within-hole variability is quite high relative to the between-hole variability.

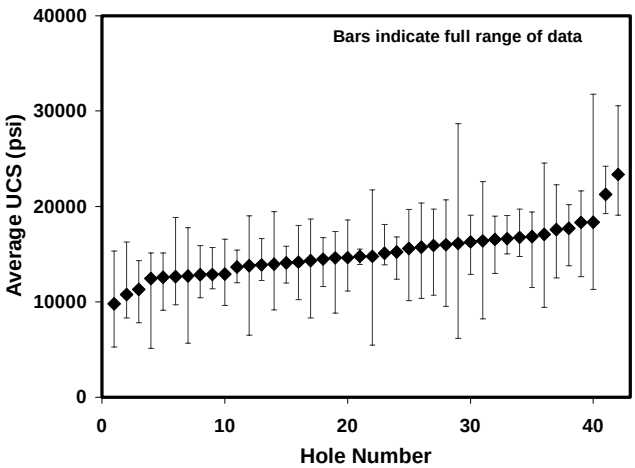

Figure 6. Within- and between-hole variability for the Eagle Sandstone data set.

A one-way analysis of variance (ANOVA) was used to compare the between-corehole variability to the within-corehole variability for each of the four data sets. Only the boreholes with two or more UCS tests were used in the analysis. Table 2 shows the results. First, the degrees of freedom (df) are calculated according to the number of holes and the total number of UCS tests in each data set. Then the sum-of-squares, indicating the total variability within each data set, is apportioned between the within- and between-hole variability. The mean square is simply the sum-of-squares divided by the df. The F-value is the ratio of the mean squares. When the F value for the model is statistically significant ($p < .05$), this indicates that the mean UCS for *at least one* of the coreholes is different from the mean UCS of the other coreholes. Without a significant F-value, it is likely that all of the holes share a common "true mean" UCS, and it would probably make little sense to look for spatial relationships within the data.

Table 2 shows that the Eagle Sandstone data set shown in Figure 6 actually has the second greatest F value (i.e., the between hole variability is actually the largest relative to the within-hole variability). The other West Virginia data set has a slightly higher F-value. The F-values are very low for both the Midwestern data sets. In the case of the Turner Mine Shale, it seems that the range of UCS values likely to be encountered in a single hole is approximately the same as that in the entire data set.

Table 2. Comparison of within- to between-corehole variability.

Source of Variability	dF	Sum of squares (10^6)	Mean square (10^6)	F-value	Pr > F
Rock Unit–Brereton Limestone					
Between-Corehole	51	3,131	61.4	1.40	0.07
Within-Corehole	108	4,725	43.8		
Rock Unit–Turner Mine Shale					
Between-Corehole	44	617	14.0	0.70	0.90
Within-Corehole	62	1,250	20.2		
Rock Unit–Shale above Eagle Coalbed					
Between-Corehole	35	1,795	51.4	3.58	<.001
Within-Corehole	162	2,327	14.3		
Rock Unit–Sandstone above Eagle Coal bed					
Between-Corehole	45	1,515	33.7	2.94	<.001
Within-Corehole	245	2802	11.4		

Computation of the variograms was the final step in the analysis. The results are shown in figure 7. Of the four data sets, only in the two strong rocks is there even a hint of a spatial relationship. Of these two, the Eagle Sandstone has the best variogram, but even here the variance is still quite high for even the closest holes. The variograms for the shales are essentially horizontal lines, indicating that there is basically no spatial relationship between the holes.

CONCLUSIONS

The lack of spatial correlation in these four data sets has some important implications. First, it means that contour plots of UCS for these units must be treated with skepticism. This would be particularly true for the shale units.

Second, since these four units represent a fairly wide range of UCS, rock types, and U.S. coal basins, it seems likely that meaningful spatial correlations for UCS are the exception rather than the rule for other rock types. The most likely explanation is the large variability in UCS values that is regularly observed within single coreholes. Perhaps it is not surprising that in layered, sedimentary rock that the vertical variations even over small distance are quite large compared with the horizontal variations.

It is possible that the UCS of some rock types does truly vary from place to place, but that exploratory drillholes are just too widely spaced to measure it. In the past studies cited in this paper, Ledvina et al. (1994) found that a corehole spacing on the order of 300 ft was necessary to accurately predict the thickness of some roof rock units. However, the large within-corehole variability in the two shale units implies that it is highly unlikely that a spatial relationship for UCS could be found regardless of the corehole spacing.

Finally, this paper has shown the power of geostatistics in helping to understand the variability of geomechanical properties of coal measure rock. Increased use of geostatistical methods could greatly improve the reliability of data used in ground control design.

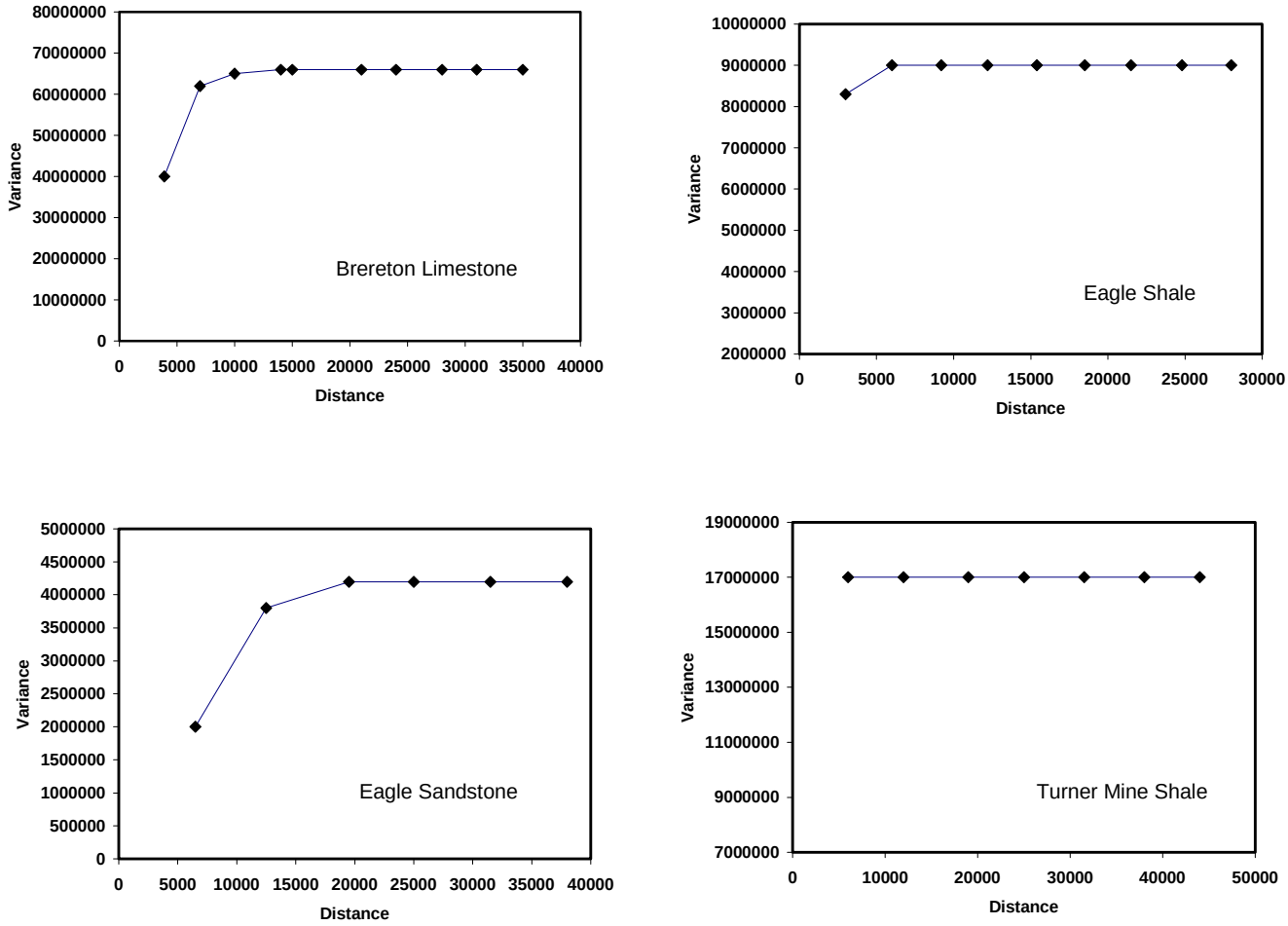

Figure 7. Variograms constructed from the four data sets.

REFERENCES

Armstrong, M. (1998). Basic Linear Geostatistics. Springer-Verlag, Berlin, 155 pp.

Comish, S. (1993). The Westray Tragedy –A Miner's Story. Fernwood, Halifax, NS, 82 pp.

Kim, Y.C., Cervantes, J.A., Farmer, I.W. (1989). Predicting Rockfalls in an Underground Coal Mine Using Discriminant Analysis and Geostatistics. Proceedings, 7th Annual Workshop of the Generic Mineral Technology Center for Mine Systems Design and Ground Control, Blacksburg, VA, pp. 95-104.

Ledvina, C.T., Dowding, C.H., Fowler, S., Hunt, G. and Nance, R. (1994). Geostatistical Guidance of Exploration in Roof Control – How Many Drill Holes are Enough? Proceedings, 5th Conference on Ground Control for Midwest U.S. Coal Mines, Collinsville, IL, YP Chugh and GA Beasley, Eds., pp. 14-30.

Ledvina, C.T. (1991). Geostatistical Inference and Exploration of Coal Mine Roof Strata. PhD Dissertation, Northwestern University, Evanston, IL, 419 pp.

Mark, C., Molinda, G.M., Barton, T.M. (2002). New Developments with the Coal Mine Roof Rating. Proceedings, 21st International Conference on Ground Control in Mining, Morgantown, WV, Aug. 6-8, pp. 294-301.

Reifenberg, J. (1994). Geostatistical Modeling of Coal Mine Roof Quality for Hazard Mapping. Proceedings, 5th Conference on Ground Control for Midwest U.S. Coal Mines, Collinsville, IL, YP Chugh and GA Beasley, Eds., pp. 31-40.

Reifenberg, J. (1996). Geostatistical Methods for Hazard Assessment and Site Characterization in Mining. Proceedings, 15th International Conference on Ground Control in Mining, Golden, CO, Aug. 13-15, pp. 671-680.

Richard, K.P. (1999). The Westray Story—A Predictable Path to Disaster. Chapter 10--Ground Control. Report of the Westray Mine Public Inquiry, Nova Scotia Department of Labor, Vol. 1, pp. 351-385.

Rusnak, J. and Mark, C. (2000). Using the Point Load Test to Determine the Uniaxial Compressive Strength of Coal Measure Rock. Proceedings, 19th International Conference on Ground Control in Mining, Morgantown, WV, Aug. 7-9, pp. 362-371.