Chapter 2

First Estimates of Grade and Tonnages and Potential Grade and Tonnages

It is common practice in exploration to start with economic evaluations as early as possible and to update these evaluations in parallel with the physical exploration work with an ever improving data base. The purpose of this ongoing process is to have a ready base for go/no-go decisions after each exploration stage before proceeding to the next normally more expensive stage. An economic evaluation needs tonnage and grade information to work with. In an early stage, the geologist has only an tentative idea about expected grades and tonnages based on the initial geological concept and early concrete indications through observations from trenches or a limited number of drill holes. This early idea about grades and tonnages we will call grade potential and tonnage potential.

If the exploration of a possible deposit is well advanced, one can work with geostatistical methods, which take the spatial interdependence of drill hole data into account (see e.g. Wellmer 1998, Stat. Eval.) and are certainly the best way to arrive at the most reliable input data. At an early exploration stage, however, a sufficiently large data base is not available for geostatistical methods. Other cruder methods have to be applied to arrive at approximate estimates of grade and tonnage or potential grade and tonnage. Many exploration projects have a chequered history with many owners. Sillitoe (1995) examined the history of 53 Circumpacific producing base- and precious metal mines. Only a third went from discovery to the stage of producing mine in one go, meaning with one company, for the second third two attempts were necessary, and for the last third up to 11 different companies tried their exploration luck and only the last one was successful to bring the deposit into production. Consequently one frequently deals with a mixed bag of data sets. For example, there might be a property with some percussion hole data, some data from core drilling – some with good core recoveries, some with low core recoveries – some data from chip sampling in trenches and from bulk sampling in an exploration pit. Some holes might have been drilled at very oblique angles in an attempt to show large apparent thicknesses to a potential buyer or farm-in partner. One cannot afford to disregard low quality data. The competition for good exploration projects is fierce, and therefore the maximum information value has to be extracted from all data available, regardless of quality.

In this book, we are dealing only with first order-of-magnitude estimates for grade and tonnage (or the potential of both quantities) aimed at obtaining quick-and-ready economic assessments using any available data. This is common practice for exploration and mining companies at all stages of evaluation when go/no-go decisions are required. More advanced methods for larger data sets are dealt with in Wellmer 1998 (Statistical Evaluations in Exploration for Mineral Deposits) or other geostatistical textbooks for ore reserve estimation.

For this purpose of obtaining quick-and-ready-economic assessments we need in any case the true thickness from a drill hole intersection and a first idea about block sizes. Only this is briefly demonstrated in this book, primarily concerned with economic evaluations, with deriving blocks on cross sections and plan maps.

The advances of computer programmes makes three dimensional (3D-) modelling very easy. They shall not be discussed here. It should be pointed out, however, that with limited data at hand a first volume estimate based on a computer model is not "more correct" than the sectional or polygonal approach.

2.1 Estimation of Volume and Tonnage of Ore Deposits

2.1.1 Calculating the True Thickness

2.1.1.1 Drilling Perpendicular to Strike

This is the standard case. As a rule, a profile is drawn from which the true thickness can be graphically measured. For exact calculations, if the drill length is L_B (Fig. 2.1a), the true thickness (M_w) is given by

$$M_w = L_B \times \sin[180° - (\alpha + \beta)] = L_B \times \sin(\alpha + \beta)$$

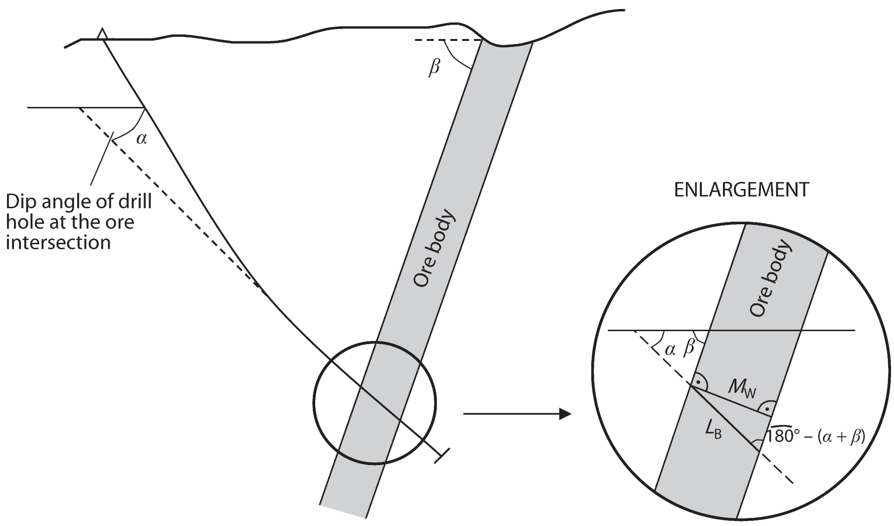


Fig. 2.1a. Vertical section to calculate the true thickness of a drill intersection

Where α is the inclination angle of the drill hole at the intersection of the drill hole with the ore body and β is the dip angle of the ore body. If the drill hole is perpendicular, i.e. perpendicular at the point of intersection, then α is 90° and the relationship will become (see also Sect. 2.2.3.3 and Fig. 2.9)

$$M_W = L_B \times \cos\beta$$

because $\sin(90° + \beta) = \cos\beta$

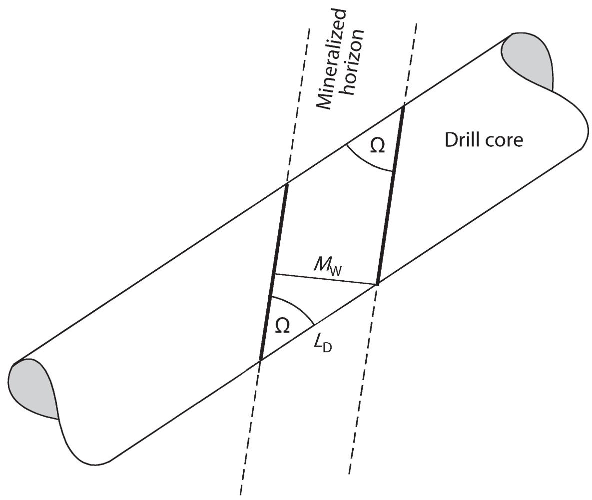
In Wellmer 1998 (Stat. Eval.) in Sect. 7.3, page 48ff and Fig. 18 about the law of perpetuation of errors, it is shown what effects errors in the angles α and β can have. If a drill hole does not intersect an ore body perpendicular, but at an oblique angle, the error for the true thickness increases dramatically at very oblique angles i.e. if the angle between ore body and drill hole is less than 30° or, respectively, more than 150°.

2.1.1.2
Drilling Oblique to Strike (see Appendix B)

The situation can be more complicated, if the drill hole runs oblique to strike. Spatial restrictions such as drilling underground or in mountainous areas often necessitate drilling oblique to strike. Sometimes, however, this method is used by promoters to give the impression of an exaggerated apparent thickness and disguise a low true thickness.

As long as one drills a stratabound horizon with clear hanging and foot wall contacts which are recognizable in drill core, the situation is simple. Let us do a thought experiment: We drill a stratabound deposit. Regardless under which angle you intersect the stratabound ore horizon you will get a core as shown on Fig. 2.1b. There is an angle between the core axis and the stratabound horizon, which we will call Ω. We do

Fig. 2.1b.
Example of a drill core which intersected a stratabound ore horizon at an oblique angle



not have to know anything about strike or dip of the ore horizon. With this angle Ω and the apparent thickness in the drill hole L_D we can determine the thickness of the ore horizon M_W, meaning the normal distance between foot and hanging wall measured at right angles, which is

$$M_W = L_D \sin \Omega$$

So Ω corresponds to the angle 180° – ($\alpha + \beta$) in the enlargement of Fig. 2.1a.

However, especially with vein or other epigenetic mineralizations, foot and hanging walls are frequently very irregular or blurred. Often core losses occur when the drill hole reaches mineralisation because of changes in rock competency. So one just knows in the drill core where the mineralisation starts and ends, but there are no obvious planes from which angles can be taken. We now have to calculate the true width from the known direction and dip of the drill hole in relation to the strike and dip of the mineralized body as best as this can be inferred.

α is the angle of inclination of the drill hole, β the angle of dip of the orebody, γ the angle between the horizontal projection of the drill hole and the dip direction (Fig. 2.2a). In addition, we need δ, the apparent angle of dip of the orebody along the drilling direction.

First we want to express the apparent dip angle δ in terms of the dip angle β and the profile angle γ via the depth h (Fig. 2.2b). The triangle AHG is oriented perpendicular to the strike of the orebody. So the angle between $\overline{\text{AH}}$ and $\overline{\text{GH}}$ is the dip angle β. Therefore

$$h = b \times \tan \beta \tag{2.1}$$

Now we consider the triangle AJG with the apparent dip angle δ. The relationship for h is

$$h = \text{c} \times \tan \delta \tag{2.2}$$

combining Eqs. 2.1 and 2.2 we get

$$b \times \tan \beta = c \times \tan \delta \tag{2.3}$$

In the horizontally lying triangle AHJ the angle between b and c is γ, therefore

$$\frac{b}{c} = \cos \gamma \tag{2.4}$$

Combining Eqs. 2.3 and 2.4 we get

$$\tan \delta = \cos \gamma \times \tan \beta \tag{2.5}$$

To determine now the true thickness M_W we go back to Fig. 2.2a.

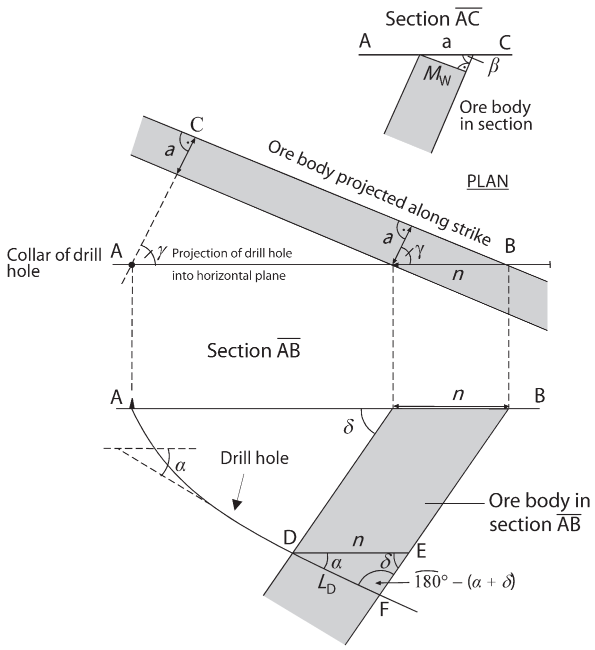


Fig. 2.2a.
Plan and section to calculate the true thickness from a drill hole running oblique to strike

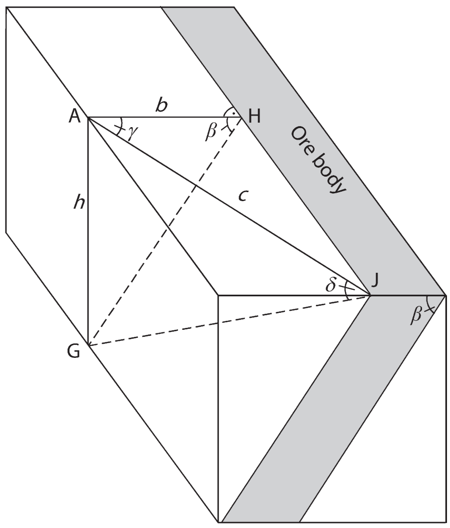


Fig. 2.2b.
Block diagram to calculate the apparent dip angle

From the profile $\overline{AC}$ (Fig. 2.2a) the true thickness M_W can be determined as

$$M_W = a \times \sin\beta \qquad (2.6)$$

where a is the apparent horizontal thickness perpendicular to strike.

From the horizontal plan in Fig. 2.2a, with n being the apparent horizontal thickness in drilling direction AB, a can be determined:

$$a = n \times \cos\gamma \tag{2.7}$$

Equations 2.6 and 2.7 combined give

$$M_w = n \times \sin\beta \times \cos\gamma \tag{2.8}$$

n can be derived from the triangle DEF in profile $\overline{AB}$ (Fig. 2.2a) by using the sinus relation, with L_D being the length of the intersection:

$$\frac{L_D}{\sin\delta} = \frac{n}{\sin(180° - (\alpha+\delta))} = \frac{n}{\sin(\alpha+\delta)}$$

$$n = L_D \frac{\sin(\alpha+\delta)}{\sin\delta} \tag{2.9}$$

Substituting Eq. 2.9 for n in Eq. 2.8; the result is

$$M_W = L_D \frac{\sin(\alpha+\delta)}{\sin\delta} \sin\beta \times \cos\gamma \tag{2.10}$$

Replacing $\cos\gamma$ by the term in Eq. 2.5:

$$\cos\gamma = \frac{\tan\delta}{\tan\beta} = \frac{\sin\delta\cos\beta}{\cos\delta\sin\beta} \tag{2.11}$$

results in

$$M_W = L_D \frac{\sin(\alpha+\delta)}{\cos\delta} \cos\beta \quad \text{or} \quad M_W = L_D R_M \tag{2.12}$$

with

$$R_m = \frac{\sin(\alpha+\delta)}{\cos\delta} \cos\beta \tag{2.13}$$

or R_m expressed only with the directly observable angles α (angle of inclination of drill hole), β (angle of dip of the target) and γ (angle of profile between drill direction and dip direction), using Eq. 2.5 and thereby not using the auxiliary angle δ:

$$R_m = \cos\beta(\sin\alpha + \cos\alpha \times \cos\gamma \times \tan\beta)$$

R_m is the thickness reduction factor. In Appendix B, curve sets for R_m are given for various drill hole inclinations (Figs. B1 to B4). At the end of Appendix B, in addition, is a diagram showing at which angle to drill if an optimum length of the intersection is to be obtained when drilling oblique to strike (Fig. B5).

2.1.2
Reserve Estimations Based on Sections

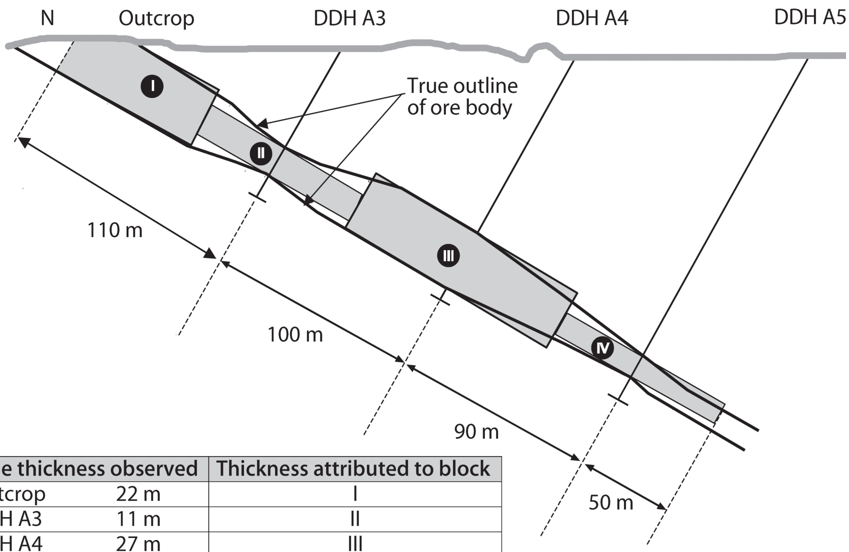
If a deposit has been systematically drilled on sections, e.g. on lines cut in the bush of northern Canada or in the rain forests of South America, reserve calculations will be based on cross-sections along these lines.

To each cross-section is assigned an area of influence corresponding to half the distance to the two adjoining sections. The limits of the blocks thus defined lie exactly halfway between the drill holes (see Fig. 2.3).

The surface area of the blocks on the section are given in Table 2.1.

If we assume the distance between neighbouring sections to be 50 m and the density of the ore to be 4.0 g/cm^3, we arrive at a tonnage on this profile of

$$T = 50 \times 4 \times 5\,595 = 1.119 \text{ million t}$$



True thickness observed		Thickness attributed to block
Outcrop	22 m	I
DDH A3	11 m	II
DDH A4	27 m	III
DDH A5	7 m	IV

Fig. 2.3. Cross-section for reserve calculations with blocks

Table 2.1.
Surface area of the blocks in Fig. 2.3

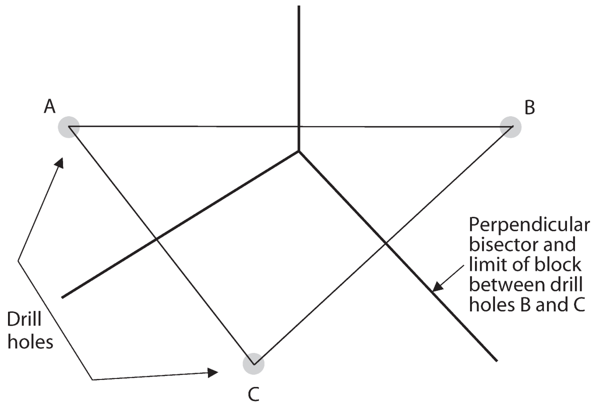
Block	Surface area (m^2)
I	$55 \times 22 =$ 1 210
II	$(55 + 50) \times 11 =$ 1 155
III	$(50 + 45) \times 27 =$ 2 565
IV	$(45 + 50) \times 7 =$ 665
Total	5 595

The important question of how far one can extrapolate from the last drill hole can best be answered geostatistically, if enough data for a geostatistical evaluation are available (Wellmer 1998, Stat. Eval. p. 223). A rule-of-thumb from experience is to use half the distance between drill holes, but seldom more than 50 m. The resources beyond this limit should be considered as resource potential.

2.1.3 Reserve Estimations on the Basis of Plan Maps

Drilling in mountainous terrains or residential areas, where suitable sites for drill holes are restricted, will result in irregularly spaced intersections. Drill holes with significant hole deviations produce the same effect. In such cases, instead of using cross sections, it is better to work with plan maps for inclined tabular deposits or palinspastic maps for folded ones.

Fig. 2.4. Construction of equidistance lines



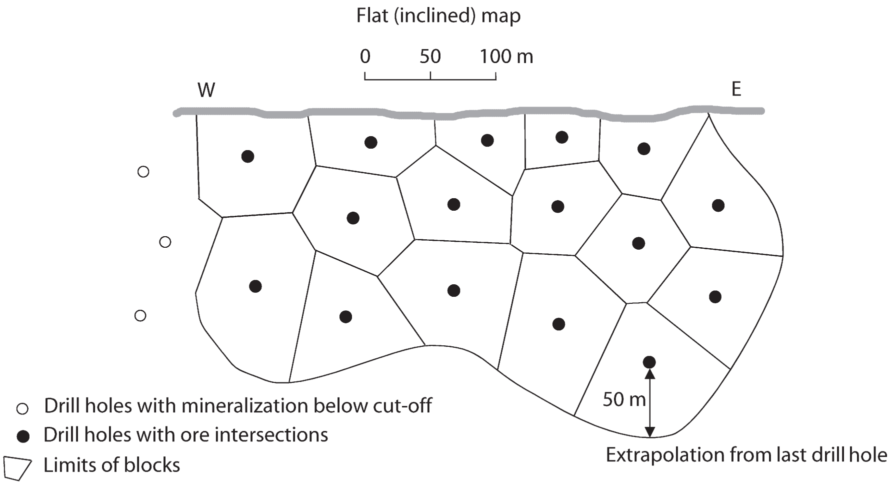


Fig. 2.5. Plan map for reserve calculation with blocks

Usually the blocks (see Fig. 2.4 and 2.5) are delimited by drawing equidistance lines to the adjoining drill holes. As Fig. 2.5 shows, applying this method creates polygons. That is the reason why this method is also called the polygon method. The block method of Sect. 2.1.2 and the polygon method definitely have weaknesses (Giroux 1990). If enough data are available and geostatistical tools can be applied, these are to be preferred (Wellmer 1998, Stat. Eval. Sect. 13.3). Block and polygon methods are, however, well suited for a first orientation. The surface area of the blocks is then multiplied by the thickness and density as in the example in Sect. 2.2.1. The construction of the equidistance lines is explained below and shown in Fig. 2.4.

By connecting adjoining boreholes with each other a net of triangles is created. The equidistance lines, perpendicular bisectors, halve the sides of these triangles and bound the polygonal area of influence centred on each hole. The western border of the deposit in Fig. 2.5 is defined by drill holes which encountered uneconomic mineralisation (grades below cutoff). How to determine cutoff limits will be dealt with in Sect. 10.1.

2.2 Grade Estimation and Weighting

Grade estimations will only be dealt with in this book if the calculations involve simple weighting with, for example, assay intervals in drill holes or with reserve block volumes. This is sufficient for a global estimate of a deposit, or potential deposit in the early stages of exploration. A global estimate is the estimate of grade (or tonnage) of the total deposit, contrary to a block estimate. As will be shown later in Chap. 11 we assume in our simplified economic calculations that the grades during each mining year are the same, meaning the grades of the global estimate. If one wants to model the deposit more in detail and simulate the change of grades from year to year, one has to use geostatistical methods for grade determinations of blocks (Wellmer 1998, Stat. Eval. Sect. 13.4).

In this chapter we will also deal with the problem of deriving grades from visual inspections. When there are old adits with visually recognizable mineralisation on a property offered for sale, it is possible to get a quick grade estimate as helpful preliminary information for a global estimate.

2.2.1 Weighting in Reserve Calculations

One of the most frequent calculations geologists have to do are weightings, e.g. for the calculation of the average grade of a drill hole from assay intervals of different lengths or of the average grade of a deposit from the combined grades of individual, unequal blocks.

If G_1 to G_n are the values whose weighted average is to be determined, and a_1 to a_n are the weighting factors, then the weighted average is $\overline{G}_w$:

$$\overline{G}_w = \frac{G_1 a_1 + G_2 a_2 + \cdots + G_n a_n}{a_1 + a_2 + \cdots + a_n} = \frac{\sum_{i=1}^{n} G_i a_i}{\sum_{i=1}^{n} a_i} \qquad (2.14)$$

Assignment. The analytical results from unequal, but consecutive intervals are provided in Table 2.2.

What is the weighted mean?

The weighted mean is

$$\bar{G}_w = \frac{2.1\times1+8.4\times1.5+12.0\times0.75+10.2\times1.25}{1.00+1.50+0.75+1.25}$$

$$\bar{G}_w = \frac{36.45}{4.50} = 8.10\%\ \text{Pb}$$

Careful consideration must be given to the choice of the correct weighting factors. The weighting in the above example assumes that the densities are constant (or the difference in densities is negligible). If this assumption is not justified, as it often happens with vein deposits in which massive sulphide and disseminated ore occur together, then the density must also be allowed for in the weighting.

Assignment. Calculate the weighted mean for the drill intersections in a barite deposit presented in Table 2.3.

The weighted average is

$$\bar{G}_w = \frac{70\times1.5\times3.7+98\times2.8\times4.2+50\times1.0\times3.4}{1.5\times3.7+2.8\times4.2+1.0\times3.4}$$

$$\bar{G}_w = \frac{1\,771.0}{20.7} = 82.7\%\ \text{BaSO}_4$$

An additional exercise will show how important it is to perform the weighting correctly.

Table 2.2. Analytical results from unequal, consecutive intervals

Analytical result (% Pb)	Sample interval (m)
2.1	1.00
8.4	1.50
12.0	0.75
10.2	1.25

Table 2.3. Drill intersections in a barite deposit

Analytical result (% $BaSO_4$)	Sample interval (m)	Density (g/cm^3)
70	1.50	3.7
98	2.80	4.2
50	1.00	3.4

Assignment.

1. *Question:* Which mistake crept into the following reserve calculation and how big is it?
2. *Case Description:* A nickel laterite deposit has been sampled by pits. The pits are 25 m apart. Each pit has therefore an area of influence of 12.5 m to each side. The lines on which the pits are located are at a distance of 50 m so that an area of $50 \times 25 = 1\,250$ m is allocated to each pit. Two different types of ore with different densities were encountered in the pits (Fig. 2.6): the laterite (L) has an in situ density of 1.25, the decomposed serpentinite (ZS) has an in situ density of 1.0 g/cm^3.
   i. The average grades of the pits were determined by weighting with the lengths:

$$\text{Pit A:} \frac{4\times1.2+4\times2.9}{8}=2.05\%\ \text{Ni}$$

$$\text{Pit B:} \frac{4\times1.1+3\times3.5}{7}=2.13\%\ \text{Ni}$$

   ii. In addition, the densities were determined by weighting with the sample lengths:

$$\text{Pit A:} \frac{4\times1.25+4\times1.0}{8}=1.125$$

$$\text{Pit B:} \frac{4\times1.25+3\times1.0}{7}=1.143$$

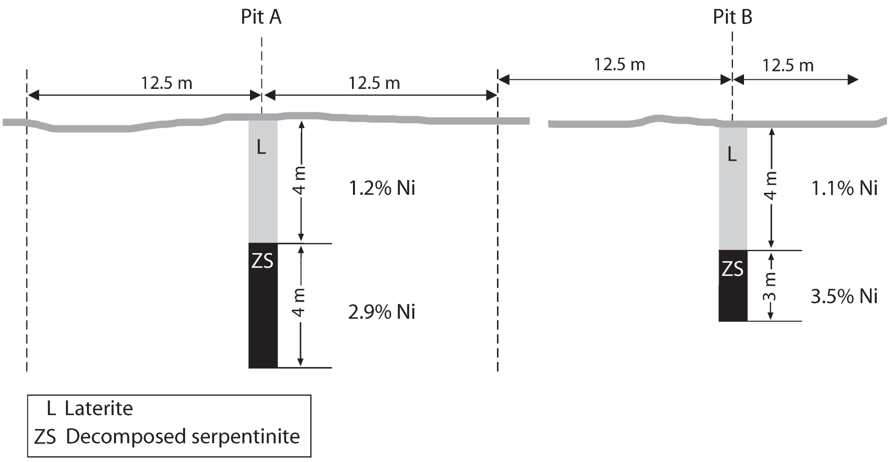


Fig. 2.6. Pit sampling in a nickel laterite deposit

iii. Since each pit has been allocated a surface area of 1 250 m^2 and the pits have a depth of 7 and 8 m respectively, the following tonnages were obtained:

Pit A: $1\,250 \times 8 \times 1.125 = 11\,250$ t with 2.05% Ni
Pit B: $1\,250 \times 7 \times 1.143 = 10\,000$ t with 2.13% Ni

iv. The nickel grade of the total tonnage was determined by weighting with the corresponding tonnages:

$$\frac{2.05 \times 11\,250 + 2.13 \times 10\,000}{21\,250} = 2.09\%\ \text{Ni}$$

3. *Correct Answer*: The following mistake was made in step *(i)*: the average grades of the individual pits were not determined by directly weighting with the densities. The correct procedure is

i. Pit A: $\dfrac{4 \times 1.25 \times 1.2 + 4 \times 1.0 \times 2.9}{4 \times 1.25 + 4 \times 1.0} = 1.96\%\ \text{Ni}$

ii. Steps *(iii)* and *(iv)* are correct. Using the correct grades step *(iv)* will result in

$$\frac{1.96 \times 11\,250 + 2.00 \times 10\,000}{21\,250} = 1.98\%\ \text{Ni}$$

The mistake leads to an overestimation of 6%. The mistake is unacceptably large for the purpose of reserve calculation, both from a purely mathematical as well as economic point of view.

2.2.2
Grade Calculations for Massive Ore Shoots

Determining grades through visual estimates is another example where correct weighting with densities is of importance. For vein-type ore deposits in which the ore occurs massive, visual grade control often plays a significant role.

Assignment. We are dealing with a steep vein which, for technical reasons, has to be mined at a minimum thickness of 1 m. In the vein a massive stibnite shoot occurs. How many percent antimony correspond to a band of 1 cm stibnite?

Stibnite has a density of 4.5 g/cm^3, the wall rock a density of 2.6 g/cm^3.

Theoretically stibnite (Sb_2S_3) contains 71.7% Sb. We assume 70%.

The thickness of the massive stibnite band has been measured at intervals of 1 m. We consider a vein surface of 1 m^2 and a mining width of 1 m.

1. With 1 m mining width and 1 cm stibnite band, the tonnage of the wall rock per 1 m^2 vein surface is

$$0.99\ \text{m} \times 1\ \text{m}^2 \times 2.6\ \text{t/m}^3 = 2.574\ \text{t}$$

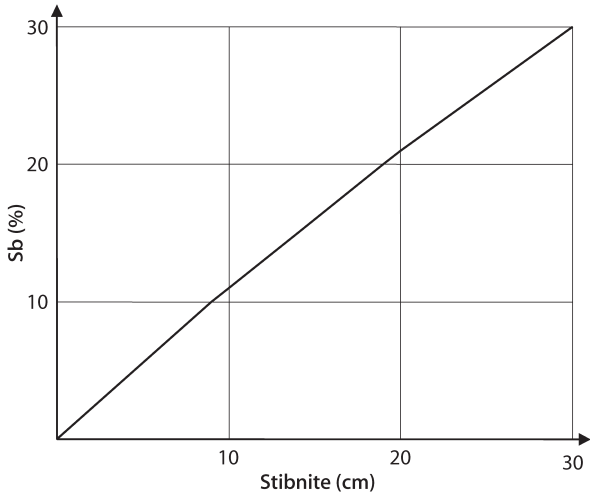


Fig. 2.7.
Graph for conversion of massive ore thicknesses (here, stibnite)

2. 1 cm stibnite per 1 m^2 vein surface corresponds to

$$1\times 10^4\ (cm^3)\times 4.5\left(\frac{g}{cm^3}\right)\hat{=}\ 45\ (kg)$$

   i.e. the total tonnage per 1 m vein surface is 2.619 t. With a conversion factor of 0.7 : 45 kg stibnite $\hat{=}$ 31.5 kg Sb
3. Conclusion: 1 cm stibnite $\hat{=}$ 31.5 / 26.19 $\hat{=}$ 1.2% Sb

Since the thickness of the lighter wall rock decreases with increasing thickness of the ore shoot, this conversion factor cannot be used as a linear function with greater ore thickness.

30 cm stibnite do not correspond with 36% Sb but with 29.8% Sb! It is better to construct a graph so that the grades can be quickly derived from the massive ore thicknesses (Fig. 2.7).

Although the ore phases often appear to be pure, a very fine intergrowth with gangue minerals is frequently revealed under the microscope. It is therefore advisable to check these conversion factors analytically and, if necessary, to correct them by means of a factor. A good example are the detailed analyses in the lead-zinc-vein mine Bad Grund in the Hartz mountains in Germany (Stedingk 2006). In the ore shoots the thicknesses of the sphalerite and galena bands were regularly measured optically and these measurements were the basis of grade control and mine planning. Whereas the predicted zinc grades agreed reasonably well with the grades of the run-of-mine ore, the lead grades were considerably overestimated. Microscopical studies showed an intimate intergrowth of galena with quartz and siderite gangue. This intimate intergrowth created the illusion of massive galena mineralisation. To bring predicted and realized grades into agreement coarse grained galena zones could be taken at face value, but the values of the visual measurements of the fine grained intergrown zones had to be divided by a factor of three. So in the mine the term "third-galena" was coined for this mineralogical phase.

2.2.3
Grade Determinations from Geophysical Downhole Logging

2.2.3.1
Introduction

In uranium exploration, it is common practice to use percussion holes, so no direct samples are obtained. However, because uranium and its radioactive decay products emit gamma radiation they can be detected and measured as counts per second "cps" in the drill holes by using down-the-hole gamma ray instruments[3]. In the evaluation of the geophysical measurements weighting plays an important role in determining grades.

Strictly speaking, uranium itself does not emit detectable amounts of gamma radiation. The gamma radiation is caused by the decay products of uranium, principally bismuth-214. In radiometric surveys, one assumes that the daughter products of the decay are in equilibrium. If this is not the case, one has to work with correction factors (see below Sect. 2.2.3.4 where correction factors are discussed). The procedure of determining uranium grades from gamma radiation cannot be used if other strong gamma emitters like thorium or potassium are present in significant amounts. Because the uranium is not measured directly, such values are not given as units of ppm or percent of U_3O_8 but as equivalent value. In the notation for this, an e is prefixed to signify that we are dealing with an equivalent value; for example, 150 ppm eU_3O_8.

2.2.3.2
Down-the-Hole Logs and Their Use

Grades are deduced from the gamma ray measurements. In consequence, it is common practice to diamond drill a hole with core after a certain number of percussion holes, usually 10, in order to be able to determine grades on core material by chemical analysis. This serves as the basis for calibration of the gamma-ray log results.

Drill hole logs are also used for other elements, such as lead, zinc, copper and iron. Fricke et al. (1987) describe a down-the-hole method which consists of introducing a radioactive source into the drill hole which induces a secondary radiation that can be measured with the help of an X-ray fluorescence device.

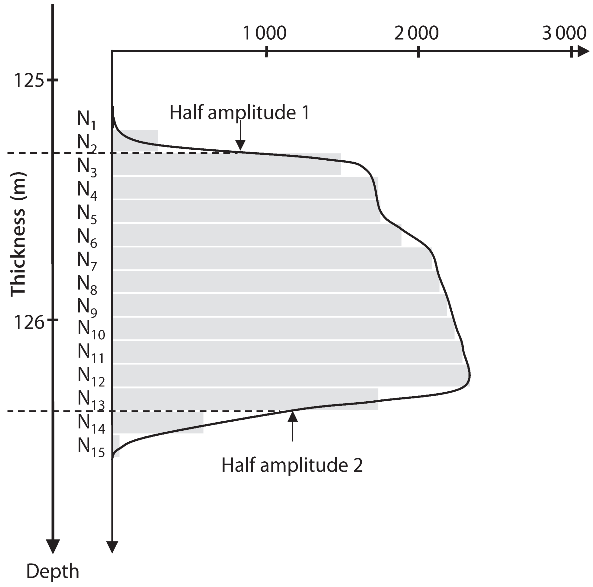
The following information can be determined from down-the-hole measurements:

a the thickness of the mineralized horizon
b the average grade of the mineralized horizon using the accumulation factor $G \times T$, i.e. the product of grade times thickness (see also Sect. 1.2.4)

This is illustrated with a gamma-ray log from an uranium exploration drill hole (Fig. 2.8). For a detailed explanation the reader is referred to handbooks available from the International Atomic Energy Agency (IAEA 1982, 1986).

[3] Gamma radiation is measured with crystal sensors which emit light flashes (scintillations) when they are hit by gamma particles. The light flashes are counted electronically in counts per second.

Fig. 2.8.
γ-log of an uranium exploration hole



2.2.3.3 Determination of Thickness

The thickness of the mineralized horizon normally is determined with the help of the called half-amplitude, where the measurements reach half of the value of the peak. It is more or less equivalent to the called half-width used otherwise in geophysics to interpret anomalies. For the log-curve in Fig. 2.8 the first peak occurs at 125.40 m. The log-value there is 1 760 cps (counts per second). Consequently the first half-value – half-amplitude – is 880 cps. At the lower end of the anomaly peak 2 occurs at 126.25 m. The log-value here is 2 440 cps. So the second half-value – the half-amplitude – is 1 220 cps. The half-value points should approximately coincide with the points of inflexion of the log-curve.

The two half-amplitude values are marked on the log-curve, and so the depth is determined. These are the lower and upper boundaries of the mineralisation which in the case of Fig. 2.8 occurs at 125.29 m and 126.4 m. So, in this case, the thickness is 1.1 m. We know from experience that the method works well when the thickness is at least 1.0 m. When the thickness is lower, corrections must be applied.

If the drill hole intersects the mineralization at right angle – for example, the drill hole is vertical and the mineralized zone horizontal – then the thickness obtained in this way is the true thickness M_w. If this is not the case, the thickness is the apparent thickness M_s which has to be multiplied by $\cos\beta$, whereby β is the dip angle of the mineralized horizon (see Fig. 2.9 and Sect. 2.1.1.1):

$$M_W = M_S \times \cos\beta$$

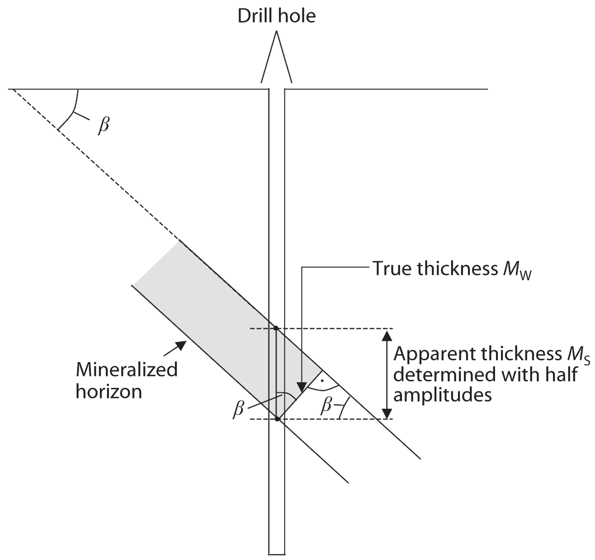


Fig. 2.9.
Calculation of the true thickness from the apparent thickness in an uranium exploration drill hole

2.2.3.4
Determination of Grade

The grade is determined with the help of the accumulation factor $G \times T$, the product out of grade and thickness. The area under an anomaly F_A is proportional to the accumulation factor $G \times T$. Basically there are three methods for determining the accumulation factor $G \times T$ which differ in the treatment of the anomaly area outside of the two half-amplitude points:

- the total area method
- the tail-factor method and
- tails cutoff method

To compare these three methods the area of the anomaly is divided into three parts:

- area 1 is the tail-end area above the half-amplitude point 1 in Fig. 2.8, i.e. squares N_1 and N_2
- area 2 is the central anomaly area between the two half-amplitude points 1 and 2
- area 3 is the tail-end area below the half-amplitude point 2, i.e. squares N_{14} and N_{15}

All three methods determine the central anomaly area 2 between the two half-amplitude width the same way, as will be shown below. With the total area method the three areas, the two tail-end areas and the central area, are treated the same way. This is the example illustrated below. With the tail-factor method the tail-end areas are taken into account by multiplying the sum of the two half-amplitude points by an empirical tail-factor which is proportional to the width considered. With the tails cutoff method, used often in practice, the two tail-end areas are not considered at all because their contribution to the grade of a mineralised horizon is only minor and is also influenced by values in the hanging and footwall of the horizon under consideration, causing "dilution".

The factor of proportionality for determining the accumulation value $G \times T$ is called the *K*-factor in the literature. Frequently a correction factor *F* has to be applied to the *K*-factor. The *K*-factor assumes ideal conditions. In actual practice it is often necessary to apply a correction factor to the *K*-factor to take into account the real diameter of the drill hole, the influence of the drilling mud etc. For details, the reader is referred to the above mentioned IAEA handbooks. For the sake of simplicity we assume that the correction factor *F* is 1 in our example. In addition, we assume that uranium and its daughter products are in equilibrium (see Sect. 2.2.3.1).

So we have the equation

$$G \times T = K \times F_A$$

The area of the anomaly F_A theoretically has to be determined by integration under the anomaly curve. In praxis, it is determined by considering single segments of the anomaly. In the example of Fig. 2.8, we choose 10 cm long segments. Rectangles are constructed, which have the same area as the log curve in this segment. In the example of Fig. 2.8 these are the rectangles N_1 to N_{15}. For these segments the measurement values are determined from the log and multiplied by the width of the segment, in this case 0.10 m, so that for each segment we have a value with the unit (cps m). The results are listed in Table 2.4. All values are added then. In our case the sum is $F_A = 2\,330$ cps m. Now the sum has to be multiplied with the *K*-factor, which determines the relationship between the U_3O_8 content and the count rate. In our case the *K*-factor shall be 1.5 ppm eU_3O_8/cps. For our example this results in

$$G \times T = K \times F_A$$

$$G \times T = 1.5 \times 2\,330 = 3\,495 \text{ ppm } eU_3O_8 \times \text{m}$$

This value has to be divided now by the thickness in the drill hole as determined in Sect. 2.2.3.3 above (It is the apparent thickness M_s as encountered in the hole). In our example the thickness was 1.1 m. So the average grade of the mineralized horizon using the total area method is

$$G = \frac{3\,495}{M_s} = \frac{3\,495}{1.10} = 3\,177 \approx 3\,180 \text{ ppm } eU_3O_8$$

In modern γ-log instruments this calculation procedure is "built in", so after determination of the half-width the instrument calculates the eU_3O_8 grade automatically. In addition, manufacturers of modern equipment provide manuals describing the conversion of γ-log readings to eU_3O_8.

If we would have applied the tails cutoff method, we would consider only the squares N_3 to N_{13} in Fig. 2.8 and Table 2.4. The sum of the areas in cps × m would be 2 226. Multiplied with the *K*-factor of 1.5 and divided by the thickness of 1.10 m we would get 3 035 ppm eU_3O_8, a difference of less than 5%.

Table 2.4.
Calculation of the anomaly area F_A

Area segment	Count rate/second	Depth interval	Area ((counts/s) m)
N_1	40	0.10	4
N_2	350	0.10	35
N_3	1 510	0.10	151
N_4	1 760	0.10	176
N_5	1 780	0.10	178
N_6	1 890	0.10	189
N_7	2 050	0.10	205
N_8	2 150	0.10	215
N_9	2 200	0.10	220
N_{10}	2 260	0.10	226
N_{11}	2 380	0.10	238
N_{12}	2 420	0.10	242
N_{13}	1 860	0.10	186
N_{14}	600	0.10	60
N_{15}	50	0.10	5
Sum = Total area			2 330

2.2.4 Grade Determination from Coverage Data Per Unit Area

For mineralization of large aerial extent and highly variable thickness, like the Deep Leads gold deposits in Australia, mentioned in Sect. 1.1.1 Fathom, and deposits like the nickel-, cobalt-, and copper-containing deep-sea manganese nodules for which thickness is insignificant, a coverage factor is given in kg metal per unit area. A coverage factor used also to be applied to the copper shale mines and uranium mines in the Erzgebirge in the former German Democratic Republic, the third largest uranium producer in the world in its time. There the term "spreading" was coined for such a grade intensity unit.

If it is necessary to calculate mining grades, the height of the necessary mining opening and the density of the extracted material have to be taken into account.

Example: In an area of the former copper shale mining district in eastern Germany the coverage (spreading) is 65 kg Cu/m^2; the density of the ore is 2.6 g/cm^3.

- *Case a:* The mining is planned to be conventional by drilling and blasting. The mining height will be 1.20 m. So, for 1 m^2 of the mineralisation the amount of run-of-mine ore will be

$$1 \times 1.20 \times 2.6 = 3.12 \text{ t} = 3\,120 \text{ kg}$$

with a coverage (spreading) of 65 kg Cu/m^2 the run-of-mine ore will have a grade of

$$\frac{65}{3120} = 0.021 \quad , \quad \text{i.e. } 2.1\% \text{ Cu}$$

- *Case b:* The mine management decides to use a specialized mining tool, a shearer, which allows the mining width to be reduced to 0.30 cm. Hence, for 1 m^2 of the mineralized area only 780 kg of run-of-mine ore will be produced:

$$1 \times 0.3 \times 2.6 = 0.78 \text{ t} = 780 \text{ kg}$$

Consequently, the grade expected is

$$\frac{65}{780} = 0.083 \quad , \quad \text{i.e. } 8.3\% \text{ Cu}$$