mining practice, the use of nomograms or specific software is common (Gy 1992, Pitard 1993, Berkman 2001).

Grade control

The practical execution of sampling depends on the conditions at the site and the precision to be attained (Gilfillan & Levy 2001). The lowest quality have samples that are picked at random from an outcrop or from an ore heap. Slits of a defined geometry (e.g. 10 cm square) are recommended if they can be cut to plan. Drillhole or reverse circulation blast hole dust and cuttings may suffer from differential settling of the (usually heavy) ore minerals and the (light) gangue minerals, or of problems with moisture. Diamond drillhole cores may have a diameter too small for representativity, core-splitting may introduce serious errors (Brooks 2008) and core loss is an ever-present risk. In spite of this, continuous sampling methods are generally preferable to discontinuous procedures. But no method is without problems and this has to be considered before starting a sampling campaign. If possible, different sampling techniques should be tested and compared before settling on a specific method. And a final advice – never underestimate the importance of accurately measuring the density of samples (Lipton 2001).

Subsampling

Subsampling designates procedures that reduce the total mass sampled (e.g. from a slit in an orebody exposure) to the few grams of powder in a small bottle that is all a modern laboratory requires for analysis. The key are stepped cycles of grain size diminution (by crushing and pulverizing), splitting (halving or quartering) and rejection of one half or three quarters of the sample. The choice of the correct splitting device is crucial, especially in cases where the ore is much heavier than the gangue (gold, uranium, etc.). Traditional riffle splitters work badly in these cases, because they are prone to cause gravity separation. Rotary splitters avoid this trap. Whatever the technique, error control is the central task in a sampling campaign. Always try to check on errors by submitting well-chosen and unrecognizable doubles of samples to the laboratory.

Geometallurgy

Geometallurgy describes procedures to model the distribution of different ore types in a mine (Petruk 2000). We have seen earlier, that mine-site processing of run-of-mine ore is an important determinant of economic success or failure of an operation. During a feasibility study, not only the variability of metal contents in ore, but also of metallurgical recovery over the life-time of the future mine, must be studied and modelled. Automated methods of mineralogical analysis, such as QEMSCAN (an electron beam technique that combines a scanning electron microscope, four X-ray detectors and a software package developed by CSIRO, Australia), provide the required data. *Geometallurgical (ore) domains* comprise parts of the orebody with similar geochemical, mineralogical, textural and processing properties. Predicting grindability and metallurgical performance is critical. The development of small-scale tests based on diamond drill cores allows multiple sampling of an orebody opposed to the traditional pilot-plant scale sample of several 100 tonnes taken from one accessible location of yet undeveloped reserves.

5.3.2 Ore reserve estimation and determination of grade

From the scale of a single mine or a mining company to national and supranational stock markets and resource planning, knowledge about the physical and economic availability of raw materials is needed for rational decisions. Therefore, international efforts to standardize measuring and reporting of mineral reserves and resources have reached a wide acceptance.

The total geological quantity of a specific raw material is termed the "mineral resource base". This includes both known and unknown quantities. However, only ore that has reasonable prospects for eventual economic extraction may be included in any estimate. Based on increasingly reliable data on quantity, quality and exploitability of minerals, McKelvey (1973) proposed a matrix ("McKelvey Box"), which differentiates between prognostic (undiscovered) resources, identified resources and reserves. Reserves are only that part of resources that is thoroughly investigated and proved to be exploitable with a high degree of confidence.

Prognostic resources

Strategic planning for the future availability of metals and minerals relies on estimates of prognostic resources. Often, dire predictions are made about the world running out of specific metals (Club of Rome: Meadows *et al*. 1974), coal or crude oil. In 1974, the reason for the prediction of depletion in the 1990s was the erroneous use of reserve instead of prognostic resource figures. Even today, this mistake is commonplace. Because of obvious reasons, quantification of undiscovered resources is notoriously difficult. Most methods proposed are based on extrapolation of identified resources (including reserves and past production) in well investigated tracts to geologically similar, less well-known parts of the globe (Singer & Menzie 2010, Gautier *et al*. 2009, Singer *et al*. 2005) or to greater depth beneath the surface (Kesler & Wilkinson 2008). Such exercises demand geological and metallogenetic maps, and mining databases.

In estimating undiscovered resources, statistics have an elementary role. Studying the negative correlation between tonnage and grade of metal ores, Lasky (1950) proposed a log-normal distribution (Lasky's Law). In a log-log diagram, the same data pairs form a line with a slope determined by an exponent D between 0 and 3 reflecting a fractal nature (equation 5.1).

Fractal distribution of ore tonnages and related grades (Turcotte 1997):

$$C_{ore}/C_{min} = (M_{min}/M_{ore})^{D/3} \qquad (5.1)$$

C_{ore} is the average grade of the tonnage M_{ore}, C_{min} the minimal grade included of the mass M_{min}, D is the fractal dimension. M_{min} may be the mass of ore exploited at the lowest-grade mine, or even source rock from which ore in a district is thought to be derived (consider the metamorphogenic model of gold deposits leached from crust with trace contents).

Based on this correlation, undiscovered resources can be estimated. A different mathematical approach (density functions) was applied by Gerst (2008), aiming at an estimate of cumulative grade-tonnage curves for continental copper resources.

Discovered mineral resources

Discovered mineral resources result from exploration and detailed follow-up work. According to increasing geological certainty, they are subdivided into *inferred, indicated and measured resources* (Figure 5.12). The last category infers that mass and grade are known with a high level of confidence. Methods for measuring confidence are typically based on drillhole or sample spacing and geostatistical criteria (Abzalov & Bower 2009). Resources are normally not acceptable as a base for commercial mining. There are, however, some projects, such as *in-situ* leaching (ISL) of uranium, which must rely on drilling only, so that mere resources and no reserves at all can be estimated until production starts.

Mineral, or ore reserves

Mineral, or ore reserves are only that part of an indicated or measured resource that can be economically mined. Only reserves justify commercial mining. Investigations supporting this attribution must include mining, metallurgical, economic, marketing, legal, environmental, social and political factors (the "modifying factors"). The modifying factors are time-bound variables. Mineral reserves are subdivided into *probable* and *proved,* the first with a lower degree of confidence.

Statistical and geostatistical methods are indispensable in the determination of different levels of confidence. The principle can be illustrated by varying density of geological observations (Figure 5.13). In a mine exploiting a simple planar orebody (e.g. a fluorspar vein, or a steeply dipping sedex ore deposit), proved reserves must be physically outlined on three or four sides. Probable reserves are those parts that are only exposed along two mine openings (or closely spaced drillholes).

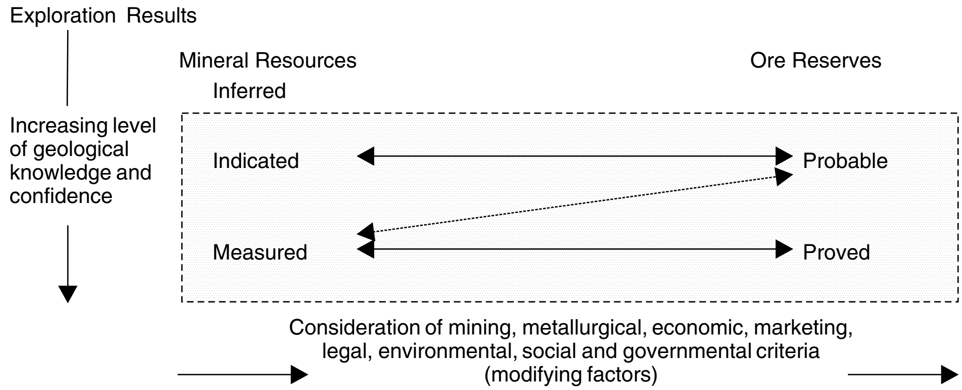

Figure 5.12 General relations between exploration results, mineral resources and mineral reserves as defined in the Jorc Code (AusIMM 2004). It is noteworthy that proved reserves can fall back to the measured resources category.

Measured resources are usually the result of detailed drilling but of insufficient quality or incomplete data for classification as reserves. Indicated mineral resources are derived from the geological orebody model and supporting physical observations.

Investigations for resource estimation include work in six stages:

1 provide a sufficient quantity of data of appropriate quality (precise and accurate);
2 develop a well-founded geological deposit model (including data that link mineralogy and metal recovery);
3 use statistical methods in order to understand the distribution of analytical data in the deposit;
4 choose a suitable interpolation model for grades, considering both the geological model and the statistics;
5 calculate tonnage and grade, either globally or for parts of the deposit;
6 prepare the report, clearly outlining resource categories and respective confidence of figures presented.

Each of these points represents a complex system of scientific and technical approaches, which influence the results. Not all uncertainties can

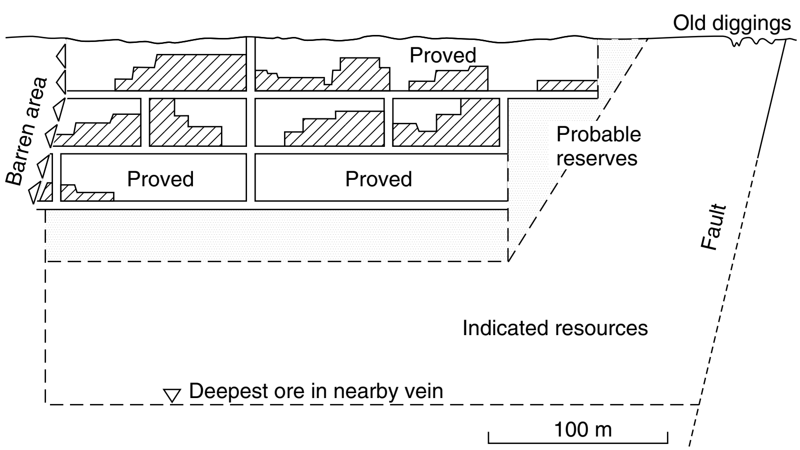

Figure 5.13 Proved and probable reserves of a vein deposit in relation to geometry of actual underground exposures, and indicated resources supported by the geological model and some drill hole intersections (not shown). Shaded areas are extracted parts of the orebody.

be resolved and some assumptions must be made. This is why the product is termed a "resource estimate" (not a calculation), even if advanced mathematical methods are used. It must always be remembered that users (e.g. banks) of the resulting figures will take them at face value and derive potentially serious financial and organizational decisions. Note also, that resource estimates are essentially numerical models of orebodies. Similar to other such models in the earth sciences, they can hardly ever be true replicas of nature (Oreskes *et al*. 1994).

Because of the economic impact, for example on financial markets and industry, all major industrial countries developed norms for estimating and classifying reserves. In Europe, a common standard is prepared by the Pan-European Reserves and Resources Committee (PERC 2008). A widely lauded and internationally adopted example is the Australian system (Joint Ore Reserves Committee or JORC Code: Edwards 2001, AusIMM 2004). The United Nations Economic and Social Council (1997) presented a proposal that was hoped to meet the requirements of private and state-controlled mining industries, as well as governmental needs for mineral inventory classification. In this system (UN International Framework, Figure 5.14), three axes are presented, including: i) the degree of geological assessment (geological axis); ii) the degree of economic viability (economic axis); and iii) the stage of feasibility assessment (feasibility axis). The latter is a new aspect specifically aimed at potential investors. Another proposal for a third axis was put forward by BGS (British Geological Survey), concerning information on the accessibility of deposits considering environmental, legal, social and political factors (Cook & Harris 1998). The intention is to visualize which part of reserves is really available for mining, in contrast to those that are blocked by other claims. In the future, mining will increasingly depend on modifying factors. Purely geological reserves unavailable for exploitation are meaningless (Weatherstone 2005).

Practical procedures of **calculating quantity and grade** of a mineral deposit are modified according to type, form, raw material contained and mode of data collection (Annels 1991). In simple cases,

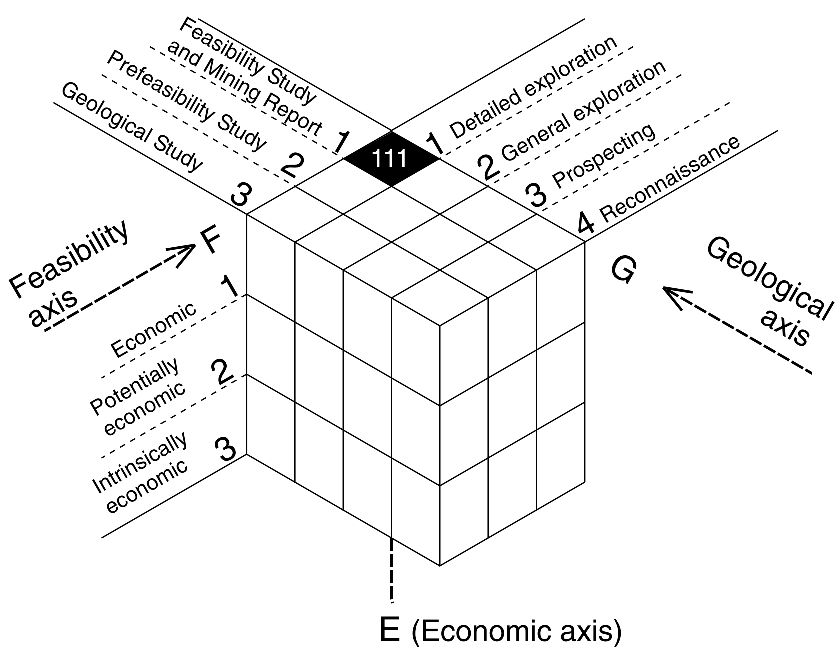

Figure 5.14 The tri-axial UN Framework Classification of mineral reserves and resources. Courtesy UN Economic and Social Council (1997). Numbered codes apply to each block. Block 111 represents proved reserves, 121 and 122 probable reserves (coordinates E-F-G). All other blocks are various resource classes.

such as homogeneous planar bodies (e.g. a seam of chromite or of coal), a simple volume-by-density formula can be applied (eq. 5.2).

Simple calculation of in situ *tonnage*:

$$Q = F \cdot D \cdot M \tag{5.2}$$

If the recoverable metal mass in a specified volume of ore is required, the formula is enlarged by the introduction of factors that take account of ore left in place (mining loss, for example in safety pillars) and the loss during metallurgical processing (eq. 5.3).

Simple calculation of recoverable metal content:

$$P = Q \cdot G \cdot A \cdot V \tag{5.3}$$

Q = ore *in situ* (t), F = surface to be mined (m^2), M = average thickness of ore (m), D = specific weight or bulk density of ore (t/m^3), P = recoverable metal content (t), G = average metal content in ore (% or kg/t), A = correction factor for mining loss, V = correction factor for processing loss.

Note that the specific weight of ore (D) must be determined using large samples, because in this case the aim is to quantify a physical property of the rock mass including joints and fissures (Bieniawski 1989, Lipton 2001), in contrast to ore rock (a specimen, for example). If appropriate, dilution of the ore by unavoidable extraction of host rock must be included. Dilution is, of course, a factor that increases costs. Polymetallic ore grade is often reported in terms of a single equivalent grade of one major metal such as gold or copper. It is usually obtained by taking the *in-situ* value (grade multiplied by price) of each of the individual metals, adding these values and calculating the grade of the same value of the primary reported metal. The result can be very misleading if the recovery of individual metals is not considered. Therefore, the preferred measure of equivalent grade is the net smelter return (NSR).

Usually due to practical considerations, parts of a deposit (single orebodies or blocks of ore) are separately submitted to ore reserve estimation. Some orebodies with a longitudinal continuity but varying contours are depicted by serial sections (wire frames) that may be based on drilling fans, such as those shown in Figure 5.11. In such cases, exploitable ore surface in cross-sections and distance between sections are measured (Figure 5.15). The ore volume can be estimated by applying eq. 5.4.

Simple estimation of ore volume from serial sections:

$$\begin{aligned}\text{Volume}\,(\text{m}^3) = {} & 0.5 \cdot (\text{F}_1 + \text{F}_2) \cdot \text{b}_{1-2} \\ & + 0.5 \cdot (\text{F}_2 + \text{F}_3) \cdot \text{b}_{2-3} + \ldots \text{etc.}\end{aligned} \tag{5.4}$$

The role of weighting

Variability of some factors of special importance, such as varying thickness of a gold vein, sample length along core, or density of ore impose the need for weighting (Wellmer *et al*. 2007). Average grade of an ore vein with varying contents of lead and, therefore, variable density can only be correctly calculated by weighting (eq. 5.5).

Weighted average content (G_w) as a function of thickness (M) and density (D):

$$G_w = \frac{M_1 \cdot D_1 \cdot G_1 + M_2 \cdot D_2 \cdot G_2 + \ldots \text{etc.}}{M_1 \cdot D_1 + M_2 \cdot D_2 + \ldots \text{etc.}} \tag{5.5}$$

The determination of the lowest grade in an orebody that can be economically mined (the "cut-off grade") is of paramount importance (Rendu 2008, Lane 1997). By definition, this is the grade where mining and processing costs are equal to proceeds from the sale of the product. The cut-off grade is the limit between ore and waste rock. Its determination is not simple, as the parameters that determine the optimal cut-off grade are not only the given geological properties of the deposit but include time-dependent factors such as varying metal prices, shallow or deep location of stopes and the cash-flow strategy of the operation. Therefore, different cut-off grades will apply during the life-cycle of a mine and at one point of time, in different parts of the mine. When setting cut-off grades, the aim will always be to maximize the profit of a mine. In recent years, maximization of the net present value (NPV) of the mining operation is the main measure of optimizing the cut-off grade (Nieto & Bascetin 2006).

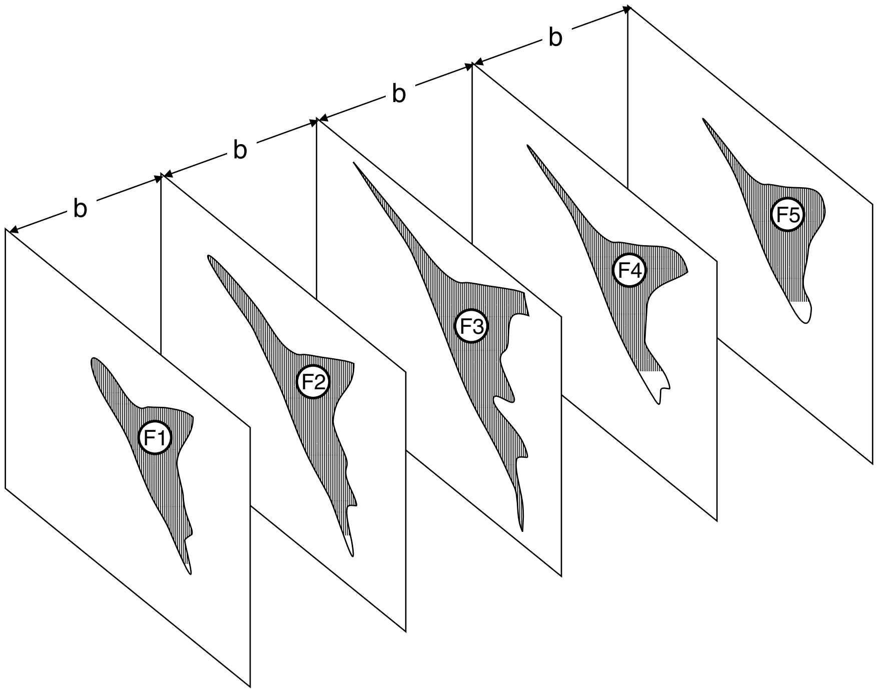

Figure 5.15 Serial profiles of an orebody (wire frames) are one of several methods to estimate the volume of ore. This drawing is inspired by the Deilmann orebody at Key Lake uranium mine, Canada.

Geostatistic modelling

Geostatistical modelling is an important part of ore reserve estimation. Geostatistics were first developed by Matheron (1971) and Krige (1981), originally aiming at a higher accuracy of gold reserve estimates. The difference between geostatistics and ordinary statistics is that geological parameters depend on the site of a measurement (e.g. the laterally changing thickness of a coal seam), whereas each throw of a dice is independent from the previous one. Commonly in geology, smaller distances between sample locations cause a higher correlation of measured values. This can be used for various predictive purposes, especially in reserve estimation. The most important step in geostatistical modelling is therefore the quantification of the spatial correlation of sample properties ("regionalized variables"):

> Semivariograms are computed in order to quantify the spatial correlation and directional properties of various parameters, such as ore grade, mineral paragenesis, thickness, etc. (Figure 5.16). Variography (or "structural modelling") is an important tool that reinforces geological understanding of a deposit (Guibal 2001). Typical uses of variograms are: i) optimizing the sampling density (e.g. the drilling grid distances); and ii) helping to define the geological model ("domaining") for resource evaluation. Domains are then subdivided into blocks for calculation of tonnage and grade of ore contained. Common computational methods employed include *kriging* (i.e. minimizing the error of estimation; Matheron 1971, Isaaks & Srivastava 1990) and *conditional simulation*, allowing extremely complex models and providing a measure of precision and probability (Abzalov & Bower 2009, Khosrowshahi & Shaw 2001).

It is very important to remember that geostatistical methods cannot replace meticulous geological data acquisition and interpretation. They are computational tools that rely on good geology and extend its reach. Erroneous applications include calculating a variogram with data that comprise

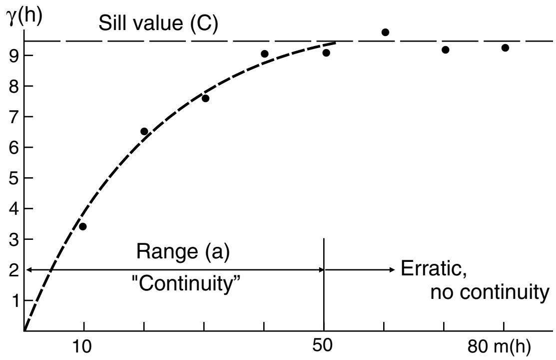

Figure 5.16 Example of a simple experimental semivariogram with a fitted model (black line), that may be used to determine the optimal sampling distance in an orebody. γ (h) is the variance of values at different sample distances (lag-distance [h] here in 10 m steps). When reaching the sill at range (a), γ (h) equals the variance of the total population (C) and the predictive value of γ (h) – the covariance – is lost. Optimal sample distance would be a/2.

distinct domains. Practical experience with geostatistical resource evaluation has shown that results (e.g. of kriging) should always be checked with different methods, including conventional "manual" ones (Sinclair & Blackwell 2002). In Australia, comparative runs with commercial software resulted in severe differences, provoking the demand that all resource announcements should be accompanied by a manual calculation (Swain 1997). Much can be learned from a comparison of reserve estimates and quantity produced. For the uranium mine Key Lake (1982–1997, cf Chapter 2.5 "Uranium"), the original reserve estimate systematically deviated from later production figures. The estimated tonnage had been too low and the predicted grades were hardly ever reached. The explanation for this disparity was an unplanned dilution of the ore by ~25% of host rock (Mistry *et al.* 1999).

Reserve management and reconciliation

The lesson of Key Lake and many similar cases is that reserve management and reconciliation is an important task. Predicted grade of reserves, *in-situ* grade of ore produced, grade delivered to the processing plant and mass of metal in concentrate must be carefully supervised (Fouet *et al.* 2009, Gilfillan & Levy 2001). The results are a measure of the overall metal recovery of the operation, but also of problems at different points in the mining process. For some time now, reconciliation results ("factors") are required information by international reserves and resources reporting codes such as JORC.

Many placer deposits of gold and diamond, but also some lode gold deposits, show a high nugget effect. A *nugget* is a large lump of gold lying about that will certainly please the finder. Of course, one lucky find would hardly be a rational cause for opening a mine. If gold in a mine is said to have a nugget-like distribution, an important part of the total content in the ore is present in erratic pockets of high concentration. This causes severe problems in ore reserve estimation. In such cases, channel sampling of the four walls of a pit may yield four different results, which in turn show little relation to the bulk sample from the pit. Common geostatistical methods cannot solve this problem, because in this case the area of influence of single samples is negligible. At the Bendigo gold mine (New South Wales, Australia), most of the gold content in each tonne of ore is present in just 5–15 very coarse gold particles (Johansen 2005). Investigations of large bulk samples of placers at Bendigo have shown that the grain size distribution is relatively stable at different total grades and can be presented in a type curve (Figure 5.17). Estimation of the grade of any sample from the mine appears to be possible by determination of gold in the lower part of the grain size curve. The method is reported to allow reliable grade estimates based on small samples like drill cores.

Reserves of a mineral deposit are a first-order control on economic evaluation, whether decisions concern the opening of a new mine, the value of company shares, the sale of a deposit, or securing a credit. Therefore, the execution of reserve estimation carries considerable responsibility. This translates into the condition that experts involved must have a demonstrated high professional standard ("competent person"), independence and ethical integrity. Australian rules are an excellent example (AusIMM 2004, 2005; Edwards 2001).

5.3.3 Valuation of mineral deposits

The term "mineral resource wealth" tempts the non-professional to consider undiscovered or undeveloped minerals in the ground as an economic value. This is not strictly correct because an income from buried minerals can only accrue when they are extracted. Only mining creates wealth for investors, miners, contractors and the whole economic space. It is true, however, that minerals in the ground are an *economic potential* and therefore can be the object of trading and speculation.

Investing in a mine is basically providing funds in the expectation of being repaid in the future and, of course, with interest. This translates in economic terms to "the *present value or worth* (P) of a sum (A) to be received or paid at some future date is such an amount as will, with compound interest at a prescribed rate (i), equal the sum to be received or paid in the future (n years)". Accordingly, the present value is less than the future income, because that must be discounted (eq. 5.6).

Present value of future income calculated by discounting:

$$P = A/(1+i)^n \tag{5.6}$$

In periods of high interest rates, the present value of future income falls rapidly to near zero. With an interest rate of 15% per year, an income (A) in 15 years from now has a present value of only 0.123 A and 0.001 A for 50 years. At common interest rates of 7.5% and a period of 15 years, the result is a present value of 0.338 A. This illustrates why the development of ore reserves (a future income) beyond a period of 15–20 years is

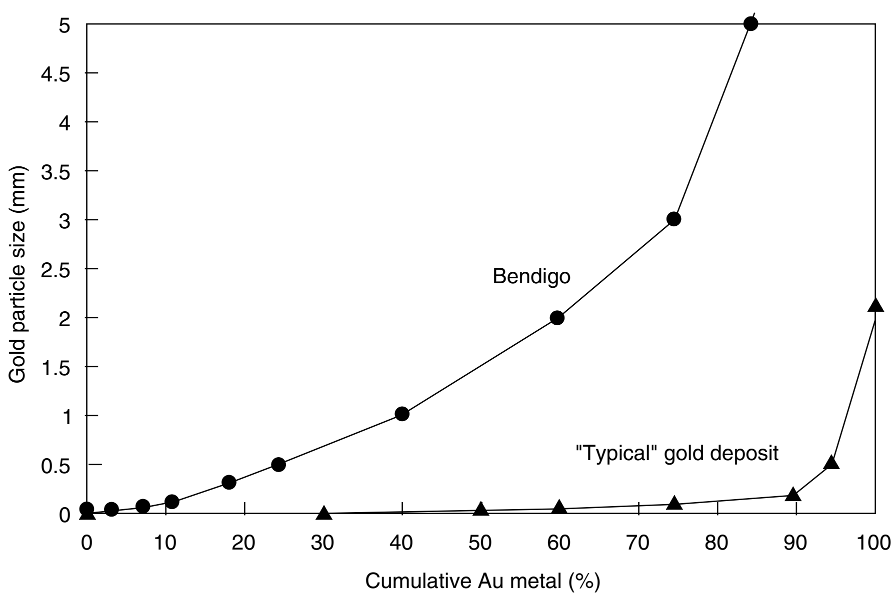

Figure 5.17 Gold particle size distribution at Bendigo, Australia compared to a "typical" gold deposit (Johansen 2005). At Bendigo, coarser (and fewer) grains control the total grade. Determinations of the characteristic grain size/metal content relations in a deposit allow improved grade estimations in spite of high nugget effects.