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How to Set Up and Develop a Geometallurgical Program

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How to Set Up and Develop a Geometallurgical Program

Abstract

Geometallurgy is a comparatively new discipline in mining that involves the fundamental integration of multiple conventional technical areas of the industry, where the outcome is a genuinely new methodology. It can be a daunting prospect to be tasked with designing and justifying a comprehensive geometallurgical program. The task could be starting a program from scratch or collating existing information to re-analyze as a pre-cursor to a program extension.

Commonly it is seen as purely the relationship between metallurgical testing and process mineralogy, although this is not incorrect, it is only part of the whole program. The discipline covers many areas across the Mine-Value-Chain. Geometallurgy is an appreciation of the prediction of rock behavior, being subject to an engineering outcome; this is in context of the engineering processes.

This paper discusses setting of strategic geometallurgy goals, how they could be used and maps out a step by step procedure experimental design. Different methods for characterizing ore at different scales is discussed. The different kinds of ore test characterization methods are discussed in context of conventional tests and the smaller scale proxy comparative tests, for comminution, flotation and leaching.

Various methods of data analysis to bring all these different technical disciplines together are also discussed. There are many valid definitions and approaches to this methodology. This paper represents just one approach to this subject.

Keywords

Geometallurgy, comminution, flotation, hydrometallurgy, mining, SEM automated mineralogy, data analysis, campaign, core imaging, ore characterization, ore body knowledge, batch flotation, impact breakage, Bond Ball Work index, leaching, comparative test, experimental design.

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" It's a long way to the top if you want to rock 'n' roll"

Bon Scott   ACDC 1975

Contents

1 INTRODUCTION

Geometallurgy is the characterization of a family of ore types in context of process performance. As such, it requires the integration of geological, mining, metallurgical, environmental, and economic parameters. It also requires the genuine integration and evolution of the paradigm of each of these technical areas into a new holistic paradigm. The ultimate goal is to develop a more sophisticated operational design, streamline and optimize the mining cycle and maximize the economic value of an ore body, while minimizing technical and operational risk

Geometallurgy is an important addition to any mining operation, large or small as it aims to improve the Net Present Value (NPV) of an ore body (Dominy & O'Connor 2016). It is known to increase site stakeholder communications and collaborations (Jones & Morgan 2016 and Suazo *et al* 2010). Creating the right kind of environment for knowledge sharing whilst concentrating on improved data acquisition and interrogation, will result in confident integration of data into the Mine-Value-Chain. All these aspects create better business optimization, better utilization of staff and better targeted key performance indicators (Chibaya 2013 and O'Connor 2016).

The benefits of a geometallurgical program is to 'de-risk' or allow for better risk management processes, either technical or operational (Chibaya 2013 and Vann *et al* 2002). The risk being the lack of knowledge of your raw materials (ore) and the expected performance and recoveries of those materials. This risk could translate into an overrun of the Capital Expenditure (CAPEX), or the money required to build and commission a mining operation to the point where it can start producing a saleable metal concentrate. Table 1 below shows an example of several recent CAPEX overruns (Noort and Adams 2006). These CAPEX overruns are a consequence of not understanding what was genuinely required to economically feasible produce revenue from these deposits.

Table 1. Recent major mining project CAPEX overruns (Source: Noort and Adams 2006)

Project	Company	Feasibility budget cost	Actual/forecast cost overrun
Ravensthorpe/Yabilu Expansion	BHP Billiton	A$1.4 billion	30%
Spence (Chile)	BHP Billiton	US$990 million	10%
Telfer Mine	Newcrest	A$1.19 billion	17.50%
Stanwell Magnesium	AMC	A$1.3 billion	30%
Boddington	Newmont	A$866 million	100%
Goro Project (Indonesia)	Inco	US$1.45 billion	15%
Prominent Hill	Oxiana	A$350 million	51%

CAPEX blowouts are just one risk that has the potential to destroy a mining operation. Figure 1 shows the rate of achievement of design capacity of plants at startup over time (McNulty 1998 & 2014). The curves shown are classified as Types 1, 2, 3, and 4. They show four types of operations. Type 1 is an operation that achieves the designed for production throughput easily within 12 months of commissioning. This is the ideal objective of all mining operations. Type 4 is an operation that never reaches the designed production throughput and takes a long time to reach its much lower maximum output. This is the worst case scenario for mining operations, as produced revenue is much lower than planned for and takes much longer to consolidate.

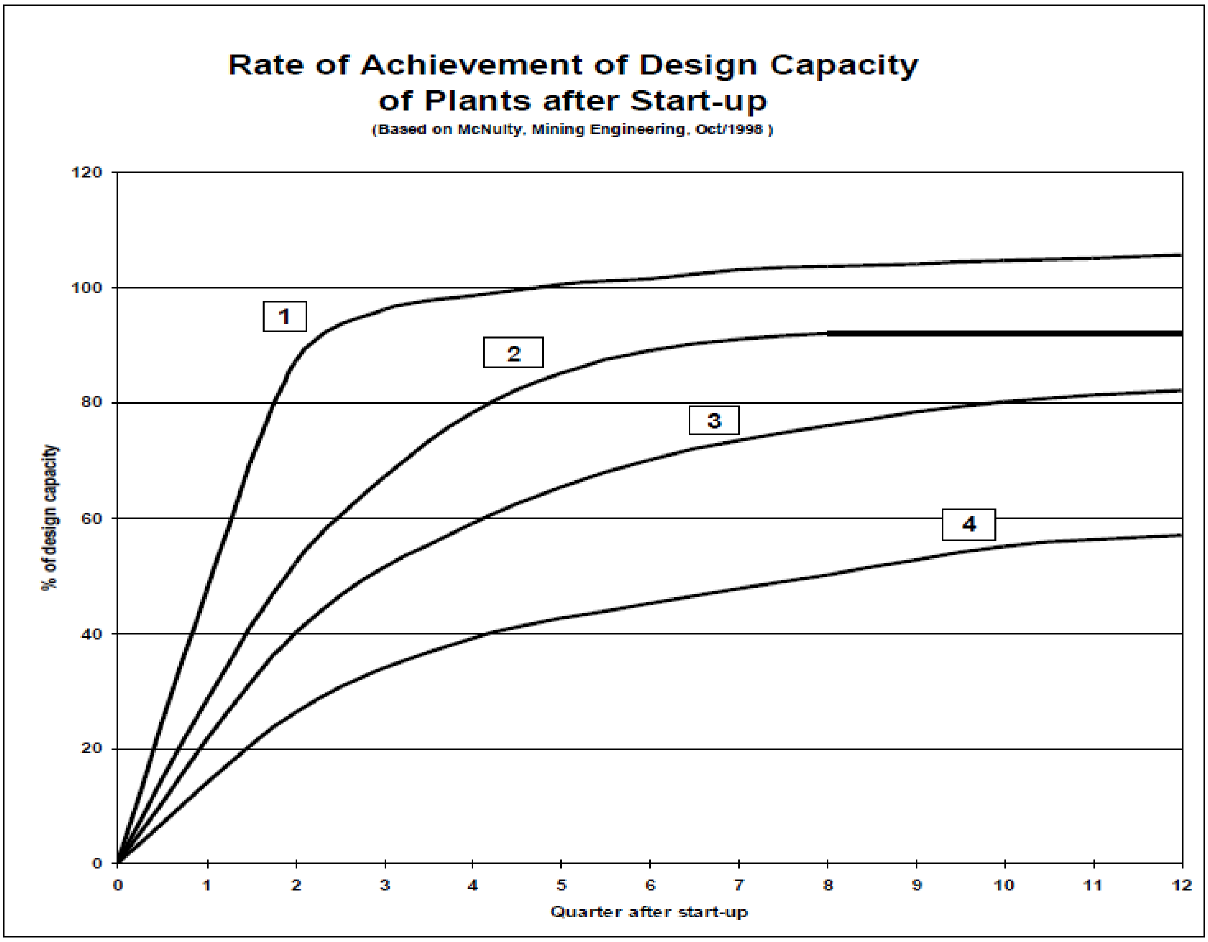


Figure 1 Start-up curve classification showing % of design capacity vs. time
(Source: *Plant Ramp-Up Profiles an Update with Emphasis on Process Development*, by T. McNulty, from Proceedings of the 2014 Conference of Metallurgists. Reprinted with permission of the Canadian Institute of Mining, Metallurgy and Petroleum).

Figure 1 above is a very real problem for capital investment due to the use of the Net Present Value tool (NPV), where possible future produced revenue is discounted as a financial risk. The longer an operation takes to pay its CAPEX loan off, the less valuable it is to investors. This is what distresses venture capitalists the most.

This however, is a common problem. A study done by the Chemical Bank (now part of JP Morgan Bank) drawn from a project database of 18 mining projects, examined the performance of these projects (Castel 1985). Questions asked and examined were:

- Was the mine completed on time?
- Was the mine completed at target budget?
- Did the mine eventually produce at design capacity?
- Was the mine producing expected cash flows?

The conclusions were as follows:

- 12 of 18 had construction delays
- 5 more than 2 years late
- 4 never satisfied their 'option' test
- 12 of 18 had CAPEX overruns
- 10 mines had greater than 20% CAPEX overruns
- 14 of 18 had difficulty operating at capacity
- 6 persisted for more than 3 years
- 12 of 18 yielded cash flow below original estimate
- 7 of 18 mines never achieved budgeted operating levels
- 7 of 18 had negative cash flows
- 9 of 18 were subject to lower commodity prices

In a survey of 258 Public-Private Projects (Flyvbjerg 2002), 86% had capital overruns, with a CAPEX average overrun of 28%. Gypton 2002 surveyed 60 mining projects over 21 years:

- Capex average overrun 22%
- No correlation between who complied original estimate
- Problems with in-house and "blue-chip", highly-reputed consultants
- 1st tier companies were no better than 3rd tier companies
- No influence from project size and location
- Bank consultants (due diligent audits) failed to identify "red flags" or CAPEX overruns
- Feasibility Study , 59% exceeded ±15% estimation accuracy (post reconciliation)
    - Of which 75% over estimate
    - Of which 25% under estimate

Another serious issue that can cause difficulty is highly variable plant performance in operation. Most process plants are designed around an average performance (throughput and recovery), based on an ore average that usually does not actually exist. This almost always translates into poor economic performance. This is a consequence of poor knowledge of process response of the ore and when the variable ores will enter the plant. Figure 2 shows an example of a challenging process plant performance.

The issue with what is shown in Figure 2 is that this mine site and process plant was designed to operate at one consistent set average, run at a steady state. That this does not happen has a direct impact on the true metal production in comparison to the planned for and predicted metal production.

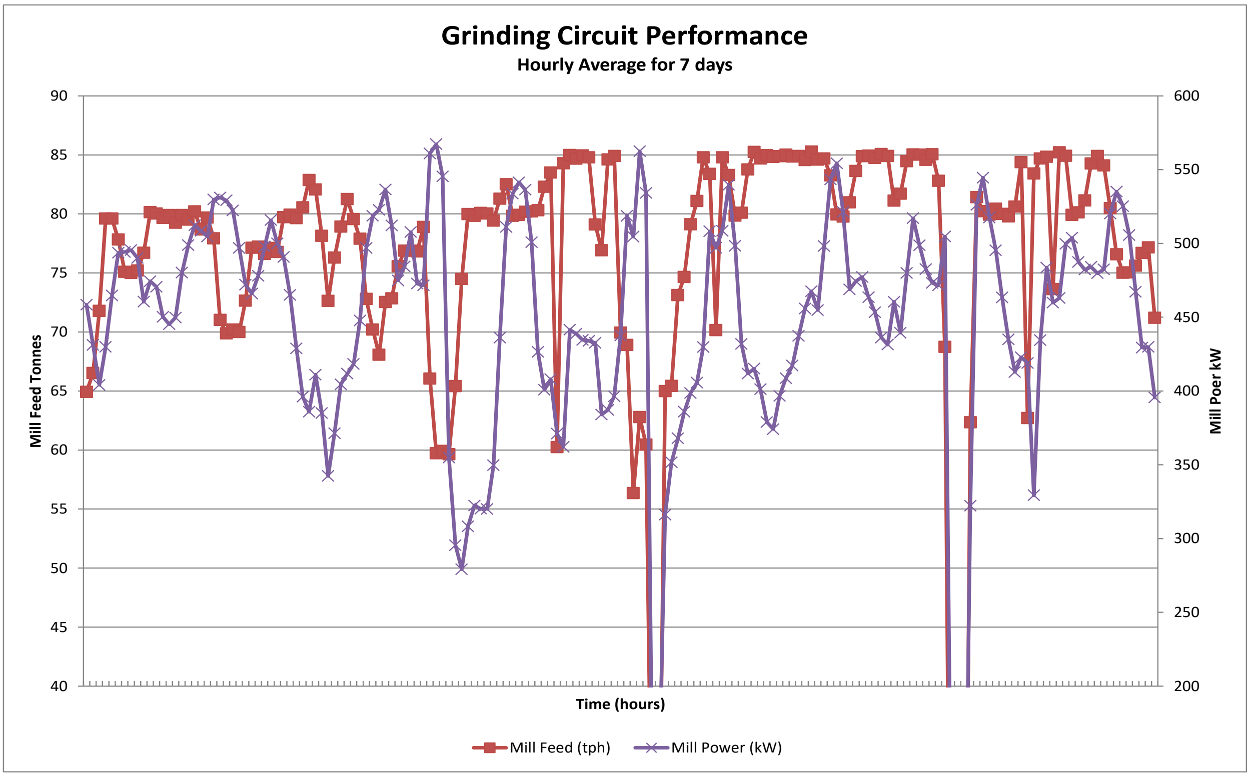


Figure 2. Mill performance at a mine somewhere in the Southern Hemisphere
(Image Copyright: Dee Bradshaw)

The examples above show challenging outcomes that have the capacity to make a mining operation fail. In the past few decades, mining operations have become larger, with a corresponding larger CAPEX required. This means that the financial risk is much higher. The next generation of mining operations is predicted to be bigger again.

Alternatively, Figure 3 below shows the Batu Hijau case study, where the long term plant performance was predicted to within 2%, and years ahead of time (F. Wirfiyata & K. McCaffery 2011 & Burger et al 2006):

- The revised function was applied to daily ore delivery data back to January 2006 and compared against the Metso PTI model output over the same period as shown in Figure 12.
- The average difference between actual and predicted was <0.4% for the WiBM corrected SMCC model and 5% for the Metso PTI model over the 3 year period.
- A sizable difference still periodically existed between Actual and the new SMCC model prediction as is particularly evident from June to about September 2008.
- On field investigation, the difference was found to be a result of a physical circuit equipment bottleneck, causing under-performance of actual mill throughput rather than over prediction by the model. This is further discussed in the Circuit Optimisation section of the paper (F. Wirfiyata & K. McCaffery 2011).
- Monitoring of the new model with the Ball Mill Work Index correction continued and in late 2008 it was decided to apply this model as the throughput predictor for Business Planning forecast purposes for 2009 onwards. The choice to proceed down this route was mainly because of the ease of application of this model within the geological block model

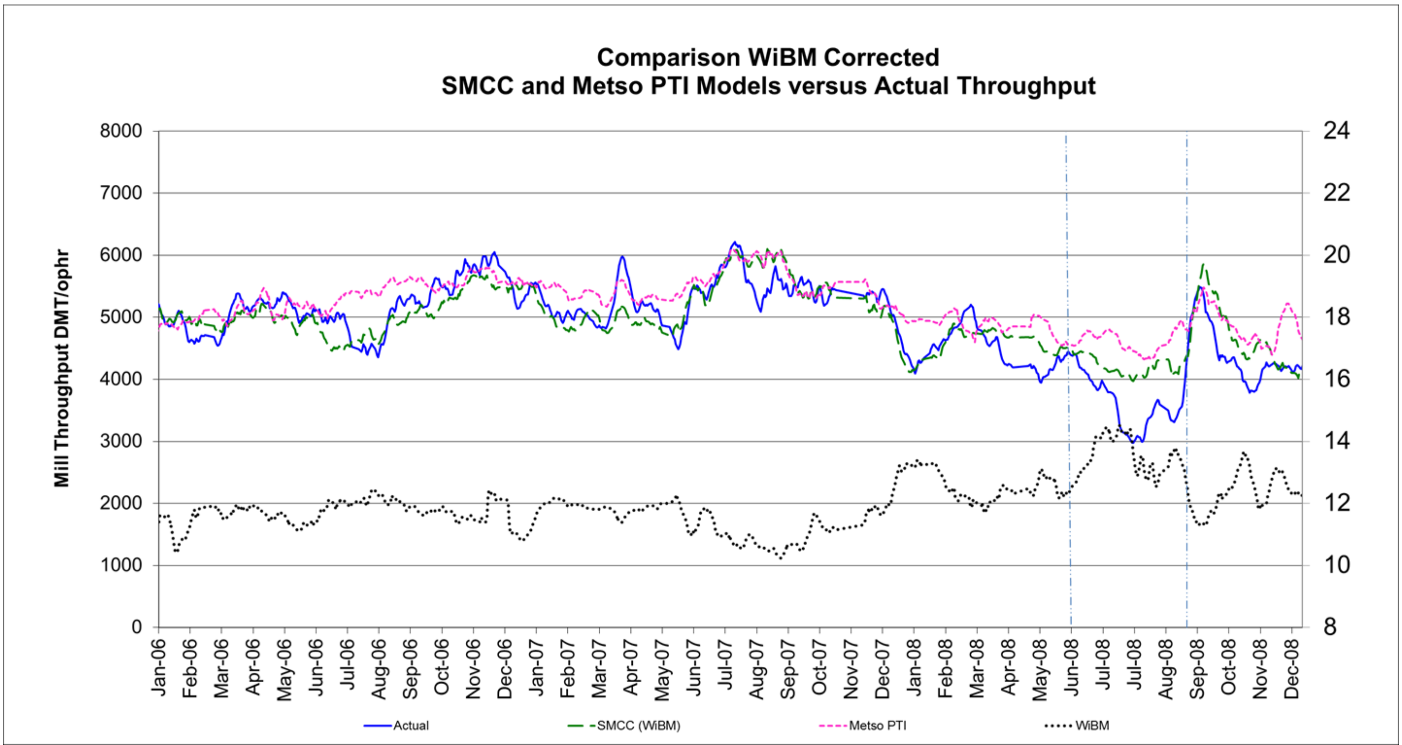


Figure 3. From "Applied Geo-Metallurgical Characterization for Life of Mine Throughput Prediction at Batu Hijau," by F. Wirfiyata & K. McCaffery. Reprinted with permission of the Canadian Institute of Mining, Metallurgy and Petroleum (Source: Wirfiyata & McCaffery 2011)

F. Wirfiyata & K. McCaffery (2011) demonstrated that Batu Hijau had systematically improved long and short term mill throughput predictions since feasibility and commissioning in 1999. This had been achieved through a combination of ongoing improvement to the geology ore hardness and grade characterization model, drill and blast optimization and, by application of these data to model mill throughput. Batu Hijau had demonstrated the ability to consistently estimate long term mill throughput within ±2% accuracy. Aside from reliably predicting expected production rates with the existing plant configuration, this predictive ability provided a solid foundation for identifying operating and design criteria for short term grinding circuit optimization and future expansions.

The difference between Batu Hijau and the examples shown in Table 1 and Curve Types 3 and 4 in Figure 1 could be a result of a successful geometallurgical campaign (a more sophisticated feasibility study cycle based on process orientated ore characterization).

An evolution of the development of mining operations is required to prevent the cost overruns in CAPEX, and the delays in production performance. There are several possible ways to do this. Geometallurgy applied to the feasibility study cycle is a practical way to do this. Part of the overall risk management in a geometallurgical program is to include statistics and modelling, specifically of aspects of data which are not normally targeted for modelling per se (Suazo *et al* 2010 and Senior *et al* 2006). A systematic and stepwise approach to running a geometallurgical program is essential. In many cases, companies have vast amounts of stored data that few have seen, or no one has utilized, normally because they are not sure how or what to do. Depending on the project stage a geometallurgy programs requires value proposition creation, gathering funds, to justify the bigger picture of business improvement with the use of those funds.

2 METHODOLOGY AND RESULTS

2.1 Campaign planning of objective goals

The question of what fundamental purpose does the geometallurgical campaign serve has to be understood. Where in the mining life cycle is the operation in question (Figure 4)? What part of the mining extraction process path is the problems that the campaign is to examine? How they interrelate with other parts of the process (Figure 5)?

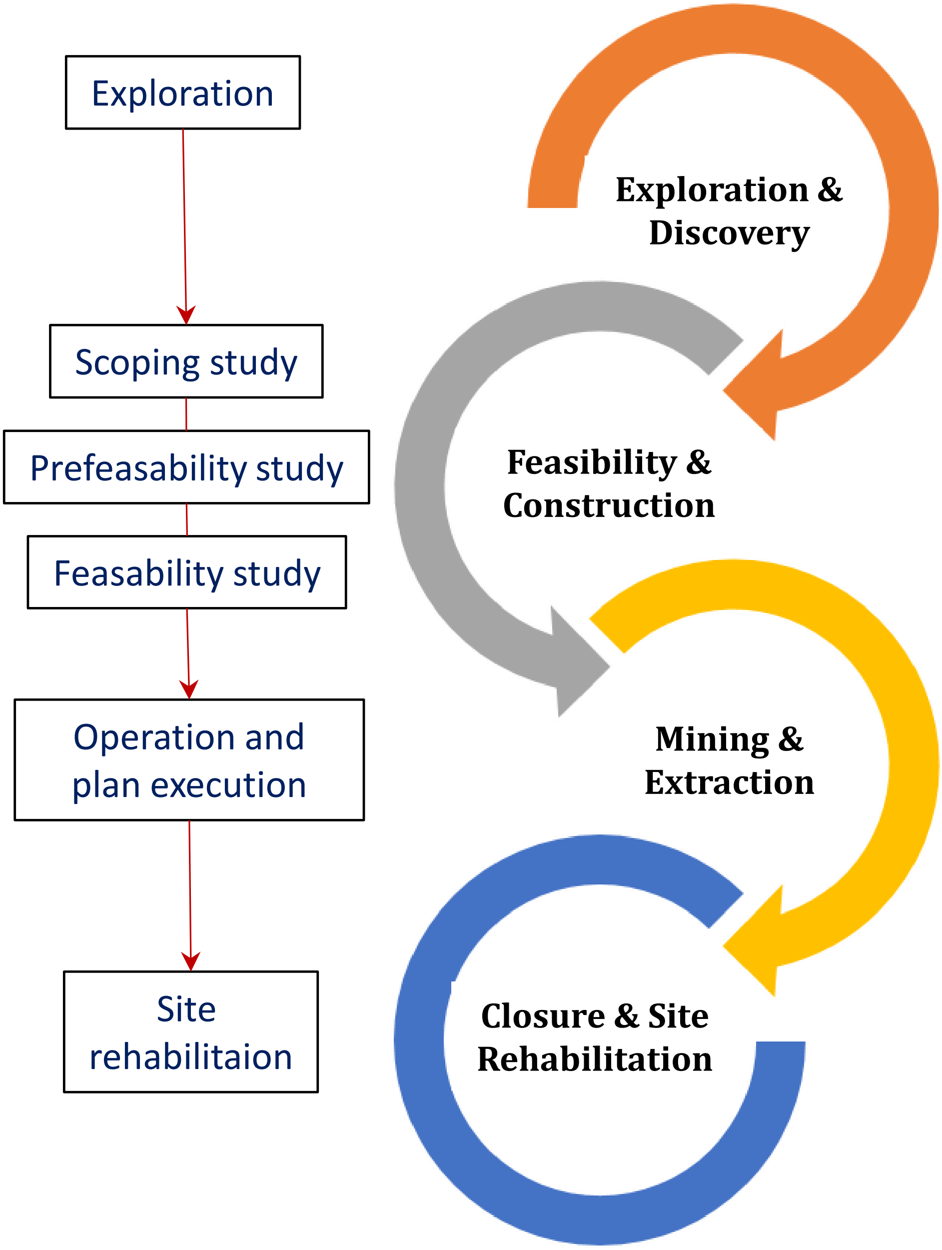


Figure 4. The mine operation life cycle
(Image: Simon Michaux)

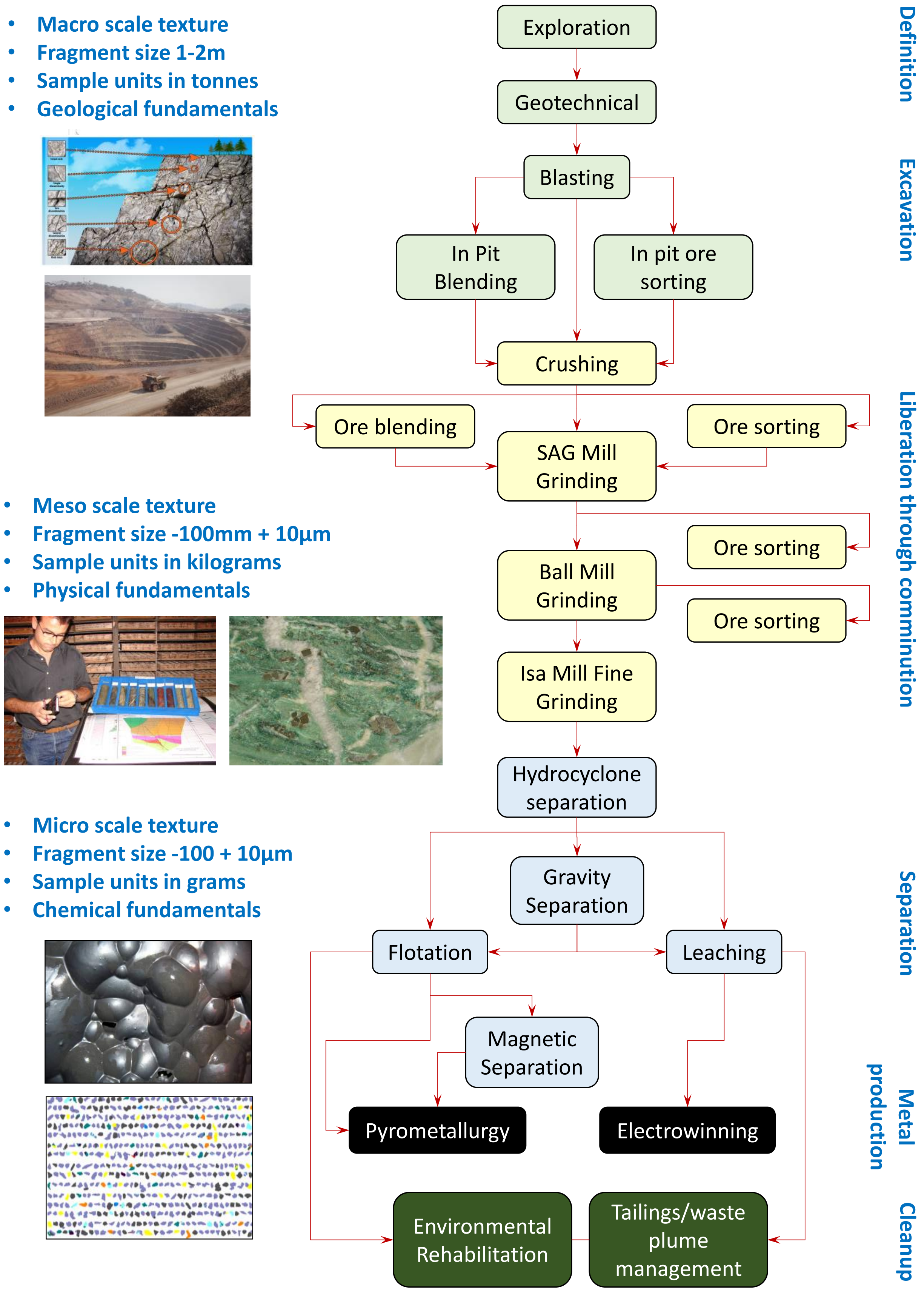


Figure 5. The mining process path
(Image: Simon Michaux, Tania Michaux, Peter Sorjonen-Ward, Pit image by Steven Harolds from Pixabay)

The decision on whether the study focus should be a valuable metal element (for example, Au; Cu or Ag), a mineral that somehow reduces production performance, or a penalty element that reduces the value of the final concentrate, needs to be considered. There are many kinds of process engineering that can be modelled successfully with a geometallurgical approach. The follow list are examples of possible focus tasks that could be integrated into a geometallurgical campaign:

Process Engineering

- Establishing the best machine efficiency process path for the flow sheet
- Choice between SAG mill or HPGR in the comminution circuit
- Recovery efficiency in flotation or leaching hydrometallurgy
- Ore type definition in context of comminution, flotation or leaching hydrometallurgy
- Choice between hydrometallurgy or pyrometallurgy in concentrate smelting and refining
- Pyrometallurgical efficiency in slag and matte management
- Penalty element impact on saleable concentrate
- Slurry rheology efficiency as a function of ore type
- Cyclone split efficiency as a function of ore type
- Impact of clays in ore processing
- Risk based process design
- Whole circuit systems-based modelling using all process areas

Economics

- Feasibility studies
- Economic viability and efficiency
- Energy efficiency
- Business model development
  - Whole mine operation optimization (Enterprise Optimization)
- Metal produced per kWh for each ore type
- Developing the mine schedule
  - Block model
  - Cut-off grade analysis

Environmental Impact

- Mine site waste plume
  - Acid mine drainage generation
  - Dust generation
  - Site water drainage pollution
- Mine closure and site rehabilitation
  - Operation legacy impact
  - Life of Mine foot print

Strategic Development

- Assessment of the true value of the resource
  - Priority for which valuable metals should be targeted
- Assessing the best time for open pit expansion cutback
- Assessing the choice between an open pit and underground operation
- Long term variability
  - Life of mine cycle
  - Mine operation footprint modelling
- Waste dump construction in context of future re-mining
- Whether to invest in this project or not in context of market forces
- Whether to shut down an operating mine in context of a challenging market environment
- Security of corporate reputation

<u>Geometallurgy for operations</u>

- Show stoppers & penalties in feed stream (clay, Cl, Fl, As, etc.)
- Predicted variability in ore hardness and recovery
- Metallurgical reconciliation
- Developing the best schedule of maintenance based on risk variability
- Plant expansion design and timing

All these tasks are to be carried out in context of the whole rock. If possible, the mineral combinations which drive and control each of these technical areas, needs to be understood and mapped out. This must be completed in a measurable and repeatable fashion, using experimental design protocol (Napier-Munn 2014). The final outcome can be clearly defined, success is quantified at the start, and experimental design can be set where success is achievable.

To achieve a single geometallurgical goal, volumes of context-based data are amassed. To study another process behavior on the same rock set may well be a comparatively small amount of test work. There is a large amount of overlap of experimental test work scope between a numbers of process behaviour, if studied in this way. With this in mind, it could be useful to collect samples large enough to be sampled several times and support more than one study.

As such, a completed geometallurgy dataset could be seen as the 'Rosetta Stone' of the rock (Figure 6). All parts of the mining process are working on the same rock but with different technical languages. This means that the rock itself becomes the data transfer point to examine multiple process behaviors and ore type mineral definitions.

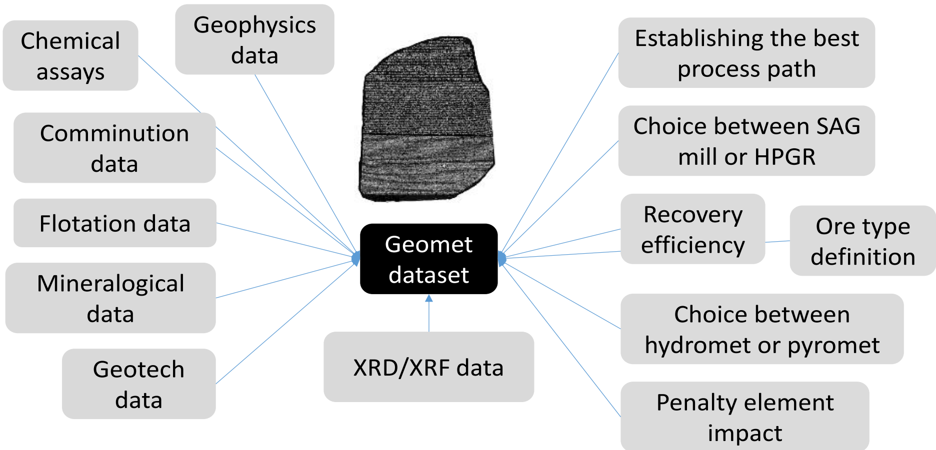


Figure 6. A well-constructed geometallurgy data set could become the 'Rosetta Stone' for analysis in that deposit (Image: Simon Michaux, and Rosetta Stone image by Kris Åsard from Pixabay)

Conventionally, the development steps for a mine are shown in Figure 7, where these steps are done separately and in isolation.

A geometallurgy program fits in and around the same steps and adds value by doing those same steps in a different way. Data is collected in a fashion that can define ore types in context of a process behaviour. This is done in a series of iterative loops, where the stability of the analytical outcome defines when data collection should stop (like the use of variography to determine if enough drilling has been done for deposit definition) (Figure 8).

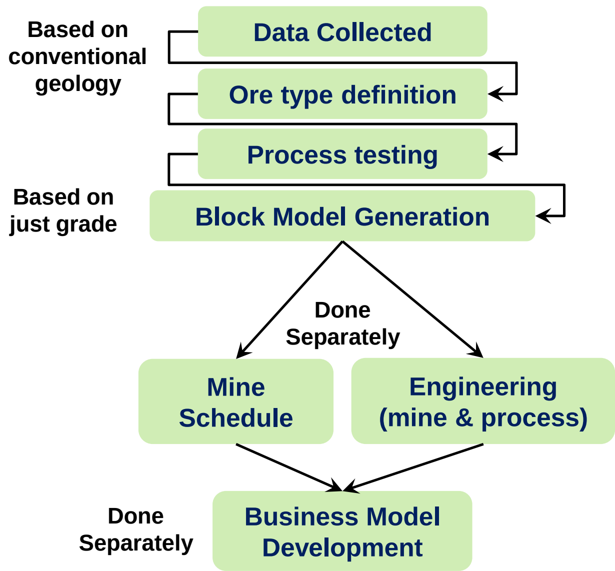


Figure 7 Conventional mine development flow sheet
(Image: Simon Michaux)

The tasks of mine schedule development and process engineering flowsheet development are worked in a systems context, where everything is optimized together (instead of separately) as shown in Figure 8.

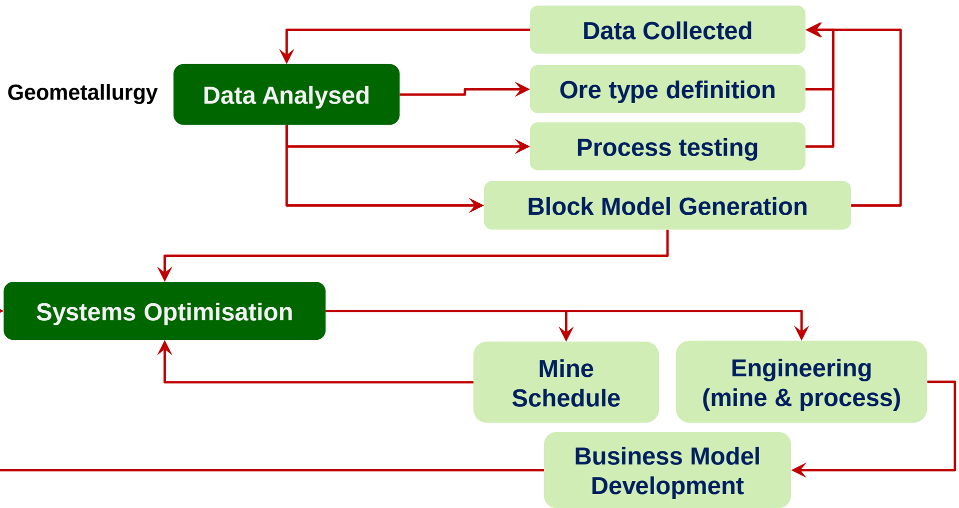


Figure 8 Geometallurgically supported mine development
(Image: Simon Michaux)

The outcomes of a geometallurgical study are variability, domain mapping and ore type definition in context of process engineering performance. These outcomes may be used for economic modelling, business development mine schedule development. All of this takes place before larger samples are taken for conventional process tests (bankable metallurgical tests), which lead to process plant design. This means a geometallurgical campaign is most cost effective in a pre-feasibility study, where it has the highest leverage of influence versus cost in the whole mine design cycle, this is front-end optimization (Mackenzie 2007) (Figures 9 to 11).

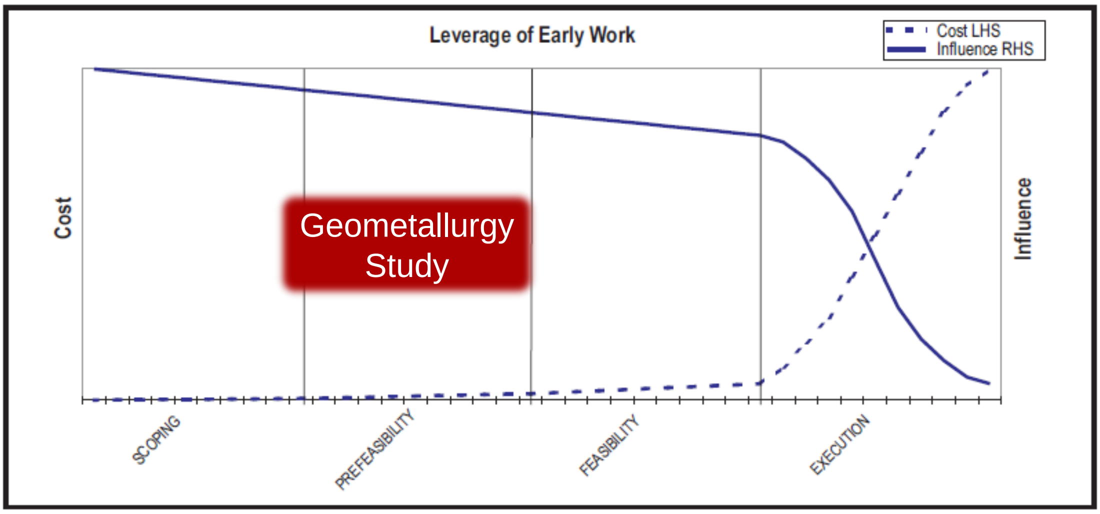


Figure 9 Leverage of early work (Source: MacKenzie and Cusworth 2007)

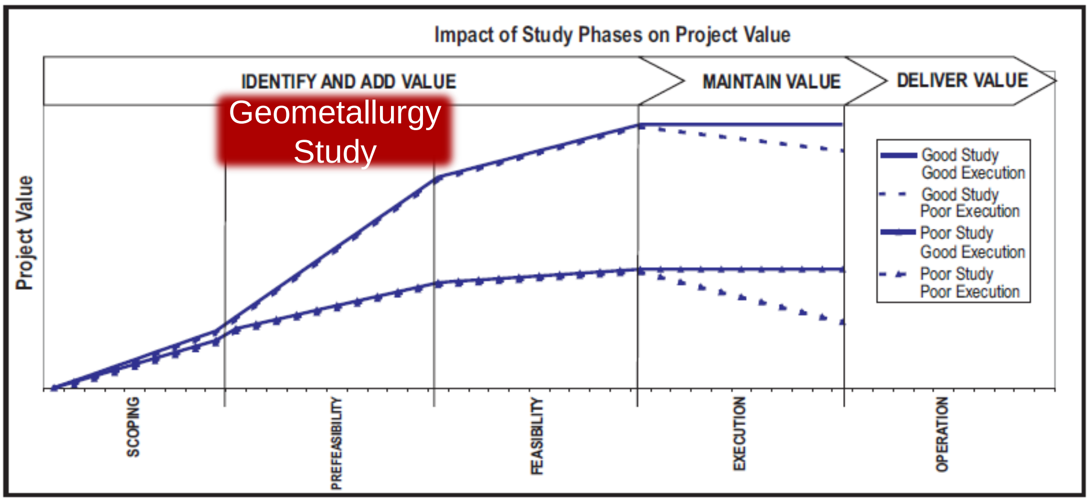


Figure 10 Impact study phases on project value (Source: MacKenzie and Cusworth 2007)

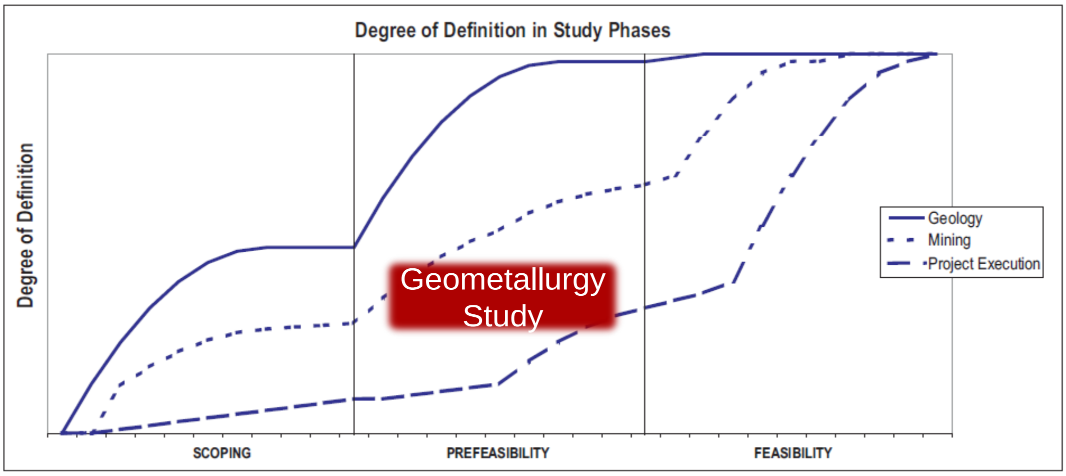


Figure 11 Degree of definition in study phases on project value (Source: MacKenzie and Cusworth 2007)

2.2 Phases in a Geometallurgy campaign

There are several phases of activity in a geometallurgy campaign and are shown in Figure 12 and 13. Depending on what the attempted goals are and scope, the following list could be a useful guide:

1. Planning and experimental design (Napier-Munn 2014)
2. Sampling from site (Gy 2004)
3. Orientation study (Section 6)
4. Geometallurgical matrix generation (Section 8)
5. Multivariate data analysis (Section 9)
6. Class definition (Section 9)
7. Process modelling by class (Section 10)
8. Mapping process domains and variability of process behavior (Section 12)
9. Hypothesis response and validation samples and analysis
10. Economic modelling and business model development
11. Assessment of study (Section 13)
12. Recommendation for a production protocol (Navarra *et al* 2018)

These different phases interact and relate to each other. While each is important, they vary in size and scope. Steps within the phases will be detailed.

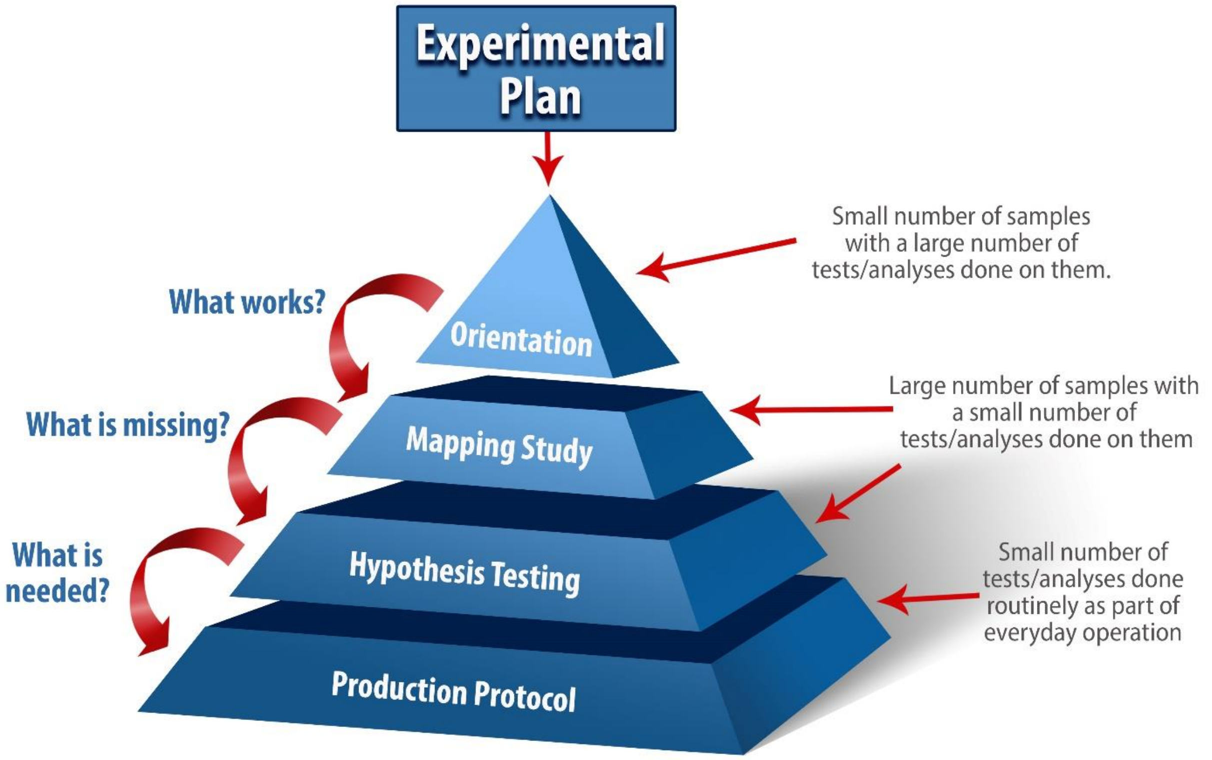


Figure 12 The different phases interrelate to each other.
(Image: Tania Michaux)

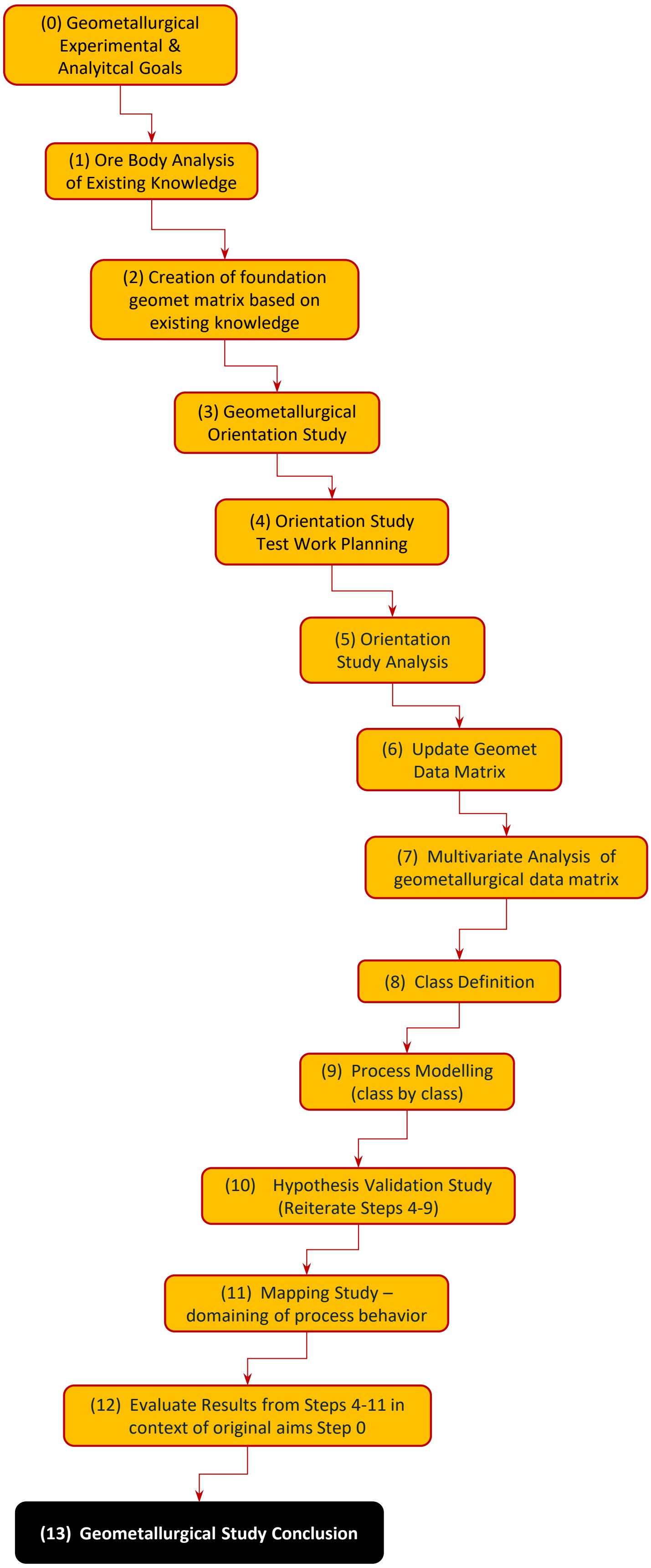


Figure 13. Steps of the geometallurgical campaign from point of genesis to completion (Image: Simon Michaux)

2.3 Planning and experimental design

For a geometallurgical campaign to be successful, it is required to be designed to a practical experimental objective and then carried out according to plan. Considerable time and thought should be dedicated to the design, where the outcome is clear and consistent with its point of genesis. A baseline, fundamental project structure is shown in Figure 14.

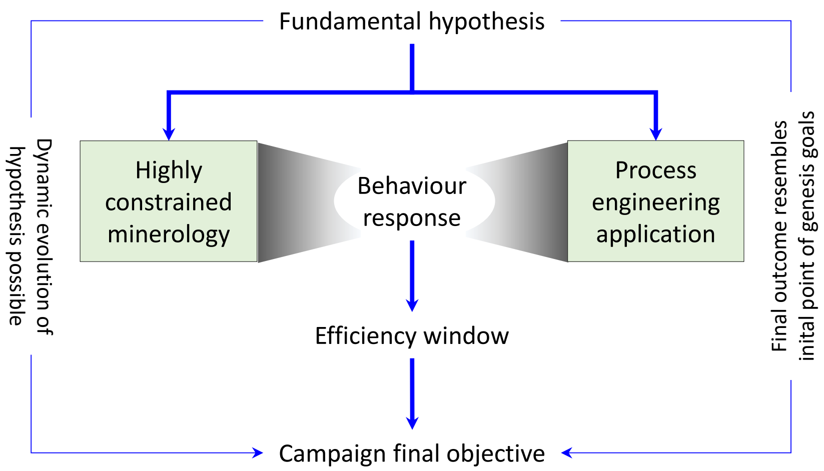


Figure 14. The fundamental structure of the geometallurgical campaign (Image: Simon Michaux)

The choice of what process behavior(s) will the geometallurgy study attempt to model is the fundamental question in experimental design (Napier-Munn 2014). Leaching for example is controlled by different mechanisms to controlling factors in comminution (for example in this case, leaching is fundamentally a chemical process and comminution is fundamentally physical process). This needs to be understood from the very beginning.

Once the target of the study is understood, the methods used to determine efficiency of that process behaviour must to be considered. What tests are used and how? What small scale tests are there and what is considered a trusted test by the industry (or a 'bankable' test – results accepted for submission to the Stock Exchange)? Then investigations into and an understanding of what mineral mapping tools are available and the form they take is required. An understanding of the precision of these methods and how the data is interpreted must be completed.

Part of the experimental design is a complete understanding of the options of data analyses selected for use. All data collected experimentally must serve a coherent purpose in the overall analysis. That analysis must be planned in the beginning of the campaign, to ensure technical limitations do not overrun the campaign objective. It is recommended that statistical methods of experimental design are used to collect data to support later methods of analysis, where the outcomes of that analysis can be judged in terms of statistical significance. Figure 15 provides a structure for the initial aims and objectives of the campaign.

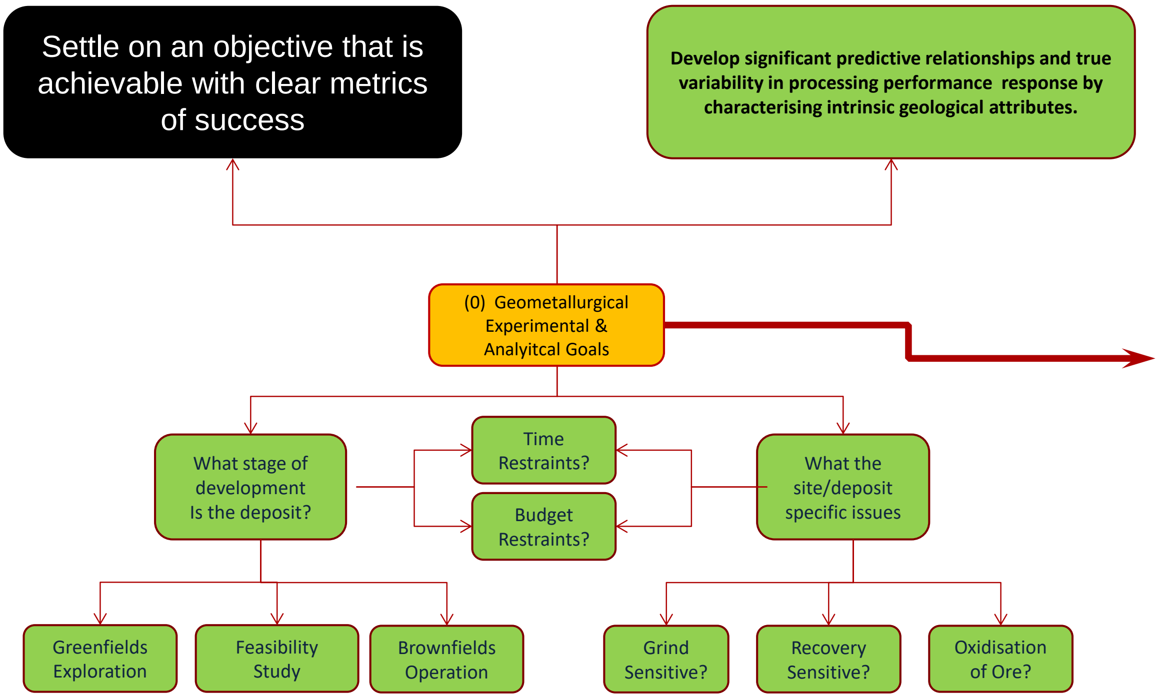


Figure 15. Step 0 - The development of the objectives and aims of the geometallurgical campaign (Image: Simon Michaux)

All concepts need to be merged together to define what samples are to be taken and what volume they should be. Sampling is often carried out poorly due to the lack of understanding of what is to be sampled, why and how. Ample literature is freely available to assist in defining a method (Dominy 2016, Krishnamurthy & Sawkar 2014, Bazin *et al* 2013, Sampling 2014, Dominy *et al* 2017, and World Conference on Blending 2017). Campaign planning of objectives to be agreed upon before sampling:

- Campaign objectives set
- Campaign mandate and scope agreed upon
- Metrics of success agreed upon
- Experimental design and analysis to be conducted agreed upon
- Test work required agreed upon (including where test work is done)
- Budget and time frame agreed upon
- Required mass of each sample understood

2.4 Sampling from site

Once the campaign objectives, experimental design, and scope are finalized, existing orebody knowledge (OBK) needs defining. This is to ensure that previous work done is not repeated. Assessment of historical reports, experimental test work and client personnel professional knowledge. A discussion with an exploration geologist who worked on drill core/site and any process engineering staff that have characterized the target ore types. Any work done on the process design flowsheet should be understood through discussion with the engineers who developed it. Any site-specific issues are required to be understood as early as practical.

<u>Site geologist intuition is to be listened to and considered</u>

- What ore types do they think are relevant?
- What are the weathering horizons and clay content understood in this deposit?
- What deposit geology structures are most significant?
- What drill holes traverse and map these structures?

Site metallurgist intuition is to be listened to and considered

- What is the process range found so far in this deposit?
- Axb & BMWi extremes
- Flotation recovery performance
- Leaching recovery
- What minerals are considered penalty elements
- Are there site specific assay models that have been developed?
- Are there any site mineralogy relationships, associations or ratios already established?

Figure 16 shows a set of ideas to start this process.

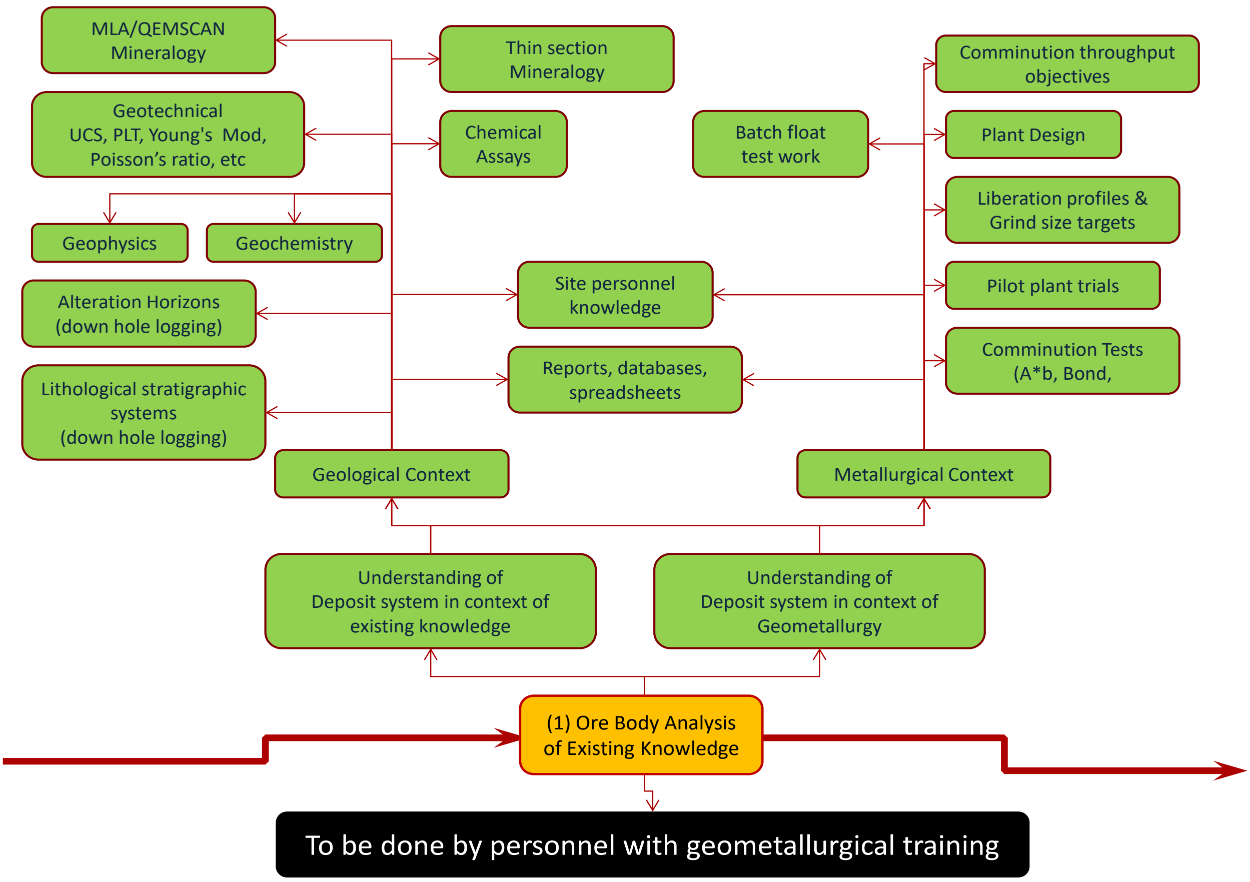


Figure 16 Step 1 - Assessment of existing site deposit knowledge
(Image: Simon Michaux)

The data and information collected from this examination will have to be assessed whether it is useful in context of the planned geometallurgical campaign. The quality assurance and quality control (QA/QC) needed to build a geomet data set, is quite different to conventional methods due to the much smaller samples masses available for testing. For example, when metallurgical test work was completed, did the sample comprise constrained mineralogy, or was it drawn from a combination of multiple sources? How was ore-type definition decided? Were the samples collected in a form where the spatial position was known, and the results can be mapped back into the deposit model?

Once the existing deposit data is understood from a geometallurgy paradigm, the foundation data matrix can be assembled. Ideally, the samples collected for a geomet campaign would be drill core alone. However, this is not always possible and other drill products need to be utilized. Most mining operations have extensive drill core libraries/archives either on site or in a storage facility, where half of the drill core has already been logged lithologically and then sent for elemental assay. Core remaining at site (library core) could be sampled from drill holes that have existing analytical interpretations, such as: geophysics and geotechnical characterization (Point load, UCS compression test, Brazilian Disc, Fracture Frequency, RQD, etc.) All of this is potentially useful data depending on your initial objectives.

A series of further tests could be carried out on diamond drill core that already has good context data collected. With this in mind, a data set is to be set up with architecture to allow the import of existing data that passes QA/QC, the results from new test work and the analytical outcomes of the campaign. This is to be executed in a fashion where all data types can be used as model inputs, where the result is able to predict the process behavior of constrained mineralogy in a spatially defined way. A useful approach to do this is to limit the sample volume to the same scale as chemical assays (or a multiple of). For example, chemical assays are samples from half drill core in 2m length intervals (or 1m interval with RC chips). Such a small volume that is routinely tested as part of existing operational protocol is a practical method to identify specific mineralogy and have test work consistent with the rest of the data set. In the dataset, there will be many different kinds of data that are not conventionally reported in the same matrix due to method of collection, and the spatial geometry associated with the result. With the understanding that this is a work in progress, professional judgment is required from this point onwards in how to achieve this.

It is recommended that each sample in the geometallurgical campaign has a sample identification label that can be used to manage the data set in a consistent manner throughout the whole campaign. This will be used in data QA/QC procedures, analysis, and later the mapping of the results back into the deposit. It will also assist in the later compositing larger samples according to the geometallurgical analytical outcomes to provide more meaningful results from larger scale test work. Figure 17 shows a possible starting point (Stage 2).

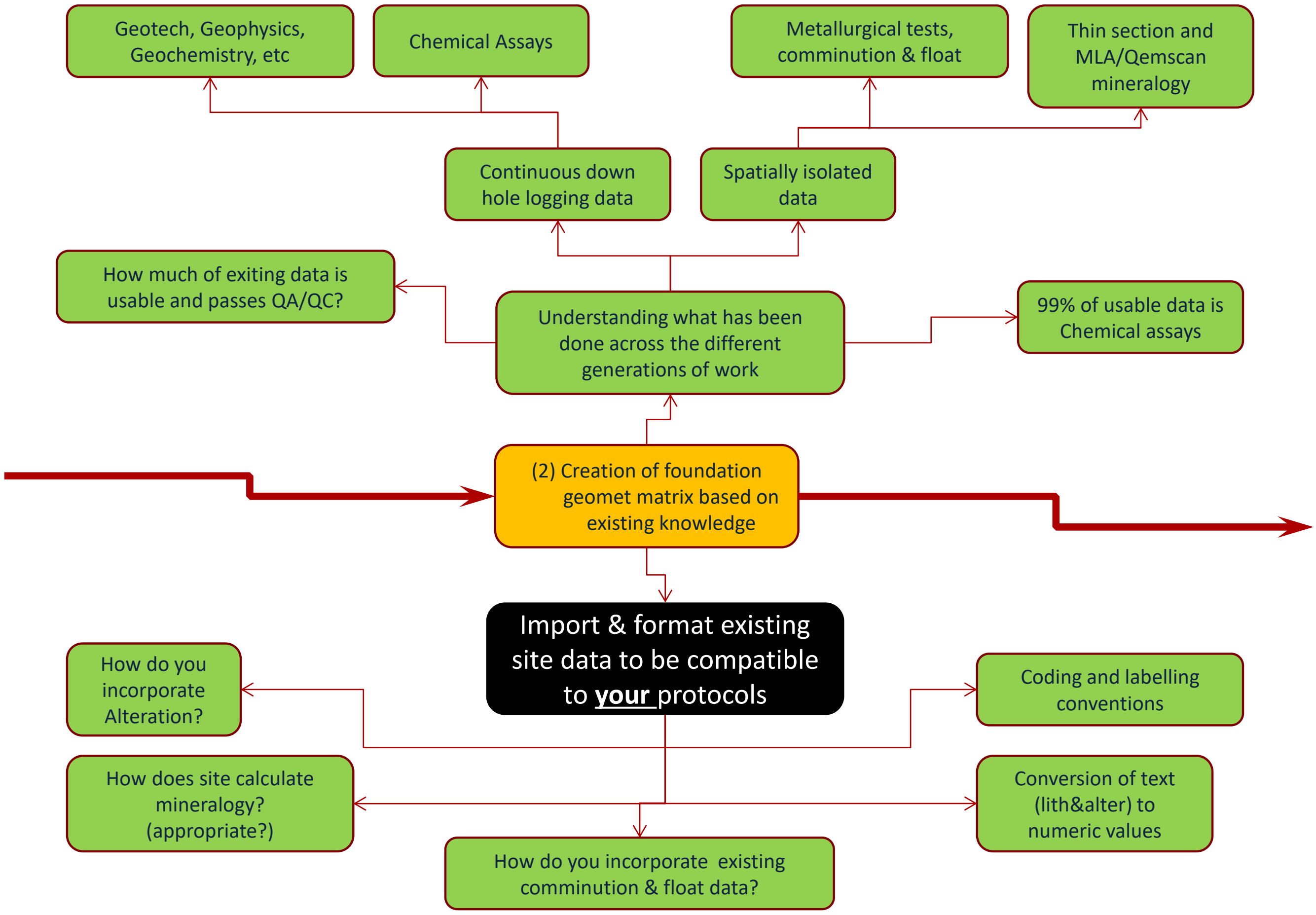


Figure 17 Step 2 - Development of a dataset which will be the foundation of the geometallurgical dataset (Image: Simon Michaux)

Once the foundational geomet data set has been established, collection of samples for further test work can be considered. Communication with other site stakeholders is key at the point, collaboration is key to success. A geometallurgy study is most effective when using drill core samples; planning destructive test work procedures with limited material requires good knowledge of bankable metallurgical testing, or proxy testing (small-scale) (Schouwstra *et al* 2017, Kuhar *et al* 2011, Chauhan 2013). Each test sample has to be of tightly constrained mineralogy and also needs to be able to be accurately spatially positioned in the deposit. Samples can be taken from the exploration campaign or from a mine site core yard library.

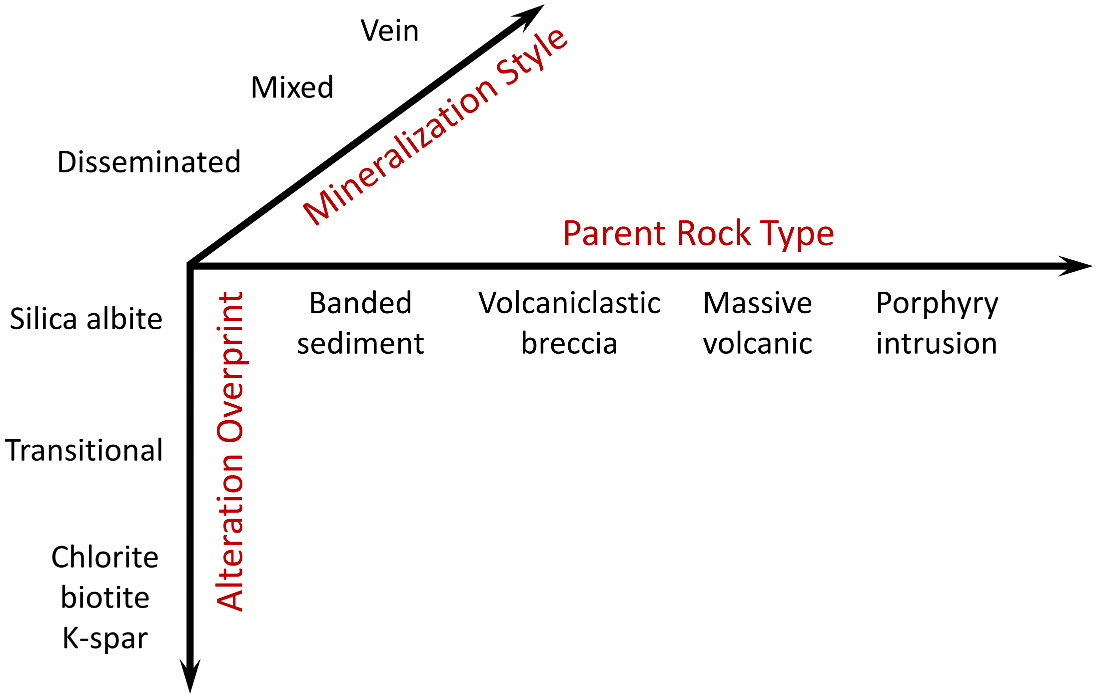


Figure 18. Selection and definition of end member rock texture samples
(Image: Simon Michaux)

While examining drill core in the site core library, assess end member present in this deposit. Isolate ten to twenty ore type extremes. Physically collect samples of each end member, where each sample is consistent in lithology and rock texture (2-6m length). If veins and intrusions are in the sample, make sure they are consistently all through the sample, not just one or two isolated places (within reason, professional judgment required, be representative). These samples are for the Orientation Study.

Ensure there is enough sample volume for each end member, to undertake all of the planned orientation study test work. Each end member mineralogy will need a successful (passed QA/QC) larger scale 'bankable' test (industry accepted in feasibility studies) as well as a number of the planned smaller scale proxy tests.

Once the end member rock texture extremes have been sampled, collect other drill core samples that transverse all of the geological structural features of the deposit. (Texture incorporates mineral species, grain size and shapes, weathering and alteration (Clark 1991, Lund *et al* 2015, Perez-Barnuevo *et al* 2013, and Lang *et al* 2018)). Ideally, this would be continuous sections of half core from four to five drill holes. This would represent drill core sections of the deposit that cross the different zones of the deposit, where all the end members found so far can be put in context of how they relate to each other. This often results in something like 1500-2000m of half drill core. Where possible, collect samples from the biggest available core size possible to increase sample mass per meter (usually HQ or NQ core size).

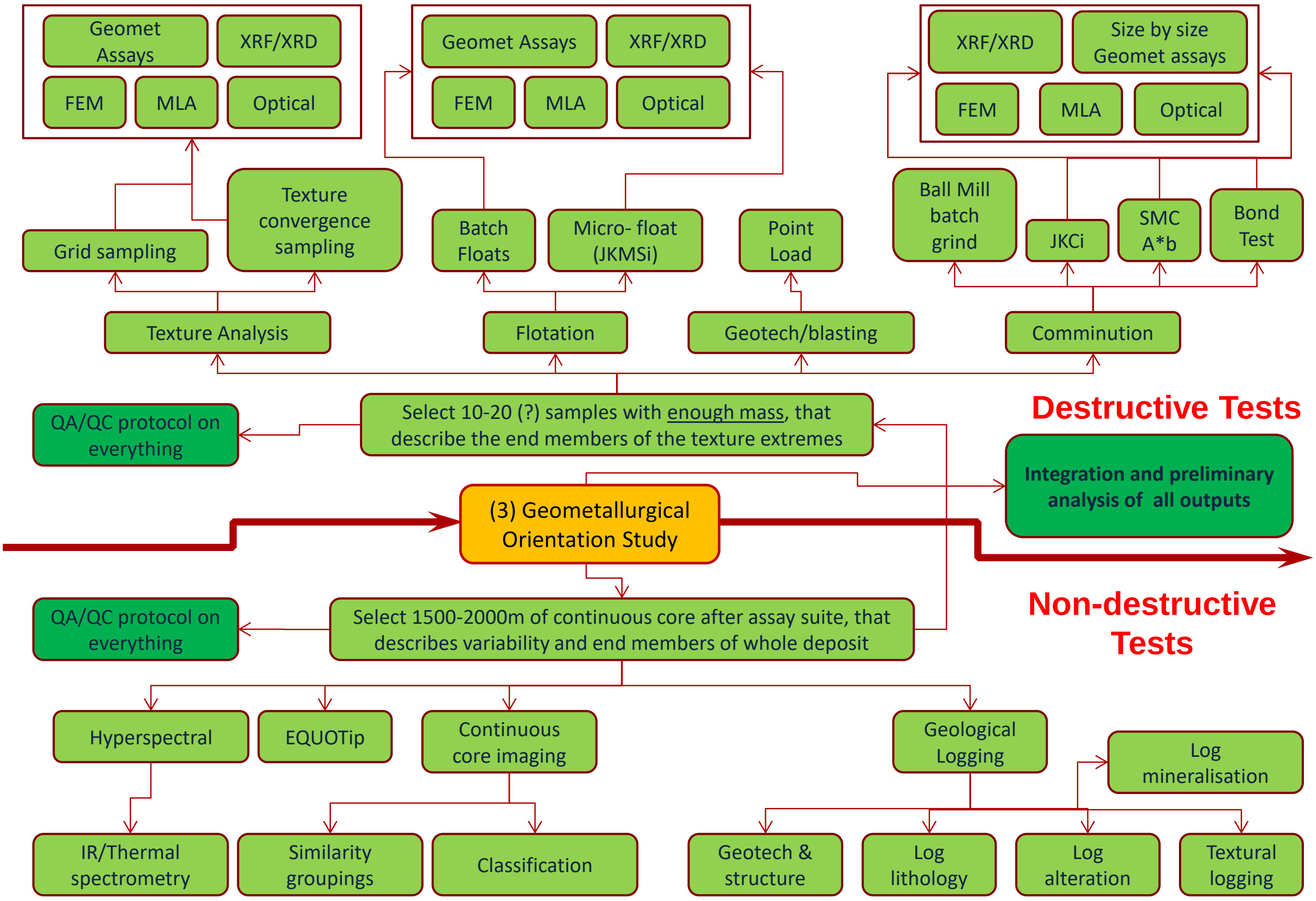


Figure 19 Step 3 - Experimental test planning ideas for the geometallurgy experimental campaign (Image: Simon Michaux)

Poor quality core is not to be rejected in sample. Traditionally metallurgical sampling (comminution in particular) has been carried out on the good quality pieces of core, whereas the core is often broken up due to the friable nature. This is often where the clays, heavily altered minerals and poor recovery performing ore types are found (fines in particular in the bottom of the tray). The comminution of these samples might be faster (often softer ore is friable) but the recovery performance is likely to be poor. As this represents part of the deposit, it needs to be mapped and modelled if this ore type is in large quantities in the drilling library. These samples are to be what the Mapping Hypothesis set and based on. It is with these samples that ore variability could be assessed and mapped. The final scope of test work will probably be smaller than noted in Figure 19 due to practical constraints. The purpose of this exercise is to consider many things, then establish was is best for the geometallurgical campaign objectives.

2.5 Orientation study

The purpose of the Orientation Study is to investigate and plan in detail the process behavior and response of the extreme rock textures or end members of the deposit. The full range of measurements of the process response (for example flotation recovery, or comminution response) that might be encountered in this deposit, and then understand the mineralogical controls which influence.

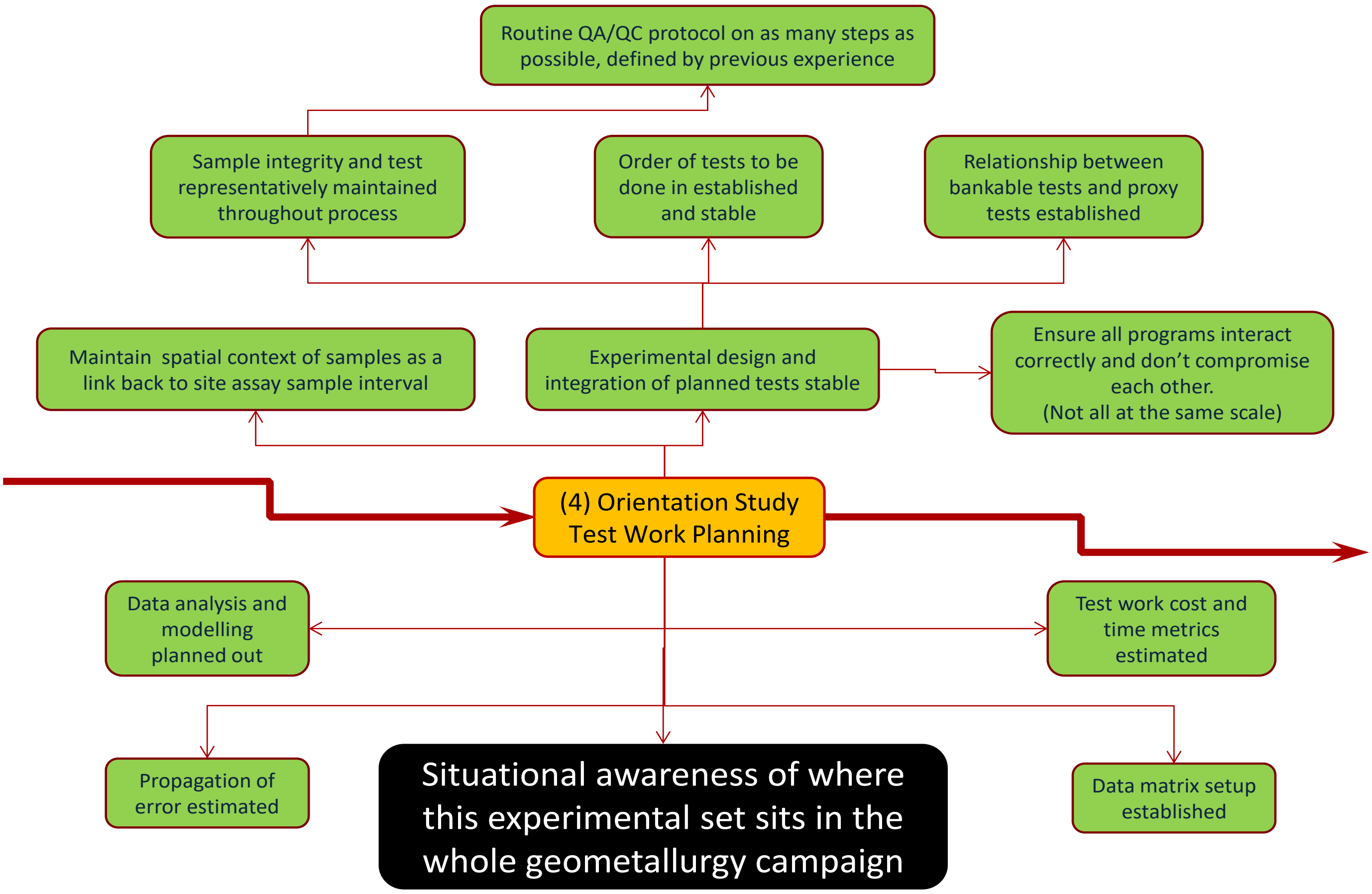


Figure 20. Step 4 - Planning of Orientation Study testwork
(Image: Simon Michaux)

3 ORE MINERALOGICAL CHARACTERISATION

Rock texture is a method of describing the structure and the style of assembly of how mineral grains are grown together as the rock formed. Ore is a classification of a rock that has some economic value and are defined as a mineral or aggregate of minerals that form a rock which can be mined at a profit. They can therefore be classified, like everything else in earth sciences, into conveniently simplified categories. There are classifications that are based on simple criteria, such as the minerals or metals contained (Pb-Zn, Cu-Au, PGM,etc).

Rock texture has been a complicated phenomenon to measure and then classify in any consistent context. Texture can be related to a 2-dimensional description i.e. an image, or it could relate to the physical texture. At times, these can be miscommunicated. It is a key component to geometallurgy. Understanding rock texture is often required to understand the ores potential process behaviour response and likely recovery. Where the relationship between the materials of which a rock comprises is studied. The broadest and simplified geological textural classes are but certainly not limited to (Whitten & Brooks 1972 and Craig 1994):

- Crystalline (in which the components are intergrown and interlocking crystals)
- Fragmental (in which there is an accumulation of fragments by some physical process)
- Aphanitic (in which crystals are not visible to the unaided eye)
- Glassy (in which the particles are too small to be seen and amorphously arranged)

The geometric aspects and relations amongst the component grains or crystals could be referred to as the crystallographic texture or preferred orientation. Textures can be quantified in many ways. The most common parameter is the grain size distribution (2D). This creates the physical appearance or character of a rock, such as size, shape, arrangement, and other properties, at both the visible and microscopic scale.

The approach of how to characterize mineral rock texture can be complex. Measurements at different scales will provide different kinds of results. An optical microscopy analysis on a 20mm thin section will provide different data outcomes compared to the same sample analyzed in an automated mineralogy QEMSCAN on 100μm particles. This implies that the needed answer depends on the appropriate question. The same dilemma exists in the geotechnical mapping of the macro scale (1m to km scale). The size of the measurement gives a different answer (Figure 21). A very small sample (say a 50mm drill core) provides a sample of the intact rock, with no jointing, which could be used to measure intact rock strength (UCS compressive strength or Tensile Strnegth). A larger sample of say a scan line mapping of 500mm could return intact rock with one or two faults. A scanline mapping of 100mm could provide a rock mass with a distribution of faults that allow the estimation of *insitu* block size.

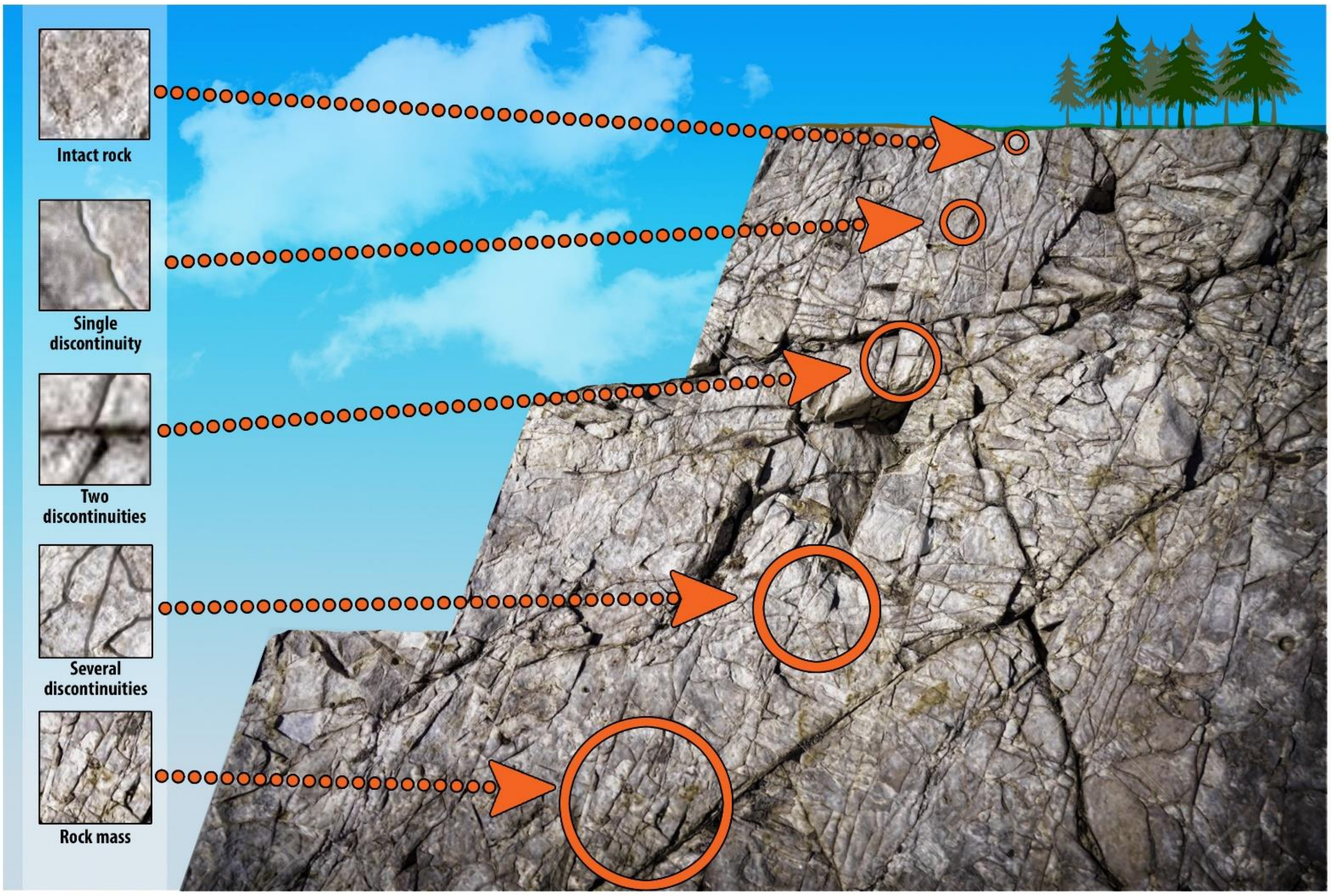


Figure 21. Rock texture at macro to meso scale depends on scope of measurement
(Image. Tania Michaux)

What data needs to be collected? A geometallurgy study is not necessarily the same thing as in other studies, thus does not have the same experimental architecture. Usually, the target is the understanding of a specific mineral, mineral of interest (MOI), for example chalcopyrite. That being said, it is not just the target mineral but what it is associated with and in what mineral assemblage is present. It may not be necessary to map/analyse the entire sample. This could be relevant when planning the number of tests. A geometallurgy campaign often requires many more such tests than a conventional study. Figure 22 illustrates different examples of mapping the same mineral texture. The MOI is chalcopyrite, from a comminution aspect, the host mineralogy is a priority, from a floatation aspect, the immediate mineral associations also extremely important. Mineral association is key information when understanding floatation performance, liberation potential and recoveries.

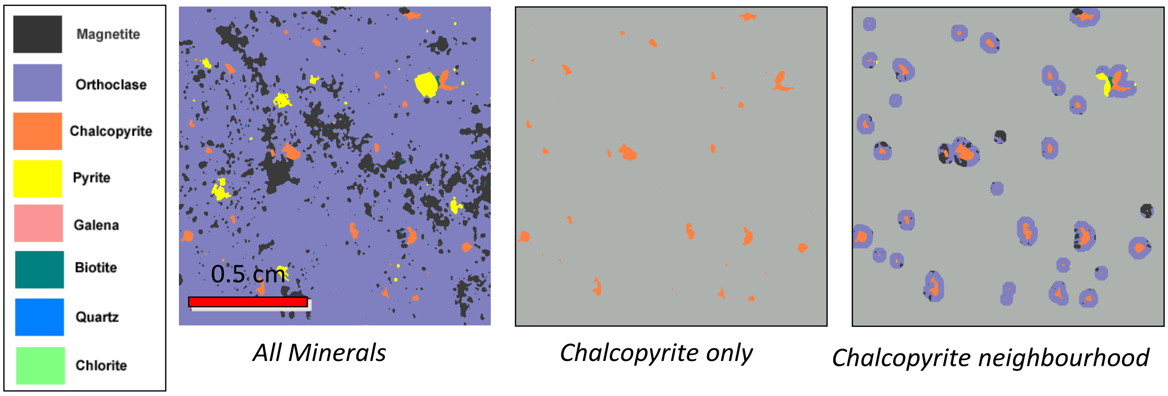


Figure 22. What is really needed in rock texture characterization?
(Image and copyright: Steve Walters, JKMRC, AMIRA)

Also, different process behaviors are influenced by different volumes of the rock, and are best modelled by different scales of characterization as noted in Figure 23 (AMIRA P843 2009).

Figure 23. Scales of characterization
(Image: Simon Michaux)

3.1 Characterization of Meso-textures

Meso texture is scale of examination measured in centimeters (cm). A number of new technological methods have been developed, that allow the characterization of drill core samples, where core is in the drill tray. Before the samples are crushed and sub-sampled, the core could be imaged to characterize the meso-scale texture.

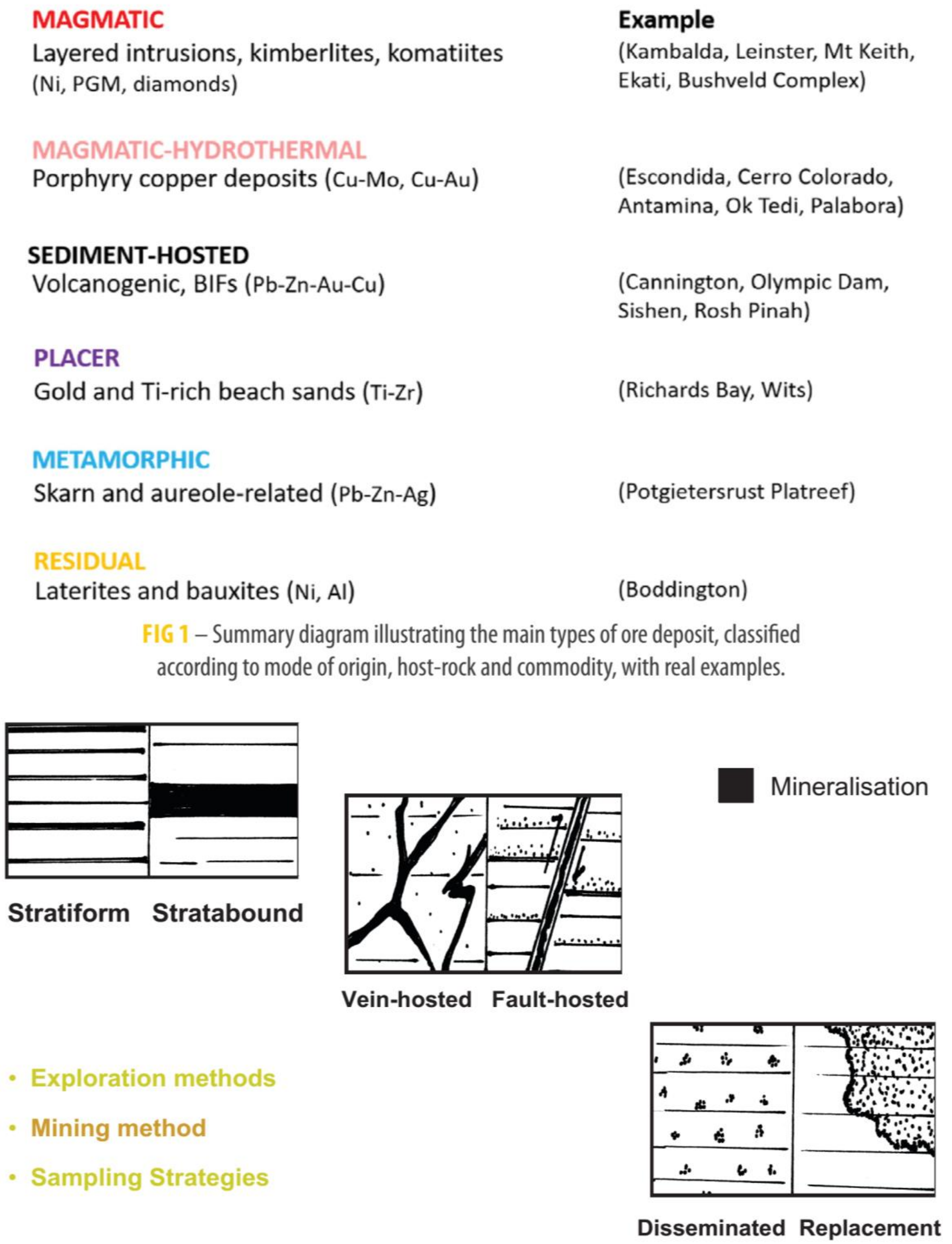


Figure 24. Some examples of meso textures
(Source. Butcher 2019, Image: Alan R Butcher)

Examples of different meso scale rock textures can be seen in Figures 25 to 28.

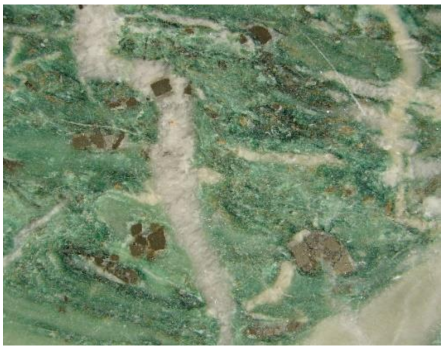


Figure 25. Meso texture scale polished section sample
(Image: Marinia Sorjonen-Ward)

Figure 26. Meso texture scale drill core sample 1
(Image: Marinia Sorjonen-Ward)

Figure 27. Meso texture scale drill core sample 2
(Image: Marinia Sorjonen-Ward)

Figure 28. Meso texture scale drill core sample 3
(Image: Marinia Sorjonen-Ward)

3.2 Core Imaging

If the study scope and budget allows core imaging of the samples, some interesting opportunities can develop. Historically, only micro-scale rock texture was able to be measured with relative ease. Through technology advancements, multi-sensory capabilities, optical camera's, x-ray imagery (CT) and gamma can be carried out on drill core and data collected (Vatandoost et al 2008 and Shreeve 2018). The data is in the form of imagery most recently, elemental. The images collected can be filtered in their raw form into similarity groupings, or classified first then grouped, using specialist software (Developed at Julius Kruttschnitt Mineral Research Centre JKMRC).

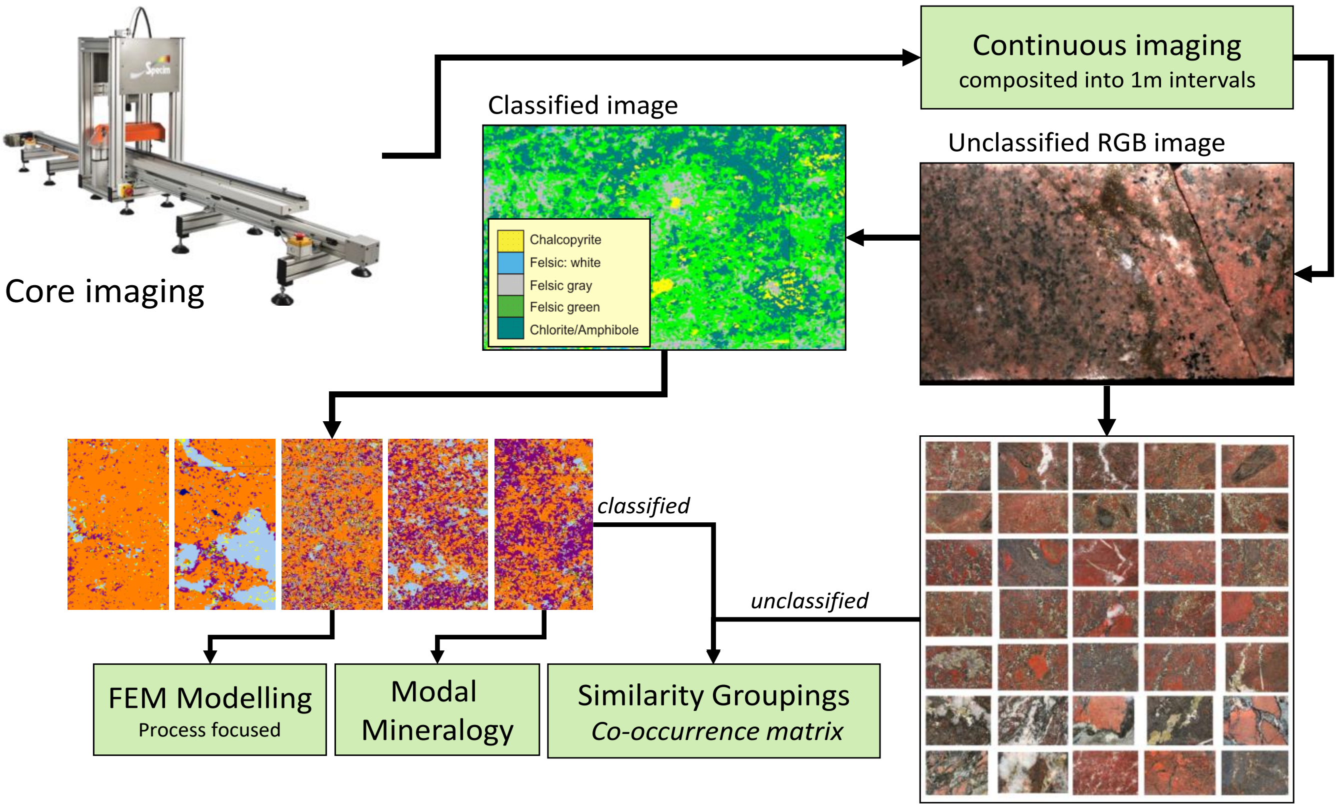


Figure 29 Core imaging
(Source: AMIRA 2009, Copyright AMIRA)

The core is imaged with appropriate cameras in appropriate lighting. The images (with core depth and intervals recorded) are then processed with appropriate software. Similar images can be grouped with other like images (grouping lithologies in context of how they visually appear). This could be a fast and cheaper version of hyperspectral imaging. An example of this is shown in Figure 30 to 36, which show seven of the similarity groupings found by the software for this case study. Figure 37 shows an example of a classification tree of images.

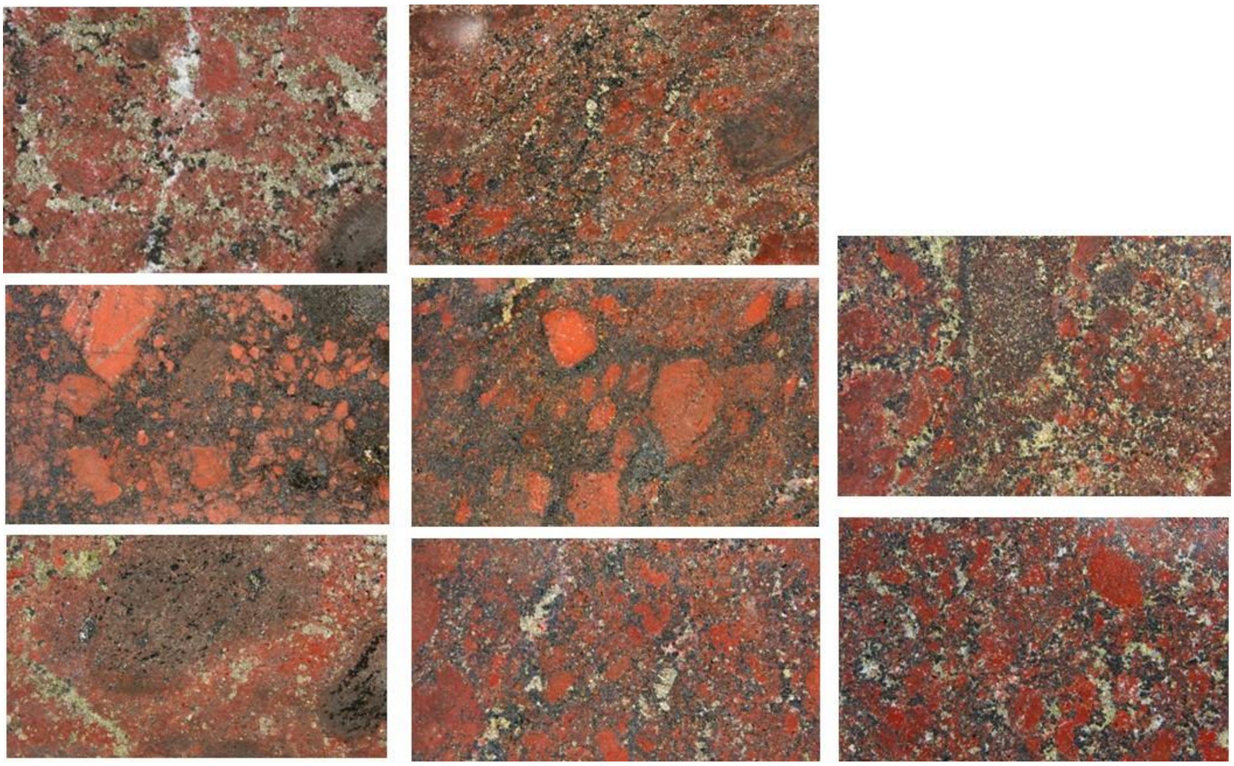


Figure 30. Similarity group class 1 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

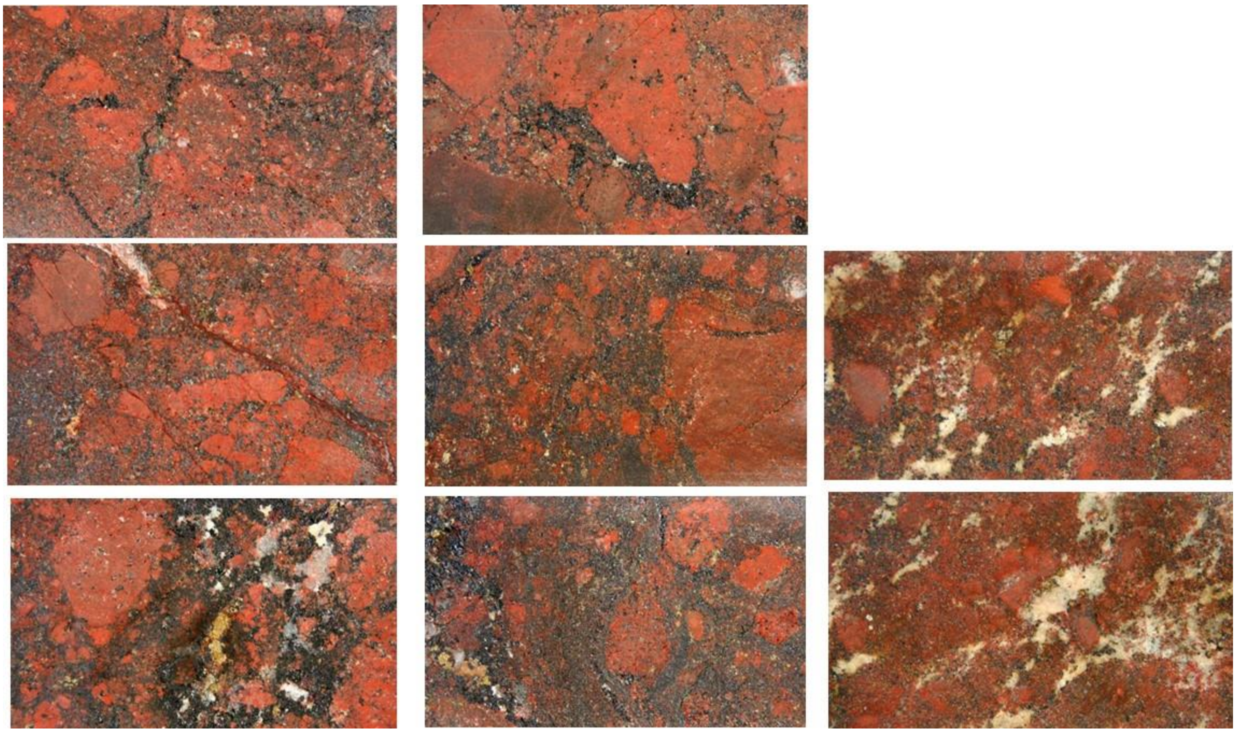


Figure 31. Similarity group class 2 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

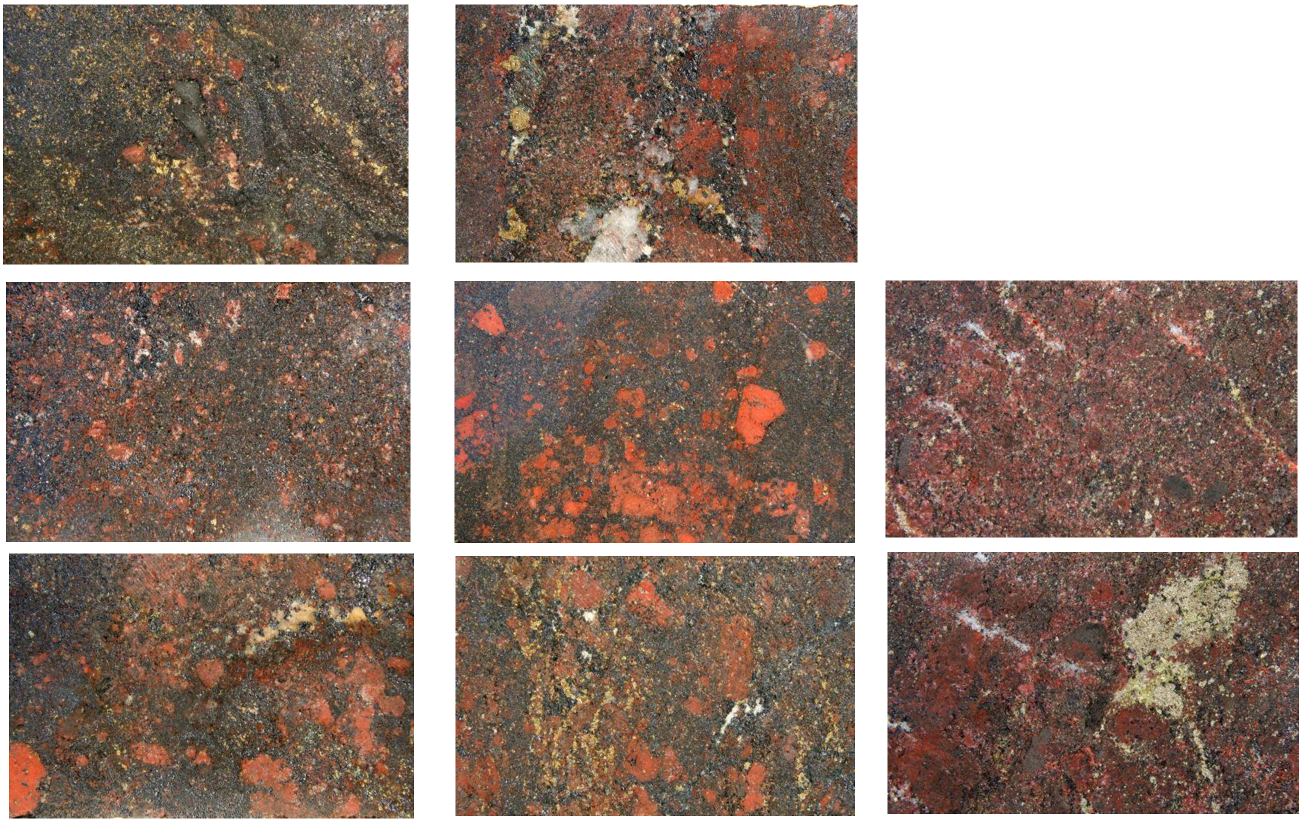


Figure 32. Similarity group class 3 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

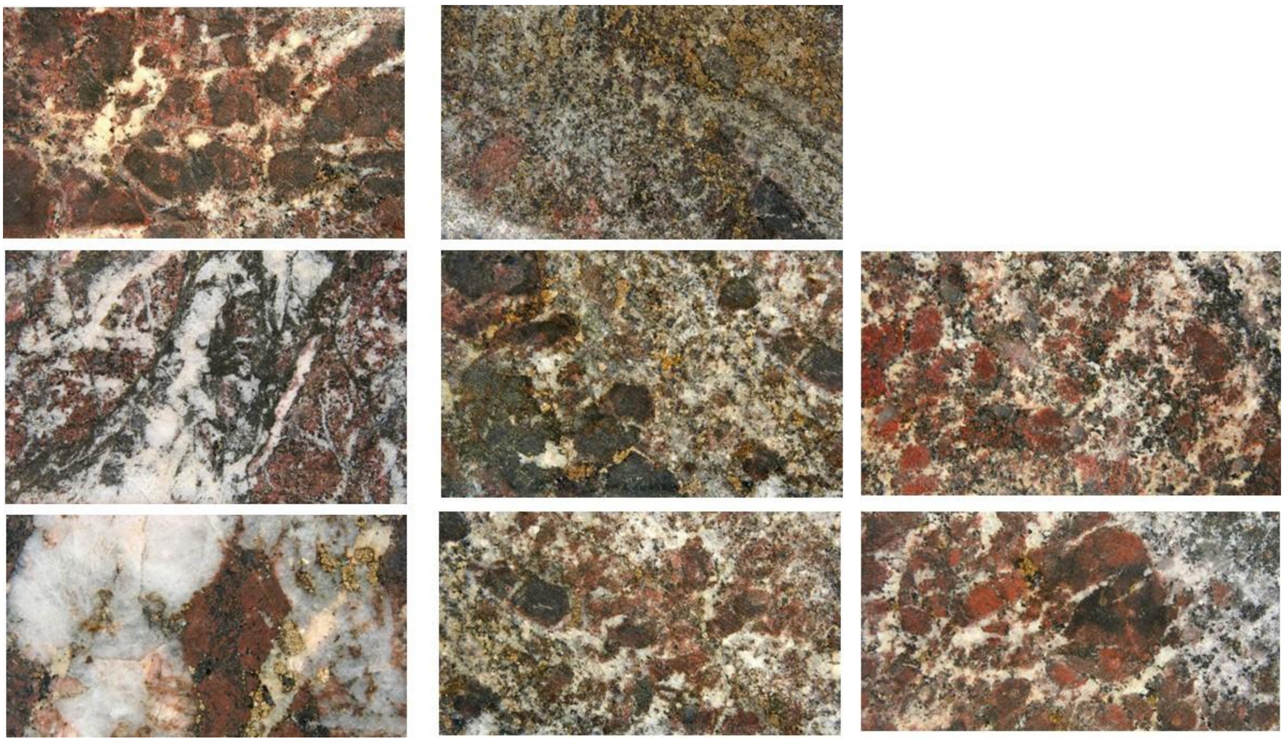


Figure 33. Similarity group class 4 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

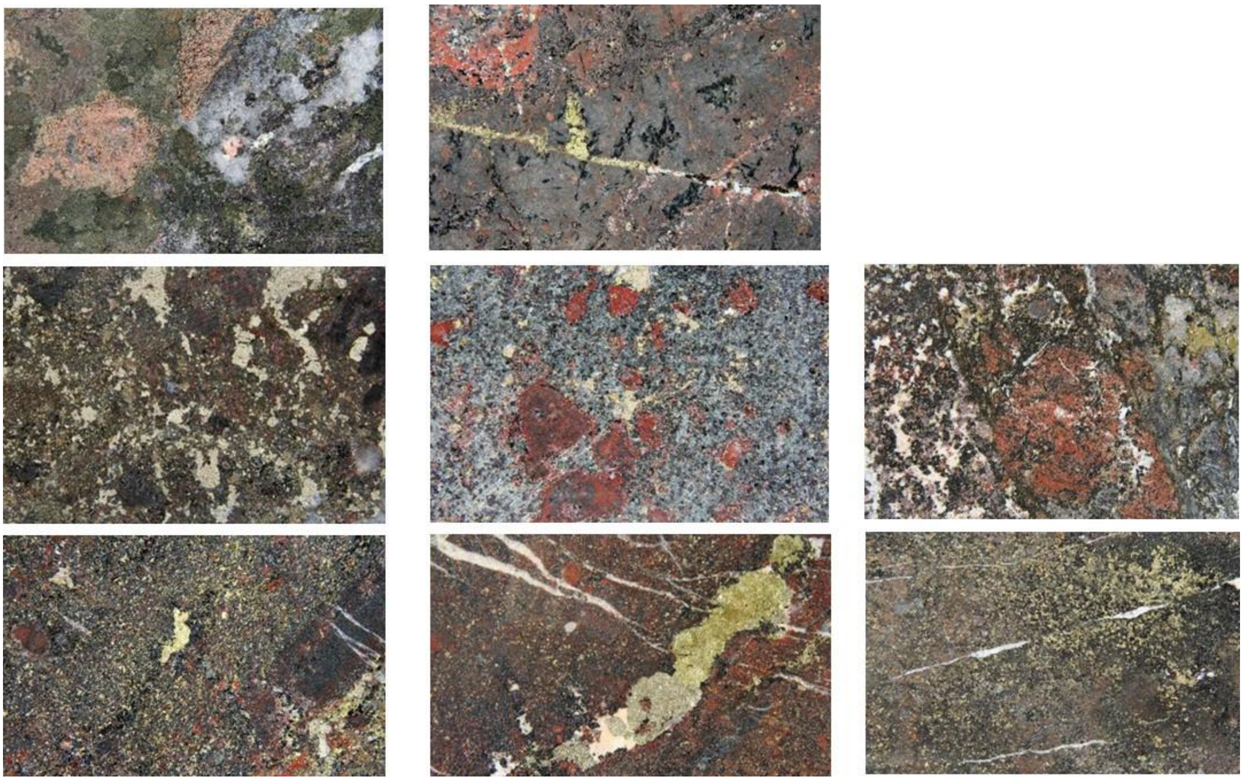


Figure 34. Similarity group class 5 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

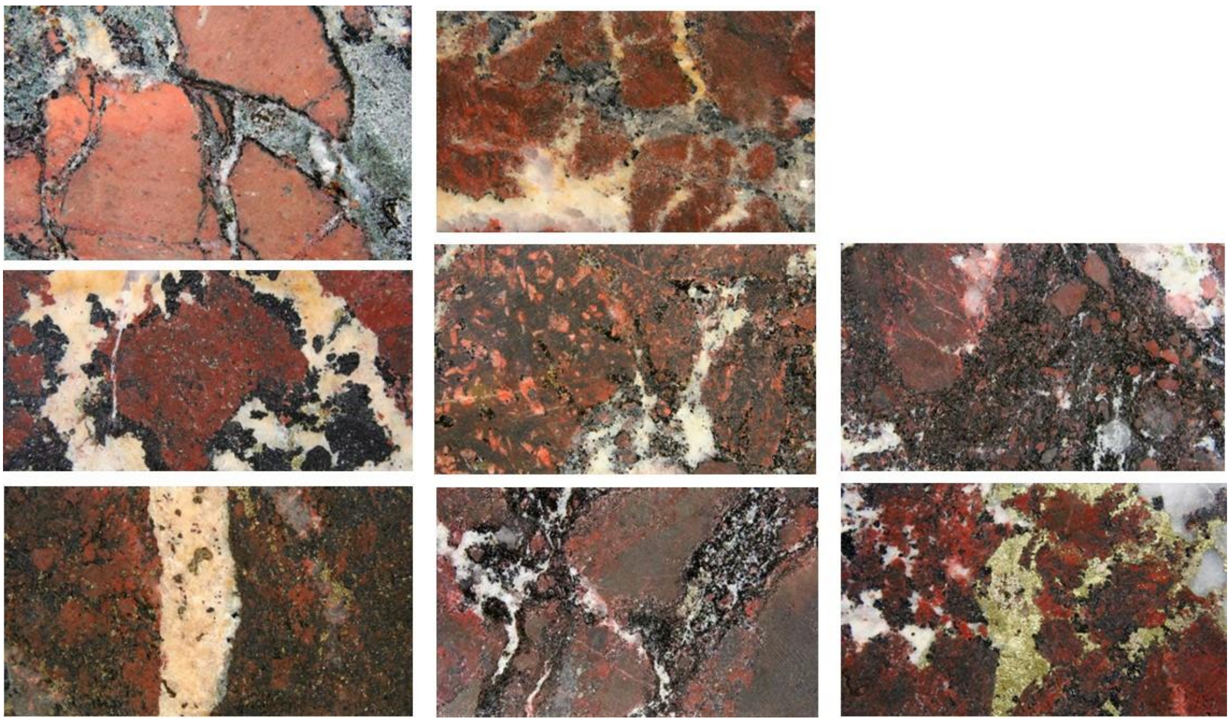


Figure 35. Similarity group class 6 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

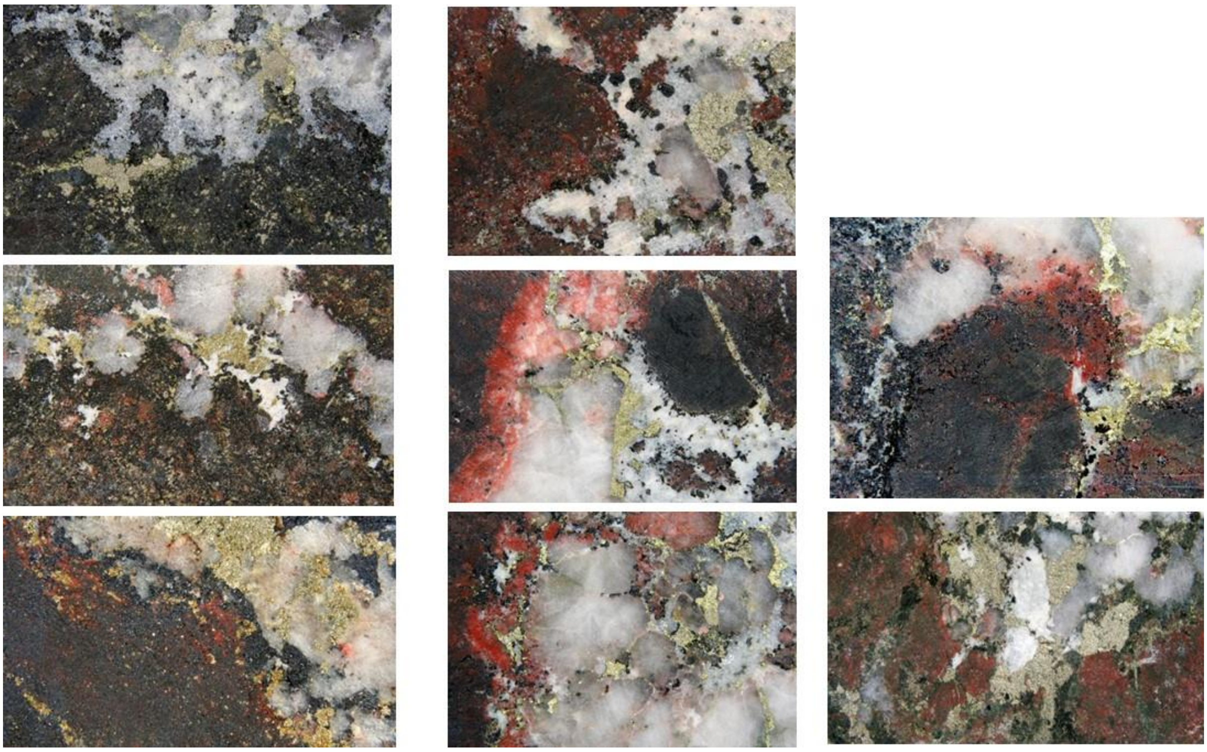


Figure 36. Similarity group class 7 of rock texture images from First P843 Case Study
(Source & Copyright: AMIRA P843, Ron Berry)

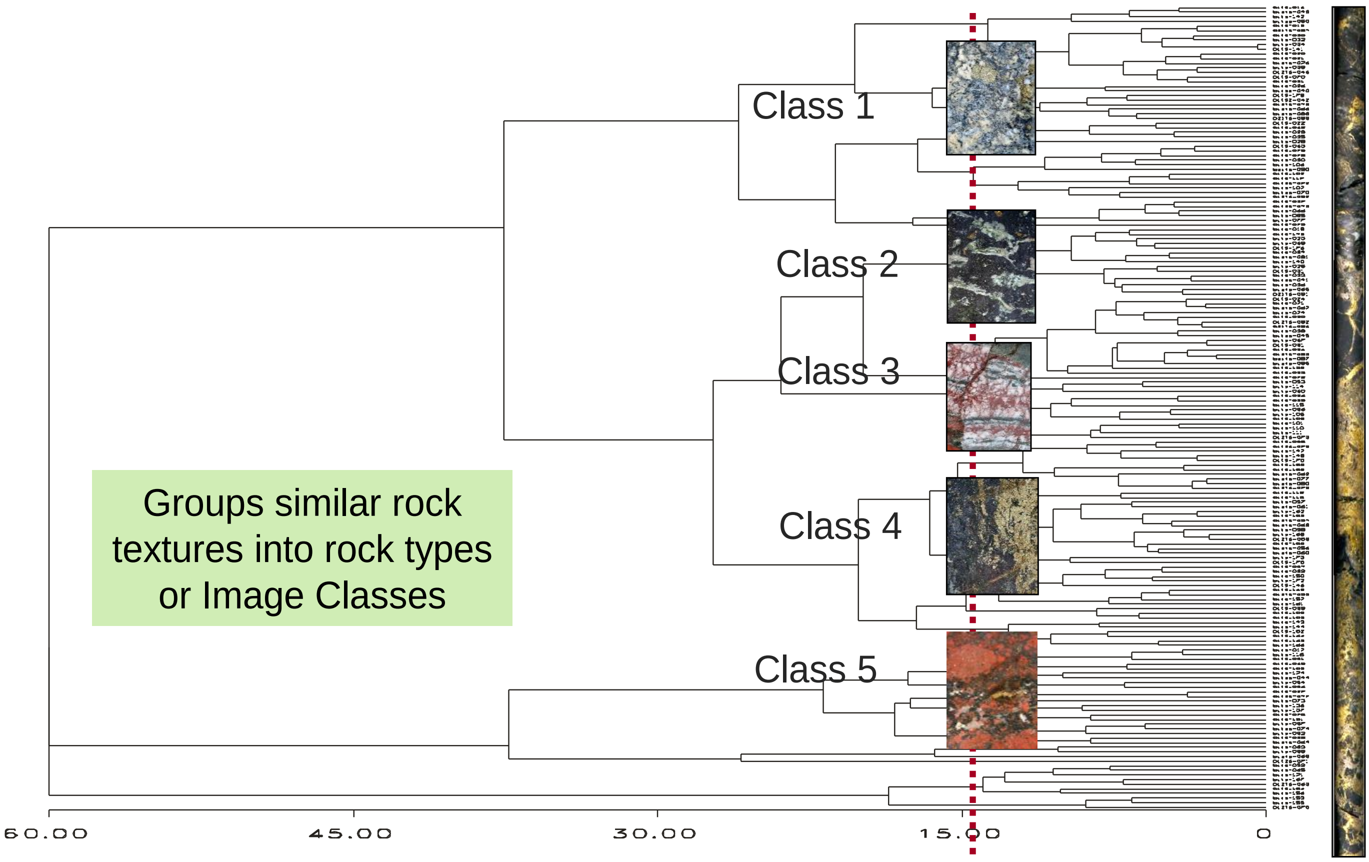


Figure 37. Grouping of similar looking rock types from drill core imaging
(Image: Khoi Nguyen, Ron Berry, Copyright: AMIRA)

In some examples (lithology based specific cases where the signature rock looked very different), the signatures of process behaviour can be diagnosed by proxy (with less precision). This represents a very fast and early diagnosis of the core for previously understood signatures. This method also provides a possible link between methods like automated mineralogy, optical microscopy and larger scale methods of characterization like geophysics or chemical assays.

3.3 Hyperspectral characterization

Hyperspectral imaging is one of these methods and shows great potential for future work. Hyperspectral imaging, like other spectral imaging, collects and processes information from across the electromagnetic spectrum. The goal of hyperspectral imaging is to obtain the spectrum for each pixel in the acquisition area, with the purpose of finding objects and identifying mineralogy, or simply elemental information. The human eye sees colour of visible light in mostly three bands (red, green, and blue), spectral imaging divides the spectrum into many more bands. This technique of dividing images into bands can be extended beyond the visible.

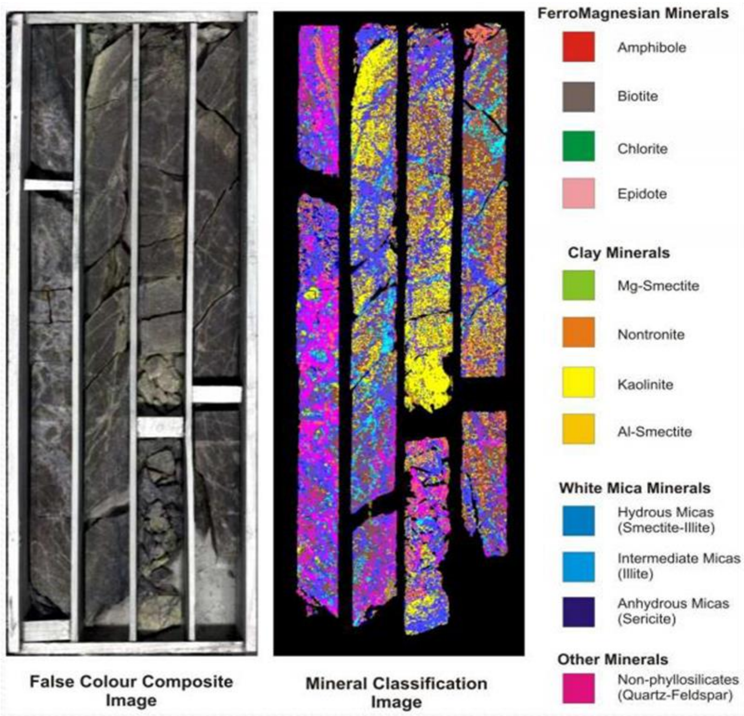


Figure 38 Hyper spectral imaging of drill core
(Source: AMIRA 2009 & GeoTek https://www.geotek.co.uk/sensors/hyperspectral/)

Hyperspectral image characterization provides a unique opportunity, again if the budget and scope allow. This technique method is considered to be the tool of the next generation geologist. The kind of precision of diagnosis of minerals and rock weathering normally only was possible by an experience geologist logging the core. Now, this method is able to provide repeatable measurement. While this method is not able to pick up many of the subtitles of mineral diagnosis that only a geologist can observe, this form of characterization could train an inexperienced geologist into an experienced one. Figure 39 shows an example of this with data laid out. Each tray of core (in a continuous form in the same drill hole) is show as a colored (characterized to mineral type) horizon line across each column. Thus, each column shows many cores tray in a continuous form vertically. Figure 205 shows a continuous 290m length of core, in 73 trays of core, with each tray a 4m section. The Hyperspectral instrument measured each tray (4m of core) in a single measurement pass. Figure 49 shows 10 different spectra on the same 290m length of core (and one column with the original grey scale image). Each tray of core (4m length) is represented by small colored line. The lines are then stacked as a function of depth. Each column represents a different spectrum.

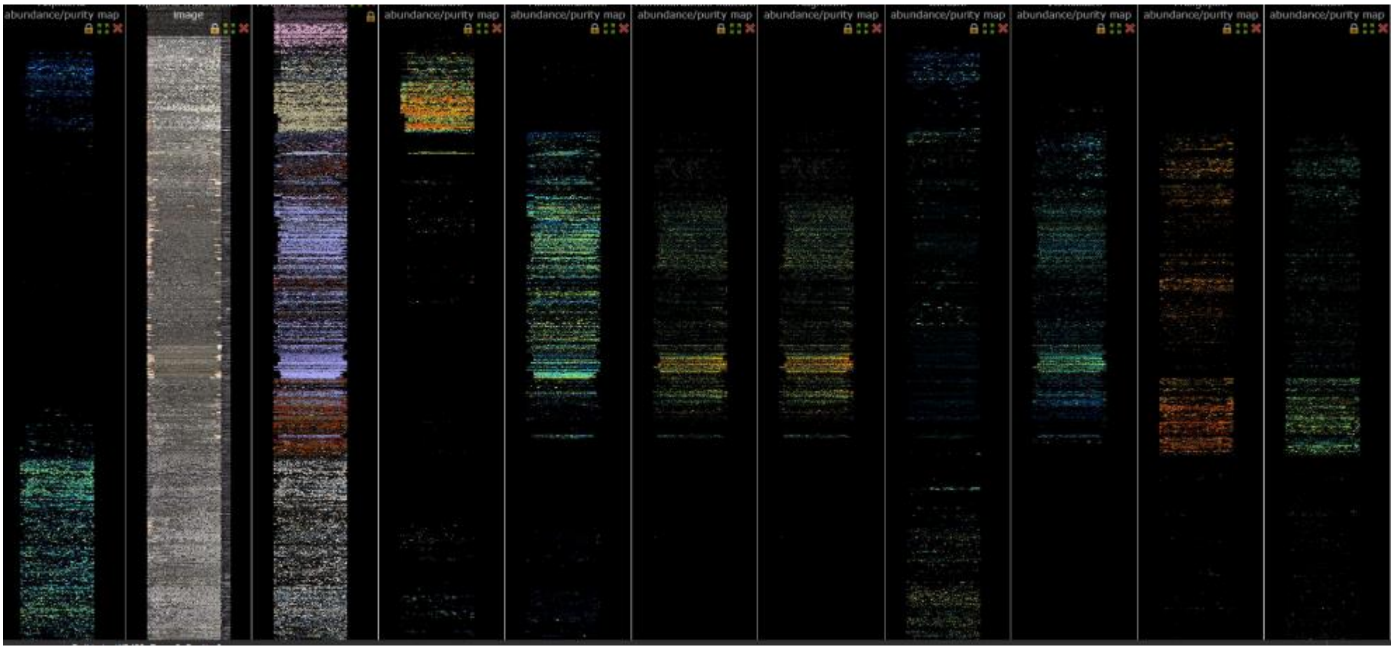


Figure 39. Hyperspectral imaging of kilometres of continuous drill core
(Image: Simon Michaux)

Clear domains can be seen. The challenge is then to understand what the domains mean and what they relate to. The target process behaviour may be related to only some of these signatures and perhaps not directly. Careful and competent analysis is required to assess this sophisticated tool.

3.4 Bulk mineralogy characterization

There are many occasions in which measurement of the bulk mineralogy is useful. In particular, the characterization (mineral phases e.g. quartz) and proportions of gangue mineralogy is useful in understanding process behavior. In mining, gangue is the often commercially worthless material that surrounds, or is closely mixed with, a wanted mineral in an ore deposit. It is thus distinct from overburden, which is the waste rock or materials overlying an ore or mineral body that are displaced during mining without being processed, and from tailings, which is rock already stripped of valuable minerals; a worthless rock containing valuable material. There are several low cost and effective methods in achieving this.

3.4.1 Quantitative X-Ray Diffraction (QXRD)

Understanding bulk mineralogy of the sample and deposit itself is a necessary step. XRD is a versatile analytical method to analyze material properties like phase composition and proportions of a powder sample. Identification of the phase s is achieved by comparing the X-ray diffraction pattern obtained from the sample being measured with a reference database. With one single measurement one can identify mineral phases (qualitative) and their concentrations/proportions (Quantitative). X-rays are generated in a laboratory diffractometer using x-ray tubes with a suitable anode material (Cu, Fe, etc).

For example, XRD can distinguish between a sample that contains Fe_2O_3 and Fe_3O_4. It can also distinguish between minerals like calcite, aragonite and vaterite (Chang-Zhong et al 2015, Ruffell & Wiltshire 2004, Tammishetti et al 2015, Potts 1987 and Rollinson 1993).

XRD measurement requires a sample is of crystalline nature and is pulverized and homogenized, pressed in a pellet or back loaded in a sample holder. A powder diffractogram is obtained by counting the detected intensity as a function of the angle between incoming and diffracted x-ray beam. In the diffractogram clay minerals have their distinguished diffracted peaks at very low 2theta angles and analysis of clays required good low angle performance and resolution on the XRD equipment. (FWHM = Full Width Half Maximum).

Amorphous minerals, i.e. sample not having any crystals = lattice plains are particularly difficult to measure and does require expert sample preparation, internal amorphous standards and knowledge to analyze and process the data.

3.4.2 X-ray Fluorescence measurement (XRF)

XRF is an elemental analysis and can chemical elements are present and what are their concentrations. For example the sample contains iron (Fe) and calcium (Ca) in a measured proportion. An X-ray fluorescence (XRF) spectrometer is an x-ray instrument used for routine, relatively nondestructive chemical analyses of rocks, minerals, sediments and fluids. In a laboratory X-ray fluorescence (XRF) spectrometer the emission of characteristic "secondary" (or fluorescent) X-rays from a material that has been excited by being bombarded with high-energy X-rays (Fitton 1997, Potts 1987 and Rollinson 1993).

It works on wavelength-dispersive spectroscopic principles that are similar to an electron microprobe (EPMA). However, an XRF cannot generally make analyses at the small spot sizes typical of EPMA work (2-5 microns), so it is typically used for bulk analyses of larger fractions of geological materials. The relative ease and low cost of sample preparation, and the stability and ease of use of x-ray spectrometers make this one of the most widely used methods for analysis of major, minor and trace elements in rocks, minerals, liquids, solids, oils and sediment.

Virtually any solid or liquid material can be analyzed with and EDXRF or WDXRF spectrometer. There are “standardless analysis software’s” available commercially which can be used when analyzing samples. Using of such a software give good indication of the elements and their concentrations. The best results one can obtain with matrix specific calibrations. For rocks and minerals, typical commercial instruments require a sample constituting at least several grams of material, pressed pellet between 15-20 grams, although the sample collected may be much larger. For XRF chemical analyses of rocks, samples are collected that are several times larger than the largest size grain or particle in the rock. This initial sample then suffers a series of crushing steps to reduce it to an average grain size of a few millimeters to a centimeter, when it can be reduced by splitting to a small representative sample of a few tens to hundreds of grams. This small sample split is then ground into a fine powder by any of a variety of techniques to create the XRF sample. Care must be taken particularly at this step to be aware of the composition of the crushing implements, which will inevitably contaminate the sample to some extent. Ideally powdered sample has grain size at 10µm scale but best results are obtained using glass beads where all grains have been melted in a flux. Sample size for a compressed powder pellet XRF measurement is 10 to 50 grams, but less can be used depending on mineralogical circumstance.

3.4.3 Hand Held XRF measurement

Portable XRF analyzers are used around the world for monitoring ore grade in active open pit and underground mines. Specifically, XRF analyzers are used for direct analysis of blast hole cuttings in mines, as a preliminary guide on direct rock faces underground, open pit faces, and within Ni mine labs after sample preparation. Advantages of using a hand held unit could include:

- Rapid indication of potential Au mineralization through XRF analysis of pathfinder elements in soil, drill cuttings, and drill cores
- Priority sample selection, maximization of analytical budgets, and improved drill target generation by prescreening samples with XRF
- Improved ore deposit understanding, modelling, and vectoring with less dilution and better Au recovery via XRF mapping of structural features and identification of mineralized and altered zones
- Inexpensive and rapid rock typing by using XRF for lithogeochemistry

Figure 40. Hand held XRF unit being used on drill core
(Image copyright: GTK)

3.5 Characterization of Micro-texture

Ore micro-texture is the mineral assembly form at the mineral grain scale (100µm). Understanding what form the ore's micro-texture takes can be the solution to why a particular process behavior responds the way it does. A flotation bubble will interact with a particle, which would be made up a portion of micro-texture. Of particular interest to flotation are the following basic types of textures. Starting at the most basic level (Figure 41 LHS), an ore texture can be evaluated as to whether it is equigranular or inequigranular. At the most fundamental level, ores can be considered either equigranular (all grains are the same size) or inequigranular (not all the same size). Mineral grain size is defined by the form of mineralization (Figure 41 RHS).

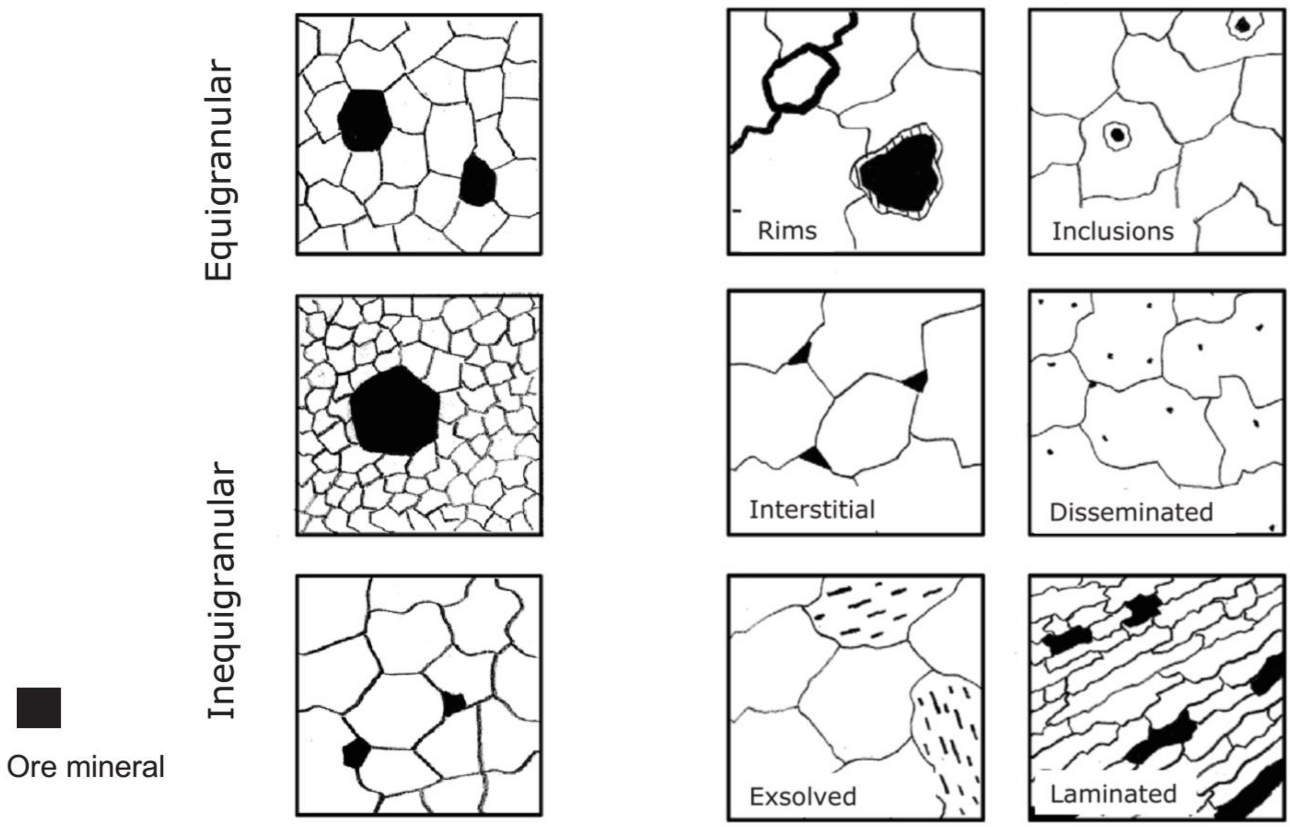


Figure 41. Basic ore textures, at a grain size scale (LHS), textures that influence flotation performance (RHS) (Source. Butcher 2019, Image: Alan R Butcher)

It is also important to understand the formation process of the ore texture in context of geological events that have produced the final outcome. Figure 42 illustrates some of the geological processes that can change rock texture. Possible changes of the three pristine ore textures is considered (one equigranular and two inequigranular – Figure 41 LHS), which undergo progressively more aggressive modification from left to right (oxidation, hydrothermal alteration to metamorphism).

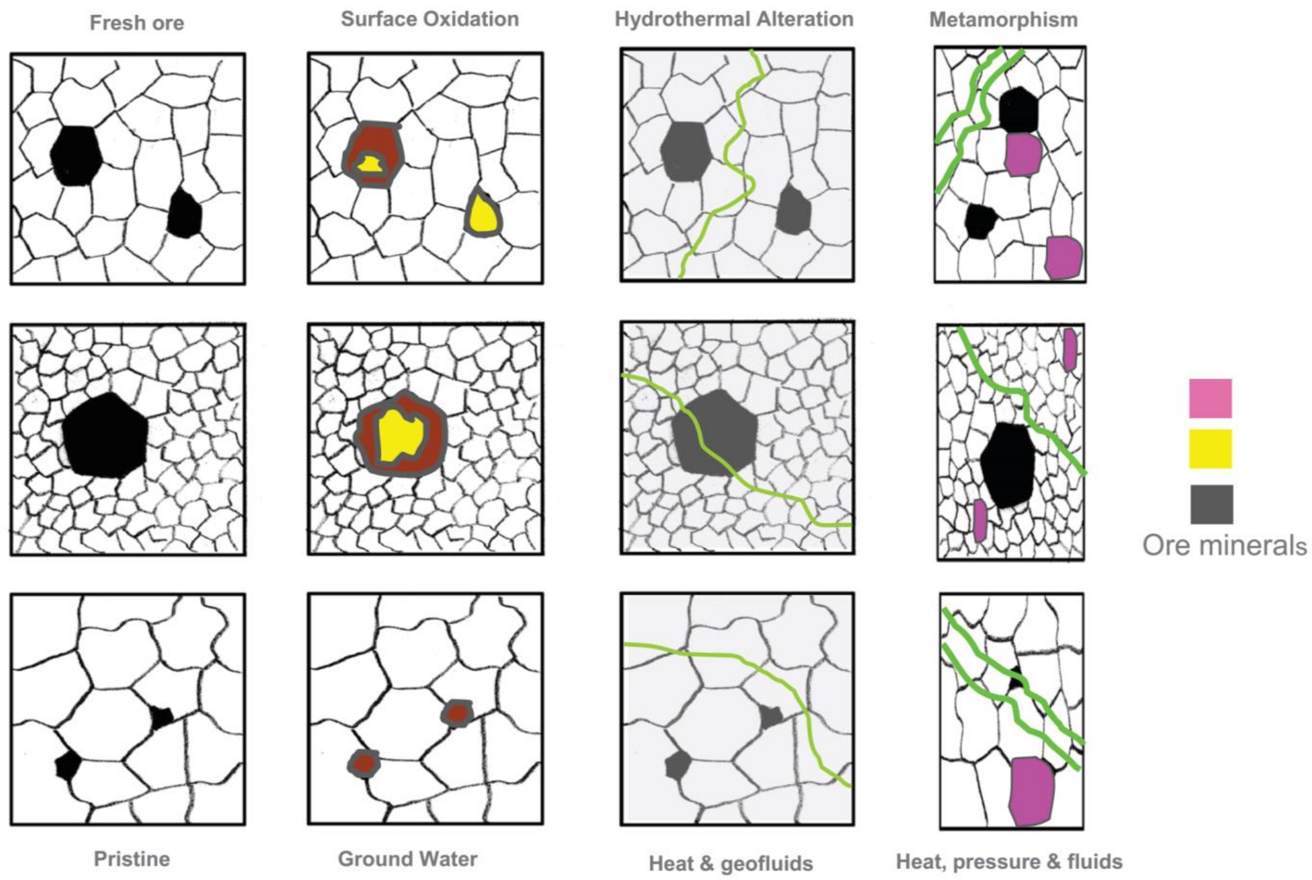


Figure 42. Basic ore textures, at a grain size scale (LHS), textures that influence flotation performance (RHS) (Source. Butcher 2019, Image: Alan R Butcher)

To characterize a rock texture, a mineralogist would examine the following (Butcher 2019):

- complete inventory of all known mineral phases present
- detailed modal analysis for major minerals (>5%), minor phases (>1% to
- <5%) and trace phases (<%)
- textural information on both gangue and ore minerals, which includes:
  - grain size estimates(used to determine optimum liberation grain sizes and predict or prevent problems such as poor liberation, over-grinding of the valuables, production of slimes)
  - mineral associations(important for optimizing the separation process, say galena from sphalerite, chalcopyrite from sphalerite)
  - grain boundary relationships(curved, straight or irregular boundaries will affect behavior during crushing and grinding as this imparts preferential breakage mechanisms – for example along or across boundaries)
  - elemental deportment information (important for tracking how metals behave during processing
  - photomicrographs of typical textures.

Figure 43 shows examples of ore microtextures.

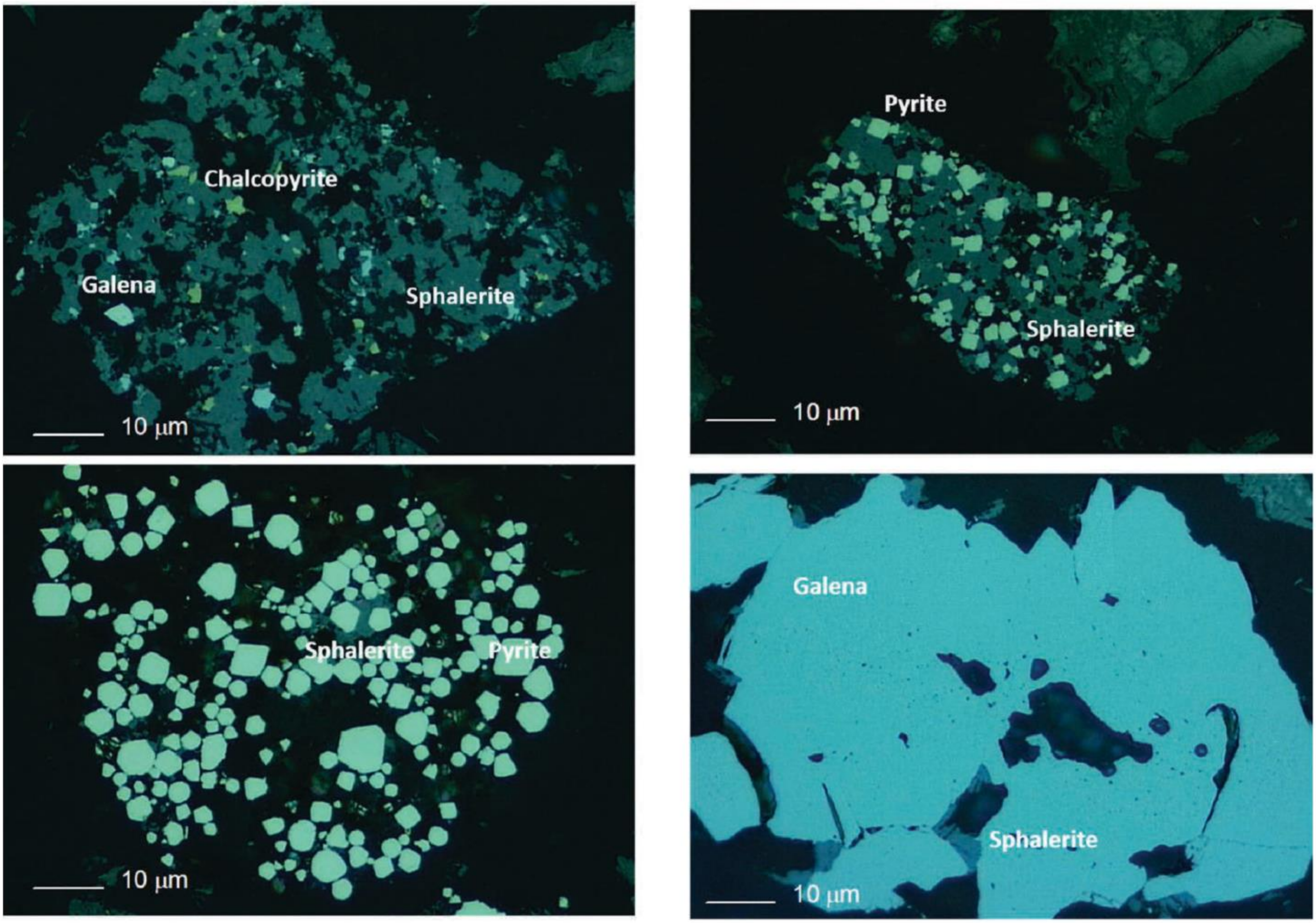


Figure 43. Examples of different ore micro-textures in optical photomicrographs, taken using a digital camera attached to a reflected polarising petrographic microscope, of processed particles from an Australian lead–zinc operation (McArthur River), which contain textures that historically would have been too fine for conventional flotation, and now require combinations of staged and fine-grinding methods to separate the galena and sphalerite. (Source. Butcher 2019, Image: Alan R Butcher)

Figure 44 shows the mineral mapping of three particles with the chalcopyrite colored blue.

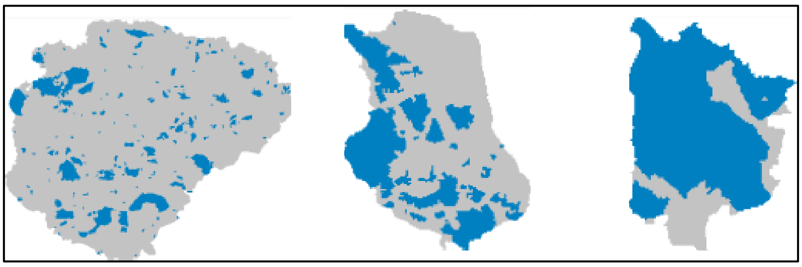


Figure 44 Chalcopyrite micro-texture of three very different forms
(Image. François Vos)

If there were three separate samples, with most of the particles in each sample were characterized by one of the images in Figure 44, the process response would be very different for all three samples. The flotation or leaching recovery of each of the above particles would be quite different. Measuring micro-texture in this form allows deterministic conclusions to be drawn which could be used to guide the rest of the geometallurgical campaign.

Figure 44 was generated using automated mineralogy (TIMA/MLA etc.). Automated mineralogy as a characterization tool has advanced considerably. Reliable instrumentation, continuously updated software capabilities and faster data acquisition. Samples are mounted into polished resin blocks and are mapped using a scanning electron microscope (SEM). This method can be good for mapping micro-textures, but it is not as effective for gangue mineralogy in some cases, or with mineralogy with very similar back scattered electron grey scales. However, it is best practice to use complimentary analysis to establish good mineral chemistry with probe work and LA-ICP-MS or Raman, depending on the MOI (Hrstka *et al* 2018, Aylmore *et al* 2018, and Anderson *et al* 2014).

Figure 45. In the foreground is the new FEI Quanta 650 field emission scanning electron microscope, in the background is an older MLA equipment. (Image and copyright: GTK)

As gangue mineralogy is also of interest to many geometallurgical analytical questions, automated optical microscopy is useful. This is a cheaper and faster method compared to automated mineralogy, but does require expert knowledge and a well-trained user. Figure 36 is an example of a desktop optical microscopy with a resin block and a second image using cross-polarized light (thin section), clearly exhibiting gangue mineralogy in an array of reflected colors.

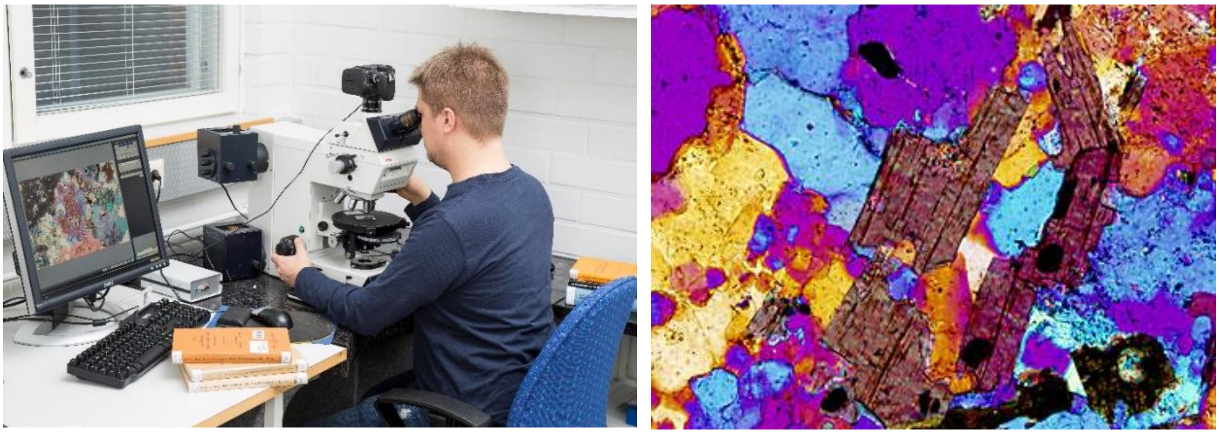


Figure 46. Optical microscopy. Picture shown is an Optical Microscope Nikon MM 40
(Image: GTK)

4 CONVENTIONAL PROCESS CHARACTERIZATION TESTS FOR PROCESS BEHAVIOR

Multiple tests of the target process behavior should be carried out for each of the end member samples. Larger scale tests that are trusted by industry in feasibility studies are referred to as 'bankable' tests. These tests are considered the industry standard when characterizing ore for process engineering design. Table 2 below shows a list of these tests and reference for each for further examination.

Table 2 Larger scale 'bankable' ore characterization tests for process engineering

Process Engineering Field	Process Behaviour Characterised	Test	Reference
Geotechnical	Compressive Strength	UCS	Brady & Brown 2006
	Tensile Strength	Brazilian Disc	Brady & Brown 2006
	Fracture Toughness	KiC	Brady & Brown 2006
Comminution	Impact Breakage	Drop Weight Test (DWT)	Napier-Munn *et al* 1996
	Bed Breakage	Lab Scale High Pressure Grinding Roll (HPGR)	Wills & Napier-Munn 2005
	Grinding	Bond Ball Mill Work Index (BMWi)	Napier-Munn *et al* 1996
		Bond Rod Mill Work Index	Napier-Munn *et al* 1996
	Fine Grinding	Laboratory Scale Isa Mill	Wills & Napier-Munn 2005
	Abrasion	Bond Abrasion Index (Allis Chalmers Abrasion Test)	Wills & Napier-Munn 2005
Flotation	Flotation Recovery	Batch Flotation Test	Runge 2010
Hydrometallurgy Leaching	Leaching Recovery	Column Leach Test	
Acid Mine Drainage	Acid Mine Drainage (AMD)	Static & Kinetic Tests	Parbhakar-Fox 2013, 2015, 2016

The outcomes of tests are used as characterization inputs into process engineering. The geometallurgy campaign objectives are to understand the mineralogy that controls process behavior response and then map variability of that process response. This means that the feed samples mineralogy is required to be known and consistent throughout the sample volume. Each of these larger scale bankable tests requires a relatively large amount of mass. Many past conventional metallurgical test work campaigns have not been that precise in ensuring the sample origins are consistent and understood in terms of source mineralogy. Sampling large masses like what is required for the tests in Table 2 from drill core often meant that the final sample was from mixed mineral sources.

4.1 Blasting in Geometallurgy

The prediction of blast induced fragmentation has usually been done with a combination of empirical models. What happens in the blast defines and influences the efficiency of every downstream process.

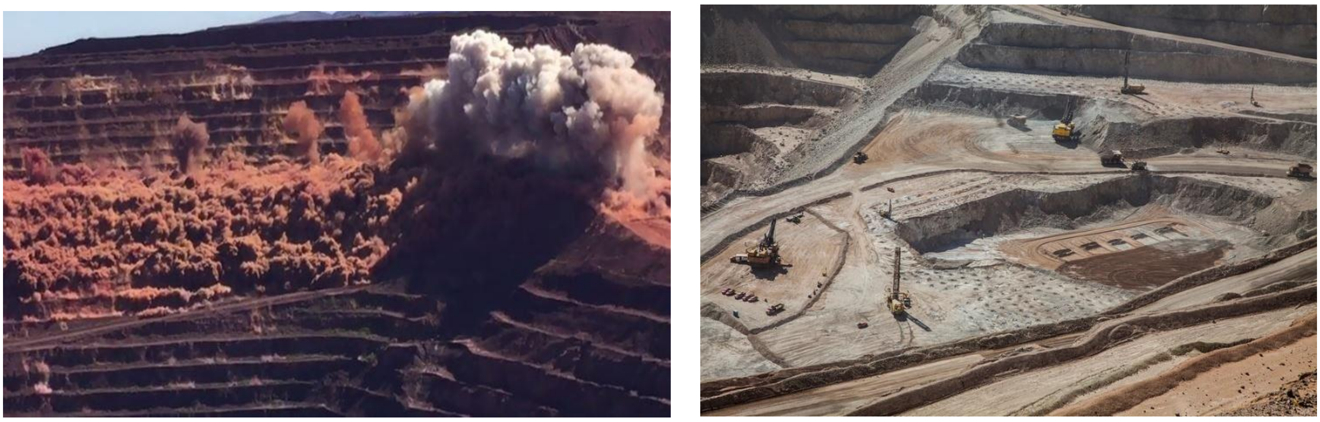


Figure 47. A production scale blast in a mine
(Images: Codelco Flickr)

There are several blast fragmentation models, which are used as circumstance dictates, to create a composite size distribution. The Kuz-Ram model predicts the coarse end of the size distribution of fragmentation produced with reasonable accuracy and in a practical manner. This model is based on an empirically derived equation that predicts mean fragment size from powder factor, explosive mass per blasthole, relative weight strength of explosive and rock blastability constant (Kuznetsov, 1979; Cunnigham, 1983 & 1987). The fine end of the fragmentation size distribution is predicted using one of several fines (fine fragmentation less than 1mm) estimation models, used by the JKMRC (Djordjevic 1999, Kanchibotla & Valery 1999). These models use the following inputs to predict how a volume (termed average in this section) of rock will fragment in the applied blast:

Rock Inputs
•UCS strength
•Tensile Strength
•Youngs Modulus
•Rock Density
•P-wave Velocity

Design Parameters
•Burden x Spacing
•Hole length
•Charge length
•Stemming length
•Inter hole timing

Explosive Inputs
•Explosive Density
•Explosive VoD
•Relative Weight Strength

In 2010, a method was developed to predict blast fragmentation (an approximate estimate only) from drill core (AMIRA 2010b and AMIRA 2010c). This method was developed by the W.H. Bryan Mining Centre (University of Queensland) in to the iFrag software. A drill that has been characterized appropriately (fracture frequency and well log geophysics parameters) that would pass through a volume of rock that later would become a production scale blast bench is selected. The drill hole is then modelled like an explosive borehole and fragmentation around it is estimated in a 1m interval layered fashion, where zones of fragmentation are modelled (as in Figure 48). The explosive design and explosive products used is kept the same to demonstrate the difference in rock type. From this model, a crushed envelope and a breakage envelop around the drill hole is estimated (Figure 49). These results are then scaled up to a volume of a production scale bench. It is to be remembered that this approach was designed to be done on drill core during a pre-feasibility study, well before a 'bench' is assessable to characterize let alone blast.

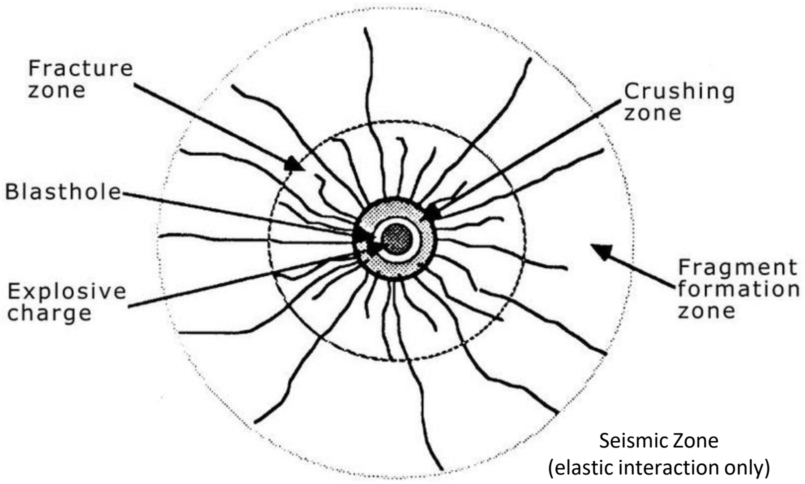


Figure 48: Zones of Fragmentation around an explosive blast hole (plan view)
(Source: Michaux 2006)

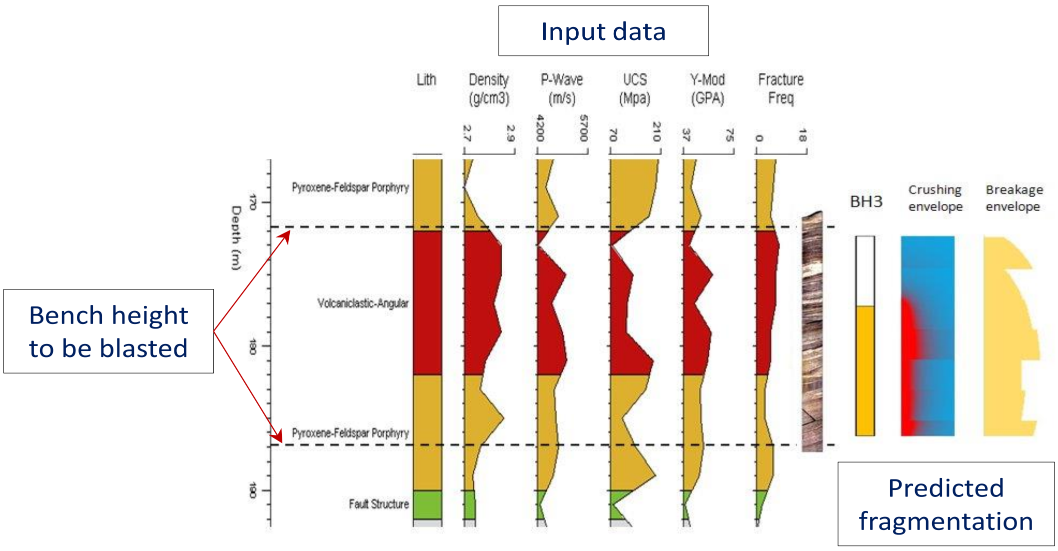


Figure 49. Layered composite fragmentation of BH3 prediction from drill core using iFrag
(Source and copyright -AMIRA 2010b, Italo Onederra)

The model inputs into the iFrag software are:

Rock Inputs
•Tensile Strength (from EQUOtip)
•Young's Modulus
•Rock Density
•P-wave Velocity
•Fracture frequency (sliding window of 10m)

Explosive Inputs
•Static

Design Parameters
•Static

To compare the two approaches, a drill hole was selected from the client database (Case Study R) and the relevant data was collected. A section of this drill hole was used for the comparison. The drill holes was segmented into four 10m intervals to simulate 4 blasted benches, stacked on top of each other (where the third bench BH3 is shown in Figure 50). The fragmentation around the drill hole in each virtual bench is then estimated using the layered composite iFrag model (shown in Figure 51 as 'Composite' for each BH).

The same volume of rock is used to predict the fragmentation subject to the same explosive blast design but using the conventional Kuz-Ram model (Cunningham 1987) and the crush zone model (Kanchibotla and Valery 1997). This is shown in Figure 51 as 'Average' for each BH.

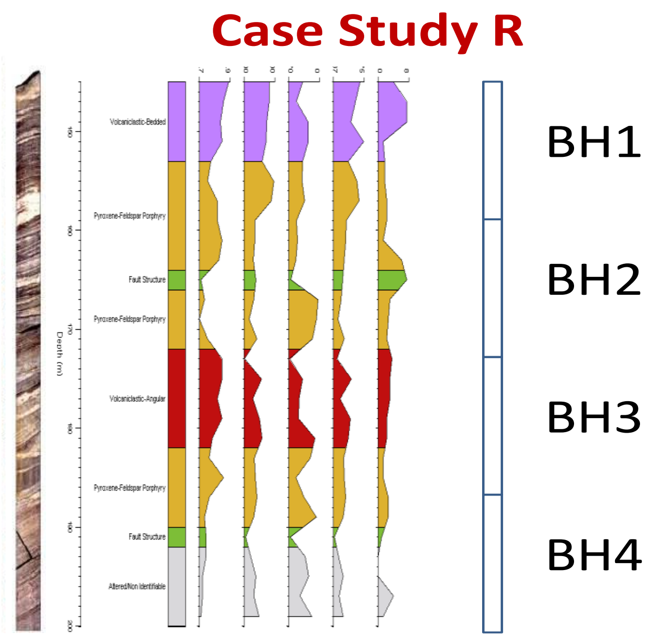


Figure 50. Blast induced fragmentation (Source and copyright: AMIRA 2010b, Italo Onederra)

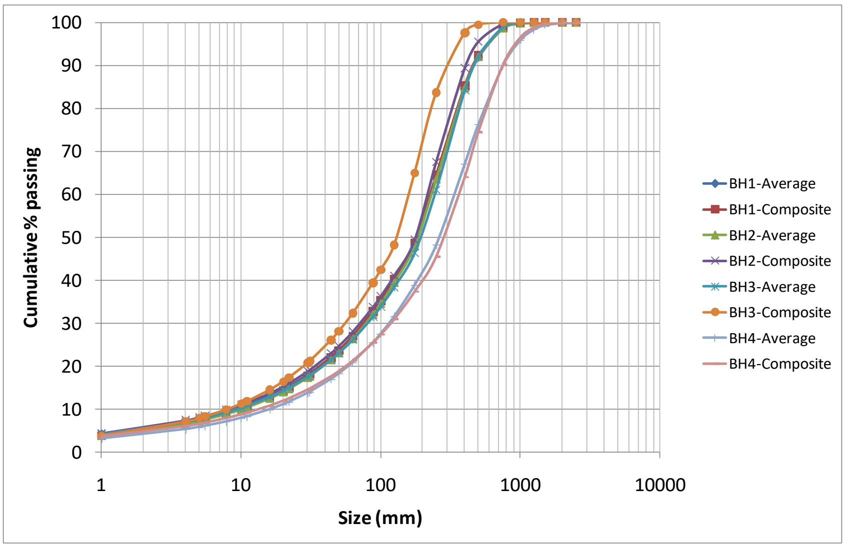


Figure 51. Comparison of the layered composite fragmentation prediction from drill core to the conventional average volume approach using Kuz-Ram and CZM (Source and copyright: AMIRA 2010c, Italo Onederra)

This section is to show that it is possible to apply geometallurgy to blasting problem solving and that it can be used in systems based operations prediction if the input data is available. Whether this is a worthwhile exercise has to be considered in context of the overall aims of the geometallurgical campaign. In practice, the primary crusher can break down any larger oversize fragments reasonably easily and at relatively low power cost. The fines produced in blasting though would heavily influence all downstream processes.

4.2 JKMRC Drop Weight Impact Breakage Test

The Drop Test was developed by the JKMRC (Julius Kruttschnitt Mineral Research Centre) to model the rock breakage process in comminution machines. JKMRC comminution models are based on two sets of parameters.

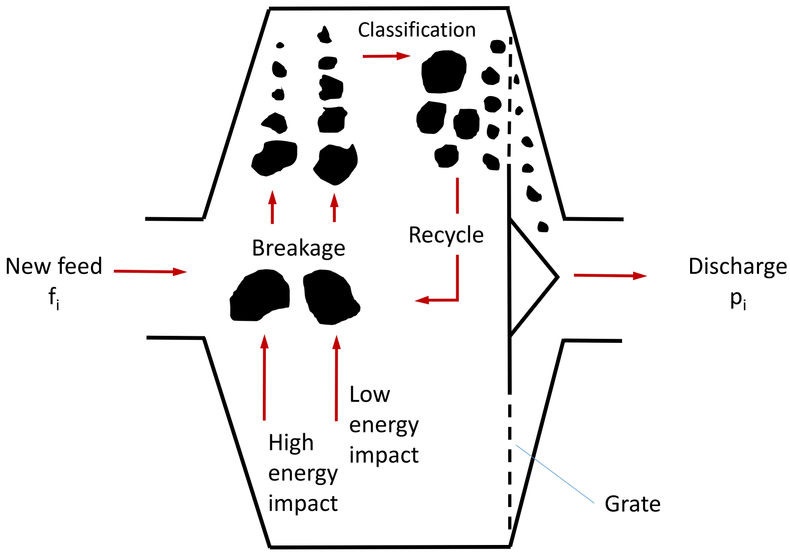


Figure 52. Schematic diagram of AG/SAG mill process mechanisms
(Source: Napier-Munn *et al* 1996, copyright: JKMRC)

As the JKMRC methods aspired to model the ore characteristics separately from the processing machine characteristics, these parameters were to be measured independently. To be useful, these parameters needed to be experimentally measured in a way that the simulation models can exploit. The Drop Test has been the most successful way to characterize the high energy impact of mineralized ore this to date.

The principle behind the JKMRC Drop Weight Tester is simple but very effective. A known weight is dropped from a known height onto a single particle of a known size. Thus a known energy is applied to a single particle. This is achieved with the Drop Tester (Figure 53).

The objective of the Drop Test is to characterize the resistance to size reduction an ore has over a range of applied energies by measuring an energy breakage curve used for comminution engineering. Individual breakage events (points on the energy breakage curve) can be expressed as a specific comminution energy level, Ecs (kWh/t), or as breakage or appearance functions. The principle of the drop breakage process is well known (Napier-Munn et al, 1996). The full Drop Weight Test (DWT) procedure is described as follows. Five different particle sizes are prepared into groups of relevant sizes to provide a representative sample mass to be measured. These groups of particles are then broken individually at known energies (defined by a known mass dropping from a known height onto a particle of known mass), which are then collected and the size distribution is measured. A DWT sample has a starting mass of approximately 200kg.

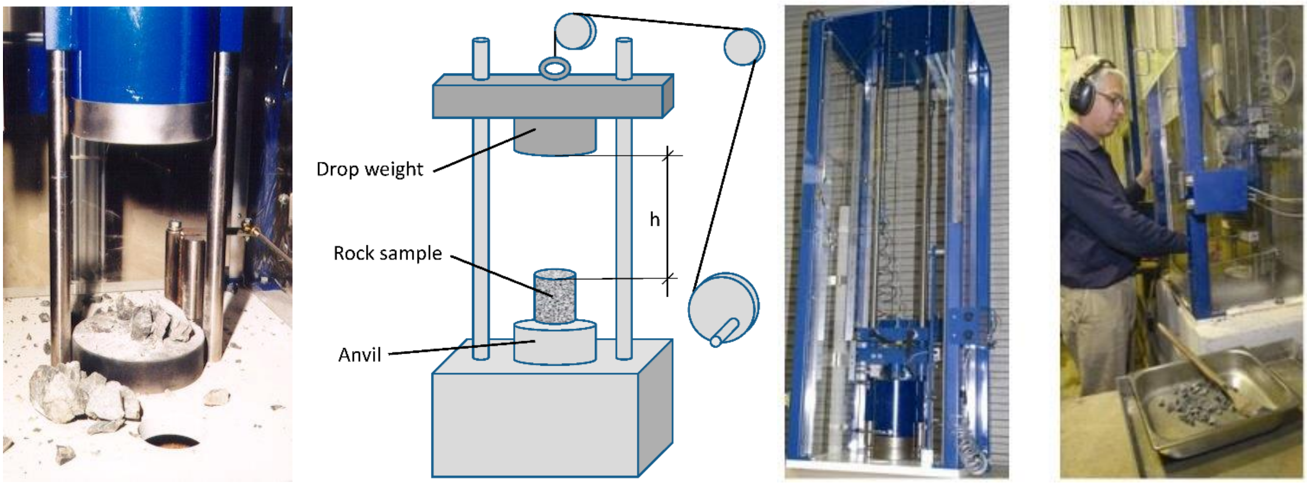


Figure 53 The JKMRC Drop Weight Test
(Source and image copyright: JKTech, https://jktech.com.au/)

After breakage, each of the samples are sieved to measure the size distribution. The t_{10} of each size distribution is calculated where the t_{10} is the percent passing $1/10^{th}$ of the initial mean particle size (calculated with a spline function).

Product size distribution Relative size distribution showing t markers

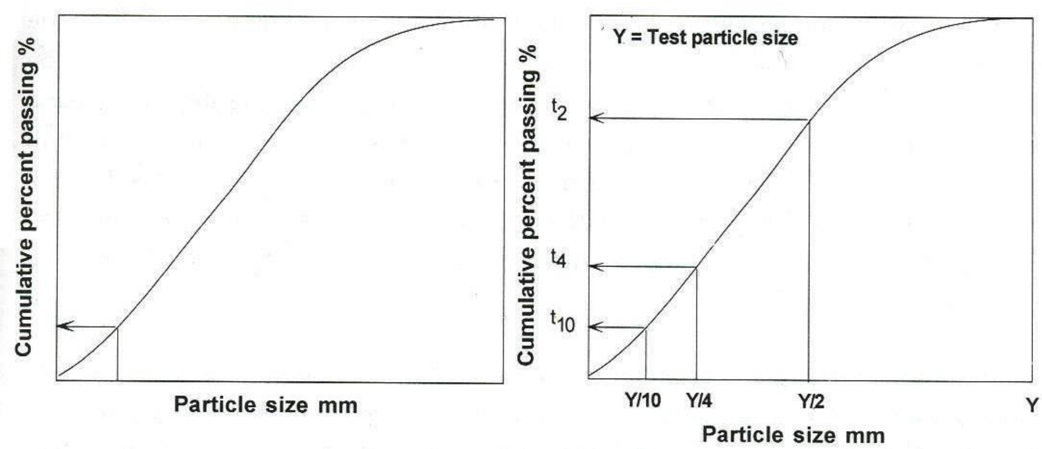


Figure 54. Calculation of t_{10} from a size distribution
(Source: Napier-Munn *et al* 1996, Copyright: JKMRC)

The parameter of t_{10} is significant as it can be used to describe the whole family of t curves from t_2 (1/2 of the mean initial size) to t_{75} ($1/75^{th}$ of the mean initial size), shown in Figure 55.

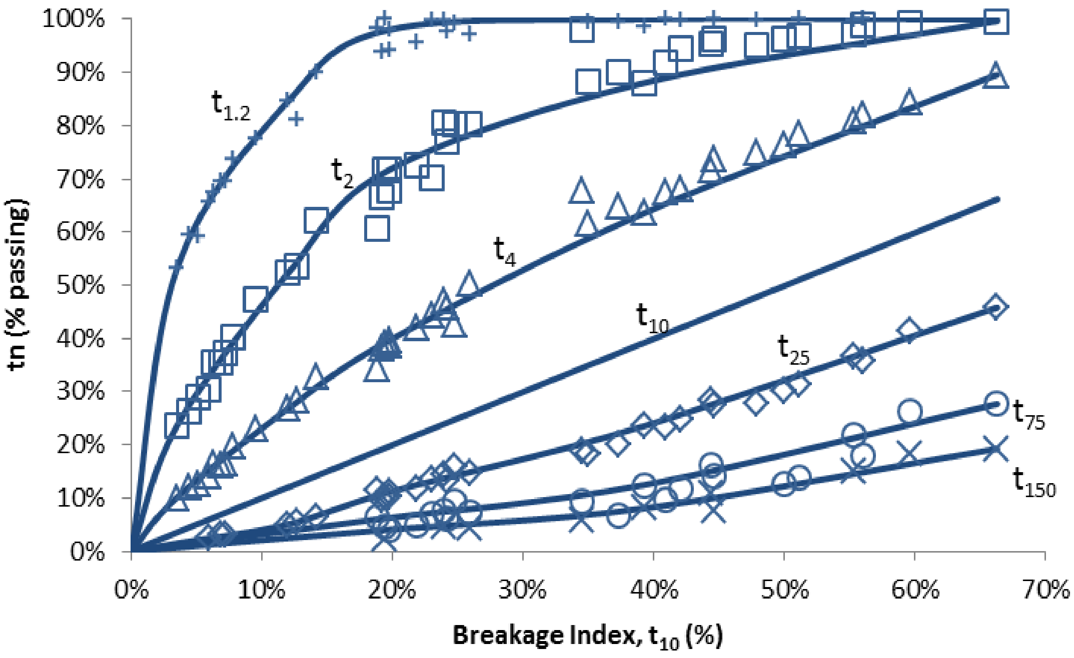
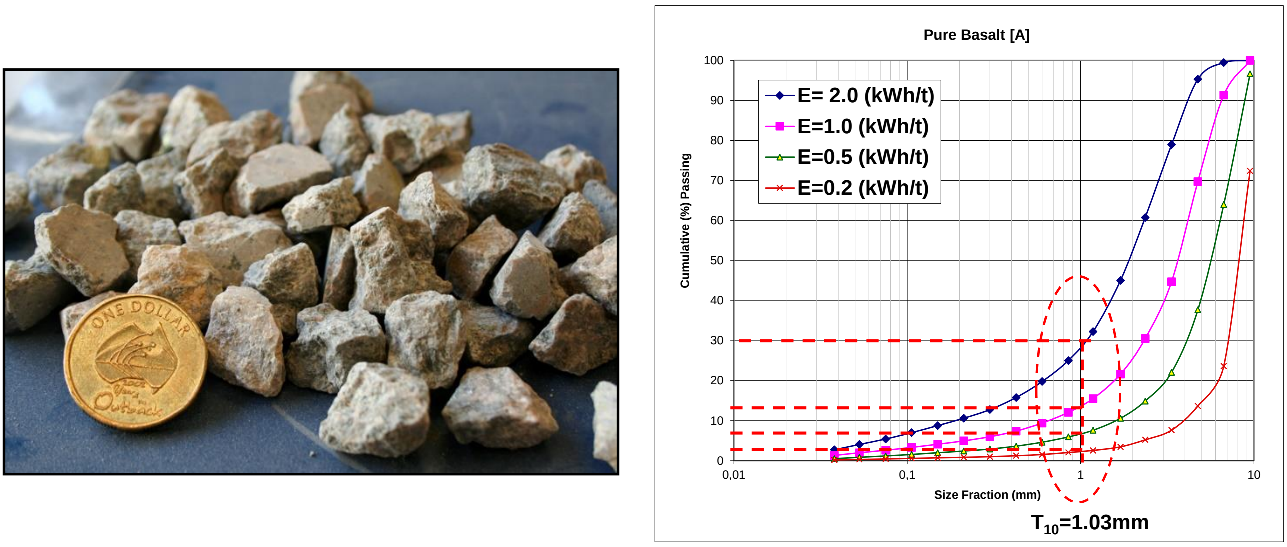


Figure 55. The t family of curves and degree of breakage, t_{10}
(Source: Napier-Munn *et al* 1996, Copyright: JKMRC)

Figure 56 shows an example, a basalt sample of 50 particles of size fraction -11.2+9.5mm would have a geometric mean size of 10.35mm. This in turn would give a t_{10} of 1.035mm. If these 50 particles were broken in the manner shown in Figure 56, the percentage volume of the sample that is then below 1.035mm is the t_{10} value.



Corrected E (kWh/t)	2.0	1.0	0.5	0.2
T_{10} (mm)	28.90	13.95	6.85	2.35

Figure 56. Size distribution of 4 basalt samples broken at 4 different energies
(Image: Simon Michaux)

In the example shown in Figure 56, the sample broken at 0.2 kWh/t has 4.4% of the sample volume broken to a degree finer than the t_{10} size of 1.035mm. Alternatively, the sample broken at 2 kWh/t has 38.1% of the sample volume broken to a degree finer than the t_{10} size of 1.035mm. This procedure is performed for all four breakage energies and the energy breakage curve is fitted.

Ecs (units of kWh/t) is the applied specific comminution energy, and A and b are the ore impact breakage parameters (Napier-Munn et al, 1996). The parameters A and b are calculated by fitting the above equation to the t_{10} and Ecs data, to form the energy breakage curve (Figure 57 and 58).

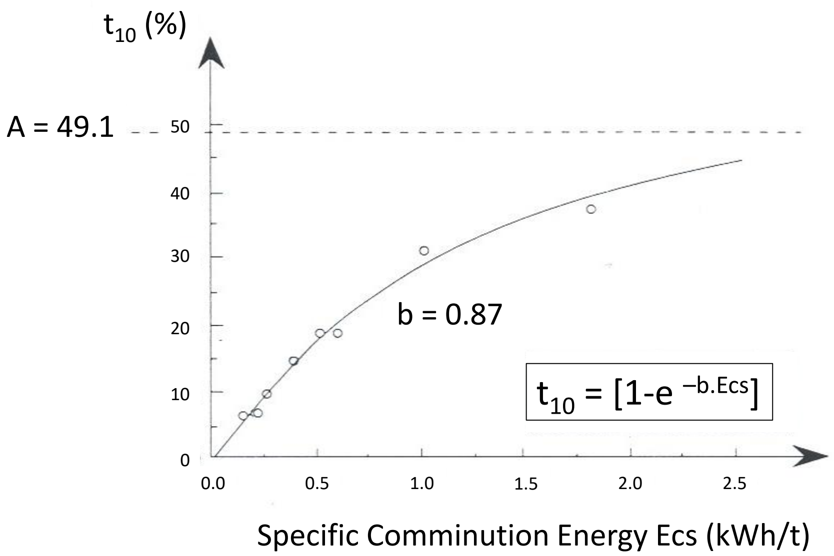


Figure 57. The energy breakage curve shown mathematically
(Source: Napier-Munn *et al* 1996, Copyright: JKMRC)

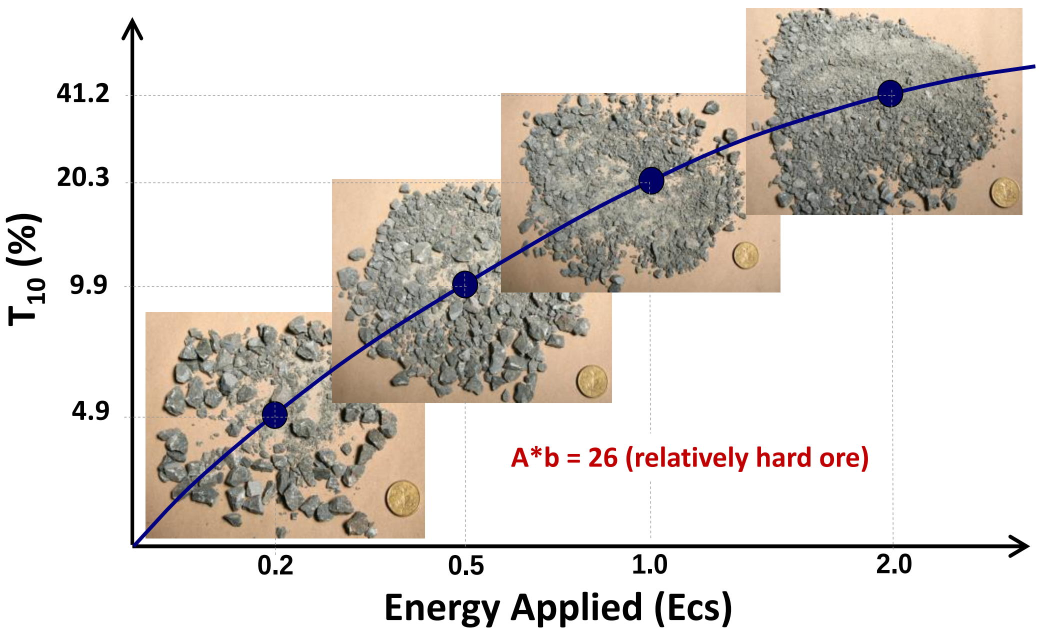


Figure 58. The energy breakage curve shown graphically
(Image: Simon Michaux, copyright: AMIRA)

The energy breakage curve is influenced by the size of the fragments tested (Figure 48). For this reason the full Drop Test examines five size fractions ranging from 63mm down to 13.2mm. If the objective is to characterize the ore to design a production scale mill, then the full Drop Test would be required to capture this spread of curves across the particle sizes.

Thus the t_{10} parameter can be used to describe the breakage phenomena size distribution as a single number. The amount of breakage, or breakage index, t_{10} is related to the specific comminution energy with the formula:

$$t_{10} = A[1-e^{(-b.Ecs)}] \quad \text{Equation 1}$$

The parameter A is the t_{10} asymptote sill or maximum of the breakage curve (Figure 59). It indicates the natural breakage limit of the rock that would not be exceeded no matter how much energy was used to break it. The parameter b represents the shape of the energy breakage curve as it approaches the asymptotic maximum (Figure 60). It represents how quickly the rock reaches this maximum.

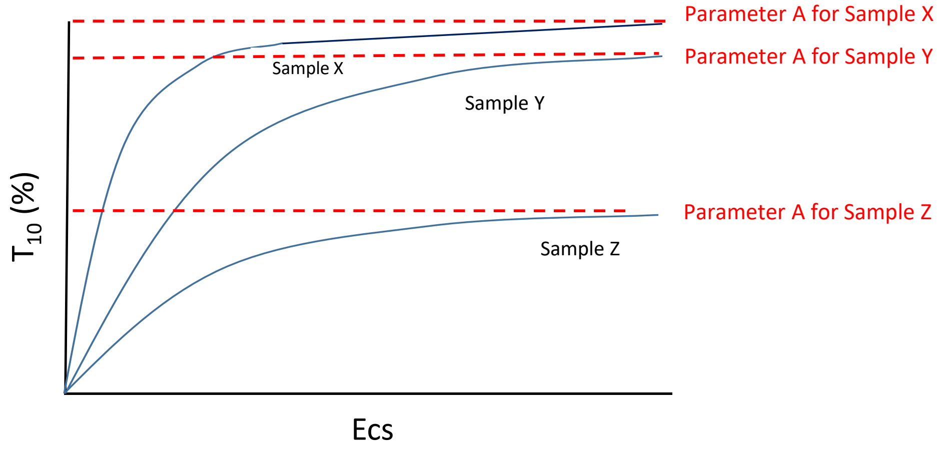


Figure 59. The A parameter in the JKMRC Axb impact breakage characterization method - conceptual example (Image: Simon Michaux)

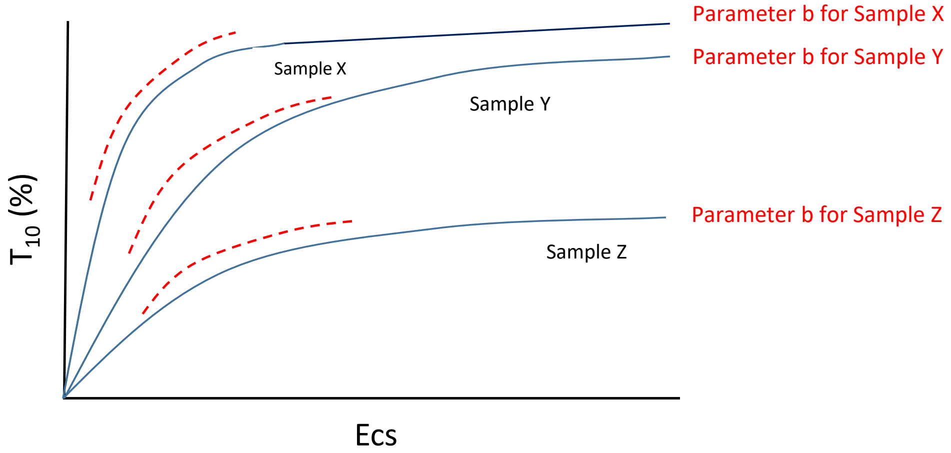


Figure 60. The b parameter in the JKMRC Axb impact breakage characterization method - conceptual example (Image: Simon Michaux)

The parameter of Axb (A times b) is used by the industry as a comminution dimensionless ranking parameter. It is possible that both A and b parameters could be related to rock texture. This has yet to be established and is the subject of further work.

These parameters of A and b are obtained from proprietary JKMRC Drop Test software routinely used to fit the ore specific parameters to a set of given t_{10}-Ecs results. A*b is used in the JKSimMet software and accepted by mining companies as a useful rank of ore hardness. A high A*b is an indication of a soft ore to crush, whereas a low A*b indicates a hard ore.

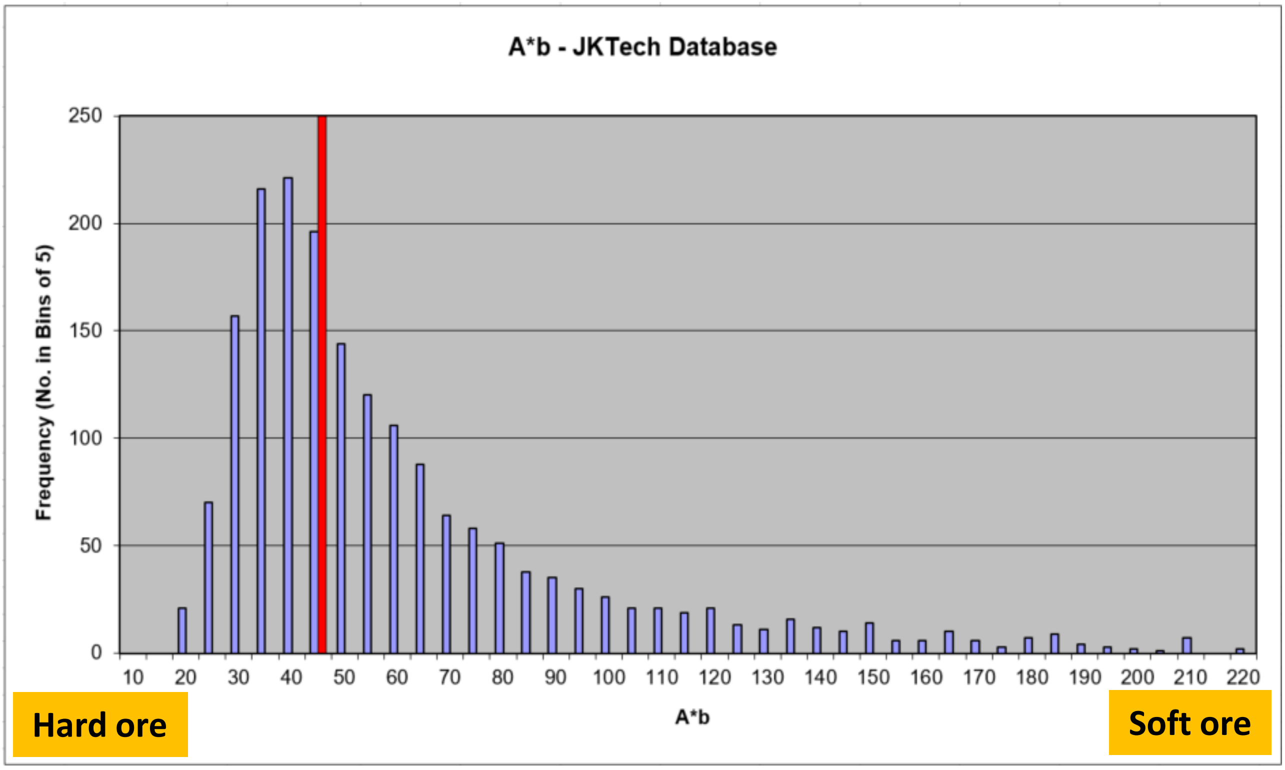


Figure 61. A measured sample referenced against the JKTech Axb Database
(Source and copyright: JK Tech Pty. Ltd)

The objectives a geometallurgy project are however to characterize small volume samples. A full Drop Test requires 200kg -70mm fragmented ore which is not practical for a geometallurgical campaign. Alternatively the A, b and A*b parameters can be calculated on a single small size fraction (which could be drawn from a much smaller volume of rock) with the understanding this is a ranking exercise and the energy breakage curve would not have the precision required for a plant survey (see the SMC test and Rotary Breakage Test in Section 5.3 and Section 5.4).

4.3 JKMRC Abrasion Test

The Abrasion Test is used by the JKMRC to quantify the ores resistance to low impact energy breakage (sometimes called attrition). The test is performed in a specially manufactured tumbling mill (300 mm diameter and 300 mm long) with four 10mm lifter bars. A representative 3kg sample of –53+37.5mm material is selected from the ore (+/-5g). The number of particles in this sample is recorded. The sample is then placed in the mill and run for 10 minutes at 53 rpm (70% of critical speed). The sample is removed from the mill and sized on a √2 sieve series down to 38µm. From this size distribution, the t_{10} is measured.

The abrasion ore parameter, t_a, used in the SAG/AG model is defined to be $^1/_{10}$th of the t_{10} parameter.

4.4 High Pressure Grinding Roll (HPGR)

A high pressure grinding roll, (often referred to as a HPGR or roller press), consists out of two rollers with the same dimensions, which are rotating against each other with the same circumferential speed (Figure 62 and 64). The special feeding of bulk material through a hopper leads to a material bed between the two rollers. The bearing units of one roller can move linearly and are pressed against the material bed by springs or hydraulic cylinders. The pressures in the material bed are greater than 50 MPa (7,000 PSI) (Figure 62). In general they achieve 100 to 300 MPa. By this the material bed is compacted to a solid volume portion of more than 80%.

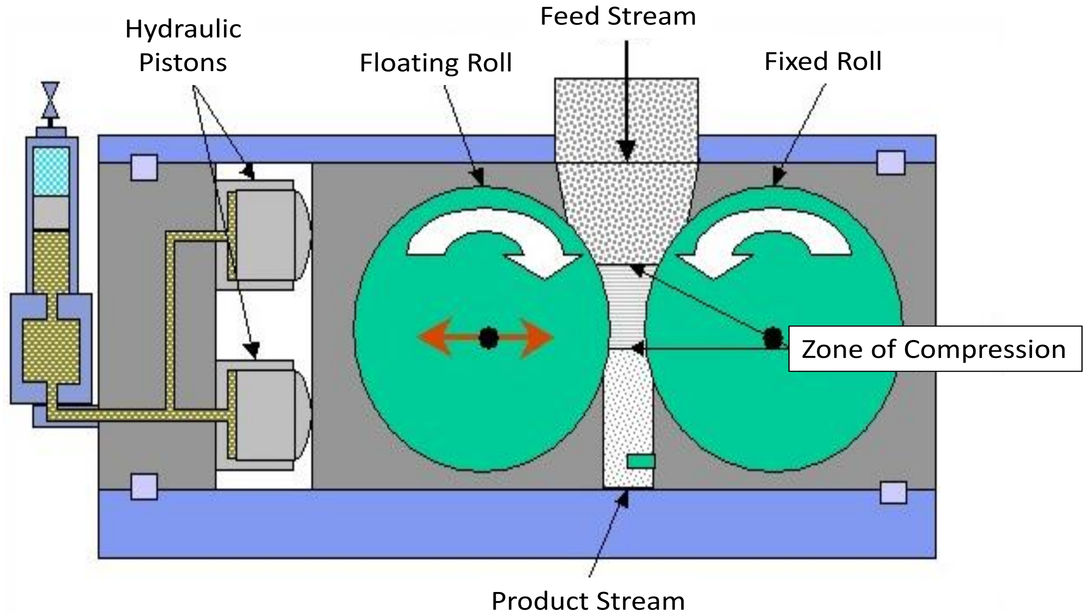


Figure 62: High Pressure Grinding Roll cross section schematic
(Image: Simon Michaux)

The HPGR has the capacity of breaking very hard rock, where a Semi-Autogenous Mill (SAG) would struggle. In terms of rock characterization (based on A*b impact breakage parameter and t_a abrasion low energy characterization), there is a SAG comfort zone (Figure 63). As this is the most fundamental process design decision (SAG mill vs. HPGR) due to CAPEX cost, this means that it will be very important to understand how hard the ore is.

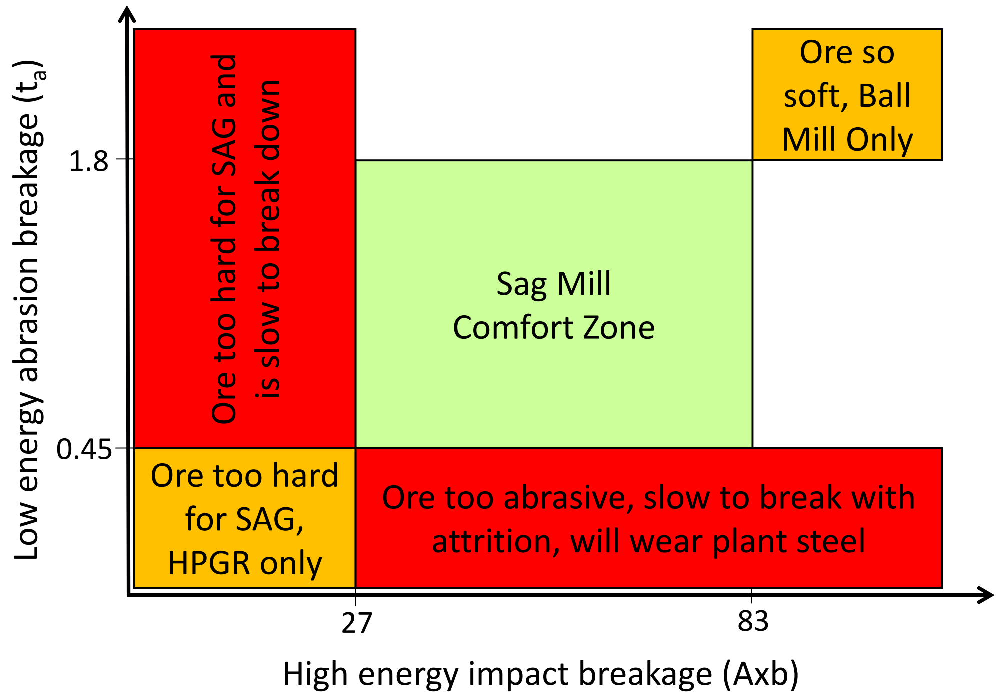


Figure 63. The SAG mill comfort zone (Source: JKTech)
(Image: Simon Michaux)

Extreme pressure causes the particles inside of the compacted material bed to fracture into finer particles and also causes micro-fracturing at the grain size level. This is often referred to as compressed bed breakage (Wills and Napier-Munn 2005). Compared to ball mills, HPGRs achieve a 30 to 50% lower specific energy consumption, although they are not as common as ball mills since they are a newer technology.

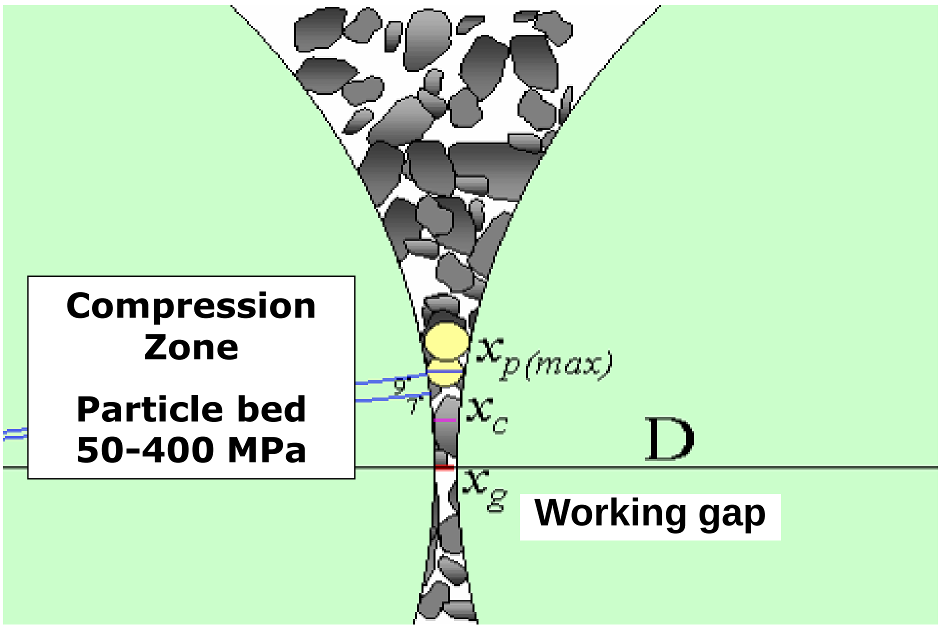


Figure 64. HPGR principle of breakage
(Image: Mike Daniel)

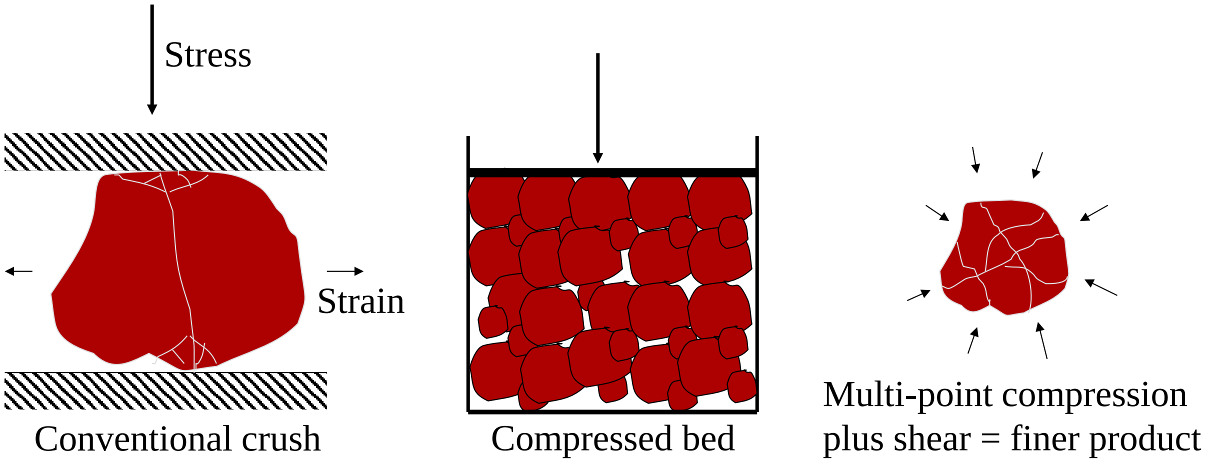


Figure 65 Compressed bed breakage fundamentals
(Image: Mike Daniel)

HPGR breakage is influenced by compressive rock strength (measured with the UCS compressing strength test). This can be empirically ranked using measurement of the A, b and A*b impact breakage parameters (Wills and Napier-Munn 2005). Bed Breakage behavior also has influence (Figure 65). This is the multi-point loading of force on each particle under pressure, instead of the single point loading of force that happens in impact breakage.

The accepted method of characterizing the HPGR response is to conduct a laboratory scale test run (Figure 66). Several test runs can be done at different rolls pressures. The feed and product samples are then sized on the $\sqrt{2}$ sieve series. The feed sample for this is 25kg of -10mm crushed ore for each run.

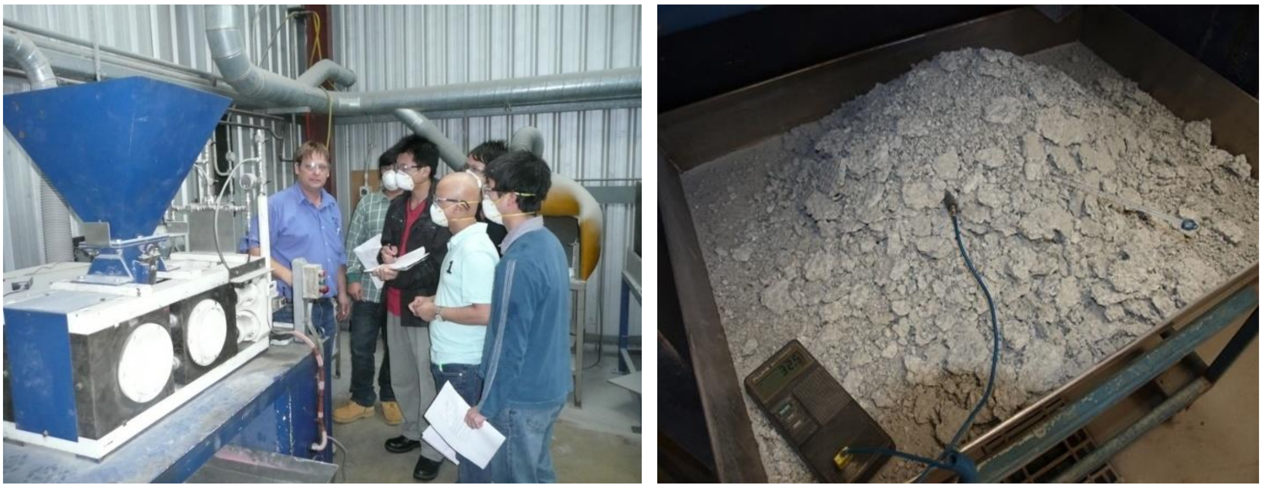


Figure 66. Laboratory scale HPGR test work
(Image: Simon Michaux)

4.5 Grinding Bond Ball Work Index

The Bond Index is used to determine the energy requirements in grinding an ore down to a target size distribution (referred to as the closing size of the test, which is defined by a P_{80}). This is a comminution design tool developed by Fred Bond (Bond 1962). Its purpose was to determine which crusher/mill is appropriate to treat the target ore (Ballantyne & Powell 2014).

Figure 67. The Bond Ball work index test, crushed feed and mill product
(Image: Simon Michaux)

The work index is a comminution parameter, which expresses the resistance of the ore to crushing and grinding. The objective of the Bond ball mill test is to calculate this numerically, and is expressed as the standard Bond Mill Work Index (BMWi). The BMWi is defined by the specific power required (kWh/t) to reduce an ore from an infinite size to a P_{80} size of 100µm. This is done through a series of consecutive grinds in the Bond mill.

According to Bond's third theory of comminution, the work input is proportional to the new crack tip length produced in particle breakage, and equals the work represented by the product minus that represented by the feed. The relationship is expressed as follows:

$$W = 10 * WI\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right) \qquad \text{Equation 2}$$

and

$$P = T \times W \qquad \text{Equation 3}$$

Where

W = work input (kWh/t)

WI = work index – a material specific constant (kWh/t)

P_{80} = size at which 80% of the product passes (µm)

F_{80} = size at which 80% of the feed passes (µm)

T = throughput of new feed (t/h)

P = power draw (kW)

To conduct a test, a prepared sample (of a specific volume of 700cc of –3.35mm material) is placed into a mill of specific size and dimensions (known as the Bond Ball mill) with a specific ball charge (number of balls at a range of set sizes, which all have a specific mass) and ground dry for 100 revolutions. The sample is then removed from the mill and sized at the closing size (defined by the grain size of the ore) to remove the undersize of the target P_{80}.

The test is based around the constant sample volume of 700cc. When each grinding cycle is complete and the undersize is removed, and is replenished with an equal mass of new feed. The length time of each batch grind is adjusted until the mass of the oversize fraction is consistently 2.5 times greater than the undersize. Under these conditions the test approximates the performance of a closed circuit of a continuous mill with a recycle load of 250%. A full Bond test has typically 7 to 8 cycles and requires approximately 5kg of sample as a starting mass.

The closing size is a controlling parameter for the test, and is related to the grain size of the target mineral to be extracted. A smaller closing size will result in more energy required to grind the rock to that target size and visa versa.

Once the standard work index (WI) measured from experimental measurement, Bond's Third law of comminution is used to calculate the specific power required to reduce a given F_{80} to the required P_{80} in an 8 foot diameter wet overflow ball mill. For a given throughput (t/h) the specific power (kWh/t) is converted to power draw (kW). Mill dimensions for a production unit are then chosen which are predicted to draw the required power, using an appropriate mill size-power correlation.

The sample mass for a Bond Ball Work Index test is stated to be 10kg, but a sample mass of 5kg can be used if the technicians are experienced. Coarse residues from chemical assay procedures can be used as can drill core or coarse fragments as test feed sample. The test requires a top size of 3.35mm in particle size.

The efficiency of the ball mill is controlled by the volume of the recirculating load (250% when operating normally). Harder minerals usually persist at the 2 to 3 mm size range and resist grinding. If a BMWi is in the higher end of the spectrum (18-25 kWh/t), it can be useful to understand what minerals are present. To characterize the feed and products, the following methods are useful:

- Chemical Assay
- Bulk QXRD
- XRF pellet
- Automated mineralogy in some circumstances

4.6 Fine Grinding – Laboratory scale Isa Mill

Fine grinding is increasingly required for an operation to be economically viable. As time has progressed, many of the easy to find and easy to extract ore deposits have been mined out. In the late 1990's a new fine grinding comminution mill was commercialized called the Isa mill. The Isa mill made it possible to mine and extract ore with much finer mineral grains (resulting in much higher cost of production due to cost of grinding). The Isa mill was developed due to a very fine grained ore was discovered in an existing deposit in Mt Isa in Australia. If the fine grained ore was to be processed, technology to fine grind had to be developed, which would later become the Isa Mill.

Figure 68 shows the difference in complexity between coarse grained, high grade ore (LHS) and the fine grained ore with refractory fine pyrite dilution (RHS) (Young *et al* 1997). Both of these samples were taken from the No 5 orebody at Mt Isa.

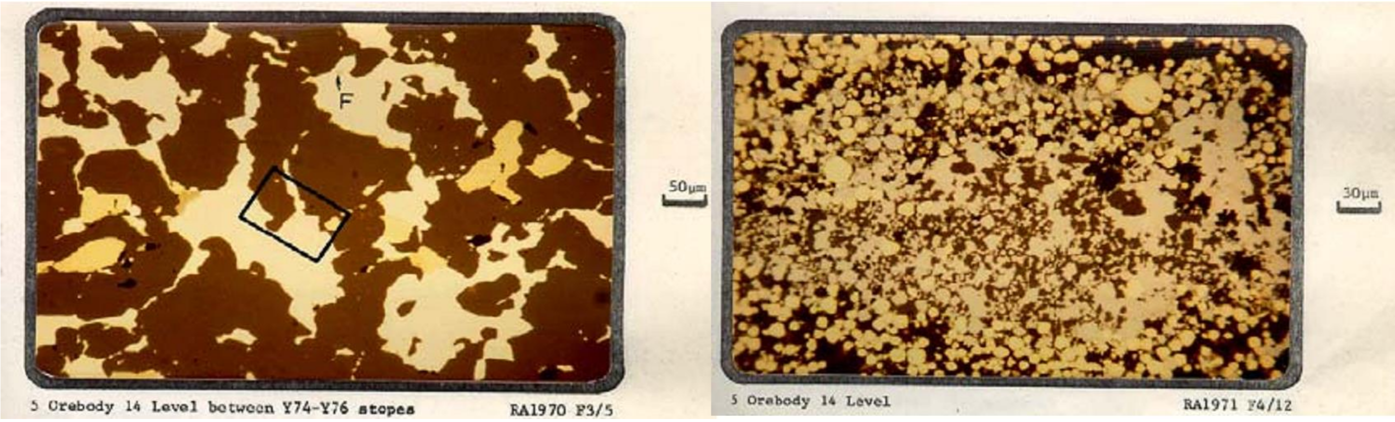


Figure 68. Mineralogy and plant closing size (Source: Young *et al* 1997)

The IsaMill is primarily known for its ultrafine grinding applications in the mining industry, but is also being used as a more efficient means of coarse grinding, with target product sizes ranging from 5 to 60 µm. Figure 69 shows what size fractions the different comminution devices work with.

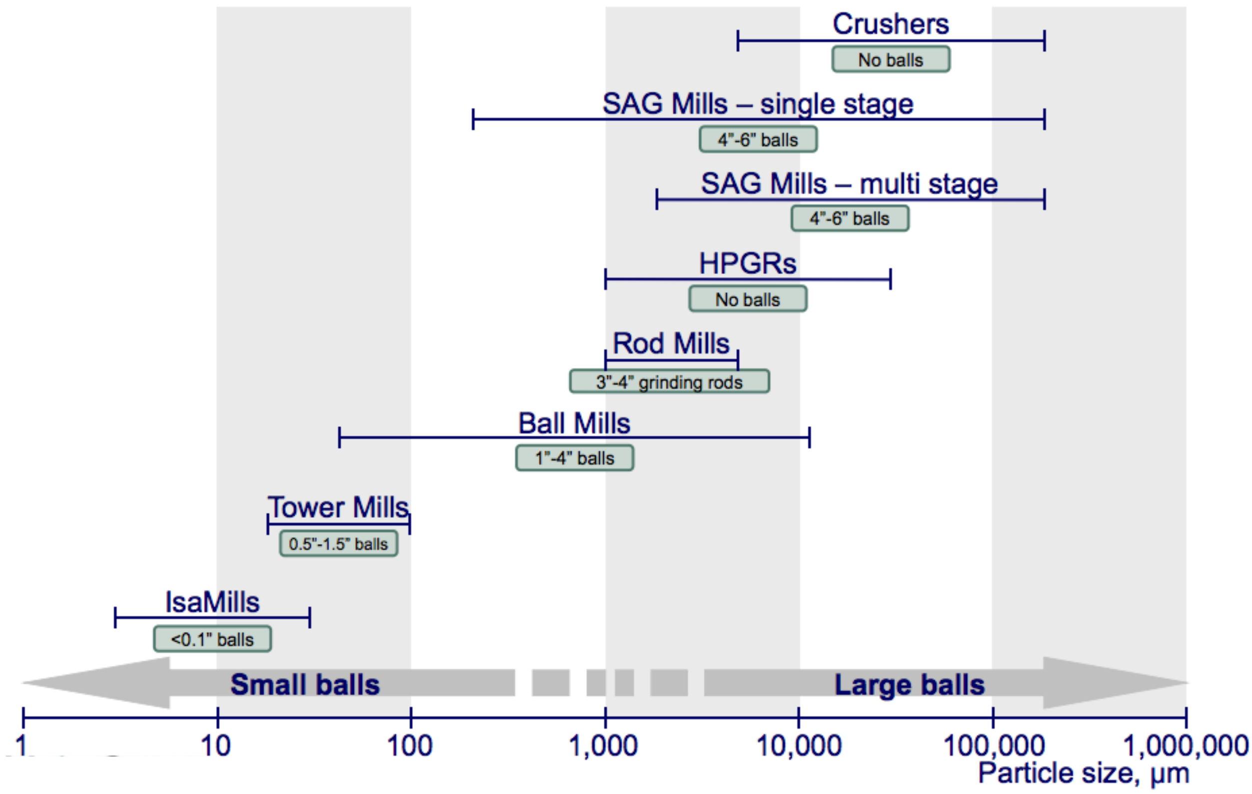


Figure 69. Fine grinding in relation to other comminution methods
(Source: Metso blogs, https://www.metso.com/blog-hub/?pageSize=13&language=English)

A currently available method for characterization for fine grinding is a test run in a laboratory scale Isa Mill (Figure 70). A feed sample was prepared in 15kg batches using the large rod mill. The particle size of each of the feeds was below 100 micron. The time required to run the rod mill depended on the hardness and the size of the initial particle and was again estimated using the 'grind, check, predict' method.

Figure 70. A laboratory scale Isa mill
(Image: Simon Michaux)

After the correct feed size is achieved, the sample is mixed with water (45% by weight sample, 55% water) in order to make a slurry of the correct consistency. The slurry is then transferred into a feed sump where a stirrer keeps it in suspension.

The IsaMill was set to be running at 1605 rpm with a flow of 2.4 liters/min. A feed sample was taken from the output hose after allowing the slurry to recirculate from the bottom of the sump, through the pump and back into the top of the sump. A specific gravity and flow rate measurement was taken at the same time as well.

When the IsaMill is set up according to the standard operating procedure (SOP), the slurry was directed through the mill. As per the SOP instructions the IsaMill should not be started until 20 seconds after the flow has been directed through the IsaMill to prevent overheating. This is the length of time needed for the slurry to reach the mill from the sump. The first sample was taken from the output hose when approximately half of the slurry had passed through the mill. Once the feed sump was close to empty the slurry flow was diverted using a multi-directional valve whereby the original feed tank now serves as the product tank and vice versa. An energy consumption reading for this cycle was recorded at the time of cycle completion. This process was repeated many times in an attempt to reach an energy saturation point for that sample.

The size distribution of the feed and products is to be measured in an optical sizer in a slurry.

As fine grinding is often happening at a particle size smaller than mineral grains, the mineralogy of the feed ore will control the effectiveness and efficiency of the process. To characterize the feed and products, the following methods are useful:

- Chemical Assay
- Bulk QXRD
- XRF pellet
- Automated mineralogy in some circumstances

4.7 Selection of grind size for the whole campaign

One of the critical decisions to make that has the capacity to make the entire geomet campaign fail if done incorrectly is the selection of sample particle grind size. This is the particle size (expressed as a P_{80}, or particle size that 80% of the sample is smaller than) to which all samples for process separation a milled down in size through grinding.

If the target minerals are very small in grain size or are embedded in complex rock texture, the automated mineralogy done on polished block mounts will be required. Figure 71 shows the possible difference in rock texture at similar grades. Each example in Figure 71 would have a different recovery and would need to be reduced to different grind size for full liberation.

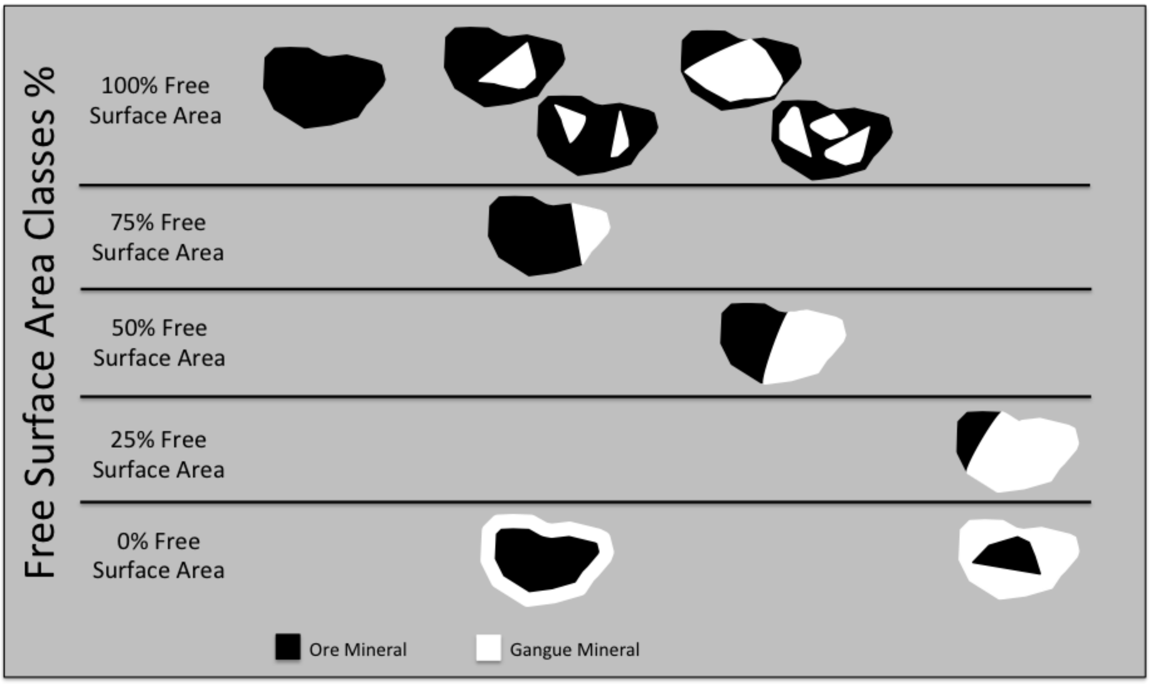


Figure 71 Different rock textures, each would have a different process recovery (Source and copyright: Cropp 2013)

The process of comminution is to subject the sample to size reduction. The particle size required for the effective liberation of the target mineral is defined by the mineralogy. For each mineral in the sample, there would be a range of mineral grain size distribution. For process separation to economically effective, enough of those particle have to be liberated or partially liberated.

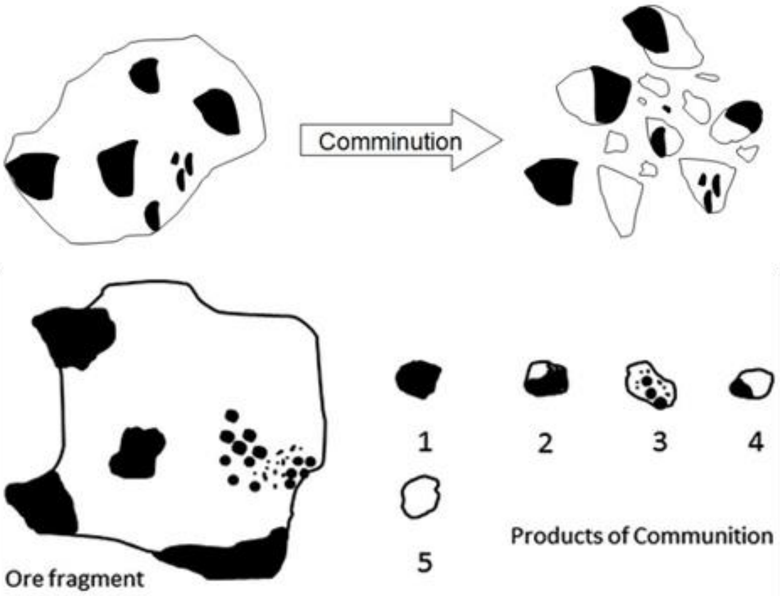


Figure 72. Particle size reduction through comminution and liberation of target minerals
(Image: Napier-Munn et al 1996, Copyright: JKMRC)

The process of size reduction through comminution is important as it has economic significance. Curry *et al*. (2014) found that the mill (defined as crushing, grinding and separation) typically accounts for between 35 and 50 per cent of the total mine costs. The size reduction to energy relationship is not linear though. The finer the grind size, the exponentially larger energy required to comminute the ore to that size (Ballantyne & Powell 2014 and Hukki 1962) (Figure 73).

The tradeoff between the cost of comminution to the target grind size and the degree of liberation and texture of the valuable mineral particles has to be considered carefully. Thus the full distribution of the target mineral needs to be understood. If there is more than one target mineral associated with different textures (with different particle sizes at liberation), then each need to be considered in context of value of the ore.

An unconventional method of defining what the size distribution of the mineral grains are inside an ore type, could be the use of electrodynamic fragmentation in a Selfrag unit (see Section 4.7.1). If done correctly, and if the mineralogy is susceptible to this methodology, all minerals could be liberated intact at their true particle size. This method has its advantages if the ore has more than one valuable mineral (possible polymetallic process path), where each target mineral would have its own unique mineral grain size distribution. For example, an ore with a Cu mineral with a P_{80} grain size of 100 micron and a Au mineral with a P_{80} grain size of 10 micron.

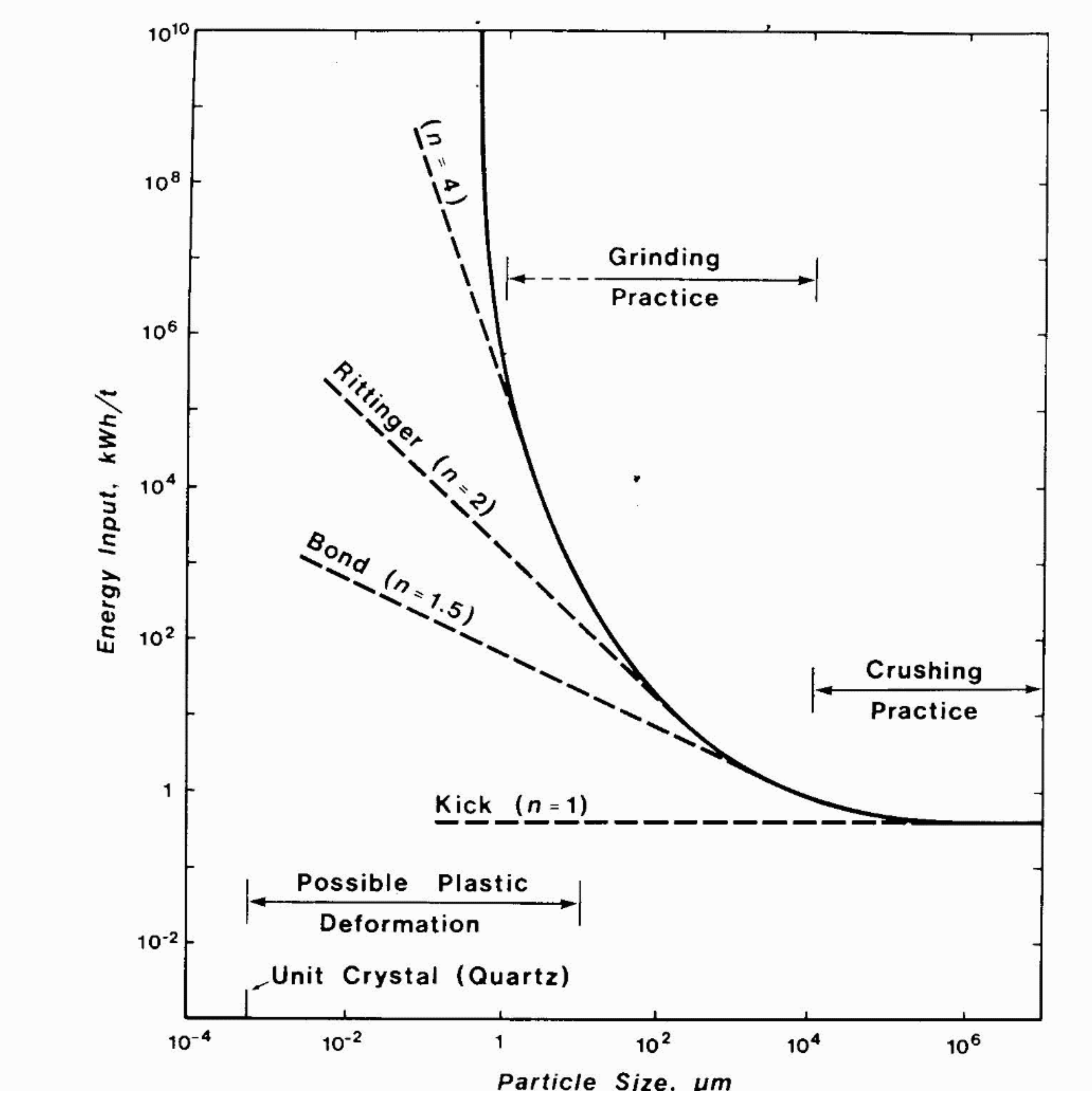


Figure 73. The Hukki energy size relationship shown as an imaginary example of the basic reduction characteristic plotted on logarithmic paper. (Source: Hukki 1961)

The grade of concentrate and recovery are the most common measures of metallurgical efficiency. Batch flotation tests are often examined together with several ores from the same deposit in a grade recovery curve (Figure 74).

Usually this is defined through a sophisticated test procedure to determine the grade recovery curve for that mineral in that ore type. The theoretical grade-recovery curve for an ore is a definition of the maximum expected recovery by flotation of a mineral or element at a given grade. This is defined by the surface area liberation of the value minerals and is consequently directly related to the grind size utilized in the process. The theoretical grade-recovery can be readily used to quickly identify potential recovery increases that can be gained through optimization of flotation circuits and whether the process is running efficiently.

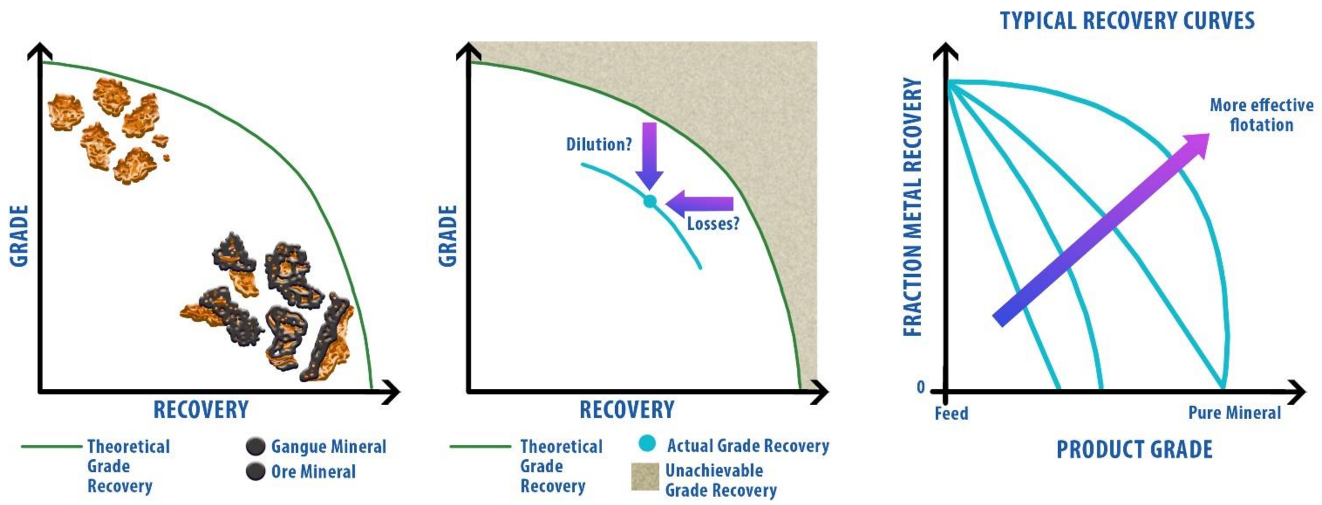


Figure 74. Theoretical grade recovery
(Source: Reproduced from Goodall 2009, Image: Tania Michaux)

Flotation performance is most commonly represented using grade-recovery curves, with a shift upwards and to the right indicating an improvement in performance. Figure 75 shows the effect of finer grind size on flotation performance by virtue of more target particles being liberated.

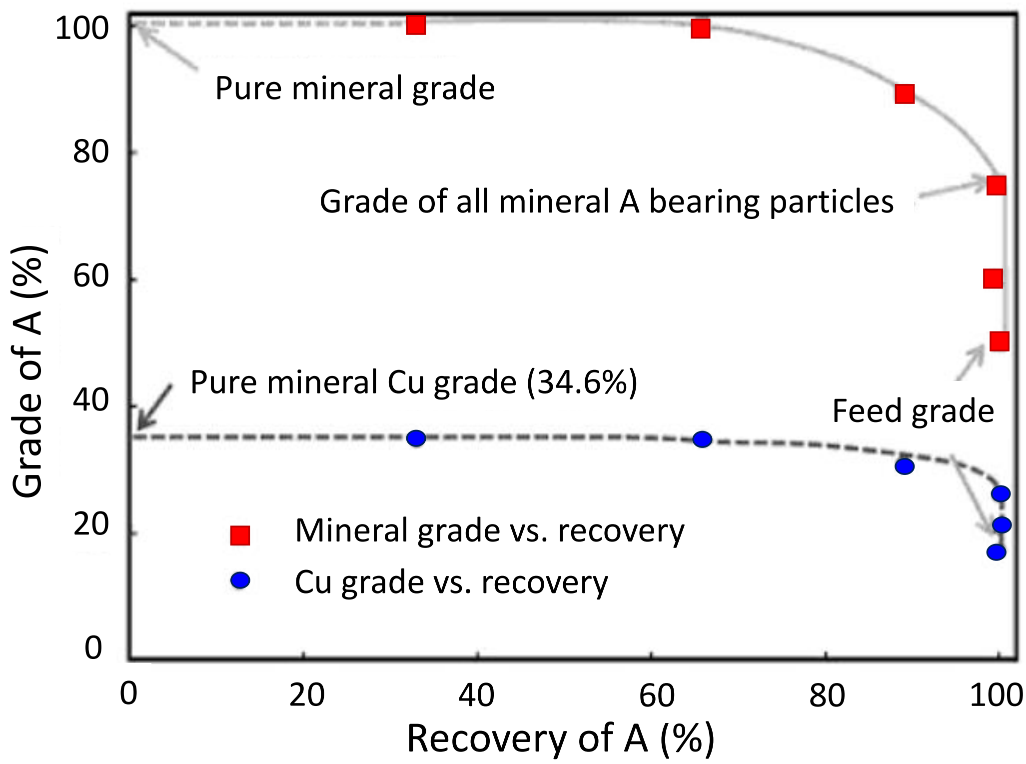


Figure 75. A grade recovery curve for perfect separation (LHS, Cu grades based on chalcopyrite, $CuFeS_2$)
(Source: Reproduced from Wills and Finch 2016, Image: Simon Michaux)

Figure 76 is an example that shows the key aspects of using the theoretical grade-recovery curve for a free gold flotation circuit.

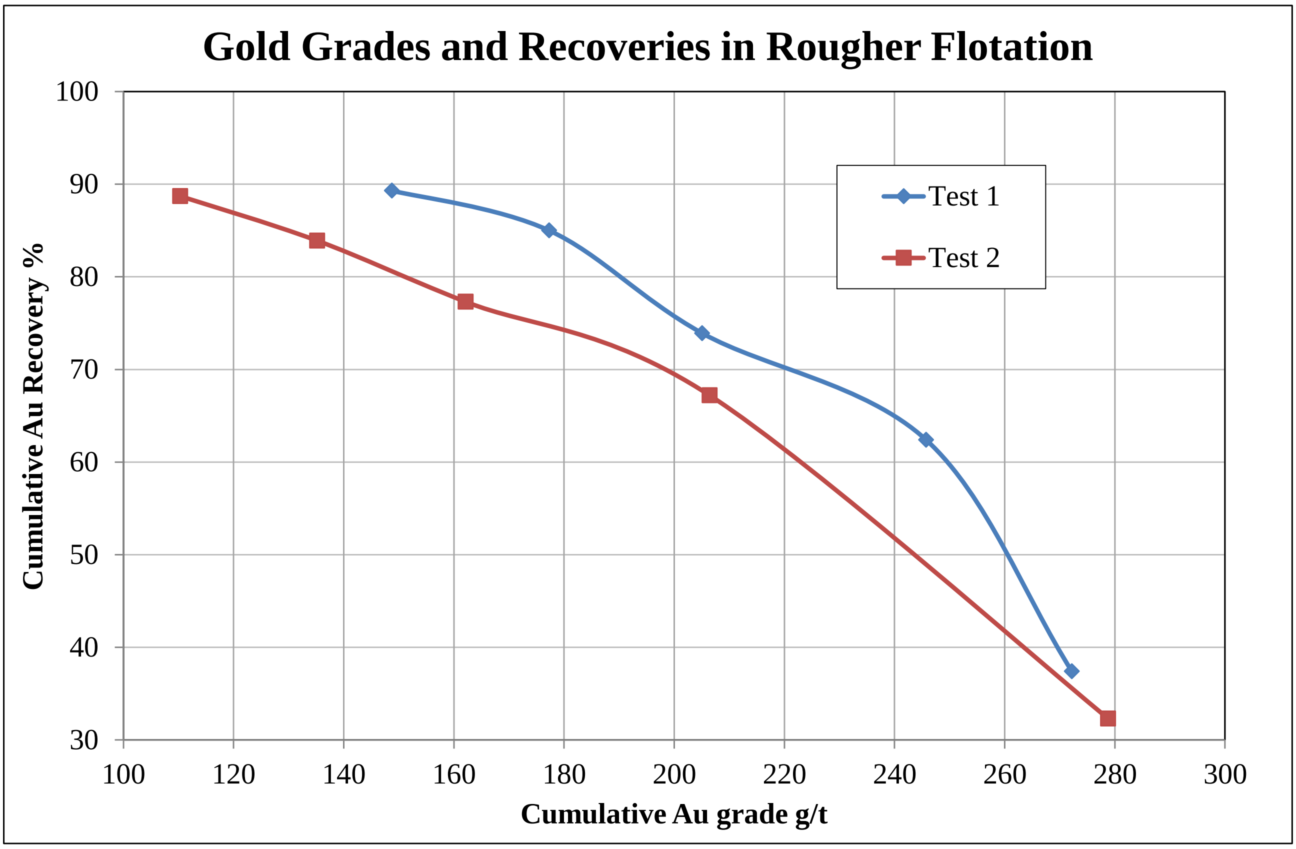


Figure 76. Experimental example of a grade-recovery curve
(Source: Tero Korhonen GTK Mintec)

To establish the theoretical grade-recovery curve for a material a mineralogical liberation study should be undertaken using tools such as automated mineralogy (QEMSCAN or MLA).

Ideally this work should be done either before or in parallel to a geometallurgical study. Given enough automated mineralogy data, a theoretical grade recovery curve could be estimated based on mineral grain size across a family of ore types.

A decisions is to be made for what P_{80} each of the process test samples are to be reduced to through crushing then grinding. As a laboratory scale rod mill produces a particle size distribution that better resembles a production scale circuit compared to a laboratory scale ball mill, all sample preparation should be done with wet grinding in a rod mill with a standard rod load. The choice of mild steel or another metal alloy for the rods is a matter of discernment in context of the ore mineralogy.

To prevent oxidization of the sample, after rod milling, the wet sample is placed in a vacuum sealed bag (ideally) and then stored in a freezer at a lower temperature than -18^{0}C. This should be done for any sample that would or could be later be subject to flotation or leaching.

The ideal situation is the definition of closing samples grind size is taken from previous work done on the same ore types.

4.7.1 Electrodynamic fragmentation

There is an unconventional opportunity to determine what the natural grain size of target minerals (possible even several minerals at once), using electrodynamic fragmentation (EDF). Electrodynamic fragmentation provides an alternative means to mechanical crushing when preparing rock samples for mineral separation. Convetnional breakage fragments a rock particle by squeezing quickly between two crusher jaws, or through impact breakage, where an external forace is applied to the particle. EDF breakage loads the particle with electricity so quickly that the induced stress field manifests inside the particle. The particle breaks apart from internal strain (Figure 77).

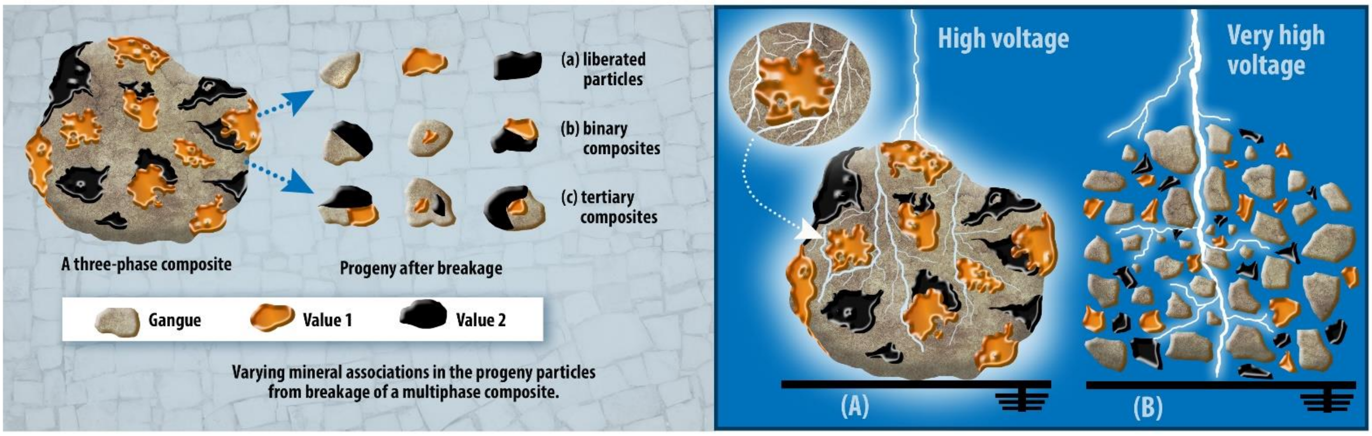


Figure 77. Conventional breakage (LHS) compared to EDF breakage (RHS)
(Image. Tania Michaux)

Fragmentation by rapid powerful electrical shocks can be used for the production of clean mineral separates for mineralogical and isotope geological purposes (Michaux 2016, Everaert *et al* 2019, Vaasjoki and Lindqvist 2001). Figure 78 shows a 3rd generation Selfrag unit, which can break small scale samples using EDF.

Figure 78. The Selfrag electrodynamic fragmentation instrument
(LHS University of Liege ULiege Gemme Dept Unit, Image- Eric Pirard)

EDF is a process that is fundamentally based on a High Voltage (HV) physical specificity. An electric pulse is arced between a cathode electrode and an anode electrode over such a short time period that it will preferentially pass through the solid sample between the electrodes than the surrounding water (Figure 79). In such a short pulse, the solid is more conductive than the water, thus is the path of least resistance.

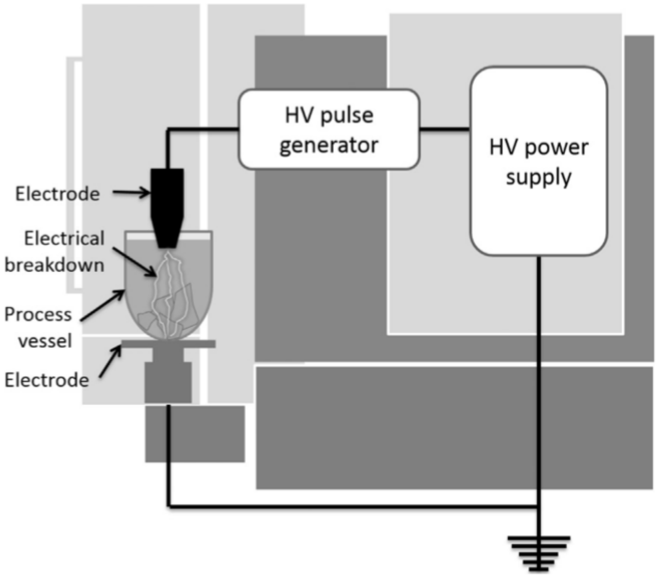


Figure 79. Schematic of the SELFRAG laboratory scale machine
(Source & copyright: Bru *et al.*, 2018, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

As the short pulse passes through the solid sample, it creates an electrostatic charged stress field with geometry defined by material texture (Figure 80 LHS and center). Electrostatic discharge occurs through solid, causing strong internal shockwaves across stress field geometry (Figure 80 RHS), defined by a difference in the dielectric material properties of the minerals in rock sample. In doing so, cracks would form along the grain boundaries using Maxwell-Wagner polarization. This results in selective breakage, where grains are fully liberated intact (Figure 81 to 85).

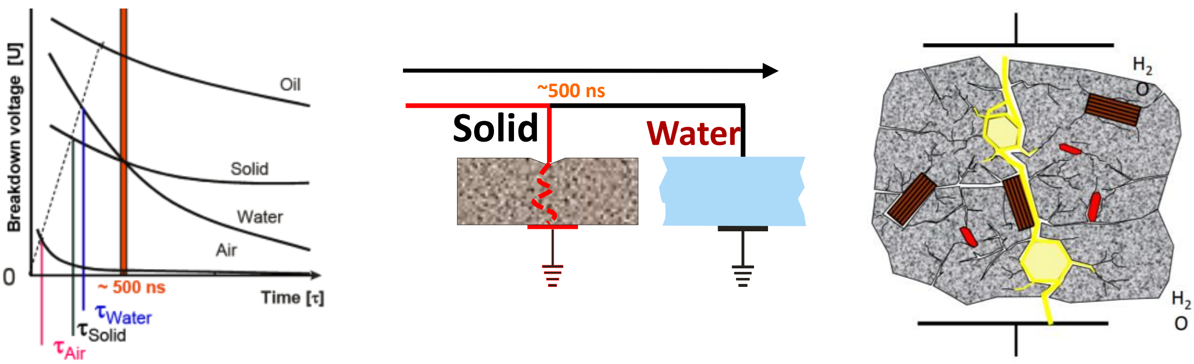


Figure 80. Electrodynamic fragmentation pulse speed
(Source & copyright: Alex Weh, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

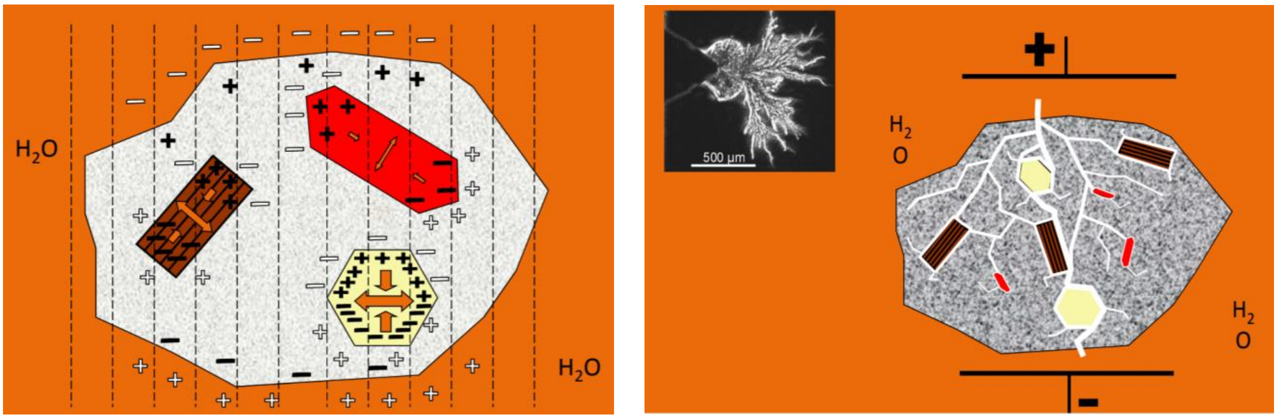


Figure 81. Maxwell-Wagner Polarization of minerals (Kaufmann 2013)
(Source & copyright: Alex Weh, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

EDF breakage happens in two basic modes. The first mode is the pulse passes through a particle and breaks the body apart in what is termed rock body breakage original particle is broken into 5 to 6 smaller pieces of similar mass (Wang 2012). The second mode is termed surface breakage, as the pulse passes along the boundary between mineral grains, which is how intact mineral grains are often liberated (Figures 82 and 83). At low voltage applications, body breakage dominates. At higher voltage applications, surface breakage dominates and individual grains are predominantly produced.

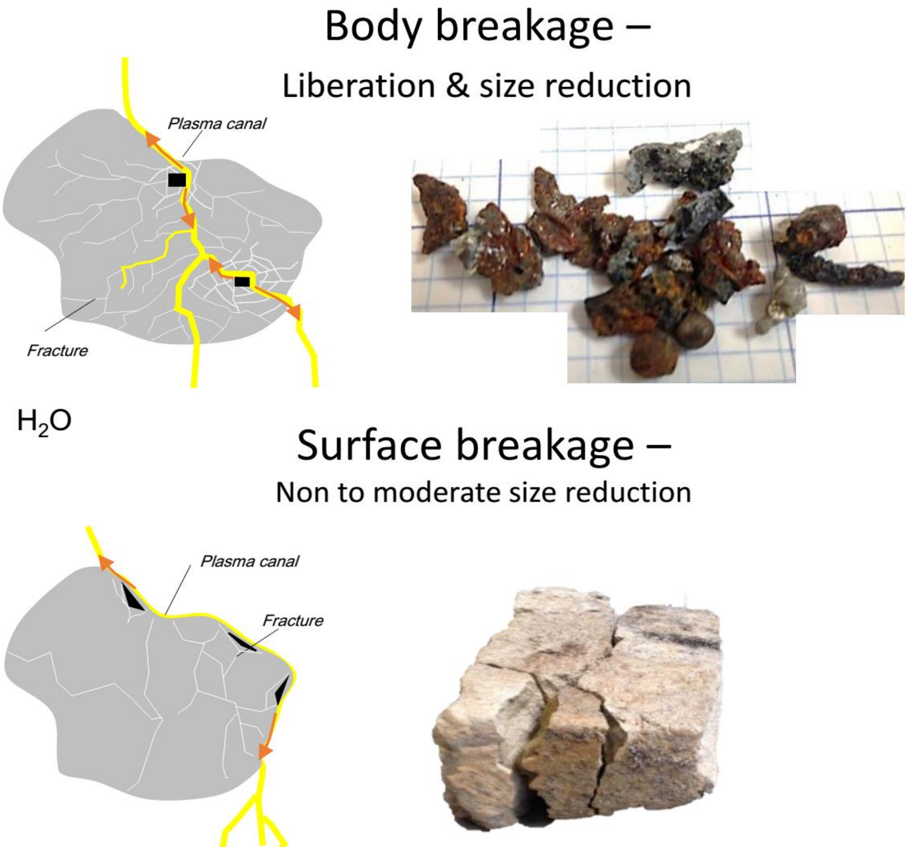


Figure 82. EDF particle body breakage and particle surface breakage
(Source & copyright: Alex Weh, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

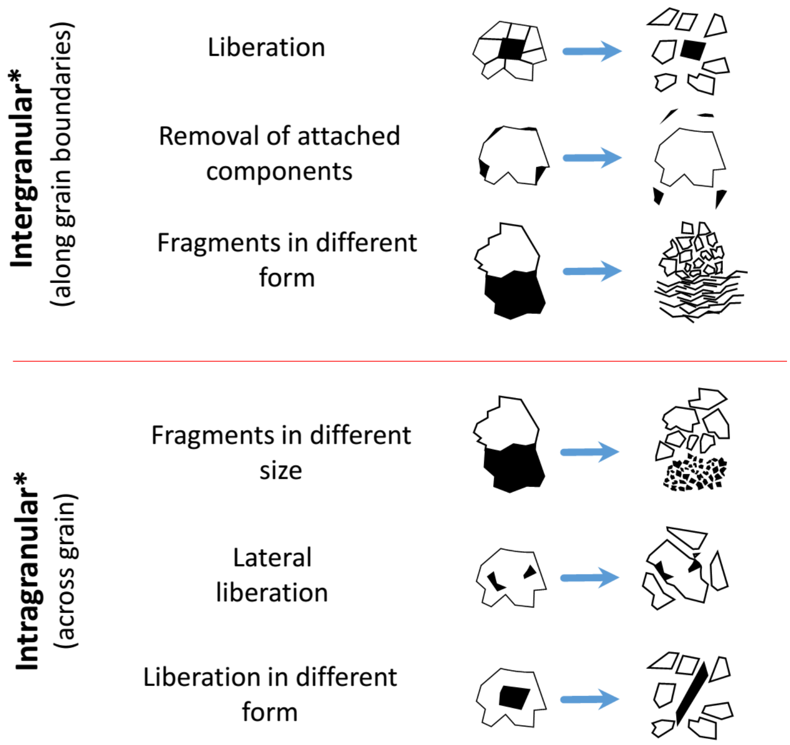


Figure 83. Definition of selective fragmentation products
(Image & copyright. Hesse & Lieberwirth)

EDF events over time can be seen in Figure 74 and 75.

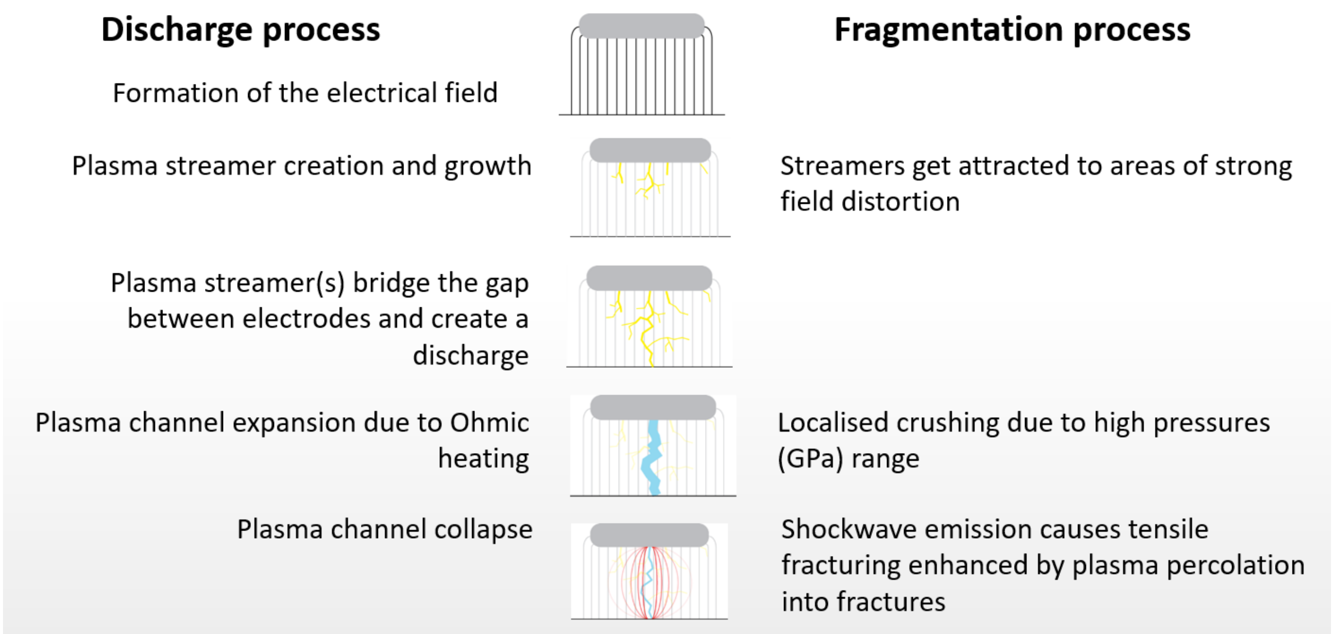


Figure 84. The electrodynamic fragmentation events over time
(Source & copyright: Alex Weh, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

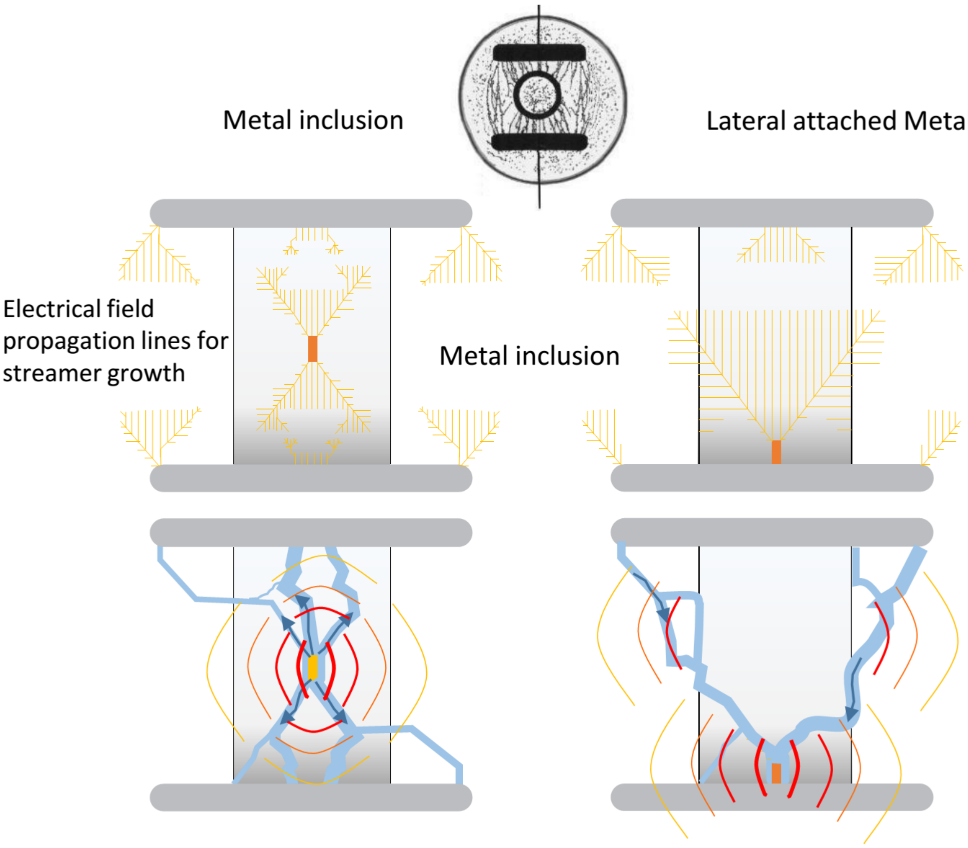


Figure 85. How the electrodynamic field geometry forms around a mineral inside a sample
(Source & copyright: Parvaz *et al* 2015, Selfrag High Voltage Pulse Power Fragmentation http://www.selfrag.com/)

This represents an unusual opportunity. EDF breakage has a very unusual signature characteristic. During trials (Michaux 2016) it was observed that whole grains were liberated completely, with their surface complete with their original texture (this was seen in the liberation of aggregate in concrete – Figure 86 and the liberation of metal particles from pyrometallurgical slag – Figure 87).

Figure 86. Clean liberation from electrodynamic fragmentation of concrete. The intact 50x50x30mm block (LHS).
The fragmented sample after one EDF application (RHS).
(Images: Simon Michaux)

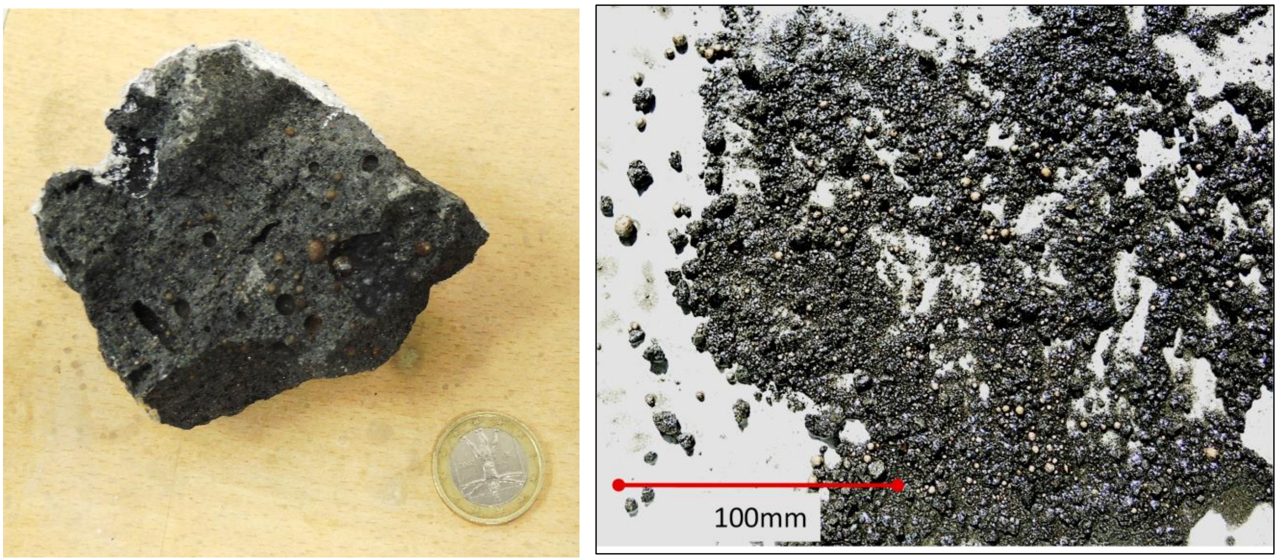


Figure 87. Clean liberation from electrodynamic fragmentation of steel furnace slag. The intact 80mm particle (LHS). The fragmented sample after one EDF application (RHS).
(Images: Simon Michaux)

Figure 88 to 90 show an example of EDF breakage of a granodiorite cut sample (Michaux 2016). The sample was broken in a single step of EDF application (Figure 88). Figure 89 shows the entire sample after being dried in an oven, then split into two sub fractions, +850μm and -850μm. Figure 90 shows the same samples as in Figure 89 but mounted on a polished block mount and then imaged under a Ziess SEM microscope. Clear particle liberation can be seen at multiple particle sizes.

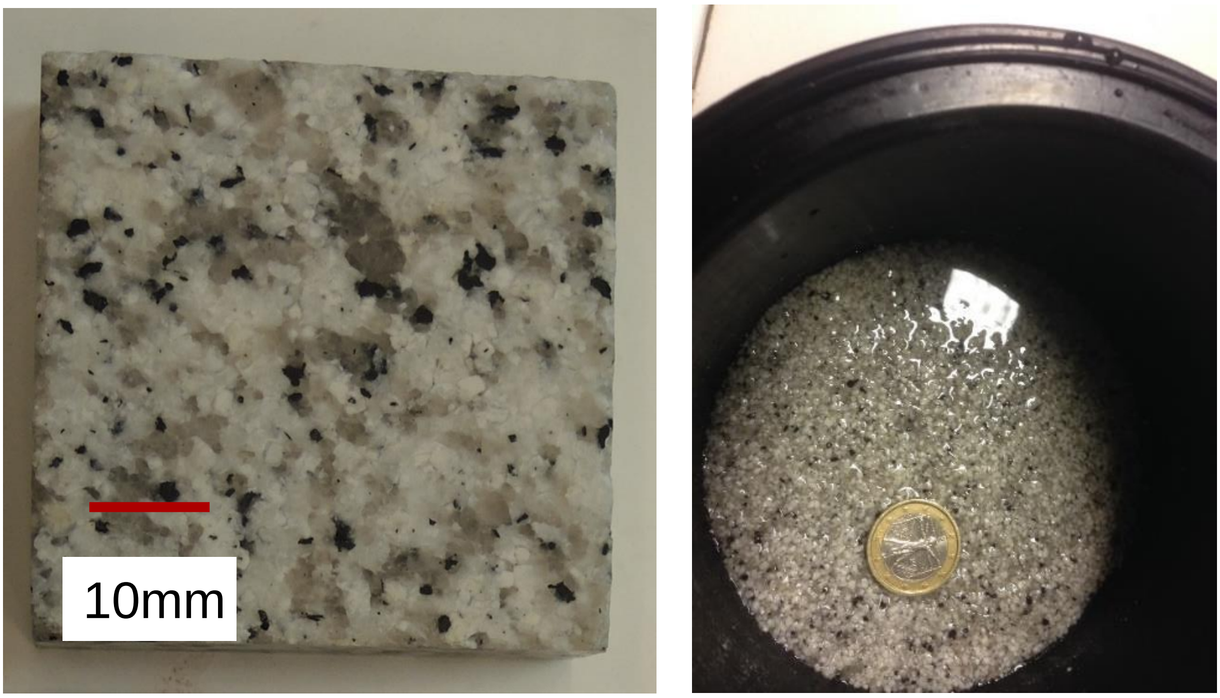


Figure 88. Granodorite sample intact, before electrodynamic fragmentation (LHS),
Same sample just after fragmentation still in the sample holder (RHS) (Michaux 2016)
(Image: Simon Michaux)

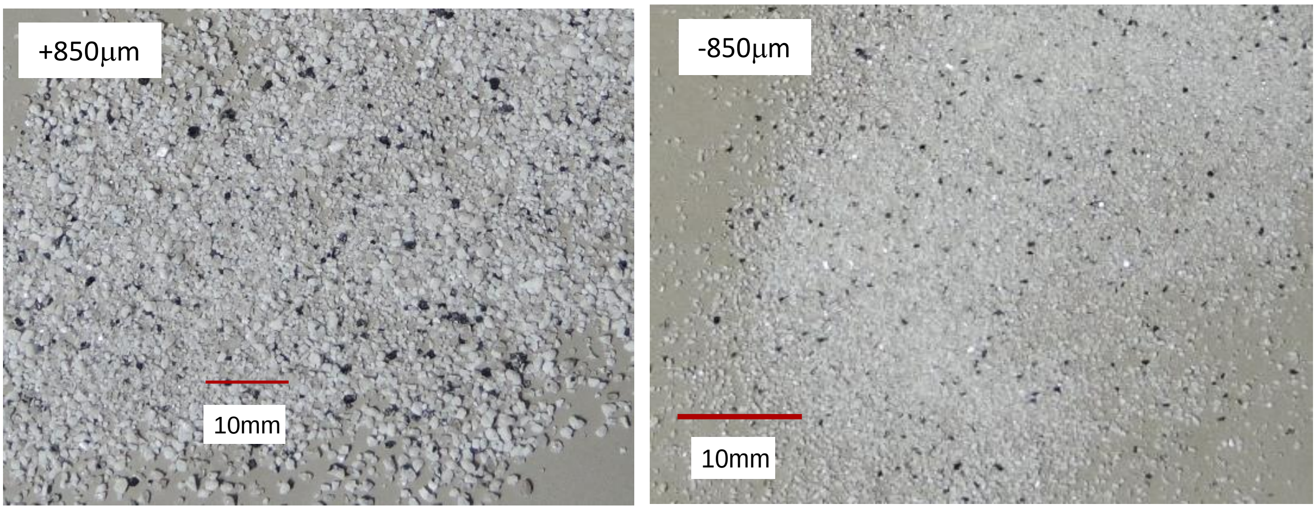


Figure 89. Granodorite sample after electrodynamic fragmentation (entire sample after oven drying) (Source: Michaux 2016) (Image: Simon Michaux)

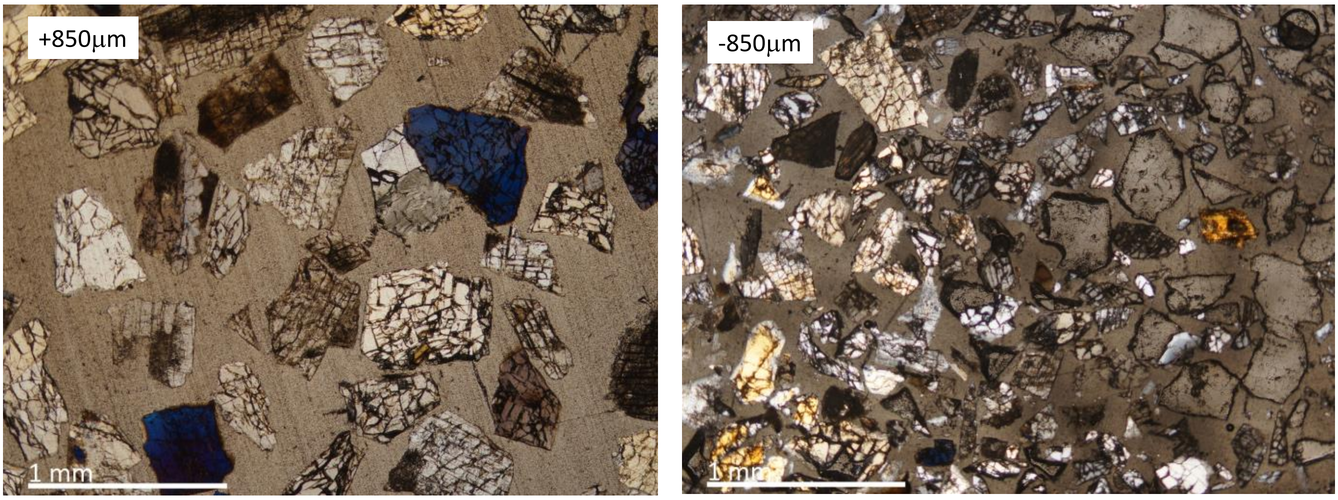


Figure 90. Granodorite sample after electrodynamic fragmentation on polished block mounts (Source: Michaux 2016) (Image: Simon Michaux)

What makes this interesting is the comparison to conventional breakage. Figures 91 and 92 show a sample that has been broken with impact breakage on a Drop Weight Tester at 1.5kWh/t (Michaux 2016). This is of a similar particle size as Figures 89 and 90. Almost all of the particles seen in Figures 91 and 92 are binary or tertiary (of have more different minerals again) in modal mineralogy. This is what is expected with conventional breakage.

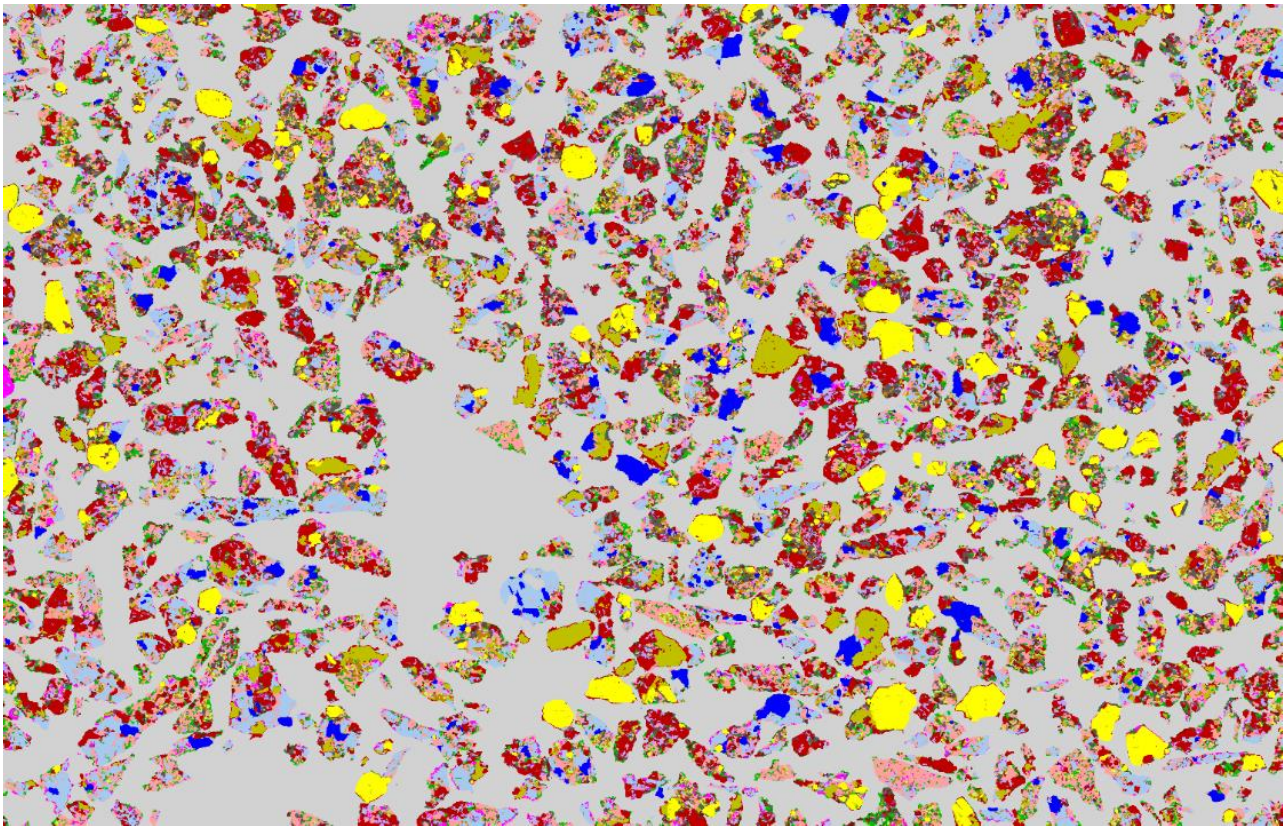
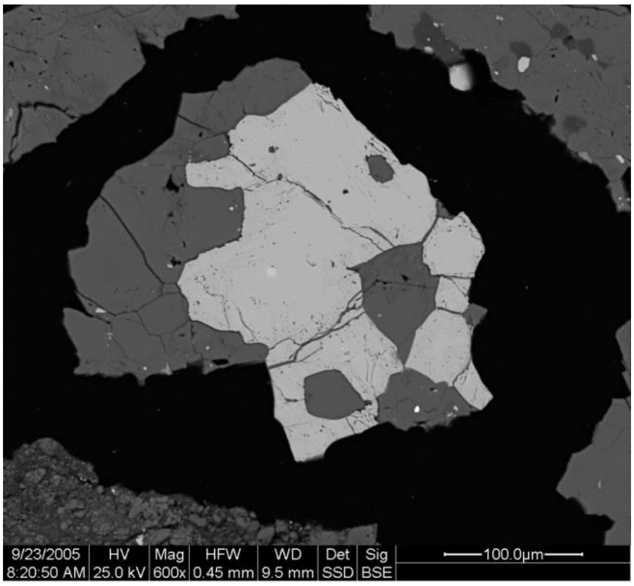


Figure 91. Samples from an IOCG style Cu-Au deposit -1.18+0.850mm fragments (MLA mineral map) (Source: Michaux 2016) (Image: Simon Michaux)



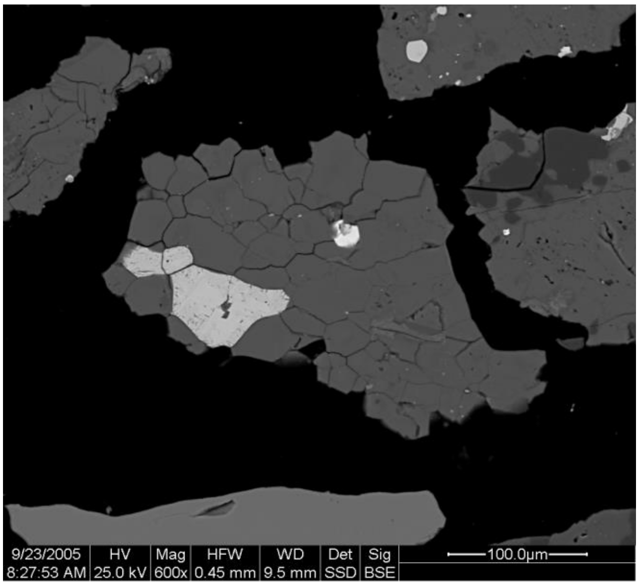


Figure 92. Samples from an IOCG style Cu-Au deposit -1.18+0.850mm fragments (BSE images) (Michaux 2016) (Image: Simon Michaux)

It is postulated that EDF fragments the rock with internal stress ("shaking the rock apart from the inside") compared to conventional breakage which fragments the rock by external application of force on the particle. The EDF opportunity is shown below in a conceptual example (Figure 93).

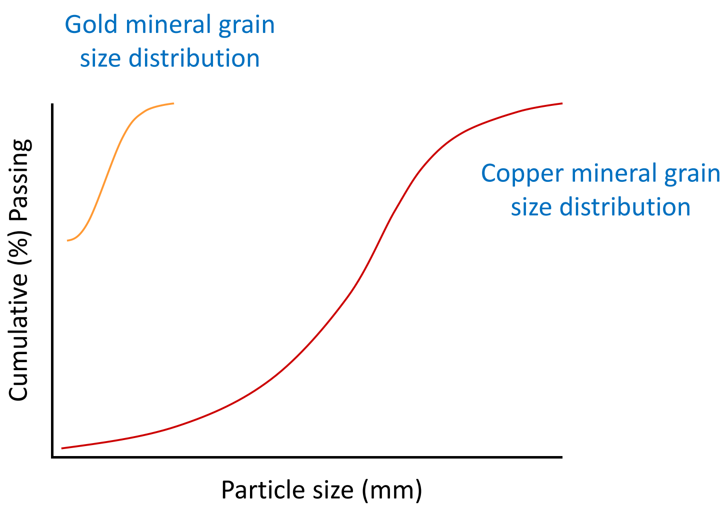


Figure 93. Conceptual example of the grain size distribution of a polymetallic ore (Image. Simon Michaux)

Gold is worth far more than copper, but is of a much finer grain size. The economic cost of comminution to grind to a finer closing size would be much higher to liberate the gold compared to the copper. The only way to resolve this interesting problem is to have accurate measurement of the liberated mineral grains for each valuable mineral in the ore.

This could be done by breaking each end member rock texture with EDF (in the Selfrag). Then appropriately characterize the products (in a batch flotation test or a column leach test for example). If enough sample can be prepared, process separation tests like batch flotation could be considered.

This is a wet process that requires high purity water (with high electrical conductivity). The sample is broken in the volume between the electrodes (an ellipsoid of 40mm in Z dimension and 50mm in X and Y dimension). This sample would be reduced to a top particle size of approximately 2mm in a very unusual breakage signature.

Thus this unit is able to break a relatively small sample of a few hundred grams of coarse crushed breakage or a single particle of rock of approximately 50 x 50 x 30 mm in size. Samples of several kilograms can be prepared in several runs that are later combined.

4.8 Flotation

Flotation (also spelled floatation), in mineral processing, method used to separate and concentrate ores by altering their surfaces to a hydrophobic or hydrophilic condition—that is, the surfaces are either repelled or attracted by water. The flotation process was developed on a commercial scale early in the 20th century to remove very fine mineral particles that formerly had gone to waste in gravity concentration plants. Flotation has now become the most widely used process for extracting many minerals from their ores.

Figure 94. Production scale flotation separation plant
(Source: Codelco Flickr, https://www.flickr.com/photos/codelco/)
(Copyright License: https://www.flickr.com/help/terms)

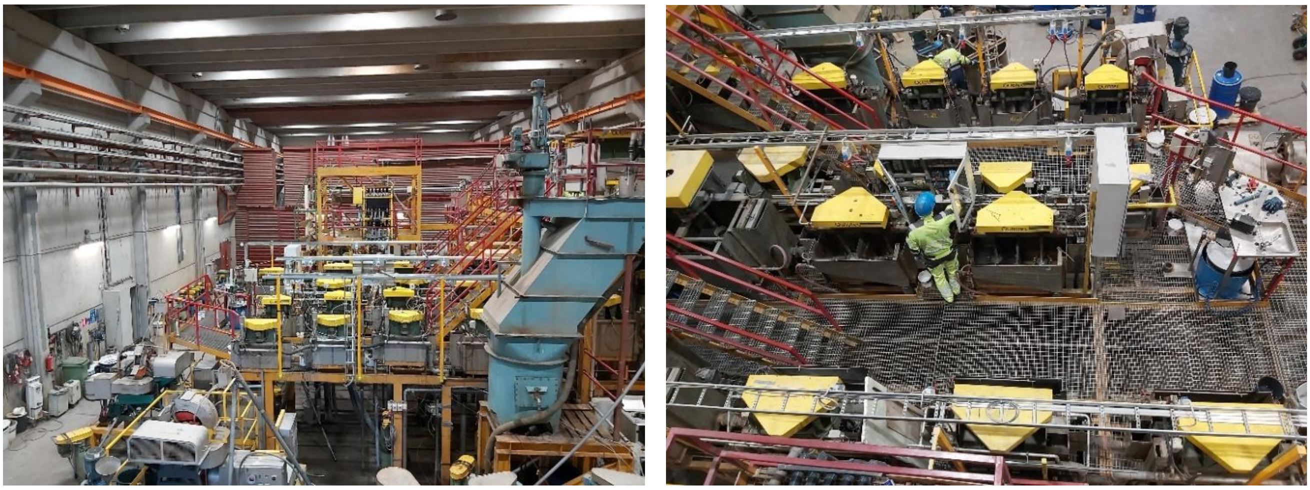


Figure 95. The GTK Mintec pilot scale flotation circuit
(Image: Arno-Matti Kirpala)

Most kinds of minerals require coating with a water repellent to make them float. By coating the minerals with small amounts of chemicals or oils, finely ground particles of the minerals remain un-wetted and will thus adhere to air bubbles. The mineral particles are coated by agitating a pulp of ore, water, and suitable chemicals; the latter bind to the surface of the mineral particles and make them hydrophobic. The un-wetted particles adhere to air bubbles and are carried to the upper surface of the pulp, where they enter the froth; the froth containing these particles can then be removed (Figure 95, 96 & 97). Unwanted minerals that naturally resist wetting may be treated so that their surfaces will be wetted and they will sink. Figure 95 shows an image of bubbles generated in a flotation experiment.

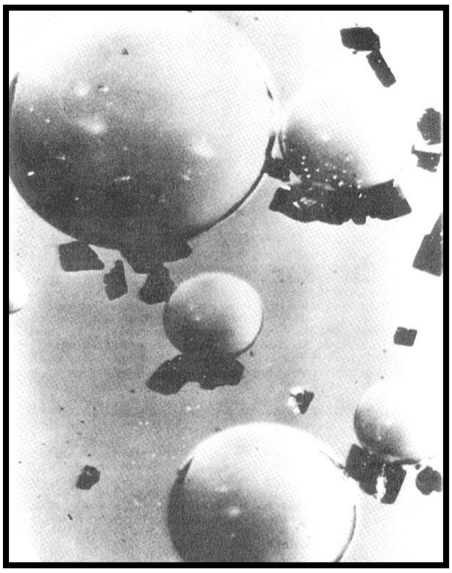


Figure 95. The flotation process photographed
(Image: Dee Bradshaw)

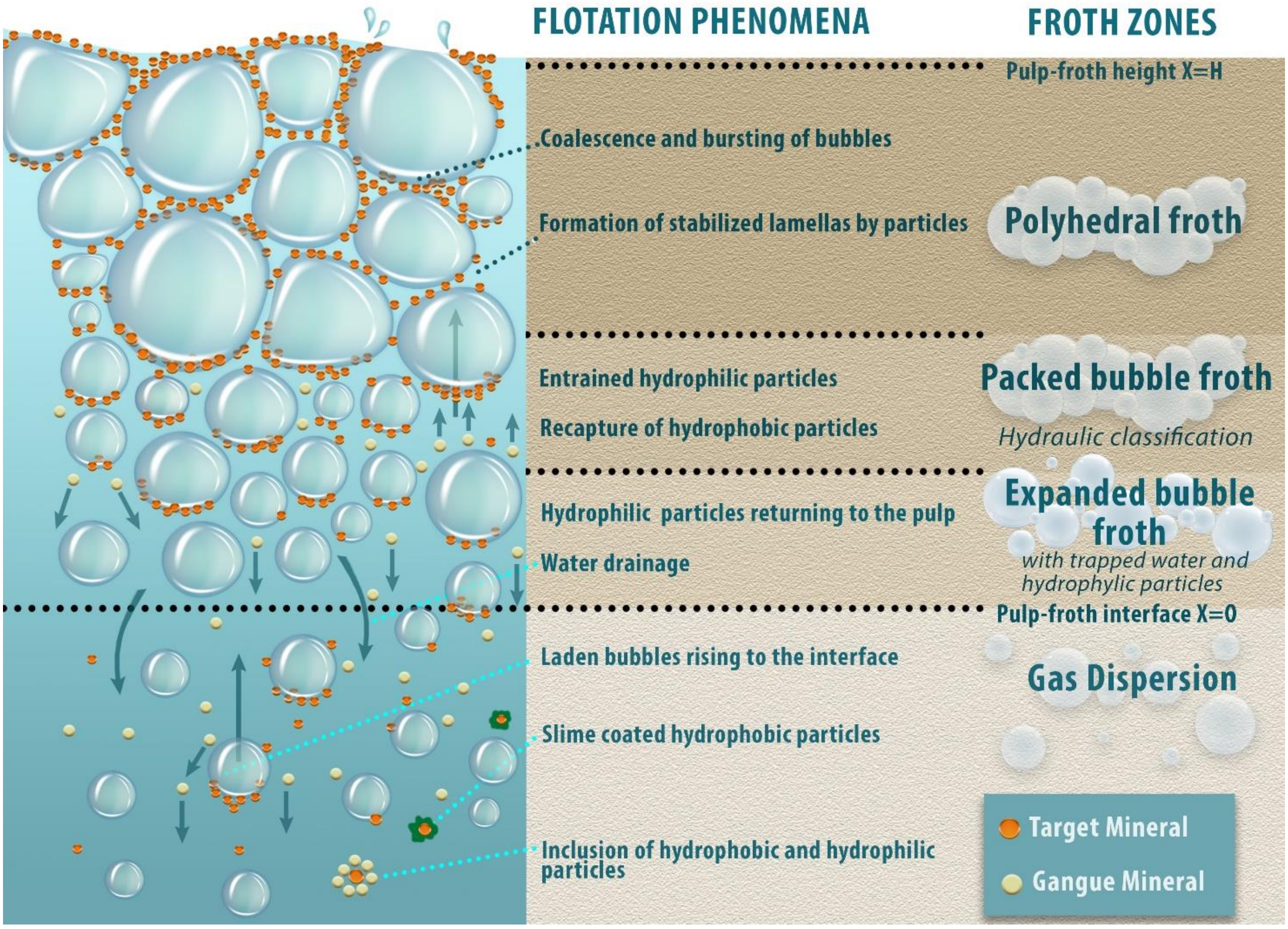


Figure 96. Flotation phenomena
(Image. Tania Michaux)

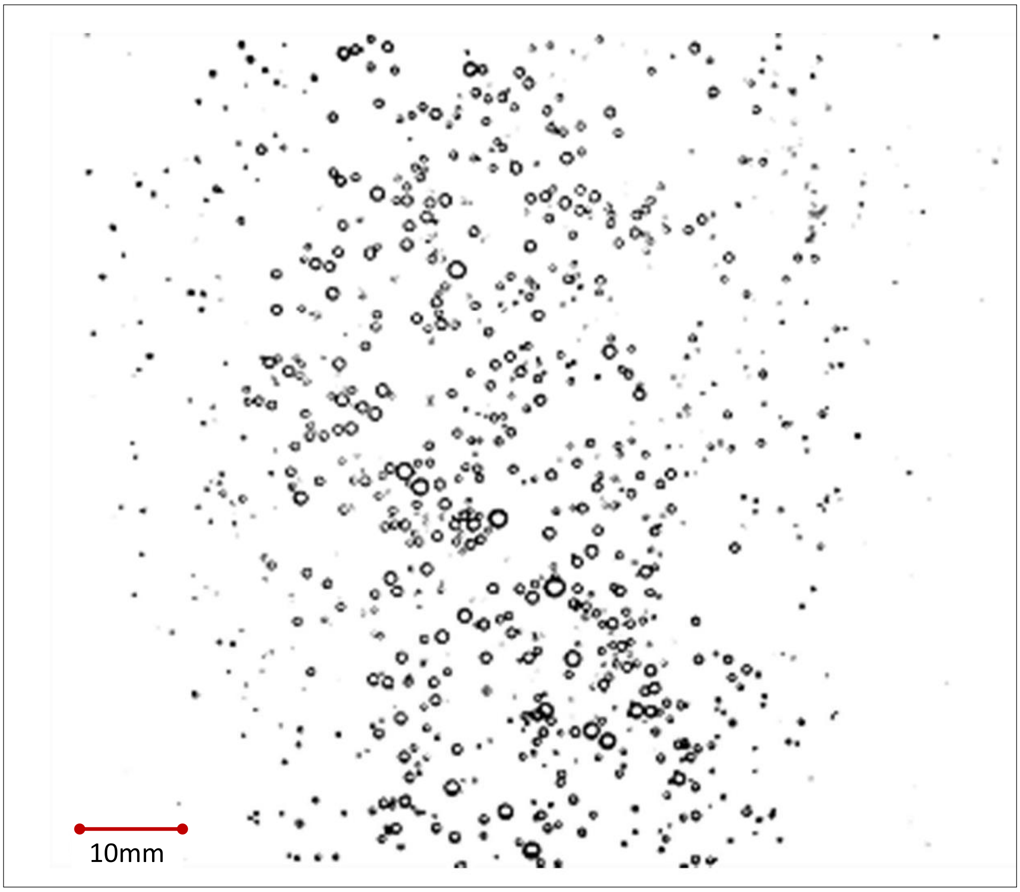


Figure 97. Bubbles generated during flotation
(Source: Nuorivaara, T. and Serna-Guerrero 2016, Copyright: AusIMM)

Figure 98. Froth flotation separation (Image: Tero Korhonen)

Flotation is the conventional method for concentrating metal out of ore, where a feed grade of approximately 1% could be concentrated to a product grade of 25-30%. Figure 98 shows this process at a production scale mine. As such, the flotation response of an ore to this separation process needs to be understood in a geometallurgical context.

There are three categories of models for the prediction of flotation (empirical models, probability models, and kinetic models).

- Empirical models are too specific to their environment and usually involve a trial and error feedback approach to optimization.
- The probability models basically consider the probabilities of particle-bubble collision, adhesion, froth stability, etc.
- Kinetic models are established on the basis of the analogy between a chemical reaction (collision of molecules) and an important flotation mechanism, i.e., the collision between either hydrophilic (or hydrophobic particles) and air bubbles in the pulp volume.

To examine and model the flotation process, the kinetic approach is certainly the oldest and most traditionally used. It postulates that the flotation rate follows a rate equation similar to those utilized to describe homogeneous kinetics. Its foundation is based on the well-known analogy existing between bubble-particle and molecule-molecule collisions. Thus, there are measurable rate constants, which describe mineral recovery in the flotation process.

The kinetic study of the flotation process includes the determination of all the factors that influence the rate of concentrate production. This rate can be defined according to various meanings, but in the operations carried out on ores, it is generally measured as absolute recovery (unity fraction) divided by the time (kinetic constant units: s^{-1} or min^{-1}). The overall absolute recovery is one metric of interest to measure. The time taken to reach this recovery is another metric to measure.

4.8.1 Batch Flotation Test Characterization

Batch laboratory flotation testing is the industry accepted method for characterizing samples in context of flotation response for modelling industrial performance. Traditionally, evaluation of batch flotation tests is based largely on a rougher recovery value achieved at a defined time. Since the cumulative recovery of a component in the concentrate is proportional to flotation time, the flotation process can be considered as a time-rate recovery process. Therefore, a mathematical flotation model that incorporates both the recovery and rate function can completely describe flotation time-recovery profiles.

The recovery in a batch flotation test is the percentage of the total target metal contained in the ore, recovered in the test (Wills and Finch 2016). Recovery is not a straight forward concept. The flotation separation method cannot separate out all minerals and textures.

This is done by a process that involves the imparting of a water repellent (termed hydrophobic) character to the target mineral particles using chemicals that are called collectors or promoters. The target minerals contain metals (like copper for example) that are naturally more hydrophobic than the surrounding rock minerals, and the metal will preferentially stick to the surface of a bubble of air, rather than reside in water. Under favorable conditions, these chemically coated particles become attached to the air that takes the form of bubbles that is agitated through the pulp. The bubbles float to the surface, taking the target minerals with it. The froth on the water surface at the top of the cell can then be wiped off, forming a concentrate (Figures 99 & 100).

Figure 99. Batch flotation bench scale test (Image: Tero Korhonen)

Figure 100. Batch flotation bench scale test (Image: Tero Korhonen)

Batch flotation starts from three fundamental types of processes, which form the basis of flotation experimental design.

- Bulk Flotation – the separation of like and unlike minerals (e.g. sulfides from non-sulfides).
- Differential Flotation – the concentration and removal of different minerals (with elements like Cu, Pb, Zn, Au, etc.) from a single ore type.
- Sequential Flotation - the sequential concentration and removal of minerals with the same element in several stages from a single ore type (the opposite to bulk flotation).

For this to be effective, the appropriate quantity of valuable target mineral needs to be liberated. To achieve this, each sample has to be ground to the appropriate grind size (See Section 4.7).

The test objectives and resulting outcomes from batch flotation should be established in planning beforehand. Feed size distribution, method used for chemical assays of feed and products, what the valuable materials are, what the penalty elements are, rates of flotation and bubble agitation, etc. Once this is known, the following can be planned:

- Cell type and size to be used in the test
- Feed sample weight required (and what mass the concentrate will probably be) for each test
- Water quality and quantity
- Slurry density
- Chemical reagents including collectors, regulators and frothers to be used

All of this is to be planned with the laboratory doing the batch flotation tests. The outcome of a batch flotation test is a mass fraction of froth measurement (% mass that is floatable) a kinetics recovery curve (similar to Figure 101).

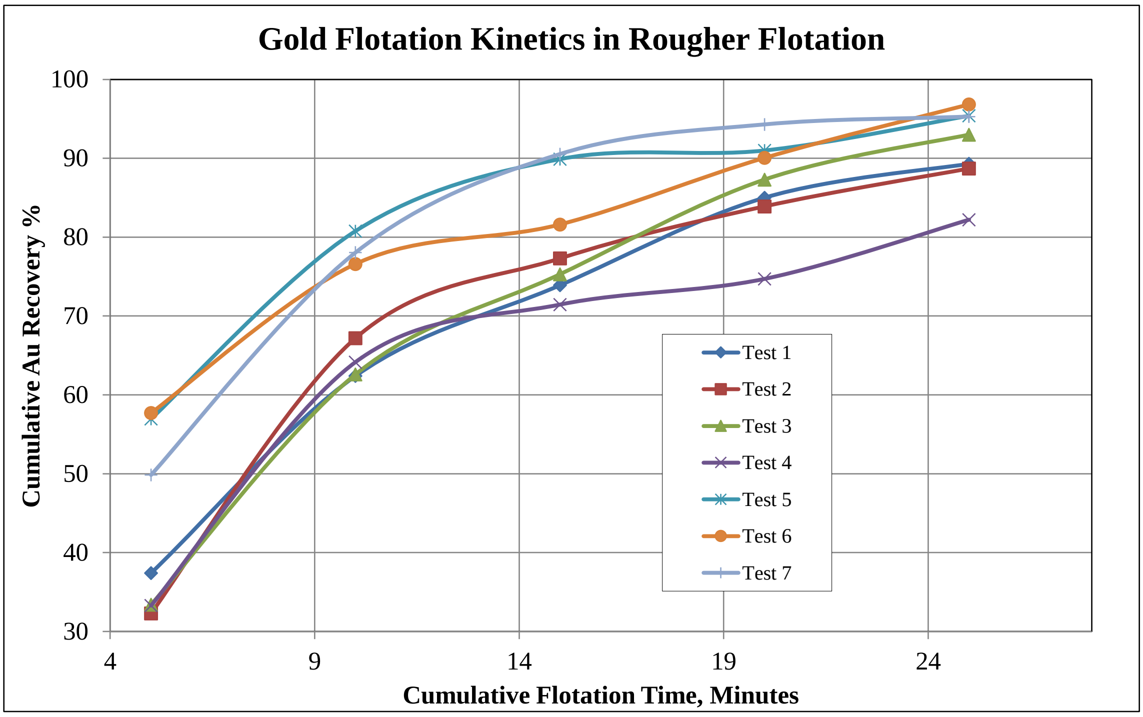


Figure 101. Flotation kinetics recovery
(Source: Tero Korhonen)

The presence of some minerals will antagonistically reduce the effectiveness of flotation for both the absolute recovery and the time taken to reach that absolute recovery. If these minerals were subject to different process paths, the final absolute recovery would also be different.

A useful approach is to a rougher flotation followed by a cleaner flotation step. Also, many flotation circuits target more than one metal in separate processes. Figure 102 below is an example of an experimental design that allows this to be modelled. Copper flotation (rougher then cleaner) could be done, and then nickel flotation could be done on the copper rougher tails. This entails four batch flotation tests done on each ore type. The products are all characterized with chemical assays as appropriate. The concentrate is sampled at regular intervals and assayed to produce the recovery curve similar to Figure 101.

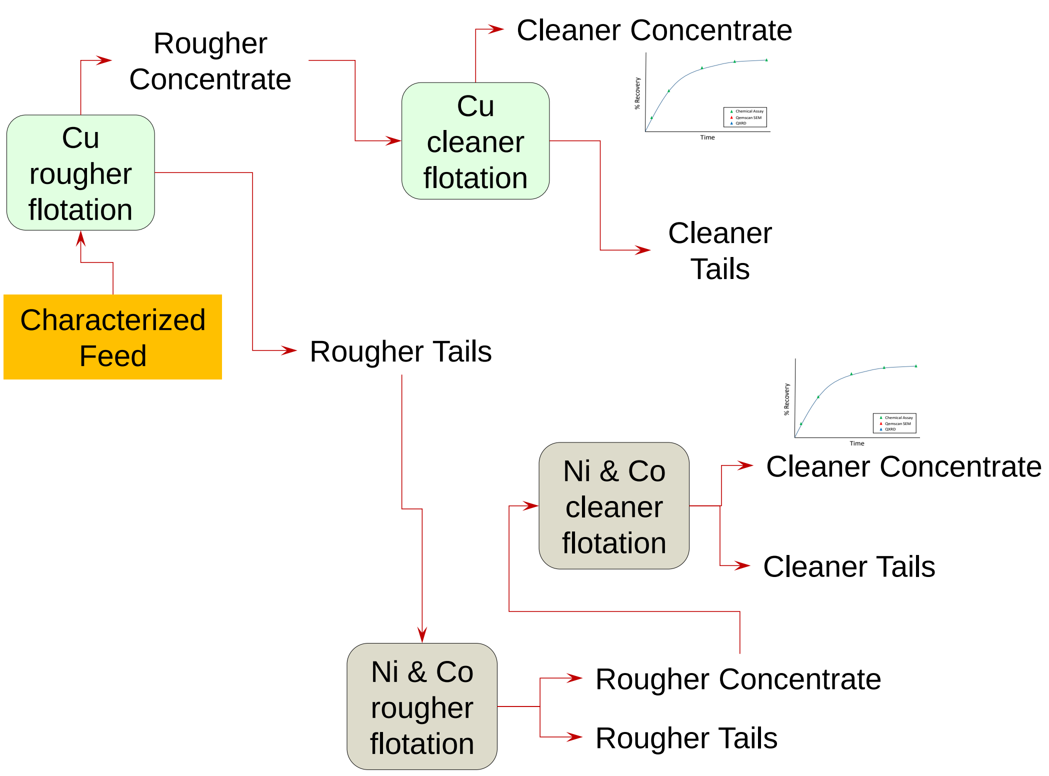


Figure 102. Flotation separation experimental design
(Image: Simon Michaux)

Sample mass requirements for a batch flotation test is about 1kg for each actual batch flotation run. As each ore type may need three to five separate tests, and sometimes sequential testing is done, thus needs larger sample feed mass, it then it is appropriate to prepare about five (5) kilogram´s for each ore type tested. This sample would then be prepared with wet grinding in a rod mill to the selected grind size (see Section 4.7). To reduce the influence of ore oxidization (which would coat each particle with a layer that would inhibit flotation), the ground sample (with some water decanted) would be vacuum packed in nitrogen, and then stored in buckets in a freezer at -18^{0}C.

4.9 Hydrometallurgy Leaching

Hydrometallurgy is a method for obtaining metals from their ores. It is a technique within the field of extractive metallurgy involving the use of aqueous chemistry for the recovery of metals from ores, concentrates, and recycled or residual materials. Metal chemical processing techniques that complement hydrometallurgy are pyrometallurgy, vapour metallurgy and molten salt electrometallurgy. Hydrometallurgy is typically divided into three general areas:

- Leaching
- Solution concentration and purification
- Metal or metal compound recovery

Leaching involves the use of aqueous solutions to extract metal from metal bearing materials which is brought into contact with a material containing a valuable metal. The lixiviant solution conditions vary in terms of pH, oxidation-reduction potential, presence of chelating agents and temperature, to optimize the rate, extent and selectivity of dissolution of the desired metal component into the aqueous phase. Through the use of chelating agents, certain metals can selectively extracted. Such chelating agents are typically amines of schiff bases.

The five basic leaching reactor configurations are in-situ, heap, vat, tank and autoclave.

1- <u>In-situ leaching</u>

In-situ leaching is also called "solution mining." The process initially involves drilling of holes into the ore deposit. Explosives or hydraulic fracturing are used to create open pathways within the deposit for solution to penetrate into. Leaching solution is pumped into the deposit where it makes contact with the ore. The solution is then collected and processed.

2- <u>Heap leaching</u>

In heap leaching processes, crushed (and sometimes agglomerated) ore is piled in a heap which is lined with an impervious layer. Leach solution is sprayed over the top of the heap, and allowed to percolate downward through the heap. The heap design usually incorporates collection sumps, which allow the "pregnant" leach solution (i.e. solution with dissolved valuable metals) to be pumped for further processing. An example is gold cyanidation, where pulverized ores are extracted with a solution of sodium cyanide, which, in the presence of air, dissolves the gold, leaving behind the nonprecious residue.

3- <u>Vat leaching</u>

Vat leaching, also called agitation leaching, and involves contacting material ore, which has usually undergone size reduction and classification, with a leach solution in large vats.

4- <u>Tank leaching</u>

Stirred tank, also called agitation leaching, involves contacting material, which has usually undergone size reduction and classification, with leach solution in agitated tanks. The agitation can enhance reaction kinetics by enhancing mass transfer. Tanks are often configured as reactors in series.

5- **Autoclave leaching**

Autoclave reactors are used for reactions at higher temperatures and pressures, which can enhance the rate of the reaction. Similarly, autoclaved enable the use gaseous reagents in the system.

After leaching, the leach liquor must normally undergo concentration of the metal ions that are to be recovered. Additionally, undesirable metal ions sometimes require removal.

- **Precipitation** is the selective removal of a compound of the targeted metal or removal of a major impurity by precipitation of one of its compounds. Copper is precipitated as its sulfide as a means to purify nickel leachates.

- **Cementation** is the conversion of the metal ion to the metal by a redox reaction. A typical application involves addition of scrap iron to a solution of copper ions. Iron dissolves and copper metal is deposited.

- **Solvent Extraction** is a mixture of an extractant in a diluent is used to extract a metal from one phase to another. In solvent extraction this mixture is often referred to as the "organic" because the main constituent (diluent) is some type of oil.

- **Ion Exchange** is where chelating agents, natural zeolite, activated carbon, resins, and liquid organics impregnated with chelating agents are all used to exchange cations or anions with the solution. Selectivity and recovery are a function of the reagents used and the contaminants present.

- **Gas reduction.** Treating a solution of nickel and ammonia with hydrogen affords nickel metal as its powder.

- **Electrowinning** is a particularly selective if expensive electrolysis process applied to the isolation of precious metals. Gold can be electroplated from its solutions.

Metal recovery is the final step in a hydrometallurgical process. Metals suitable for sale as raw materials are often directly produced in the metal recovery step. Sometimes, however, further refining is required if ultra-high purity metals are to be produced. The primary types of metal recovery processes are electrolysis, gaseous reduction, and precipitation. For example, a major target of hydrometallurgy is copper, which is conveniently obtained by electrolysis. Cu^{2+} ions reduce at mild potentials, leaving behind other contaminating metals such as Fe^{2+} and Zn^{2+}.

4.9.1 Column Leaching Characterization

Leaching hydrometallurgical response can be characterized by using a Column Leach test. This test requires 20-30kg of ore crushed to a top size of approximately 15-20mm.

Leach parameters such as, amount of sulfuric acid in the cure solution, cure technique, particle size of the material, leach solution's flowrate application, leach solution's acid strength and height of the lifts, can be evaluated using column leach tests results. This can be done by putting prepared ore aggregate in a column vessel with 12" inches leach diameter.

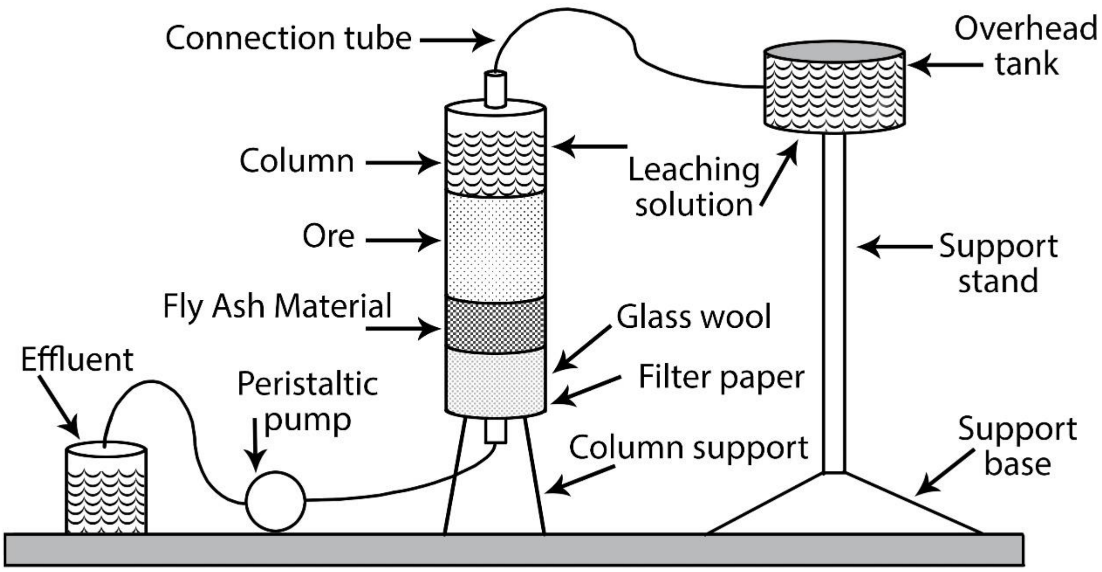


Figure 103. The Column Leach Test
(Image: Tania Michaux)

Figure 104. Laboratory scale column leach experimental setup
(Source: CSIRO Mineral Resources, Copyright: CSIRO)

The column is then placed under a drip of cyanide solution made to concentration (of around 0.1% or 1g/litre), where the flow rate from the top should be adjusted to give a constant drip approximately 2ml/min. This example is for gold leaching.

After 24 hours, and every 24 hours thereafter, the pH of the underflow solution should be taken and adjustments to the solution flow can be made if necessary. After the approximate number of days (this can be anything from 10 days to 70 days), stop the test and flood the column with hot water to clean out any residual solution. The ore in the column is laid out and dried, then sent for chemical assay characterization. The pregnant effluent collected from under the column an also sent for chemical assay. A final cyanide sample is taken to determined residual cyanide by titration as a QA/QC measure.

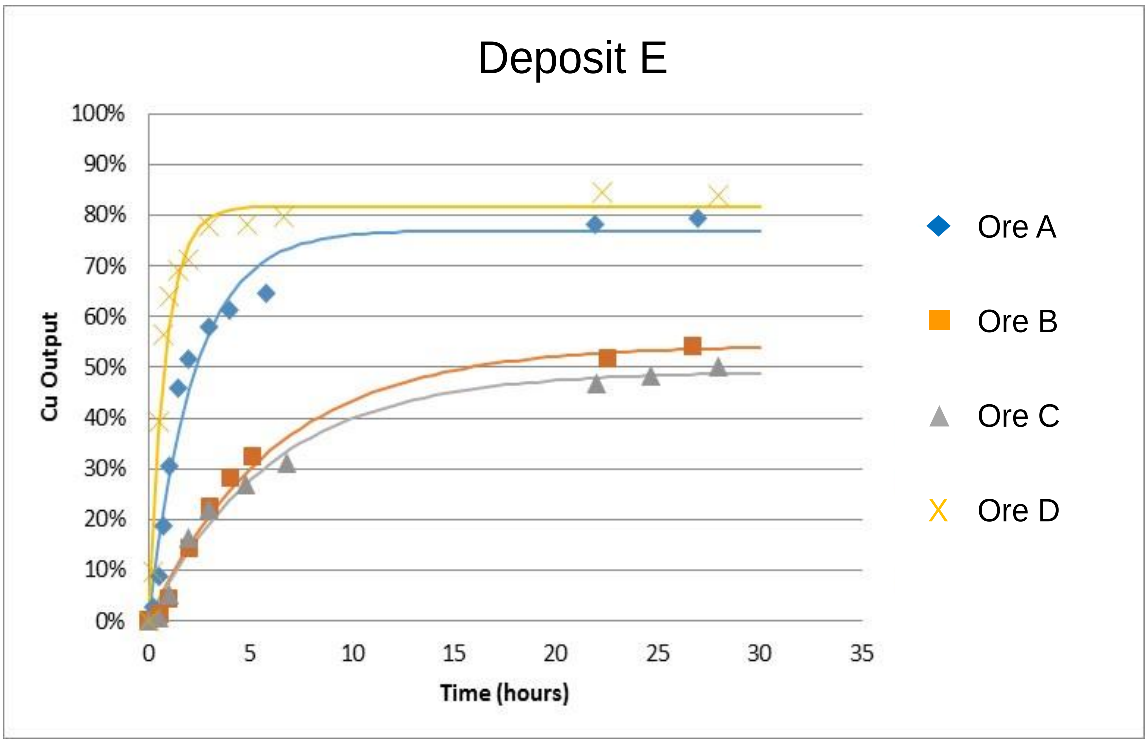


Figure 105 Leaching recovery over time for different ore types (Chart: Simon Michaux)

Sample mass required for a column leach is 20 to 30kg of -30mm crushed aggregate.

4.10 Gravity Separation

Gravity separation is an industrial method of separating two components, either a suspension, or dry granular mixture where separating the components with gravity is sufficiently practical: i.e. the components of the mixture have different specific weight. All of the gravitational methods are common in the sense that they all use gravity as the dominant force.

One type of gravity separator lifts the material by vacuum over an inclined vibrating screen covered deck. This results in the material being suspended in air while the heavier impurities are left behind on the screen and are discharged from the stone outlet. Gravity separation is used in a wide variety of industries, and can be most simply differentiated by the characteristics of the mixture to be separated - principally that of 'wet' i.e. - a suspension versus 'dry' -a mixture of granular product.

Table 3 Characteristics of metals and their minerals (Reproduced from: Klein 1999, and Gaines *et al* 1997)

Metal	Composition	Mineral	Formula	Specific Gravity	Hardness	Colour & Characteristics
Antimony	Sulphide	Stibnite (antimony glance)	Sb_2S_3	4.5 to 4.6	2.0	lead-gray, metallic luster, slightly sectile
Cobalt	Arsenide	Smaltite	$CoAs_2$	6.4 to 6.6	5.5 to 6	Tin-white, metallic, opaque
	Sulpharsenide	Cobaltite	CoAsS	6.0 to 6.3	5.5	Silver-white to reddish, cubic, metallic
Copper	Metallic	Native	Cu	8.8 to 8.9	2.5 to 3	Copper-red, malleable and ductile
	Oxide	Melaconite (black oxide)	CuO	5.8 to 6.2	3	Black
	Oxide	Cuprite (red oxide)	Cu_2O	5.8 to 6.1	3.5 to 4	Red, brittle
	Carbonate	Malachite	$CuCO_3Cu(OH)_2$	3.9 to 4.0	3.5 to 4	Green, brittle
	Carbonate	Azurite	$2CuCO_3Cu(OH)_2$	3.7 to 3.8	3.5 to 4	Azure-blue
	Sulphide	Chalcopyrite	$CuFeS_2$	4.1 to 4.3	3.5 to 4	Brass-yellow, brittle
	Sulphide	Bornite	Cu_3FeS_3	4.9 to 5.4	3.0	Copper-red to brown, brittle
	Sulphide	Covellite	CuS	4.59		Indigo-massive
Copper	Sulphide	Chalcocite (copper glance)	Cu_2S	5.5 to 5.8	2.5 to 3	Blackish lead-gray, often tarnished, sectile
	Antimonide	Tetrahedrite (gray-copper)	$Cu_8Sb_2S_7$	4.4 to 5.1	3 to 4	Flint-gray to iron-black
	Sulpharsenate	Enargite	Cu_3AsS_4	4.4	3.0	Grayish-black, brittle
	Silicate	Chrysocolla	$CuSiO_3 + 2H_2O$	2 to 2.2	2 to 4	Turquoise-blue, translucent, vitreous, rather sectile
	Oxychloride	Atacamite	$Cu_2ClH_3O_3$	3.75	3 to 3.5	Bright bottle green, brittle
Gold	Metalic	Native	Au	15.6 to 19.3	2.5 to 3	Gold-yellow, malleable and ductile
	Telluride	Sylvanite	$(Au.Ag)Te_2[Au:Ag = 1:1]$	7.9 to 8.3	1.5 to 2	Steel-gray to silver-white to yellow, brittle
	Telluride	Calaverite	$(Au.Ag)Te_2[Au:Ag = 6:1]$	9.0	2.5	Pale bronze-yellow, massive
	Telluride	Petzite	$(Au.Ag)_2Te[Au:Ag = 1:3]$	8.7 to 9.0	2.5 to 3	Steel-gray to iron-black, brittle
Graphite	Carbon	Plumbago	C	2.0 to 2.23	1 to 2	Iron-black, dark-steel gray, greasy, flexible
Iron	Metallic	Native	Fe	7.3 to 7.8	4 to 5	Steel-gray to iron black malleable
	Oxide	Hematite	Fe_2O_3	4.9 to 5.3	5.5 to 6.5	Steel-gray, red
	Oxide	Limonite	$2Fe_2O_3, 3H_2O$	3.6 to 4.0	5 to 5.5	Brown, opaque
	Oxide	Magnetite	FeO, Fe_2O_3	5.16	5.5 to 6.5	Iron-black, magnetic
	Carbonate	Siderite	$FeCO_3$	3.8	3.5 to 4	Gray-brown, reddish, brittle
	Sulphide	Pyrite	FeS_2	4.9 to 5.1	6 to 6.5	Brass-yellow, brittle
	Sulphide	Marcasite	FeS_2	4.85 to 4.9	6 to 6.5	Pale bronze-yellow, brittle
	Sulphide	Pyrrhotite	Fe_5S_6 to $Fe_{16}S_{17}$	4.58 to 4.64	3.5 to 4.5	Bronze-yellow to copper-red, easily tarnished
	Sulpharsenide	Mispickel	FeAsS	5.9 to 6.2	5.5 to 6	Silver-white, brittle
	Titanate	Ilmenite	$FeTiO_3$	4.5 to 5	5 to 6	Iron-black, slightly magnetic
	Tungstate	Wolframite	$(Fe, Mn)WO_4$	7.2 to 7.5	5 to 5.5	Brownish-black, brittle
	Chromate	Chromite	$FeCr_2O_4$	4.3 to 4.57	5.5	Iron and brown black, brittle
	Manganate	Franklinite	$(FeZnMn)O(FeMn)_2O_3$	5.1 to 5.22	5.5 to 6.5	Iron-black

Gravity dressing is done by means of water and air. Gravity separation is one of the efficient methods of separating minerals from barren rocks by using the difference in densities.

The most notable advantages of the gravitational methods are their cost effectiveness and in some cases excellent reduction. Gravity separation is an attractive unit operation as it generally has low capital and operating costs, uses few if any chemicals that might cause environmental concerns and the recent development of new equipment enhances the range of separations possible. Types of gravity separators are:

- Conventional jigs
- Pinched sluices
- Reichert cones
- Spirals
- Centrifugal jigs
- Shaking tables
- Dense medium separation
- Chute separation

Gravity separation is the most well-proven and accepted technique of concentrating minerals and has been used as a primary form of mineral concentration for centuries. Due to its high efficiency and low cost, gravity separation is always the first consideration in any mineral processing program. In the case of gold and PGE, gravity separation can quickly generate a precious metal concentrate that can be sold direct to refineries, which gives immediate payment for the work carried out.

Possible reasons to consider using Gravity Separation:

- To reject barren waste as an initial pre-concentration step
- To recover malleable and/or friable coarse heavy minerals from grinding circuit circulating loads. Such minerals are otherwise hard to recover after regrinding
- To pre-concentrate heavy minerals to minimize downstream processing costs
- To concentrate heavy minerals
- To clean low weight yield bulk concentrates
- To scavenging plant tailings
- To generate a precious metal concentrate that can go direct to a refinery rather than a smelter

4.10.1 Strong Gravity Separation

To characterize the ore to strong gravity separation response, centrifugal gravity concentrators used for gold such as the Knelson separator can be considered (IM 2014). The purpose of this is to examine the possibility of the extraction of precious metals (gold or PGE platinum group elements). The Knelson Concentrator relies on an enhanced gravitational force together with fluidization process to recover even very micron sized particles (Wills & Finch 2015). First water is injected into the rotating concentrating cone through series of fluidization holes. Then feed slurry is introduced through a stationary feed tube. Once the slurry is filled in each cone, create a concentrating bed and high specific gravity particles are retained in the cone and then flushed into the concentrate launders. This procedure can be completed in less than one minute.

Figure 106 below shows a possible experimental program for strong gravity separation using a Knelson unit. The heavy fraction of a Knelson separator would be quite small in volume compared to the lighter fraction (2-3% of the feed mass). This needs to be considered when planning possible further testwork of the heavy fraction. The sample products are then to be characterized appropriately.

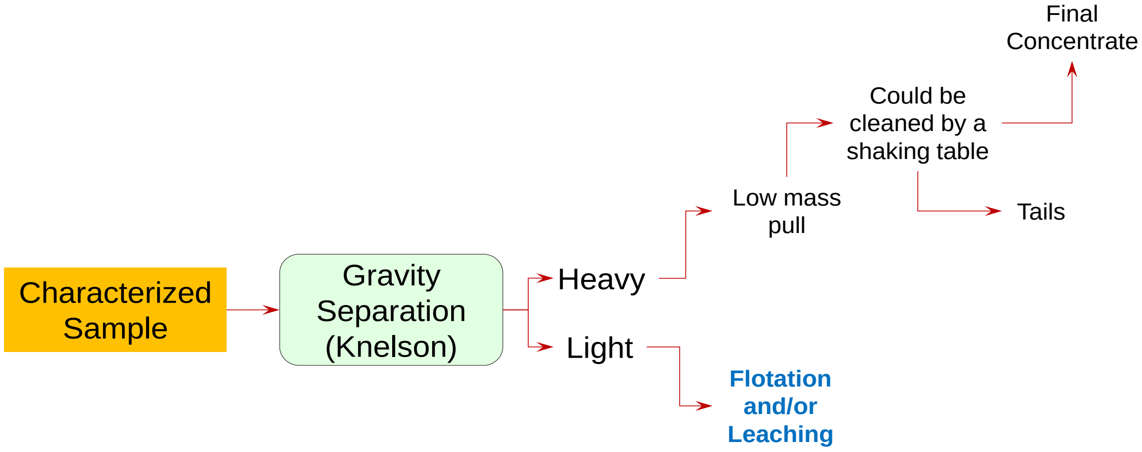


Figure 106. Strong gravity separation experimental design (Simon Michaux)

The light fraction could then be sent to flotation and/or leaching recovery. Sample mass requirements for Knelson gravity testing can be as little as 200g but should be at least 2kg. The particle size can vary but should be a sample feed F_{80} of approximately of 100μm.

4.10.2 Weak Gravity Separation

To characterize the ore to weak gravity separation response, a gravity spiral concentrator separator and/or shaking tables can be used for fine grained minerals. A gravity jig can be used for recovery of coarse grained minerals. The purpose of this is to examine the possibility of the extraction of gangue minerals that could negatively impact recovery.

Spiral concentration separators of the wet type, also called spiral concentrators, are devices to separate solid components in a slurry, based upon a combination of the solid particle density as well as the particle's hydrodynamic properties (e.g. drag). The device consists of a tower, around which is wound a sluice, from which slots or channels are placed in the base of the sluice to extract solid particles that have come out of suspension.

As larger and heavier particles sink to the bottom of the sluice faster and experience more drag from the bottom, they travel slower, and so move towards the center of the spiral. Conversely, light particles stay towards the outside of the spiral, with the water, and quickly reach the bottom. At the bottom, a "cut" is made with a set of adjustable bars, channels, or slots, separating the low and high density parts. Many things can be done to improve the separation efficiency, including:

- changing the rate of material feed
- changing the grain size of the material
- changing the slurry mass percentage
- adjusting the cutter bar positions
- running the output of one spiral separator (often, a third, intermediate, cut) through a second.
- adding washwater inlets along the length of the spiral, to aid in separating light minerals
- adding multiple outlets along the length, to improve the ability of the spiral to remove heavy contaminants
- adding ridges on the sluice at an angle to the direction of flow.

Typical spiral concentrators will use a slurry from about 20% - 40% solids by weight, with a particle size somewhere between 1.5-0.075 mm (17-340 mesh), though somewhat larger particle sizes are sometimes used. For good separation, the density difference between the heavy minerals and the light minerals in the feedstock should be at least 1 g/cm^3; and because the separation is dependent upon size and density, spiral separators are most effective at purifying ore if its particles are of uniform size and shape. A spiral separator may process a couple tons per hour of ore, per flight, and multiple flights may be stacked in the same space as one, to improve capacity.

A shaking gravity table is a mechanized gold pan, that operates with a high degree of efficiency and continuously. The table is comprised of a deck, in somewhat of a rectangular shape, covered with riffles (raised bars running perpendicular to the feed side of the table), mounted in a near flat position, on a supporting frame that allows the table to slide along the long axis of the table.

Figure 107 shows a laboratory (or bench) scale gravity shaking table. Figure 108 shows a pilot scale (5 tonne per hour) gravity shaking table.

Figure 107. Bench scale shaking table gravity separation
(Image: David Bastin & Walter Rivera, ULiege University of Liege)

The mechanism is attached to the table, and it moves the table along the long axis a distance adjustable between 10mm and 30mm and then back to the starting position between 200 and 300 times per minute. This reciprocal movement is faster on the reverse stroke than it is on the forward stroke. This shaking movement helps transport the concentrates or heavy material to the concentrate end of the table. A very important operating variable of a shaking table is the tilt adjustment. Normally, the feed side is lower, and the concentrate end is higher on the table, which creates an upward slope where the heavy material will ascend, while the light density material will not, and consequently, flow over the riffles. The tailing (low density) side is usually near level to lower than the feed side.

The gravity concentrating tables remove the high density material from the low density material, since the high density material will reside behind the riffles and allow the low density material to flow over the top of the riffles with the wash water, to the tailings discharge. As a general rule, the frequency and stroke relationship are similar to screens, short stroke, high frequency is better for fines (-180µm), while a shorter stroke and lower frequency is better for coarse material (-3+0.180mm).

Figure 108. The gravity circuit section of the GTK Mintec process pilot scale plant, gravity spirals and shaking table (Image: Arno-Matti Kirpala)

Mineral jigs are a type of mining equipment, also referred to as gravity concentrators or jig concentrators that are used in operations to separate different ore materials based on their densities. Usually they will process material that is a similar size after the ore has passed through a crusher or over a screening plant. The particles are introduced to the jig bed (usually a screen) where they are thrust upward by a pulsing water column or body, resulting in the particles being suspended within the water. As the pulse dissipates, the water level returns to its lower starting position and the particles once again settle on the jig bed. As the particles are exposed to gravitational energy whilst in suspension within the water, those with a higher specific gravity (density) settle faster than those with a lower count, resulting in a concentration of material with higher density at the bottom, on the jig bed. The particles are now concentrated according to density and can be extracted from the jig bed separately.

Figure 109 below shows a possible experimental program for weak gravity separation using a gravity spiral unit. The lighter fraction could potentially contain minerals like quartz, clays or talc, all of which negatively impact flotation.

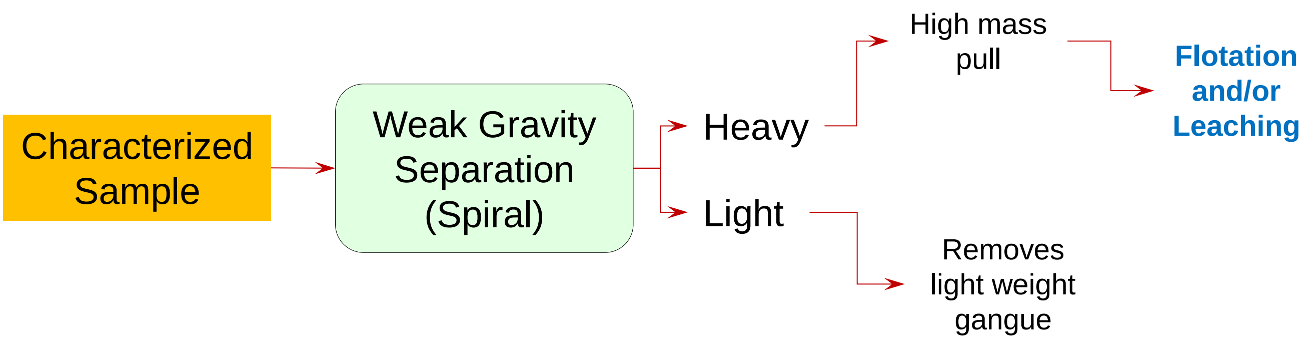


Figure 109. Weak gravity separation experimental design

The heavy fraction could be sent flotation and/or leaching recovery. In doing so, the recovery could be greatly improved once the ore had been cleaned of minerals that could reduce efficiency.

Sample mass requirements for gravity spiral testing should be at least 20 to 30kg (needed to fill the spiral). It is possible to use a micro spiral which can take a much smaller feed sample. The particle size can vary but should be a sample feed F_{80} of approximately of -1mm + 75µm.

Figure 110 below is an idealized experimental design, combining Figure 106 and 109. The strong gravity separation could remove a portion of the precious metals (Au & PGE) and/or any elements that are too heavy to separate with flotation. Then, weak gravity separation could remove the lighter gangue minerals which could reduce recovery.

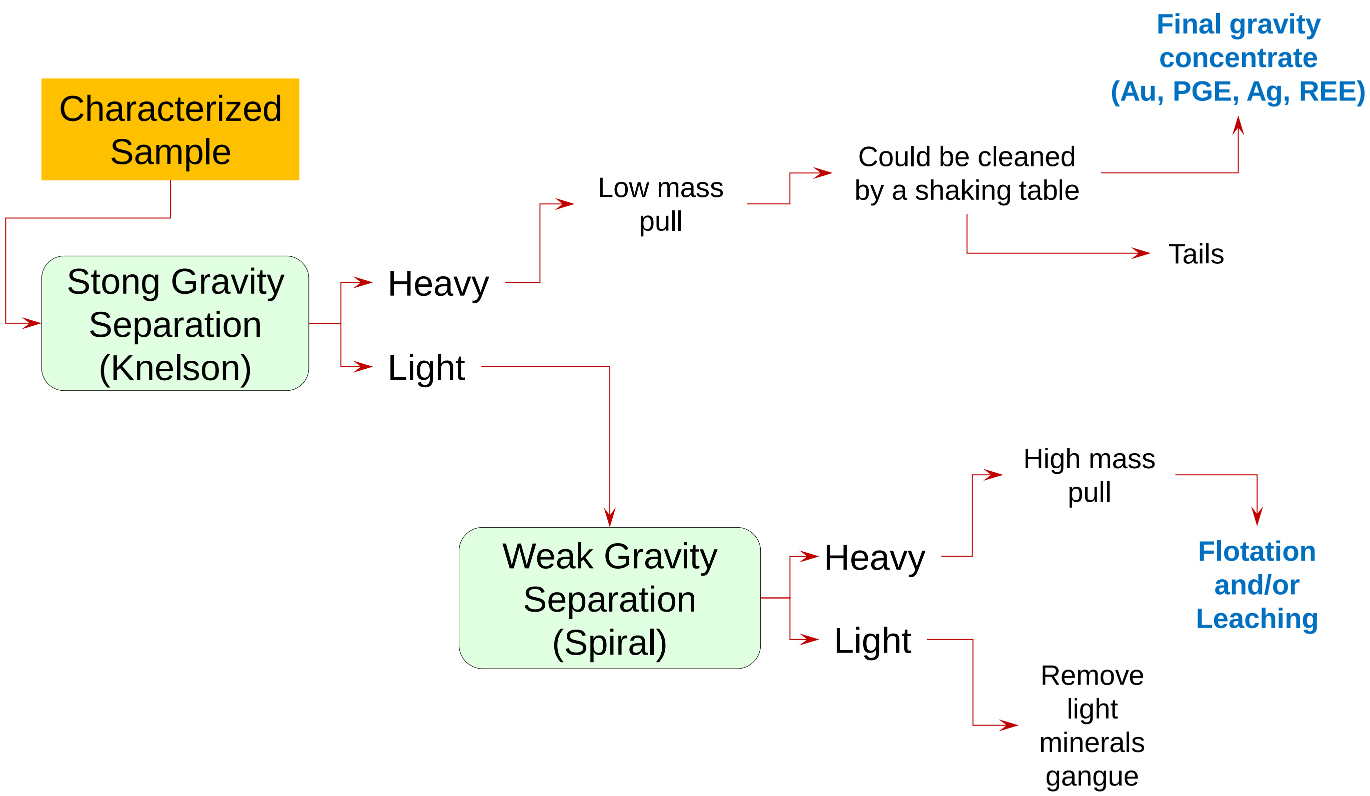


Figure 110. Ideal gravity separation experimental design
(Image: Simon Michaux)

One issue to consider with this design is the difference in sample feed between the two units, the Knelson separator and the gravity spiral. They may require different feed sizes and different sample masses to be efficient.

4.11 Magnetic Separation

Magnetic separators exploit the difference in magnetic properties between the minerals in a deposit and are used to concentrate a valuable mineral that is magnetic (e.g., magnetite from quartz), to remove magnetic contaminants, or to separate mixtures of magnetic and nonmagnetic valuable minerals (Wills 2015).

It is not practical to apply electromagnetic separation to an ore stream before flotation, due to the volumes required. What usually is feasible is to use electromagnetic separation to clean impurities from the flotation concentrate.

All materials are affected in some way when placed in a magnetic field, although with many substances the effect is too slight to be easily detected. For the purposes of mineral processing, materials may be classified into two broad groups, according to whether they are attracted or repelled by a magnet:

1. **Diamagnetic** materials are repelled along the lines of magnetic force to a point where the field intensity is smaller. The forces involved here are very small and diamagnetic substances are often referred to as "nonmagnetic", although this is not strictly correct. Diamagnetic minerals will report to the nonmagnetic product ("non-mags") of a magnetic separator as they do not experience a magnetic attractive force.

2. **Paramagnetic** materials are attracted along the lines of magnetic force to points of greater field intensity. Paramagnetic materials report to the "magnetic" product ("mags") of a magnetic separator due to attractive magnetic forces. Examples of paramagnetic minerals which are separated in commercial magnetic separators are ilmenite ($FeTiO_3$), rutile (TiO_2), wolframite ($(Fe, Mn)WO_4$), monazite ($(Ce, La, Nd, Th)PO_4$), xenotime (YPO_4), siderite ($FeCO_3$), chromite ($FeCr_2O_4$), and manganese minerals.

Figure 111 shows a laboratory scale (also called bench scale) magnetic separation unit in operation.

Figure 111. Bench scale magnetic separation process testing
(Image: David Bastin & Walter Rivera, ULiege University of Liege)

Table 4. Magnetic separation characteristics of minerals (Source: reproduced from 911Metallurgist.com)

Specific Gravity	Non-Conductors (High Tension Pinned): Magnetic	Non-Conductors (High Tension Pinned): Weakly Magnetic	Non-Conductors (High Tension Pinned): Non Magnetic	Conductors (High Tension Thrown): Highly Magnetic	Conductors (High Tension Thrown): Magnetic	Conductors (High Tension Thrown): Weakly Magnetic	Conductors (High Tension Thrown): Non Magnetic
Over 8.0							Gold Copper
8.0							
7.5					Ferberite	Wolframite	Galena Cassiterite
7.0							
6.5							
6.0			Scheelite			Columbite Tantalite	
5.5							
5.0	Monazite	Bastnasite		Magnetite Ilmenite (high iron)	Ilmenite Davidite	Samarskite Euxenite Hematite Chromite	Pyrite
4.5	Xenotime		Zircon Barite				Molybenite
4.0	Garnet		Corundum Celestite Perovskite				Rutile Chalcopyrite Brookite Limonite
3.5	Siderite Staurolite	Epidote Olivine	Kyanite Topaz Sphene				Diamond
3.0		Apatite Hornblende Tourmaline	Sillimanite Flourite Anhydrite Mica (Muscovite)				
2.5		Mica (Biotite)	Beryl Feldspars Calcite				Graphite
2.0			Quartz Gypsum Chrysotile Sulphur				
Under 2.0							

4.11.1 Low-intensity magnetic separation - ferrite magnet LIMS

Low-intensity separators are used to treat ferromagnetic materials and some highly paramagnetic minerals. Minerals with ferromagnetic properties have high susceptibility at low applied field strengths and can therefore be concentrated in low intensity (<~0.3T) magnetic separators. This is usually a dry process but also can be a wet process. This often depends on the particle size of the feed.

4.11.2 Medium intensity magnetic separation - permanent magnet MIMS

Permanent magnetic drum separators combine the attributes of a high-strength permanent magnetic field and a self-cleaning feature. These separators are effective in treating process streams containing a high percentage of magnetics and can produce a "clean" magnetic or non-magnetic product. The magnetic drum separator consists of a stationary, shaft-mounted magnetic circuit completely enclosed by a rotating drum. The magnetic circuit is typically comprised of several magnetic poles that span an arc of 120 degrees. When material is introduced to the revolving drum shell (concurrent at the 12 o'clock position), the non-magnetic material discharges in a natural trajectory. The magnetic material is attracted to the drum shell by the magnetic circuit and is rotated out of the non-magnetic particle stream. The magnetic material discharges from the drum shell when it is rotated out of the magnetic field.

4.11.3 High intensity magnetic separation – electromagnetic HIMS

High-intensity magnetic separators are effective in collecting paramagnetic particles. These focus on the separation of very fine particles that are paramagnetic. This is a wet process.

The current is passed through the coil, which creates a magnetic field, which magnetizes the expanded steel matrix ring. The paramagnetic matrix material behaves like a magnet in the magnetic field and thereby attracts the fines. The ring is rinsed when it is in the magnetic field and all the non-magnetic particles are carried with the rinse water. Next as the ring leaves the magnetic zone the ring is flushed and a vacuum of about – 0.3 bars is applied to remove the magnetic particles attached to the matrix ring.

In a geometallurgical experimental campaign, there would most probably not be enough mass in any given flotation concentrate to test directly with magnetic separation. What can be done though is the magnetic separation of a given ore type in a series of increasing magnetic intensities. After each separation step, the product concentrate could be characterized appropriately. The comparatively non-magnetic tails of each separation could then be subject to a stronger intensity magnetic separation. Once the last separation step had been completed, the tails could be characterized appropriately, and then a mineral reconciliation could be done across the whole electromagnetic process.

In doing so, each ore type could be characterized in context of what minerals are present that would be susceptible to magnetic separation. This has several possible useful outcomes. Some minerals that are magnetic (see Table 4) are also antagonistic to pyrometallurgical refinement. Thus removing them would increase the value of ore concentrate. Also, some magnetic minerals can also be closely associated with valuable metals (e.g. platinum embedded in magnetic pyrrhotite).

Figure 112 below shows an experimental design to determine the ore response to magnetic separation, using a laboratory scale electromagnetic unit. Each separation step was to be done at a different applied current. The ferrite LIMS could be estimated at a 1 Ampere current. A permanent magnet MIMS estimated with a 2 Ampere current. An electromagnetic HIMS could be estimated with a 3 Ampere current.

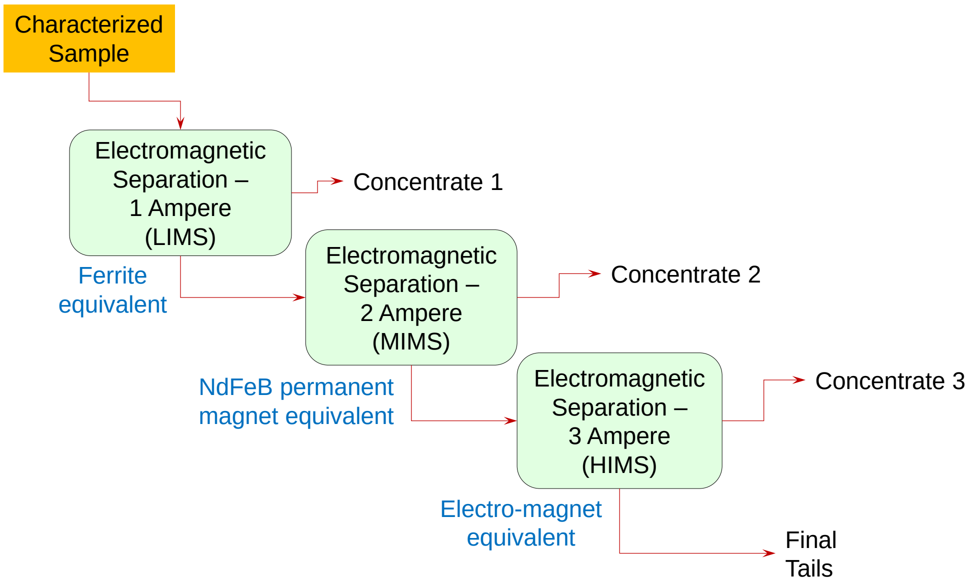


Figure 112. Electromagnetic separation experimental design
(Image: Simon Michaux)

Sample mass requirements for magnetic testing should be at least 2 to 5kg. The particle size can vary but should be a sample feed F_{80} similar (the same) as what is feed to flotation and leaching. This would be defined by the selected grind size (see Section 4.7).

4.12 Ore Sorting

Modern, automated sorting applies optical sensors (visible spectrum, near infrared, X-ray, ultraviolet), that can be coupled with electrical conductivity and magnetic susceptibility sensors, to control the mechanical separation of ore into two or more categories on an individual rock by rock basis. Also new sensors have been developed which exploit material properties such as electrical conductivity, magnetization, molecular structure and thermal conductivity. Sensor based sorting has found application in the processing of nickel, gold, copper, coal and diamonds.

There are many ways of improving recovery processes for a wide variety of ores using sorting equipment. Sensor-based ore particle sorting has the potential for a powerful method for pre-concentration or waste rejection early in the comminution process.

This process has the potential to upgrade the value of the ore by increasing the grade, and/or remove a penalty mineral that would negatively impact the recovery of the valuable target metal (Figure 113). This is usually done dry, with crushed ore feed of size -100mm. The viability of sorting is ore specific and is not traditionally used. That being stated, technology is evolving quickly and at some point sorting will become viable. When this happens sorting technology has the potential to make whole mining operations viable where before they were not.

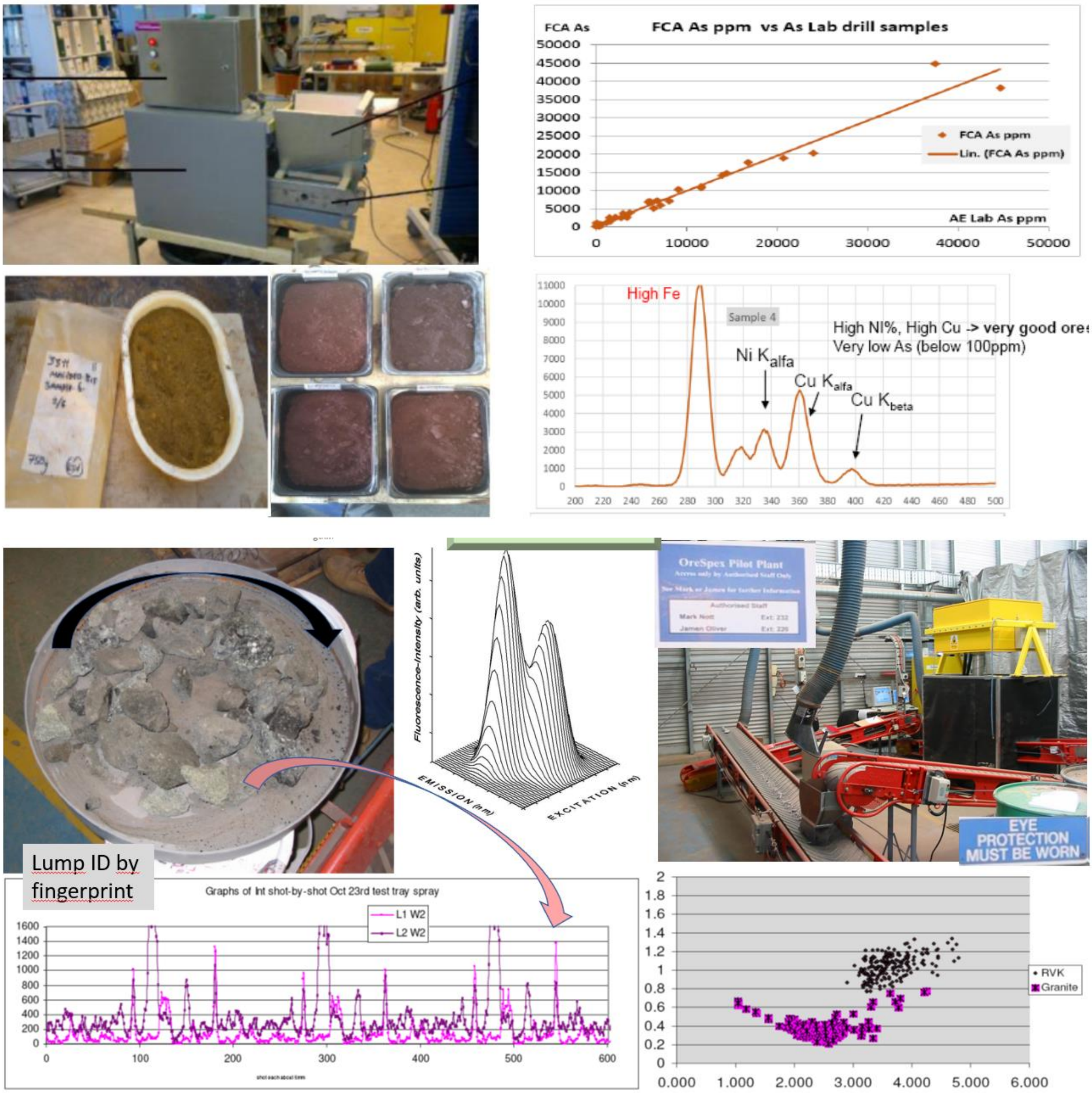


Figure 113. Sorting of ore using an XRF characterization technique (Source and Images: IMA Engineering Finland)

There are several ways for ore to be sorted. One method uses air jets to separate out the target sub-fraction. The ore characterization technology (based around the principles of XRF) analyzes each and every rock particle on-line for unique physical and chemical properties and separates the particles by high pressure air jets. This maximizes recovery or upgrade and is suitable for various minerals and applications. A shortcoming of this method is the use of air jets consumes a considerable amount of power (therefore is expensive). This often is enough to make this method uneconomical.

To determine whether sorting would be effective, an assessment study on the range of end member rock textures within the ore being extracted needs to be done. As each rock texture would respond differently and to an efficiency curve, this fits the geometallurgy paradigm well. If sorting is viable, it should be considered as a separation process in the same context as flotation or hydrometallurgy leaching. As such ore should be characterized so all effective processes can be optimized together.

4.13 Pyrometallurgy

Following separation and concentration by mineral processing, metallic minerals are subjected to extractive metallurgy, in which their metallic elements are extracted from chemical compound form and refined of impurities. This is done with the products from flotation and leaching hydrometallurgy.

Metallic compounds are frequently rather complex mixtures (those treated commercially are for the most part sulfides, oxides, carbonates, arsenides, or silicates), and they are not often types that permit extraction of the metal by simple, economical processes. Consequently, before extractive metallurgy can affect the separation of metallic elements from the other constituents of a compound, it must often convert the compound into a type that can be more readily treated.

Pyrometallurgy is a branch of extractive metallurgy. It consists of the thermal treatment of minerals and metallurgical ores and concentrates to bring about physical and chemical transformations in the materials to enable recovery of valuable metals. Pyrometallurgical treatment may produce products able to be sold such as pure metals, or intermediate compounds or alloys, suitable as feed for further processing. Examples of elements extracted by pyrometallurgical processes include the oxides of less reactive elements like iron, copper, zinc, chromium, tin, and manganese.

Pyrometallurgical processes are generally grouped into one or more of the following categories:

- calcining,
- roasting,
- smelting,
- refining.

Most pyrometallurgical processes require energy input to sustain the temperature at which the process takes place. The energy is usually provided in the form of combustion or from electrical heat. When sufficient material is present in the feed to sustain the process temperature solely by exothermic reaction (i.e. without the addition of fuel or electrical heat), the process is said to be "autogenous". Processing of some sulfide ores exploit the exothermicity of their combustion.

The efficiency and effectiveness of any given pyrometallurgical process is often related to the character of the feed material. Impurities and toxins in the feed material are subject to penalty rates and directly affect the value of an ore concentrate. To characterize the feed, the following methods are useful:

- Chemical Assay
- Bulk QXRD
- XRF pellet
- Automated mineralogy in some circumstances

5 SMALL SCALE PROXY TESTS

Also, a geometallurgy campaign will require multiple measurements of different kinds of the same small volume. This means that small scale tests are required. The AMIRA P843 and P843A projects were successful in developing many such tests (AMIRA P843 2008a & 2008b, AMIRA P843 2009, AMIRA P843A 2010a, 2010b & 2010c, AMIRA P843A 2011a & 2011b, AMIRA P843A 2012a & 2012b and AMIRA P843A 2013). To reach the geometallurgy campaign objectives, a large number of successful tests are required (where the larger scale bankable tests usually are done in much smaller numbers for each campaign). A series of tests that require only very small sample masses, are inexpensive and have a fast turnaround times. Table 5 below shows a list of some of these comparative tests.

Table 5. Small scale comparative proxy tests for process engineering ore characterization

Process Engineering Field	Test	Reference
Geotechnical	Point Load Index (PLT)	Brady & Brown 2006
Comminution	RBT SMC JKCi SPi	Wills & Napier-Munn 2005 Wills & Napier-Munn 2005 JKTech Wills & Napier-Munn 2005 Kojovic *et al* 2011 Wills & Napier-Munn 2005 SGS 2018
Flotation	Separability Indicator (JKMSI)	Vos & Bradshaw 2014 Chauhan 2013 Morgan *et al* 2012
Hydrometallurgy Leaching	CSIRO Diagnostic Leach Test	Kuhar *et al* 2011 Li *et al* 2016
Acid Mine Drainage	Geo -environmental modelling Rinse Rinse /paste pH Carbonate staining	Parbhakar-Fox 2015 Parbhakar-Fox 2016 Parbhakar-Fox 2013
Meso scale mineralogy	Hyperspectral image analysis	Schodlok *et al* 2016
Rock Hardness	EQUOtip hardness	Keeney & Nguyen 2014

It is to be remembered that a proxy test is an estimate of a correlation between a physical/chemical/mineralogical phenomenon and a test response. This correlation can be related to the larger scale tests as shown in Section 4. Each ore type family will respond differently to this relationship. Also, there is the problem of what the test results actually mean and the same phenomena behavior at a production scale. There are all sorts of difficulties in scaling up tests between laboratory scale and production scale. It is for this reason to be mindful of what tests are being used where and for what purpose.

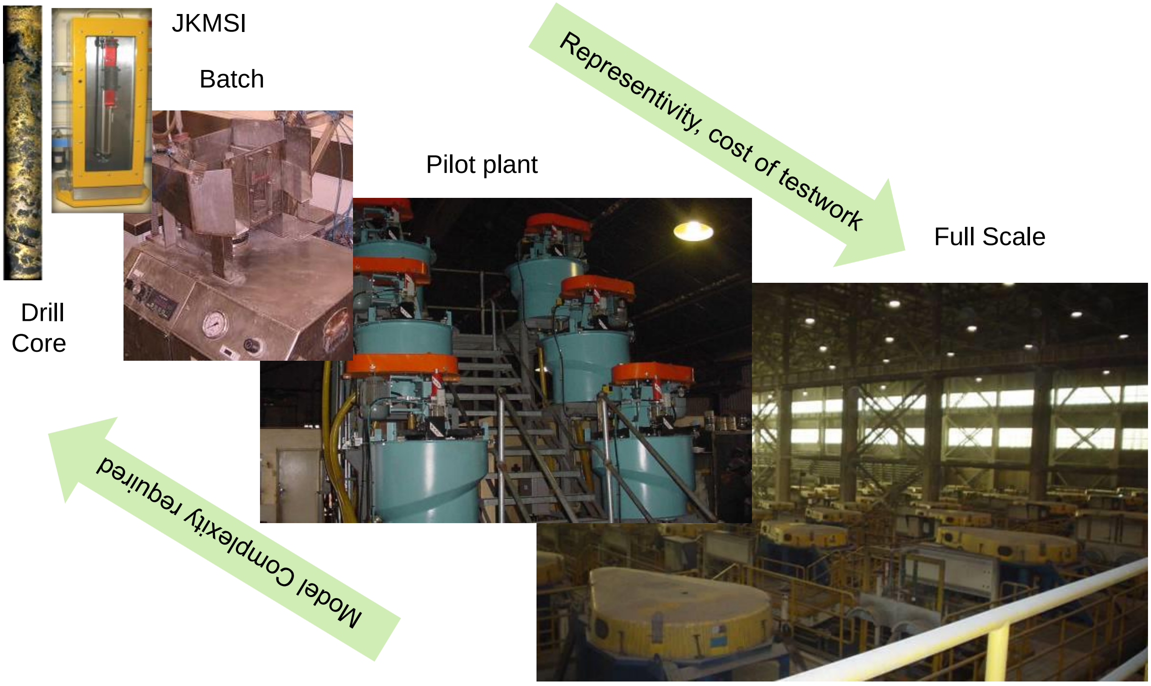


Figure 114. Proxy tests and scale of precision
(Image Copyright: Dee Bradshaw)

One of the purposes of the Orientation Study is to examine and establish the best combination of small-scale tests that describe the target process behavior and develop a working relationship to the relevant bankable tests. The approach should be that every deposit is different and unique. Some tests will work well, whereas others will not, in a deposit specific fashion.

5.1 EQUOtip test

This is a hardness tester developed for steel liners, which is based on the concept of the Schmitt hammer. Now used on drill core as a non-destructive test (Keeney 2014 and Verwaal 1993) that is to be done on intact drill core while it still sits in the core tray. Can be done on lots of drill core easily and can be used as a useful domaining tool. The EQUOtip output is also useful as a Geometallurgy model input.

While the core was still in the tray, the EQUOtip test was done across all core at 2cm intervals (Keeney 2014 and Verwaal 1993). The EQUOtip is a non-destructive test that was originally designed to measure the hardness of steel (Figure 115). It operates very similar to a Schmitt Hammer (Figure 116). While it does not directly correlate with rock hardness as it is empirically measured, it is an excellent model input for comminution.

Figure 115. EQUOtip Impact Device
(Source: AMIRA P843, AMIRA 2008b, Copyright AMIRA)

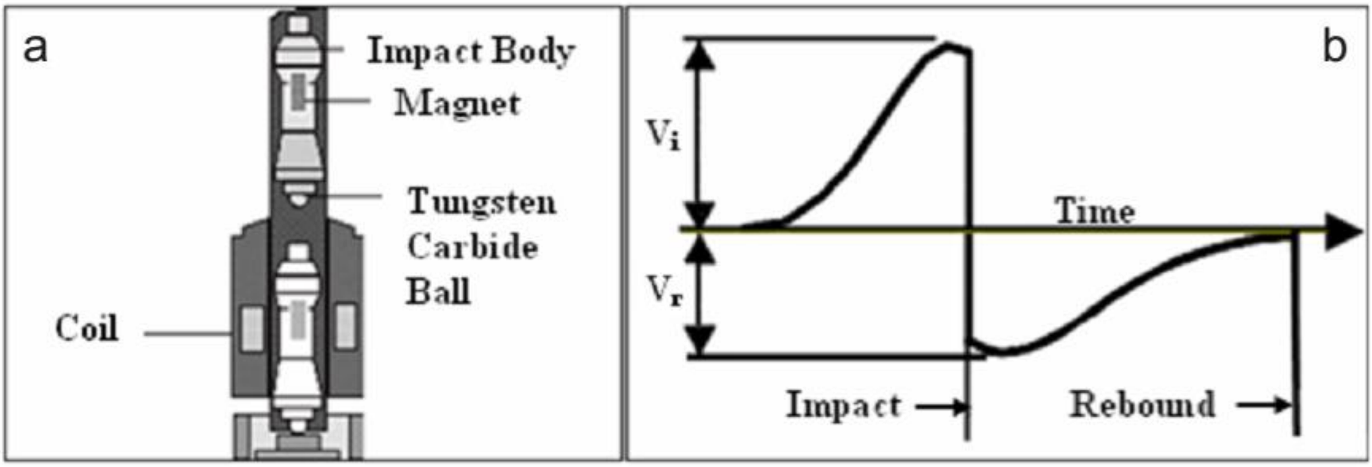


Figure 116. (a) Cross section of EQUOtip impact device and (b) voltage signal generated by movement of the impact body (Source: AMIRA P843, AMIRA 2008b, Copyright AMIRA)

EQUOtip is a measurement that is different to chemical assay analysis, or visual measurement like hyperspectral imaging. As such it often provides a useful contrast in domaining drill core. Figure 117 shows an example of continuous EQUOtip measurements on core. The raw data is variable, and it is required to smooth the data to the point where it is possible to discern domains.

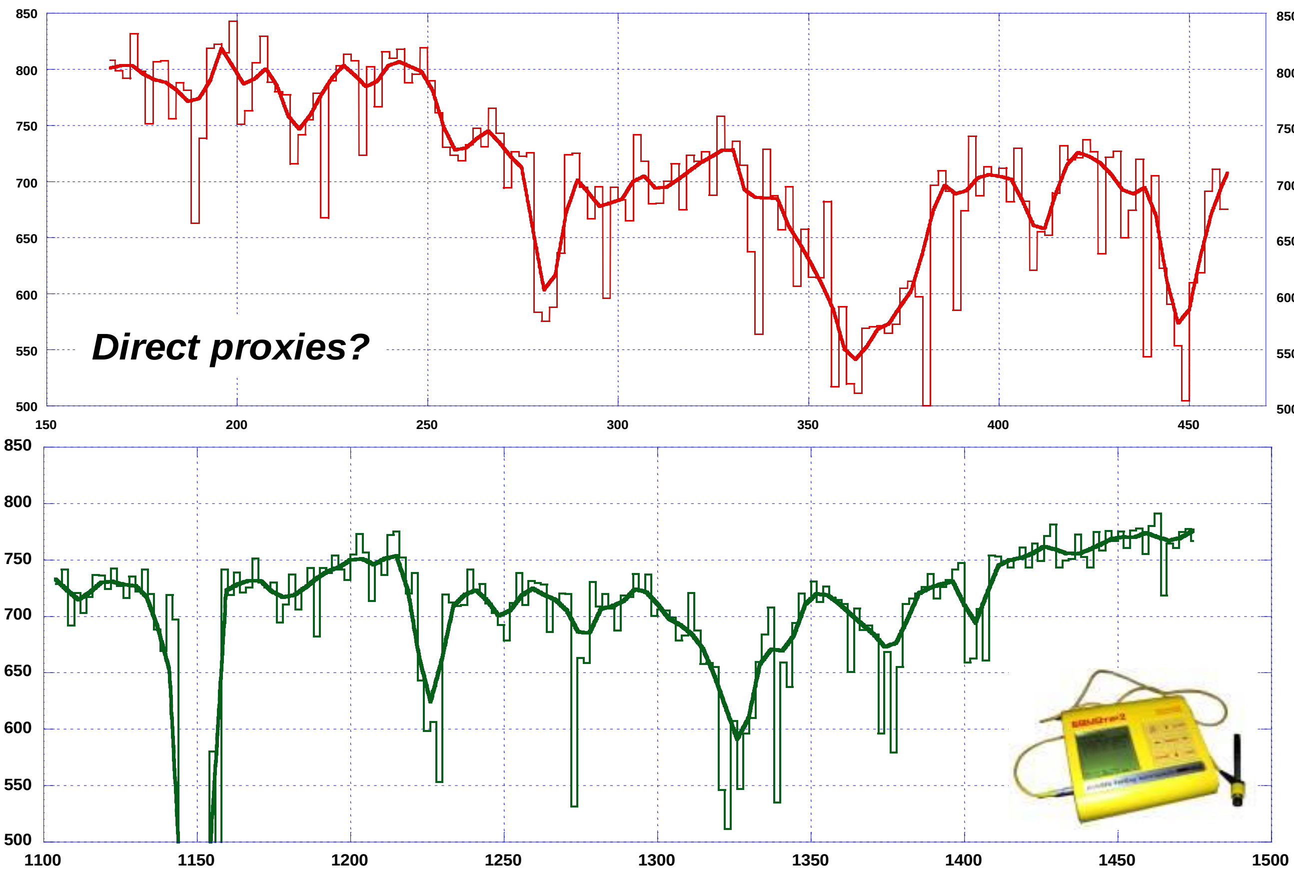


Figure 117. EQUOtip hardness tester used as a domaining tool
(Source: Amira 2008b, Image: Steve Walters, Copyright: AMIRA)

Figure 118 shows a probability plot of all EQUOtip data for 7 different deposits, each with very different geological and lithological characteristics. This is shows an interesting opportunity. What can be seen is the deposits all have a very different deposit scale signatures. If EQUOtip was done as standard practice as a test done in exploration campaigns, it could be a very fast assessment of what signature each new deposit could be against mature deposits that have been measured in the same fashion. This could be a very early indicator to some of the challenges a deposit might present if it was mined and then processed.

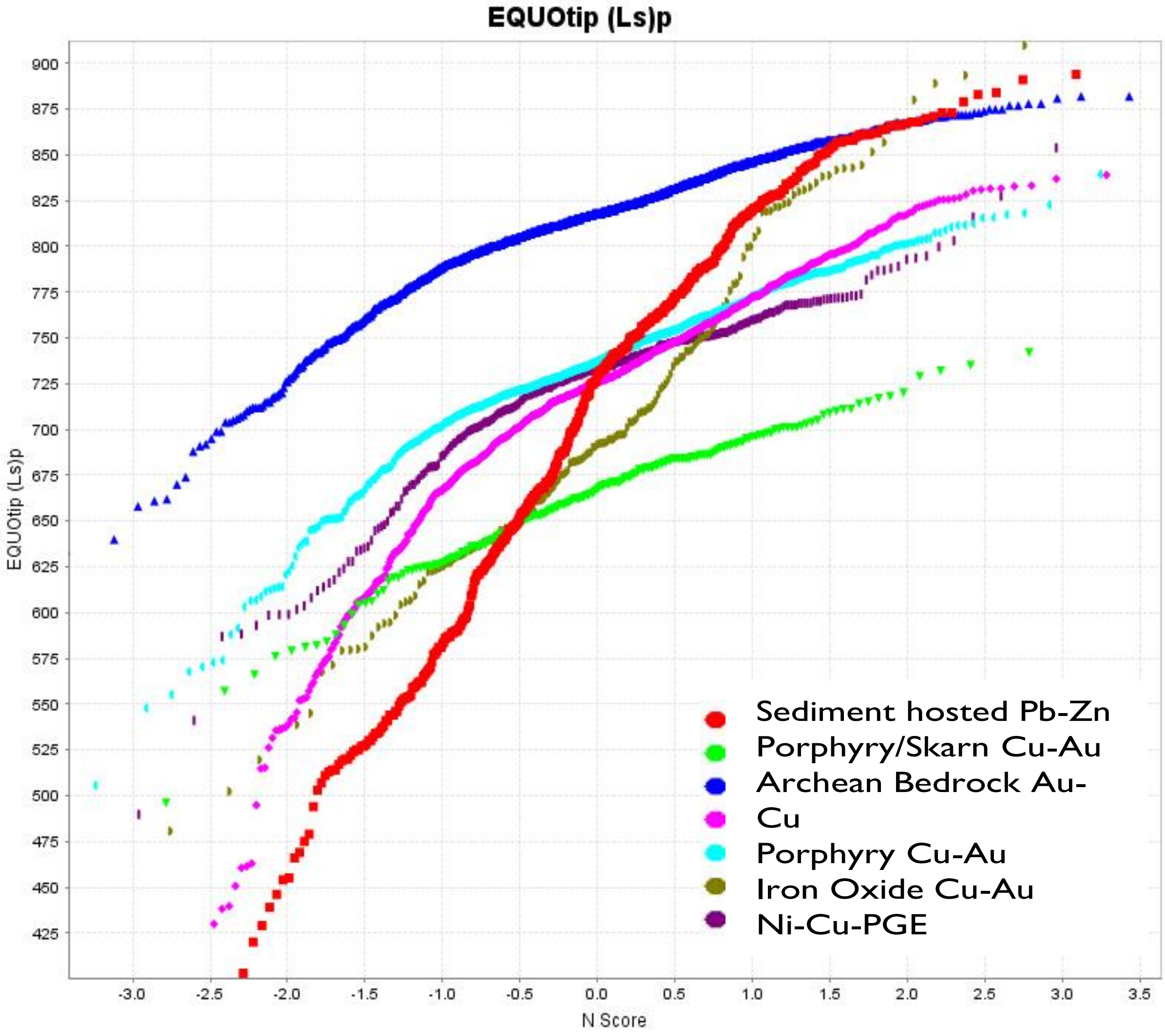


Figure 118. Deposit scale EQUOtip comparison
(Source: AMIRA 2008b, Image: Steve Walters, Copyright: AMIRA)

5.2 Point Load Index Measurement

The Point Load Index is a geomechanical test used to measure rock fragment strength (Broch and Franklin 1972) and has been correlated with rock mass parameters like UCS compressive strength and tensile strength (ISRM 1985 and Butenuth 1997). It was originally designed to test 50mm diameter cores, but there is an empirically derived correction model that allows the testing of irregular shaped fragments and ½ drill core (Brook 1985 and Brook 1980).

The device comprises of two point-shaped "platens" between which the specimen rock is placed. One of the platens is effectively stationary (though its initial starting position can be adjusted) whilst the other is free to move through the application of pressure, delivered via a hand pump and piston arrangement. As the hand pump is activated the pressure and hence force applied to the rock is increased and eventually causes the rock to fail. The peak pressure applied is indicated on a pressure gauge.

A successful/appropriate measurement for the Point Load Index is based on a criterion for fragment failure. Did the fragment fail completely between the two platens? If the answer to this is no, then the result must be discarded.

The standard formula for Point Load Index calculation is as follows:

$$Is_{50} = F\ P/(D_e)^2 \qquad \text{Equation 6}$$

Where:

F = size correction factor (used for irregular particles)

$= (D_e/50)^{0.45}$ Equation 7

P = force at failure and is calculated from the pressure and geometry of the hydraulic system.

De $= (4A/\pi)^{0.5}$ Equation 8

A = minimum cross-sectional area of specimen

There are several methods of sample selection and manner of conducting the Point Load Index (Figure 119).

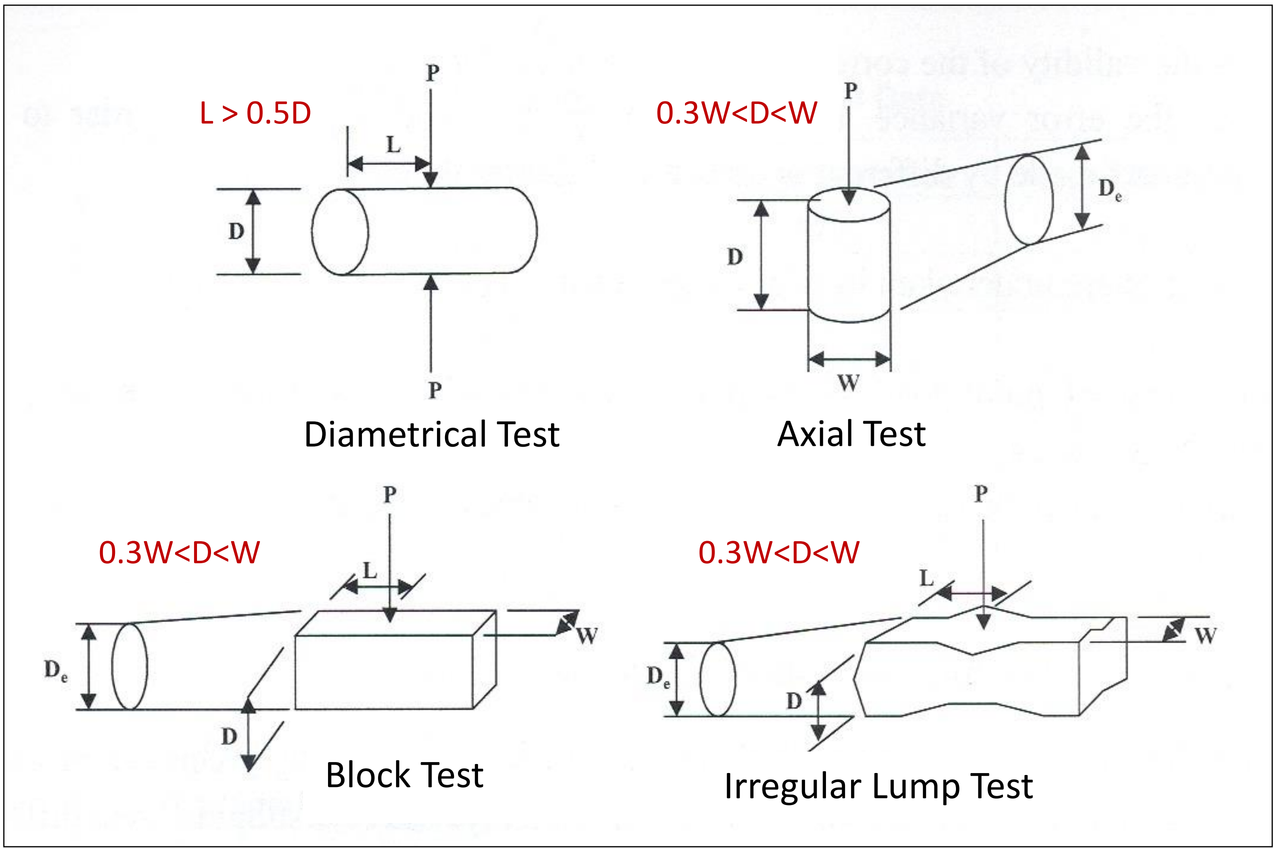


Figure 119: Sample shape requirements for the Point Load Index (Source: ISRM, 1985) (Image: Simon Michaux)

5.2.1 Diametral test

The Point Load Index was originally developed to test core rock samples with a length (L)/diameter (D) ratio greater than 1 (Figure 119).

- The core is placed into the tester so that the platens catch the sample at a point where the full diameter of the core is being loaded between them.
- The distance between the platen contact points (D) is to be recorded along with the specimen width (W) at an accuracy of +/- 2%.
- The sample is loaded over a time period between 10 and 60 seconds.
- The test is rejected if the sample did not fail through a plane passing through the two platens where the rock is held.

The full procedure for the diametrical point load test is shown in ISRM 1985.

5.2.2 Axial test

Core specimens have a ratio of length (L)/diameter (D) between 0.3 and 1.0 for axial testing.

- The sample is inserted into the tester as per shown in Figure 119 until the platens have contacted the sample.
- The distance between the platens (D) is measured along with the specimen width (W) to an accuracy of +/-2 to 5%.
- The sample is loaded over a time period between 10 and 60 seconds.
- The test is rejected if the sample did not fail through a plane passing through the two platens where the rock is held.

5.2.3 Block and irregular lump tests

Irregular lumps of rock (for example from the muckpile) can be also tested with a core correction factor (Equation 7). This is very useful, as the samples no longer have to be from drilled core. The rock particles should come from the size fraction of –75+35mm. Shape is important, as the rock has to be held in the tester where the smallest dimension is between the platens. The Diameter (D)/width (W) ratio should be between 0.3 and 1.0, but as close to 1.0 as practical.

- The sample is inserted into the tester in a manner where the rock has to be held in the tester where the smallest dimension is between the platens (Figure 119).
- The distance between the platens (D) is measured along with the specimen width (W) to an accuracy of +/-2 to 5%.
- The sample is loaded over a time period between 10 and 60 seconds.
- The test is rejected if the sample did not fail through a plane passing through the two platens where the rock is held.

Tests with irregular particle samples produce much more variation in the results than core samples.

5.2.4 Anisotropic rock samples

Rock types like shale or schist have structures at the meso-scale, which can influence th3e measurement of strength with the point load tester. Case studies have shown that strength measurements parallel to the foliation's are mush different in magnitude to strength measurements perpendicular to the foliation's in the same rock type. Table 6 shows a summary of the Point Load Index data taken from Mt Coot-tha quarry (Australia). The rock tested was a highly foliated phyllite.

Table 6: Summary of Mt. Coot-tha Point Load Index data (Source: Michaux 2006)

Test	Orientation to foliations	Mean Is_{50}	Standard Deviation	No of successful measurements
75*63mm	perpendicular	5.37	1.49	32
63*53mm	perpendicular	6.20	2.61	43
53*45mm	perpendicular	5.41	1.91	22
75*63mm	parallel	2.94	1.06	25
63*53mm	parallel	3.61	1.44	20
53*45mm	parallel	3.33	1.42	19

For cored tests, a number of diametrical and axial tests should be done as two sub sets to quantify the influence of the meso structure of the rock (to ensure that both orientations are measured). For irregular particle samples an attempt to orient the samples parallel and perpendicular to the bedding structures. Depending on how the rock is oriented in the bench will decide how to use this data. An average of the two sub sets is a practical solution.

5.2.5 Collecting samples

The Point Load Index is an approximation of strength as opposed to a direct measurement. As a consequence of this, there is a significant amount of variation in the measurements. This is not such a problem as the rock strength characterisation is also highly variable (even over a few metres). To deal with this problem, a large number of tests are required. How many depends on the variability of the rock type. After each test, some statistics analysis should carried out to determine the variability of the rock type. The t-test could be used to compare point load tests done on the same rock type. Rock density and particle mass before testing could be used to normalise the data. As a practical start, 30 successful measurements (where the force has been measured after a sample failure through a plane passing through the two platens where the rock is held) would provide enough data to approximate strength. It should also be remembered where the samples were collected. Correct sampling procedure is required, especially when spatially defining rock strength across a bench to be blasted.

In terms of sampling points on the bench or muckpile, a spatially defined grid (resolution of 1 to 2m) could be used as a sampling arrangement to collect particles (remembering shape is important and not all fragments collected will produce a successful measurement).

Sample mass requirements for a point load index is 30 to 40 drill core (NQ or HQ half core) pieces that have a length (X dimension) 2.5 times the width (Y dimension). This can be taken from 4m of drill core depending on the quality of the core (how broken up it is). The Point Load Index can also be done on fragments of rock -90+50mm fraction. The test needs 30 successful measurements that pass QA/QC requirements. This can be taken from 20kg of -90+50mm fraction rocks.

5.2.6 Use of the Point Load Index

The unit of the point load index is Mpa and is considered to be a measure of tensile failure and fracture toughness (Kic). There is a crude relationship between the Point Load Index and unconfined compressive strength (UCS), which has been used in mining research in the past. This is simply multiplying the Point Load Index by 24 to obtain the UCS. This however is not appropriate as this relationship change with rock type. The Point Load multiplication factor to estimate UCS can range from 10 to 50, depending on rock type (Thuro and Plinninger 2001). To estimate UCS from Point Load Index data, it is necessary to conduct a series of UCS tests in the same rock type to calibrate the multiplication factor.

5.3 Impact Breakage SMC test

The SMC Test is has the same objectives at the Drop Test but requires much less sample. The SMC was originally developed to measure the energy breakage characteristics of core samples and has been used as a comparison test to the Drop Test. A comparatively small amount of core is cut to a very specific size and shape (volume).

To achieve this shape and size, the mass of each fragment is measured and with the measured ore density (wet & dry method), the volume is calculated. For each fragment to pass, it must be within a statistical tolerance of an average volume. Five sets of twenty of these wedge shapes are taken for the SMC test. Each set of twenty is broken at a specified energy in the Drop Test machine as per Section 4.2. These 5 energy breakage points are then used to estimate the energy breakage curve and calculate A and b parameters.

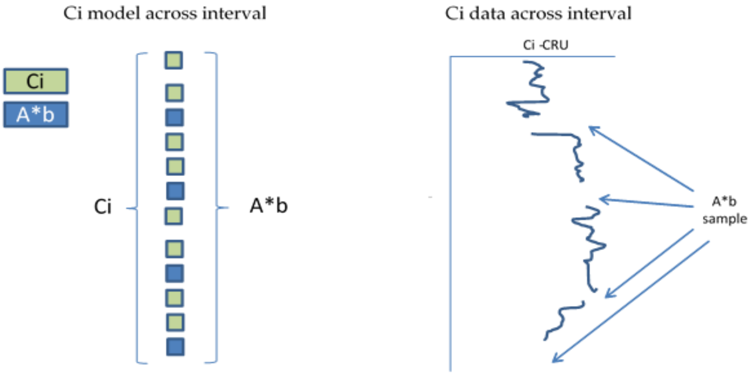


Figure 120. Sampling strategy to ensure the Comminution Index and the SMC tests were as representative as possible (Image: Simon Michaux)

The SMC Test was developed as a cost-effective means of profiling an ore body to predict its comminution circuit throughput as well as its rock mass characteristics and blasting properties. Energy requirements for crushing and HPGR options can now be determined in addition to AG/SAG mills, and with Mic, ball mills (Morrell 2004 & 2011, JKTech 2009). The results from conducting the SMC Test are used to determine the so-called drop-weight index (DWi) which is a measure of the strength of the rock" (JKTech 2009). Figure 121 shows the results of SMC test with operation data.

Table 7. SMC comminution test length of core required for the two methods (Source: JKTech SMC Test)

Core Diameter Range (mm)		Length of Core Required (m)	
		Crush and Select Method	Cut Core Method
Min	Max	Use if Plenty of Sample (Mandatory if Fragile or Friable Ore or with Frequent Breaks)	Use if Sample Limited (Requires Competent Core <70 mm Diam With Infrequent Breaks)
32.3	39.4	6.9	0.8
39.5	45.4	5.2	0.9
45.5	52.7	3.9	1.1
52.8	60.3	2.9	1.4
60.4	69.4	2.2	1.7
69.5	79.9	1.7	
80.0	89.1	1.3	

For larger diameter core, greater lengths are required for the cut core method. Half core, quarter core and crushed rock can also be used. Lengths in Table 7 must be multiplied by a factor of two for half core and four for quarter core, respectively. Predicted Comminution Circuit Specific Energy Using the SMC Test Compared to Values Measured from Over 120 Operating Plants from Around the World.

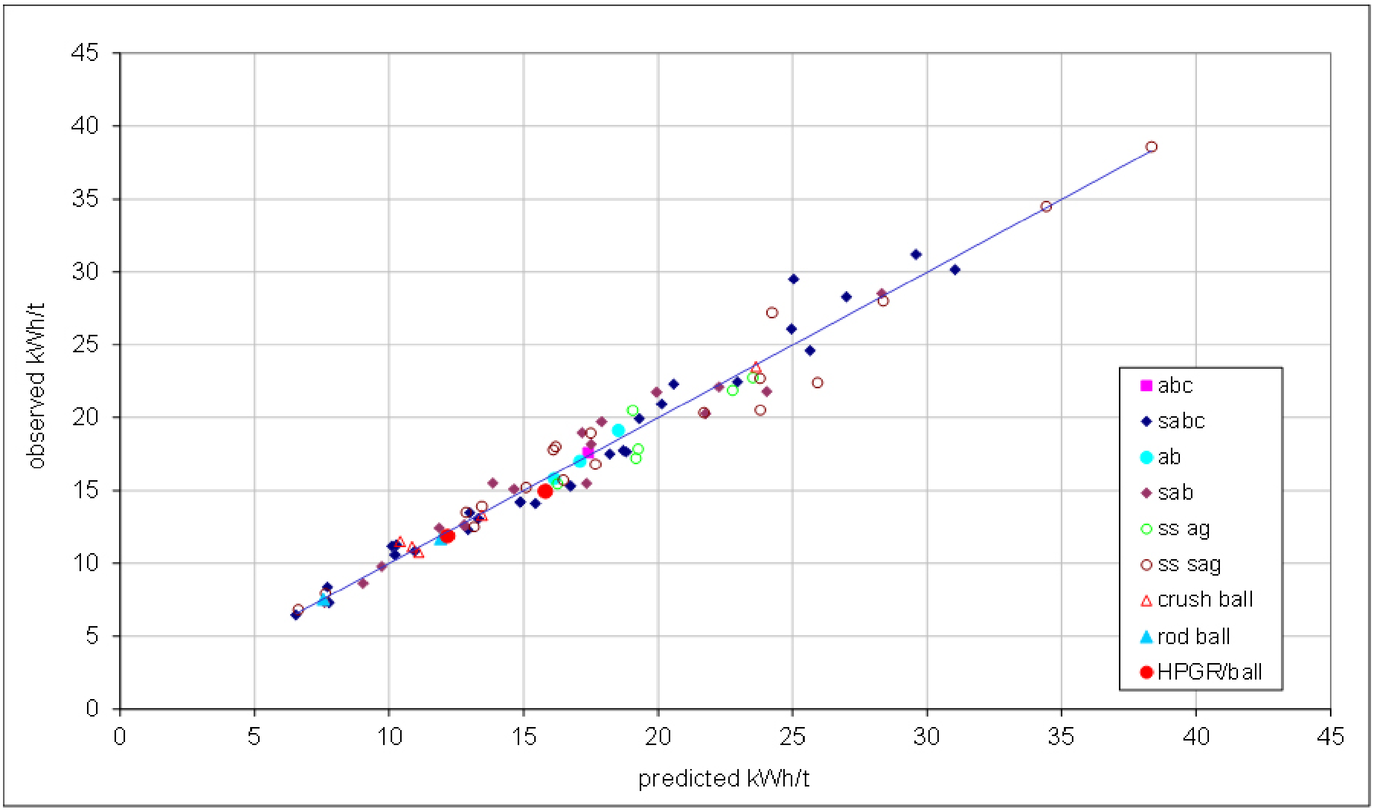


Figure 121 The SMC impact breakage test outcomes vs. operation data
(Source & Image copyright: Morrell 2011 and SMC)

The SMC Test requires significantly less sample than the JK Drop Weight Test or JK Rotary Breakage Test. Two sample preparation methods are available- the cut core method where the ore sample is competent and restricted in quantity, and the crush and select method where the ore is available in larger quantity or is friable.

5.4 RBT Impact Breakage test

The industry accepted JK impact breakage parameters, A and b, can now reliably be determined using the new generation high throughput JK Rotary Breakage Tester (JKRBT). It does this by projecting a particle (from a rotating rotor arm inside the machine) against metal anvil around the edge of the interior of the machine. The breakage products are treated the same way as in the DWT test.

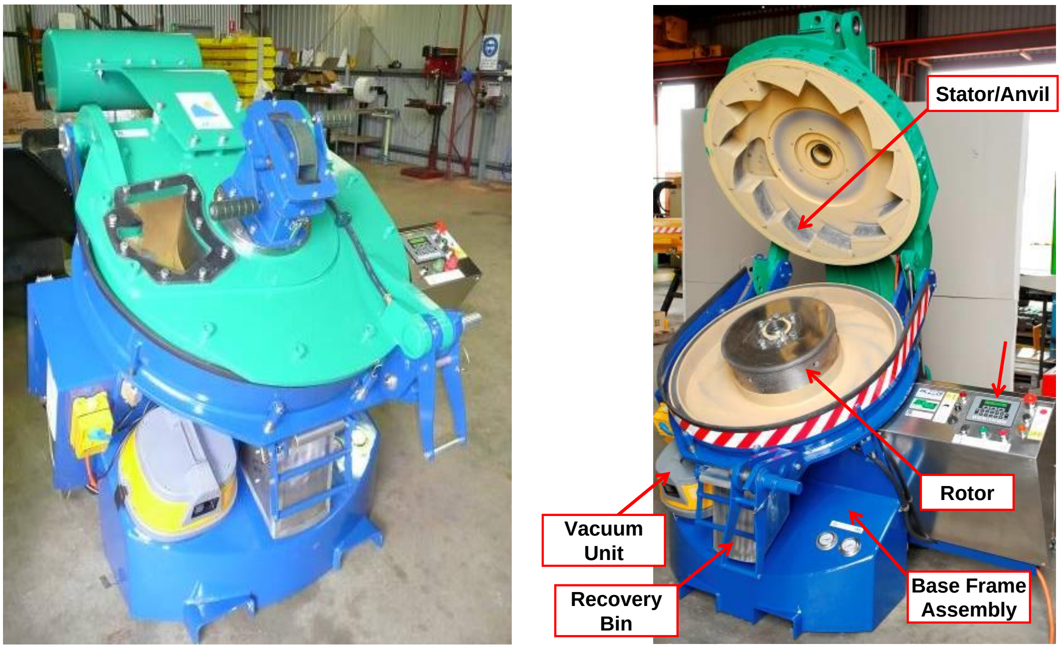


Figure 122. The JKMRC RBT impact breakage characterization test unit
(Source and Copyright: JK Tech Pty. Ltd, JK Rotary Breakage Tester)

The A and b impact parameters can be determined using the high throughput JK Rotary Breakage Tester (JKRBT). The machine is very different to the JK Drop weight tester, but outcomes are the same (See Section 4.2). Samples required are the following: 100 kg of crushed rock, -53+12.5 mm, or 100 kg of drill core". (JKTech Rotary Breakage Tester 2018). The JK Rotary Breakage Test provides these breakage parameters for use in the JKSimMet software to analyze and predict AG/SAG mill performance.

The typical minimum quantity of sample required to provide sufficient particles for testing is 100 kg of crushed rock in the -53+12.5 mm size range, or 100 kg of drill core. For diamond drill core samples, the core must be un-split and have a diameter of at least 50 mm.

5.5 Piston and Die Compress Bed Breakage (CBB)

The Compressed Bed Breakage (CBB) Piston and Die test is a small scale breakage test that can relate ore breakage to laboratory scale HPGR (Benzer *et al* 2017, Kumar 2014 and Daniel 2007). This test used a piston and die assembling in a hydraulic press (Figure 123). Approximately 300g of crushed ore (known size distribution) was put in the piston which was subject to a known pressure from the hydraulic press, thus breaking the sample in compressed bed breakage. The piston was instrumented to measure displacement over time in conjunction with applied pressure (Figures 124 and 125). Thus a known applied energy could be applied. The sample was recovered and the product sized.

Figure 123. Conventional piston and die unit in a hydraulic press
(Image: Steve Walters)

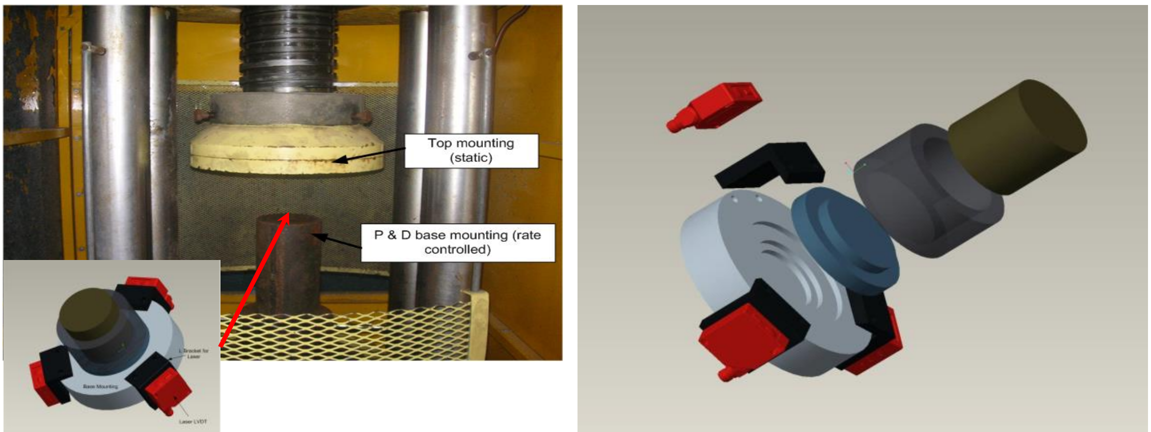


Figure 124. Instrumented piston and die test unit
(Image: Steve Walters LHS, and Rajiv Chandramohan RHS)

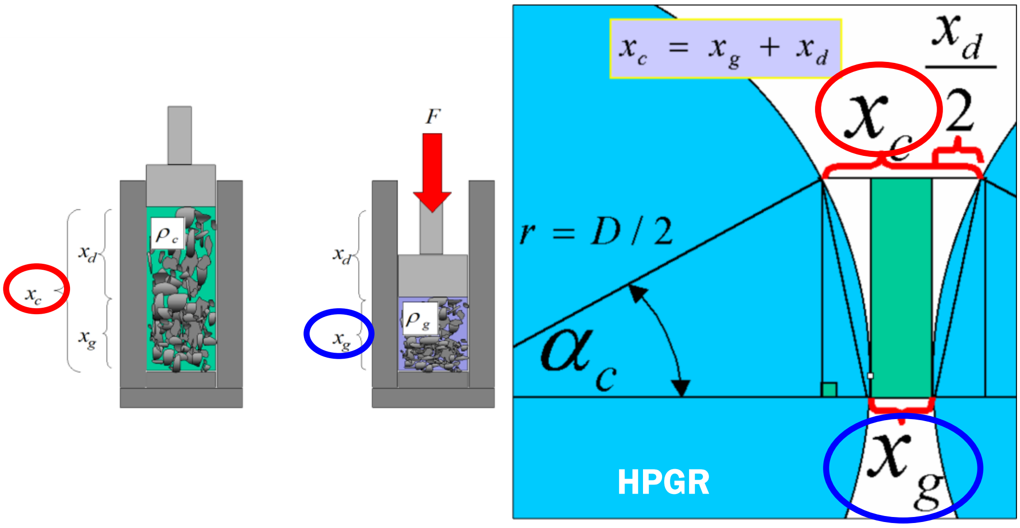


Figure 125. How the piston and die relates to the mechanism of breakage in a HPGR
(Image: Mike Daniels)

This approach is an interesting opportunity. The normalized feed size distribution of the laboratory scale HPGR is very similar in shape and form as the size distribution of the -3.35mm crushed ore of the same ore (Figure 126).

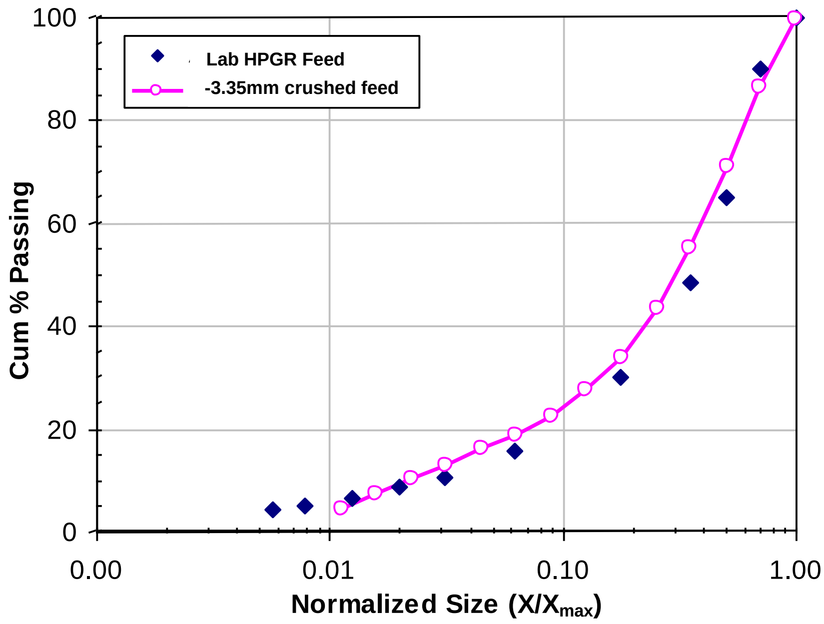


Figure 126. Lab Scale HPGR feed is very similar in character to the -3.35mm crushed ore (Image: Simon Michaux)

The products of the piston and die assembly also have a very similar size distribution compared the laboratory scale HPGR samples of the same ore (Figure 127).

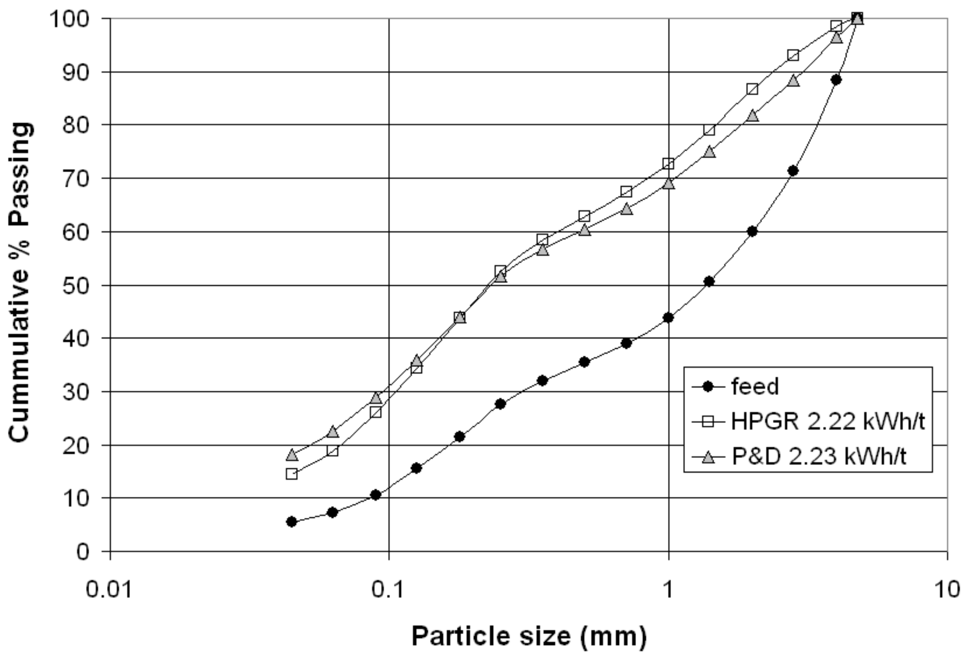


Figure 127. Lab HPGR and CBB tests can produce similar products (Daniel 2007 – P9N samples) (Image: Mike Daniels)

The CBB test can be used for the comparative ranking of different ore types possible in context of HPGR response. The feed size distribution F_{50} is compared to the piston and die product P_{50} (where P_{80} not as useful product size marker as P_{50}) using an algorithm to output an index that ranks the process performance of a HPGR crushing the ore (Figure 128).

$$CB_{50i} = Ecs/[10^{*}\ (1/\sqrt{P_{50}} - 1/\sqrt{F_{50}})] \qquad \text{Equation 9}$$

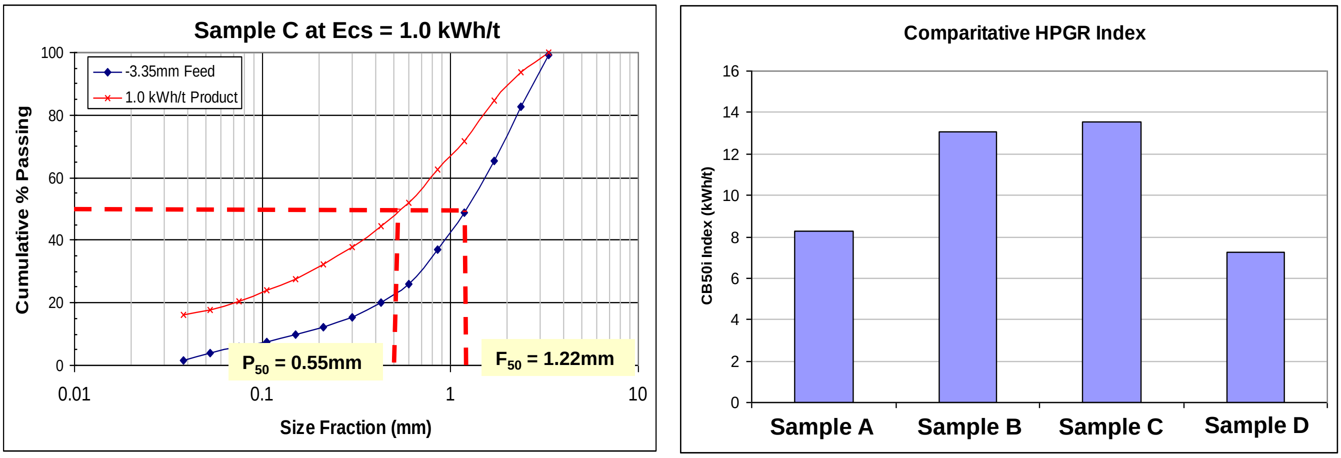


Figure 128. CBB small scale test for bed breakage (AMIRA 2009)
(Image: Simon Michaux)

Figure 128 LHS shows an example of a CBB test applied to Sample C, which produces a CB_{50i} of 7.2 kWh/t.

Another interesting opportunity involves transforming the CBB product mass size distributions into surface area distributions using the method developed in Michaux 2010. There is a fundamental relationship between surface area generated and energy applied in the breakage event (Michaux 2006). So several CBB tests could be done on the same ore, but at different applied Ecs energies (Figure 129).

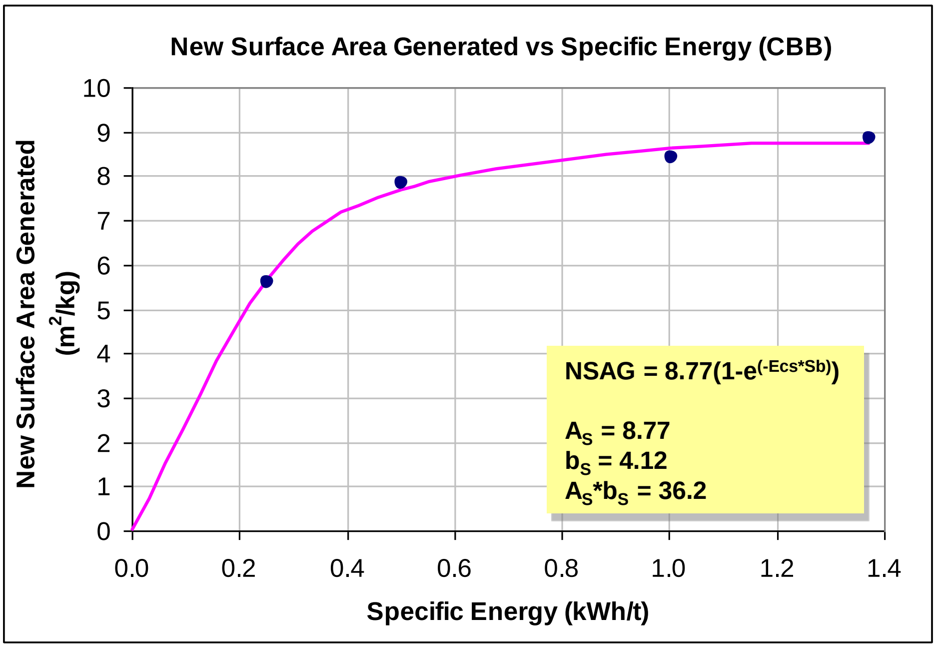


Figure 129. A series of CBB tests done at a number of applied energies (Source: AMIRA 2009)
(Image and Chart: Simon Michaux)

An exponential model could be fitted for new surface area generated (accounting for surface are in the feed product) as a function of energy is shown in Figure 117. This method has not been developed but shows promise in that it is analogous to Drop Weight A and b parameters for t_{10} vs Ecs model. This may provide more robust index relevant to HPGR, in the form:

$$CBBi=1000/A_S*b_S \text{ (kWh/t required per m}^2\text{/t of new surface area generated)} \qquad \text{Equation 10}$$

The shape of this surface area curve would be different for different rock textures.

Each CBB test run would require 200-300g of ore sample. The new surface are generated would require 4 CBB runs requiring 800-1200g of crushed -3.35mm ore sample.

5.6 Grinding - Batch Operating Work Index

A comparative grinding test was developed to rank the grinding behaviour of the small volume samples (AMIRA 2009). This test is known as the Batch Operating Work Index. In contrast to a full Bond Ball test which requires about 5kg of starting material the Batch Operating Work Index test requires only 700cc of material (crushed to 99% passing -3.35mm) as a feed sample. The procedure for Batch Operating Work Index test was as follows:

1. Select a representative sample 700cc in volume. Record its mass.
2. Size this feed sample on the √2 sieve series down to 38µm. The sample was then subject to a mass balance QA/QC step.
3. Place the sample in a standard Bond Ball mill (which has been cleaned out with compressed air and has the appropriate measured ball charge)
4. Run the mill with the sample secured in it for 358 revolutions or 5 minutes at 71.6 rpm.
5. Extract the sample from the mill and measure the mass.
6. Wet screen the sample on a 38µm sieve (with a scalping screen to protect the 38 µm screen)
7. Dry the +38 µm sample in an oven over night at 105^{0}C.
8. Size the dried sample on the √2 sieve series down to 38 µm. The sample was then subject to a mass balance QA/QC step.

9. Calculate the F_{80} from the feed size distribution using a spline function.
   - Each feed sample had its size distribution measured on the √2 sieve series using a mechanical shaker (Ro-tap). Each distribution was measured down to 38 µm. Each sample was subject to a mass balance QA/QC step.

   - The measured size distribution of the feed was used to calculate the F_{80} for each sample. The F_{80} represents the volume of sample that passes 80% of the top particle size, and was calculated from the cumulative percent passing plot of the size distribution.

10. Calculate the P_{80} from the product size distribution using a spline function. The P_{80} represents the volume of sample that passes 80% of the top particle size, and was calculated from the cumulative percent passing plot of the size distribution.

Figure 130. Feed and product samples for the Batch Operating Work Index
(Image: Simon Michaux)

11. Using the sample mass, revolutions of the Bond Ball mill, F_{80} and P_{80}, calculate the Batch Operating Work Index, using the equation below. This equation accounts for the mass of the sample in the test and the energy applied to that sample by the Bond Mill (number of revolutions).

$$OW_i = W \Big/ 10\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)$$

Equation 11

Where:

$$W = \frac{0.02*(\text{Number of revolutions of the Bond mill})}{\text{Mass of sample (kg)}}$$

Equation 12

P = The P_{80} of the product (mm)

F = The F_{80} of the feed (mm)

The sample mass and energy applied are normalized for each test. Thus the Batch Operating Work Index examines the grind behavior of the ore and different samples can be compared in a ranking fashion.

5.7 SGS SPi test

The SGS SPI test measures the energy usage for a standard size reduction. It is interpreted into an index. It measures the hardness of the ore from a SAG or AG milling perspective. It is a batch test and requires 2kg of ore material and runs in a 10cm x 30 cm mill. Time is measured (minutes) for the sample to grind 80% passing 12.5mm to 8-% passing1.70mm. (SGS 2018)

5.8 Mergan Work Index

The Mergan work index is a small scale wet grinding test developed by Niitti (1970), where the net energy consumption of the mill is continuously determined by a simple pendulum torque measurement from the mill axis. Nitty concluded that to accurately determine the grinding signature of an ore, a wet grinding test was required where a dry grinding test would be inaccurate (Heiskari *et al* 2019). Also, for a test to be reliable, it must be carried out to the same fineness as the actual grinding is done (Niitti, 1970).

The Mergan index gives good correlation to the Bond work index (the industry accepted standard for the measurement of energy required to grind to a given closing size) (Figure 131). The Bond Ball work index is a dry grinding test, but grinding done at an industrial scale is a wet process, which could make the Mergan Index more appropriate for mill design.

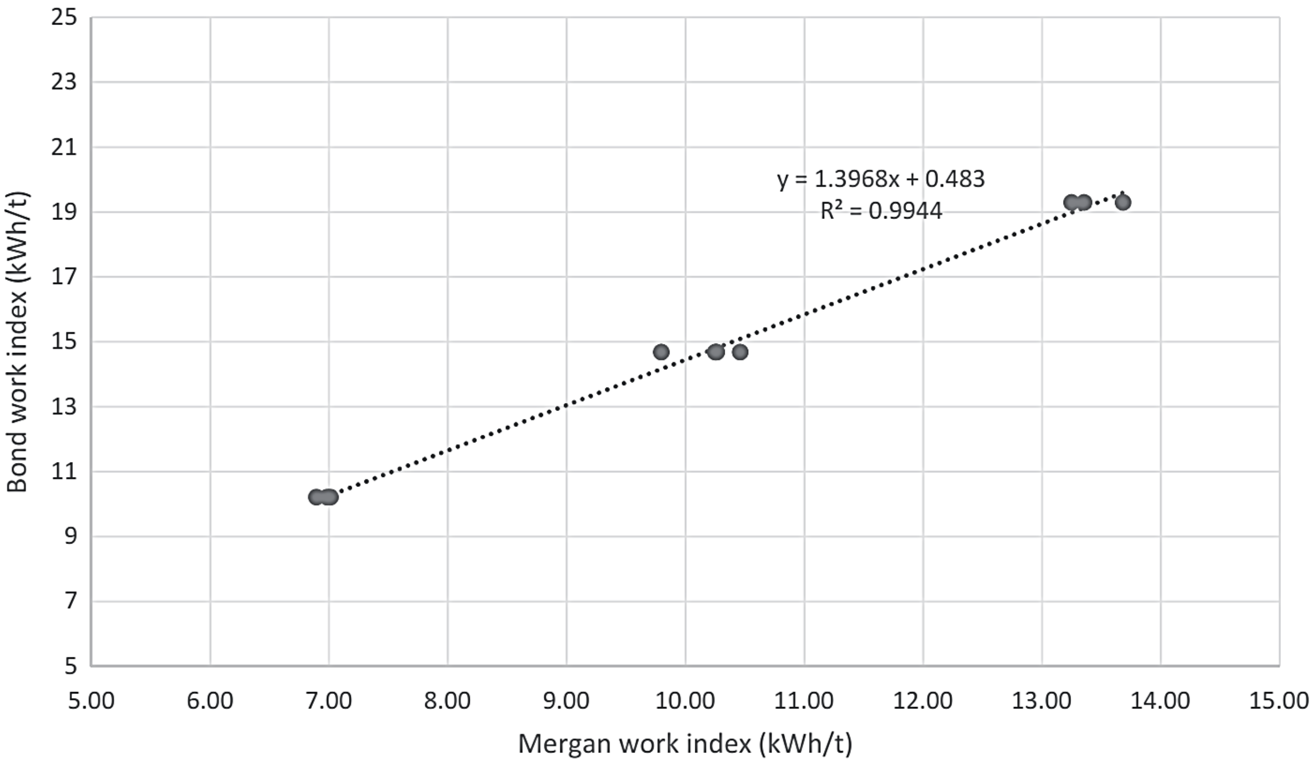


Figure 131. Comparison of the normalized Mergan work index values and the Bond work index values from the test work. (Source and Image: Heiskari et al 2019)

This test is small enough to enable taking mineralogical variability into account, and the ground product from the Mergan grindability test can be used for mineralogical characterization studies and beneficiation tests.

The selected primary sample size and the size of sub-samples are located left of Gy's Safety line and their integrity will not be compromised (Lotter *et al* 2014).

The required amount of ore, about 5 kg or 1500 cm^3 of packed material. This material is crushed ore (that has been dried) where 99% of the sample volume passes a 3.35mm sieve. The sample is then homogenized.

The dimensions of the Mergan mill are 268 × 268 mm, with a smooth inner lining (no lifters). The speed of the mill is kept at a constant 60 rpm throughout the test. The mill was equipped with a torque transducer to measure the torque of the mill. The ball charge used in the Mergan tests consisted of 318 steel balls. The ball charge was an "expanded" Bond ball mill charge. This means that the total weight of each ball size class was expanded until the total mass of the ball charge reached at least 22 kg, in this case of Heiskari (2019), 22.23 kg.

To conduct the Mergan test, the 1500cm^3 sample of packed crushed -3.35mm ore is added to the mill with enough water, so that the pulp density of the slurry was 55% solids by weight. The material is ground for an arbitrary amount of time, while the torque transducer is used to record the mill torque at 1-second intervals. After grinding, the mill is emptied, the sample is collected, the slurry was separated from the ball charge, and particle size analysis is conducted for the sample (particle size analysis was done by using a wet sieving method ultrasonic bath). After particle size analysis, the slurry was weighed and water was removed from the slurry or added to the slurry to achieve the desired slurry density of 55% solids by weight. The slurry was then returned to the mill, and grinding time was estimated based on the results of the particle size analysis. This was continued until the desired P_{80} value was reached (Heiskari 2019). The grindability may be expressed as kWh/t of the new 74-µm material produced, or as kWh/t per 1000 cm^2/cm^3 of the new specific surface area produced (Niitti 1970).

The Mergan Index is calculated as follows.

$$M - W_i = E_0 \left(\frac{\sqrt{F_{80}}}{\sqrt{F_{80}} - \sqrt{P_{80}}} \right) \sqrt{\frac{P_{80}}{100}}, \qquad \text{Equation 13}$$

where

F and P are 80% passing of feed and product (µm),
$M\text{-}W_i$ is the Mergan work index [kWh/t],
E_0 is energy consumption [kWh/t].

The calculation of (E_0), the grindability of the tested material can be calculated directly as energy consumption in kWh/t when ground from a given initial fineness to a desired final fineness. This is based on empirical data from the development of the test.

The mill power draw was obtained from the torque measurements, and the momentary power draw can be calculated from:

$$P = \frac{2\pi TN}{60}, \qquad \text{Equation 14}$$

where

P is power [Nm/s], [W],
T is measured torque [Nm],
N is mill rotation speed [rev/min].

Momentary power draw could be summed over the grinding time to obtain the energy consumption (E_0) of the grinding according to Equation 15. Once P_{80} and E_0 were calculated, the Mergan work index, M-Wi, could be calculated according to Equation 8.

$$E_0 = \frac{\sum P}{3600000\frac{M}{10^6}} = \frac{\sum P}{3.6M}, \qquad \text{Equation 15}$$

where

E_0 is energy consumption [kWh/t],
P is power [W],
M is the weight of the mill feed [g].

5.9 Comminution Index (JKCi)

The Comminution Index (Ci) is a small-scale crushing test that was developed as part of the AMIRA P843 GEM[III] project (Kojovic et al 2010, AMIRA P843 2008a & 2008b and AMIRA P843 2009). It is based on the concept of crushing drill core in very specific experimental conditions. This test can be done during sample preparation for other tests (for example chemical assay, or batch flotation). The test has a quick turnaround (~10min/test) and has proven to be repeatable and robust across a range of core sizes and geometries. Natural rock fragments can also be used as feed. The Ci has undergone preliminary trials within a major commercial laboratory involving 13,000 samples as part of routine assay sample preparation (Yildirim 2016).

The resulting size distribution from the Ci test is used to derive two parameters: Ci (CRU) which is related to the JKMRC Drop Weight Test A*b impact breakage parameter - range 0.0-6.0; and the Ci (GRD) which has a good relationship with the Bond Ball Mill Work Index (BMWi) – range 0.0 to 1.0. An orientation study is recommended to determine the optimal application of the Ci testing protocol for individual sites and sample mediums. The indices of Ci-CRU and Ci GRD are also excellent comminution based domaining tools.

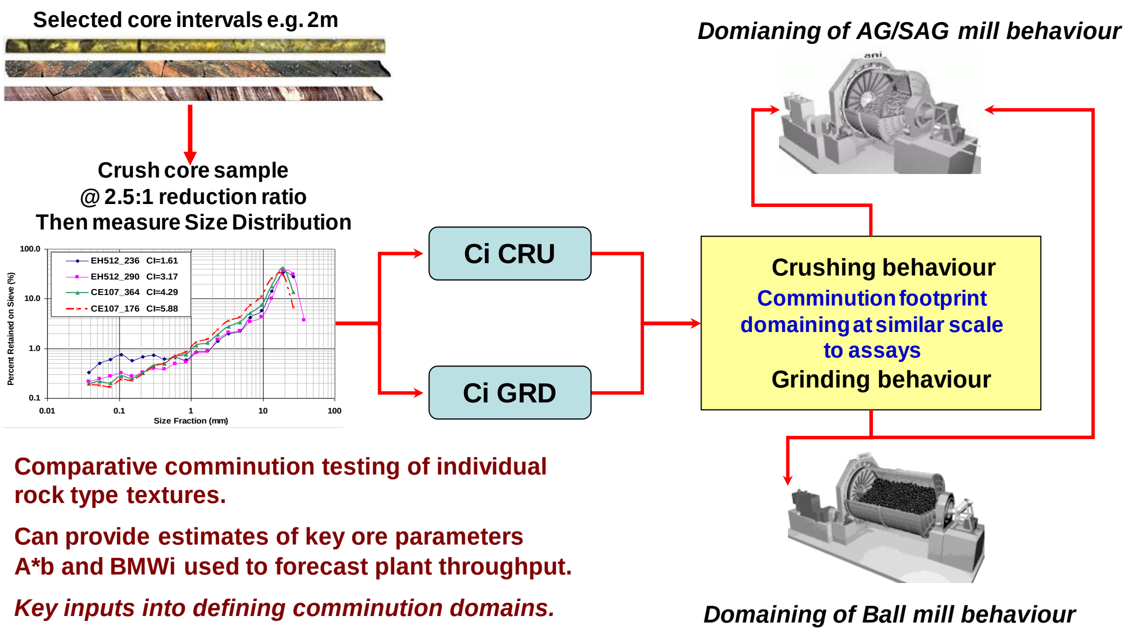


Figure. 132 The Comminution Index to domain comminution behavior

(Source: AMIRA 2008b, Copyright: AMIRA)

The circumstances where the Ci test protocol has proven to show limitations include:

- Friable core (pieces not the same size in each sample)
- High clay content in rock (plasticity and elasticity considerations)
- Variable feeding (choke vs single piece at a time)
- Crusher gap CSS is left unchecked to drift
- Natural fragments that have large variation in shape

What is an interesting opportunity is that it is able to test poor quality and friable core. This is something conventional impact breakage tests like the Drop Weight Test or the SMC test cannot do. Conventional comminution usually is done to determine the hardness of the rock, so a mill can be built that is strong enough to cope with the required throughput. This usually does not need to measure the softest parts of the deposit (usually the friable and poor-quality core). This is useful the softer core will have a higher throughput through the SAG mill but not necessarily through the ball mile or fine grinding mill. A full domain can now be done that can related to impact breakage and grinding behavior.

Figure 133 shows an example of the Comminution Index done down hole then compared to site lithology classification of the drill core section. The Comminution Index in conjunction with chemical assays and lithology logs can be an effective way domaining ore in context of comminution response as a function of rock type.

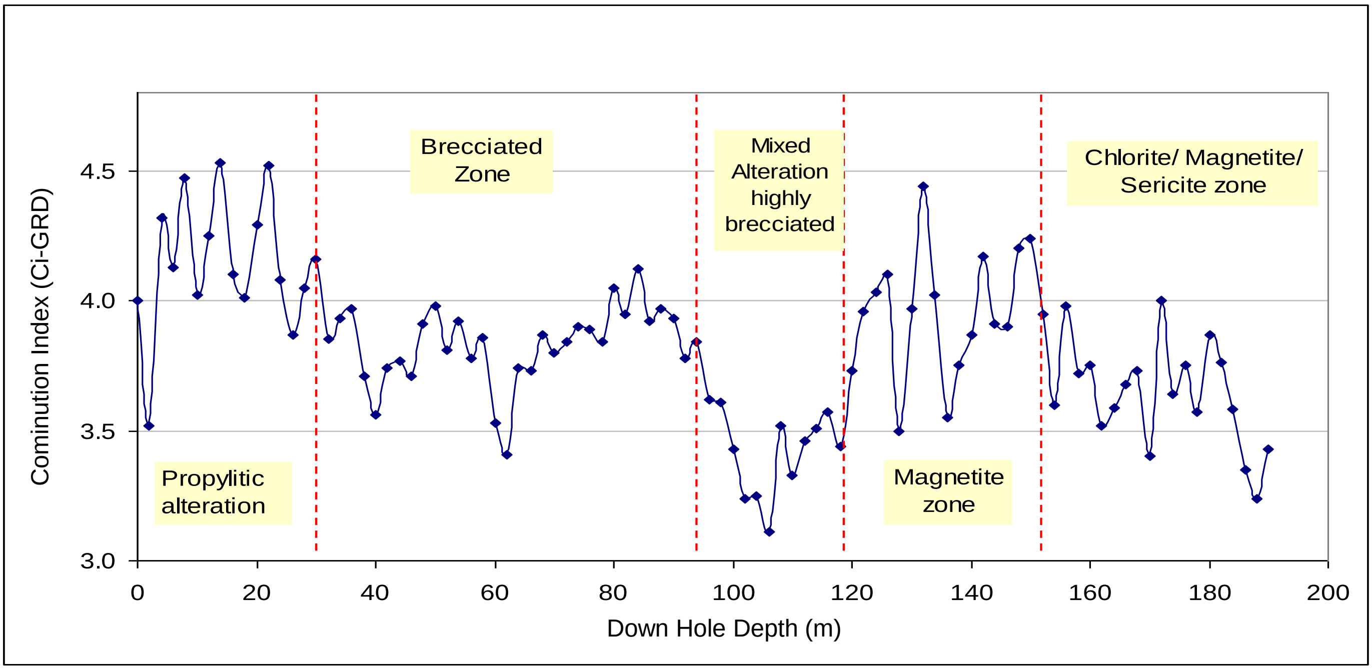


Figure 133. Comminution Index (Ci) as a domaining tool
(Image: Simon Michaux)

The outcome of a domaining study is an understanding of the domaining structure and variability of the deposit in context of the target process behavior. While this outcome is useful, further work is to be done in a non-spatial fashion to really secure understanding of what mineralogy controls the target behavior.

Figure 134 shows a 360m section of core, tested for the Comminution Index in a continuous fashion (each test is of a 2m length of half core, some HQ, some NQ in size).

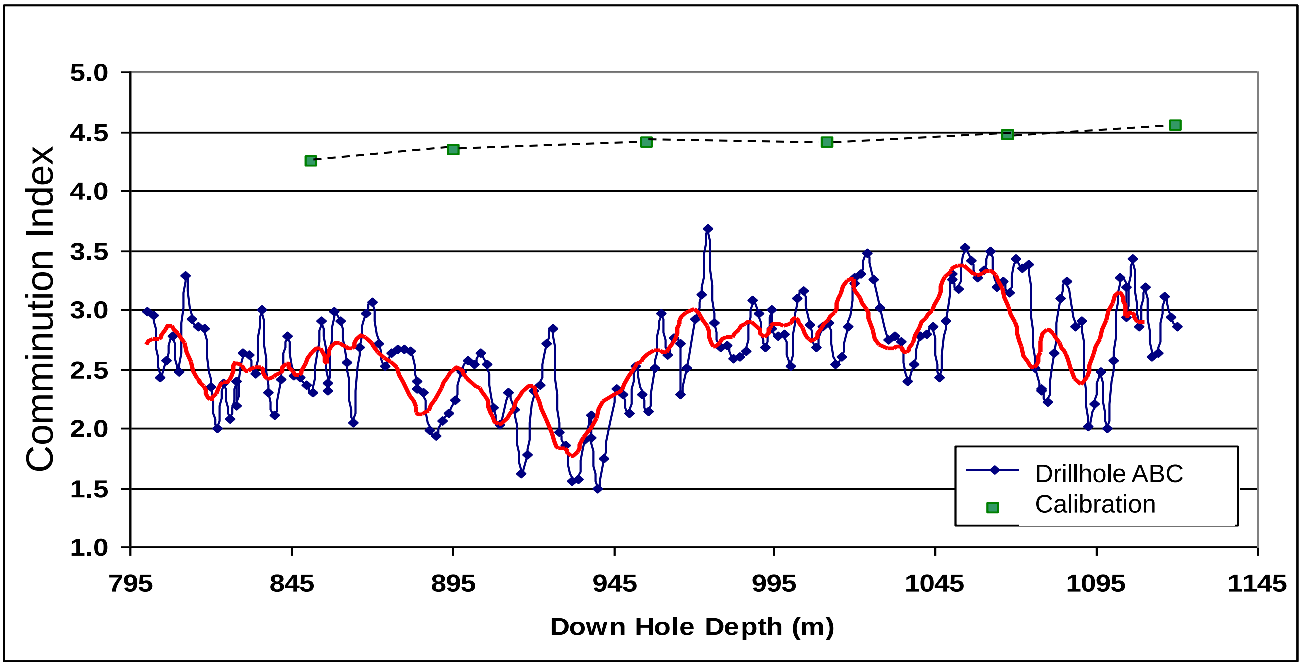


Figure 134. Comminution Index in a continuous 360m downhole section of core
(Simon Michaux)

The indices that come from the Comminution Index are used to predict comminution process parameters. The CiCRU index is used to empirically predict the Axb impact breakage parameter (Figure 135). The CiGRD index is used to empirically predict the Bond Ball Work Index (BMWi), shown in Figure 136. These are a deposit specific relationship that requires calibration (usually using the SMC tests and BMWi tests) and is not meant to replace the full Drop Weight Test for ore characterization in context of mill design.

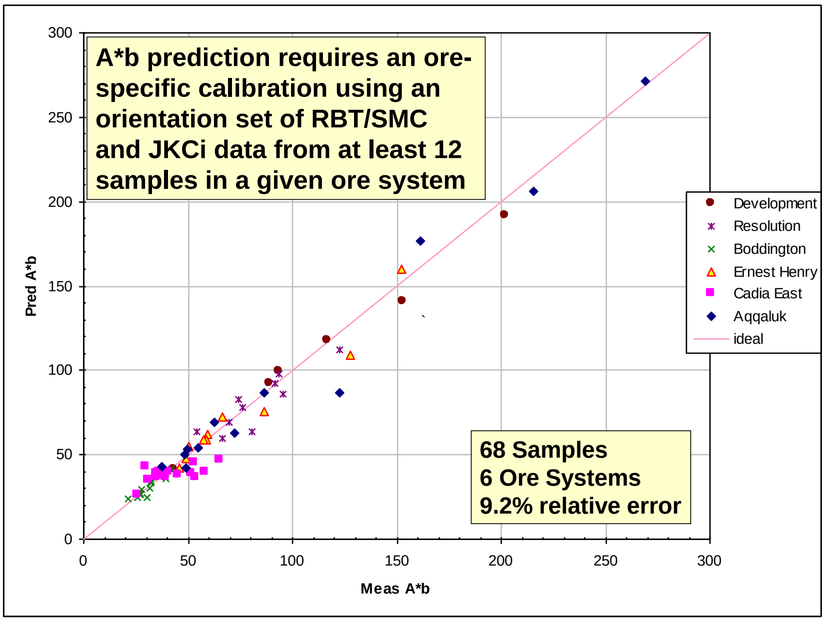


Figure 135 Comminution Index (Ci-CRU) prediction of impact breakage (Axb)
(Source: Kojovic *et al* 2010, Copyright: AusIMM)

The universal correlation of Ci-CRU with A*b shows lower level of correlation compared to BMWi estimates (Figure 136). It is also apparent that for some rock types with low A*b (i.e. hard crush) the Ci shows a greater variation than A*b. For example, in Figure 121 at a Ci value greater than 3.5, the A*b values have a limited range between approximately 20 and 50, however the Ci shows a much greater range. The significance of this relationship is still under investigation.

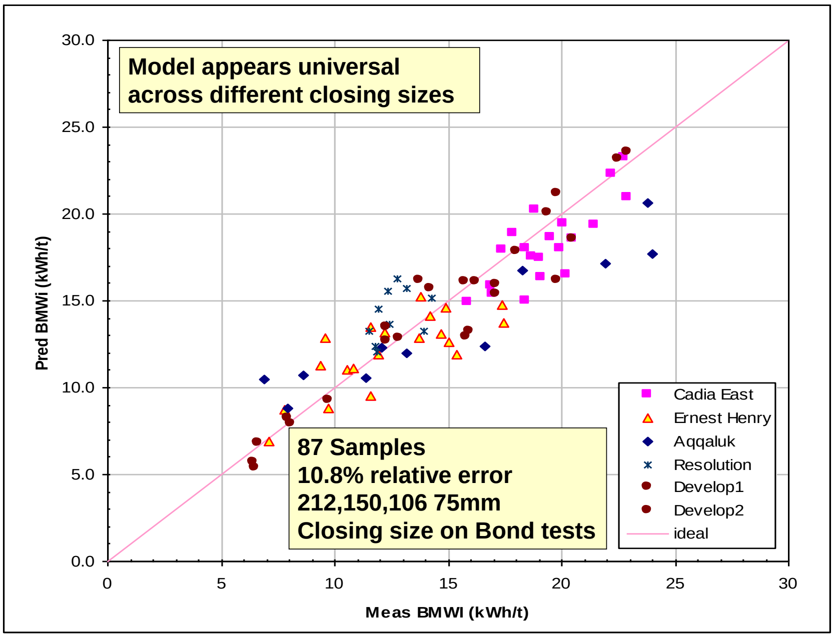


Figure 136 Quality of the prediction of bond mill work index BMWi from Ci-GRD (Source: Kojovic *et a*l 2010, Copyright: AusIMM)

Results indicate that the Ci-GRD offers a reliable index for estimating BMWi, with 8.7% average relative error across 190 samples from 7 mine sites and 10 development rock types (Figure 136). The universal correlation with BMWi is very promising, but integration with supporting information is also important at some sites for example where the mineralogy range is extremely varied, or the core is heavily broken.

To illustrate the application of the Ci in geometallurgical mapping of throughput a large-scale comparative study was carried out at an AMIRA P843 sponsor site. This involved comparing the results of a large underground bulk sample extracted from a drive and tested using conventional methods for A*b and BMWi with the results of three drill holes drilled along the drive. 443 Ci tests were conducted at two metre assay intervals along the drill core and used to estimate BMWi and A*b for each sample.

The results for the bulk ore sample treated through the existing SABC plant are compared with the Ci estimates in Table 8. The Ci based throughput estimates were based on the estimated ore hardness parameters (A*b and BMWi), actual mill operating powers during the plant trial, and surveyed grind size (P_{80}) and SAG feed size (F_{80}). As can be seen (Table 8) the mill throughput estimates are consistent with the plant trial on the bulk ore sample.

Table 8 Comparison of actual and forecast parameters and plant throughput (Kojovic *et al* 2010)

Sample	A*b	BMWi (kWh/t)	Bulk Ore Plant Throughput	JKCi Plant TPH Forecast
Bulk Ore Sample (average)	28.9	20.8	-	1417
JKCi tested Hole 1 -average	29.7	20.2	-	1458
JKCi tested Hole 2 -average	32.0	20.3	-	1460
JKCi tested Hole 3 -average	30.8	20.4	-	1451

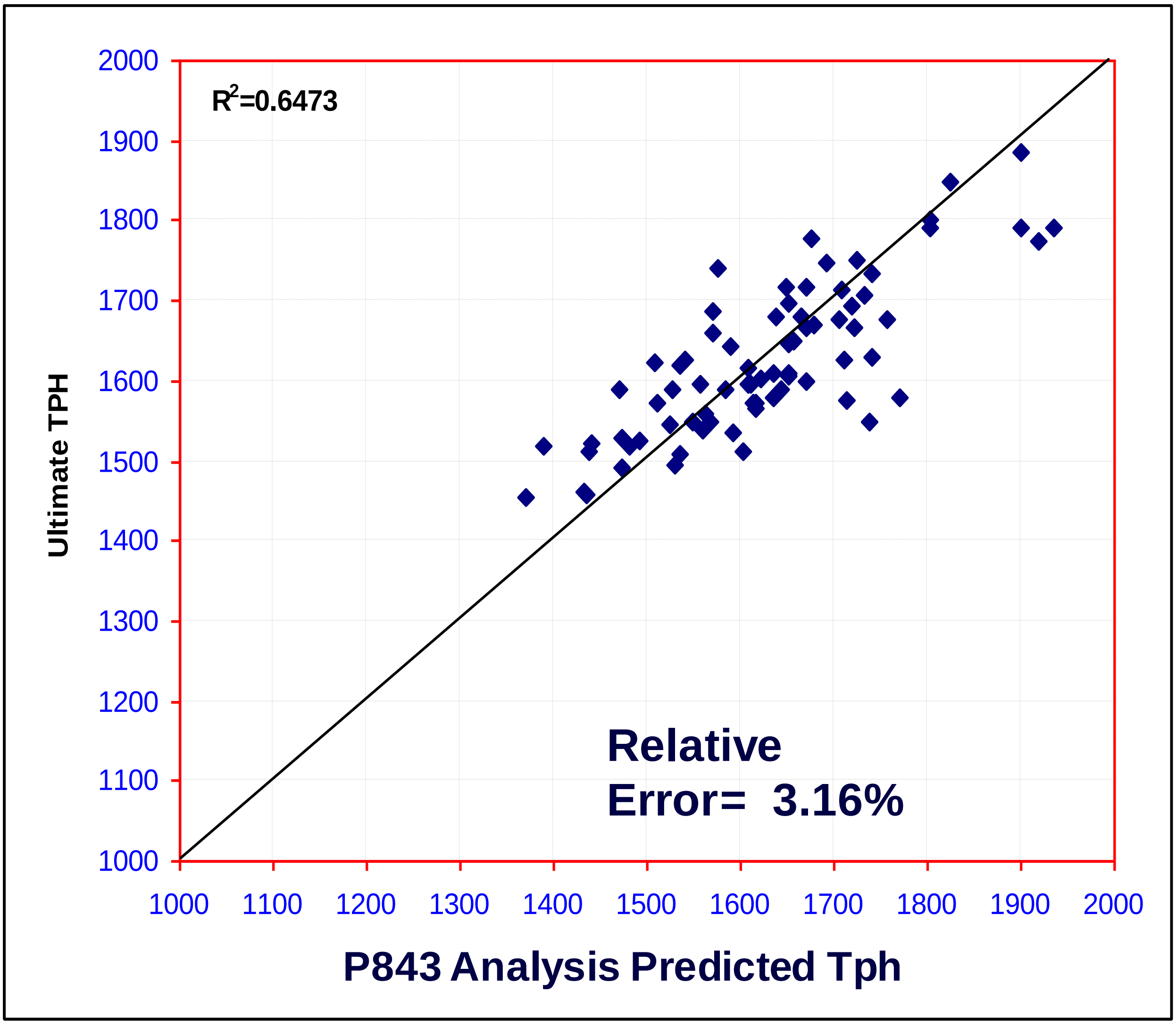


Figure 137. AMIRA P843 sponsor site Throughput Predictions vs. Underground production trial (Source: AMIRA P843 2009, Copyright AMIRA)

As can be observed, the mill throughput estimates consistent with production trial on underground ore samples (to support a mine expansion). This suggests at least in this case that the geometallugical approach can provide process attributes fit for purpose to model the mining process in a sophisticated manner.

Figure 138 shows the same data as Figure 134, but with an estimated mill throughput based on a simplified comminution circuit from the client process plant.

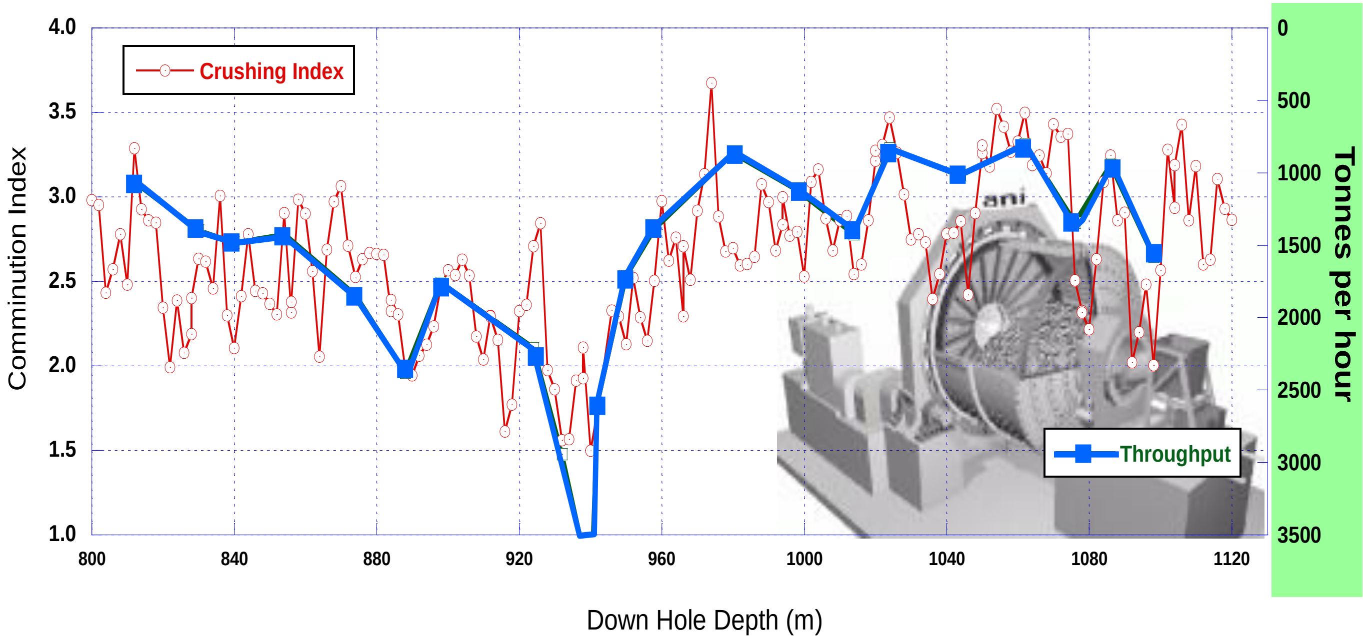


Figure 138. Estimated mill throughput using the Comminution Index in a continuous 360m downhole section of core. Mill Dimensions 11x5.5m, Ball Load 12%, 78% of Critical Speed, 25% Mill Load (Image: Toni Kojovic, Copyright: AMIRA)

The Ci had proven to be a fast and reproducible comparative comminution test designed specifically to support large-scale geometallurgical mapping and modelling. As such it is not designed as a replacement for existing higher precision tests used for ‘bankable’ engineering design. Insertion of the Ci test into routine assay preparation as shown through a large-scale laboratory trial provides an opportunity to collect the data during feasibility studies. This represents significant value adding to routine drill core testing. Estimates of BMWi from extensive Ci development and validation studies show a high level of correlation suitable for geometallurgical domaining.

Within the emerging field of geometallurgy rapid ‘front line’ testing methods such as the Ci provide the opportunity to support a new approach to definition of inherent geological variability in terms of processing performance parameters.

The sample mass required for JKTech or ALS to conduct the Ci Comminution Index is circumstance dependant. This is a destructive test that allows for the whole sample mass to be used for further testing. Drill core requirements define how the test is done. The test can be done on samples as small as 1m of drill core (PQ/HQ/NQ, whole/half/quarter core, natural fragments -40+20mm).

5.10 Comparative Comminution Hardness Test Geopöyrä

Another small scale proxy test to measure the A*b impact breakage parameter is the Geopyörä device (Matus *et al* 2020, Torvela 2020, Torvela *et al* 2020). This device is a new testing device is a variation of an instrumented roll crusher with an adjustable gap to measure breakage forces and energy applied to rock particles during the breakage process. This experimental setup can rapidly test rocks on a wide range of sizes and energy levels. The device was capable of measuring energy with great accuracy and a standard deviation of 1.07 Joules.

The device was developed at the University of Oulu and consists of two instrumented wheels of 600 millimeters diameter each, with an adjustable gap as shown in Figure 139. The steel wheels are powered by integrated electric motors placed in the frame; the available energy is range between 100 to 250 Joules. The device provides data of the impact force and measure the loss of rotational moment to determine the energy required for each particle breakage event (Torvela 2020 and Matus *et al* 2020).

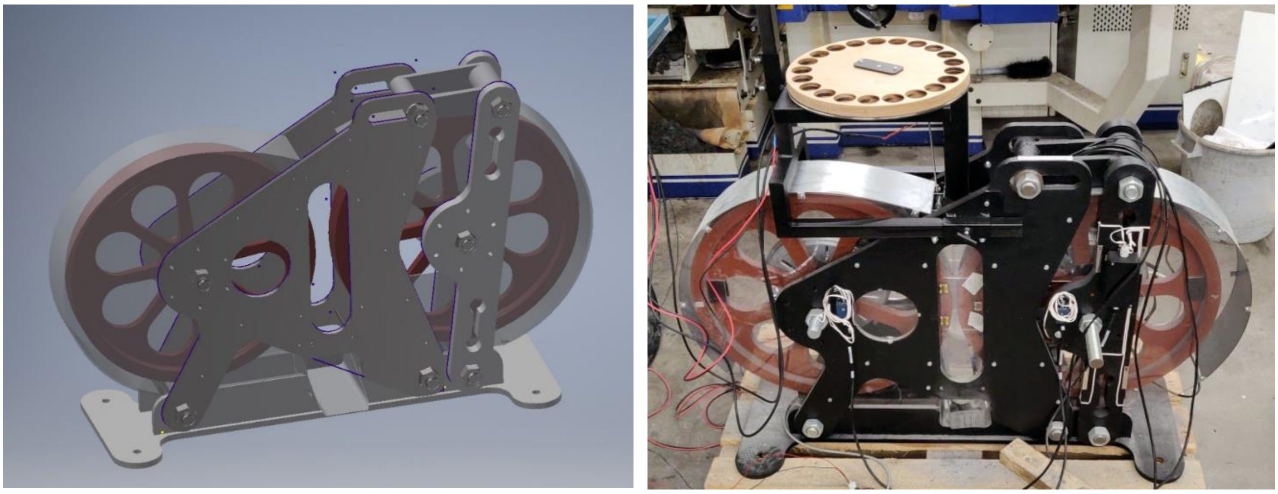


Figure 139. (LHS) 3D model of the double wheel crusher frame with wheels and wheel covers (Torvela *et al* 2020), (RHS) the constructed Geopyörä device (Matus *et al* 2020)

This device can test a variety of small discrete samples, either bulk or drill cores on a range of particle sizes and can conduct the test very quickly. The short testing time results in low operating costs, which allows a large number of samples to be tested for a better understanding of the ore variability.

Figure 140 shows a series of high-speed photographs of a Geopyörä crushing event were taken with a Sony Cyber-shot DSC-RX100 IV camera (lent by the Oulu Mining School) as part of the device development (Torvela 2020). The individual frames were cropped and combined into a single image seen in Figure 140. The series shows the particle breaking into two distinct parts at the middle while passing through the gap with a spray of dust and fragments ejected below.

Sample preparation for the standard test requires two sets of 15 particles of equal size, where each set represents an energy level, and the energy of the breakage is adjusted by modifying the gap between the two wheels. The device has been tested with particles from 31.5 to 9.5 millimeters. Particles are placed on an automatic feeder, which drops it from the top of the device to the crushing zone between the two rollers one at a time. The instrumentation and CPU record the force and energy data for each breakage event. After the crushing, the product is collected from a tray in the bottom of the equipment. This approach was developed to establish a test procedure that can be replicated and minimize operator-induced bias.

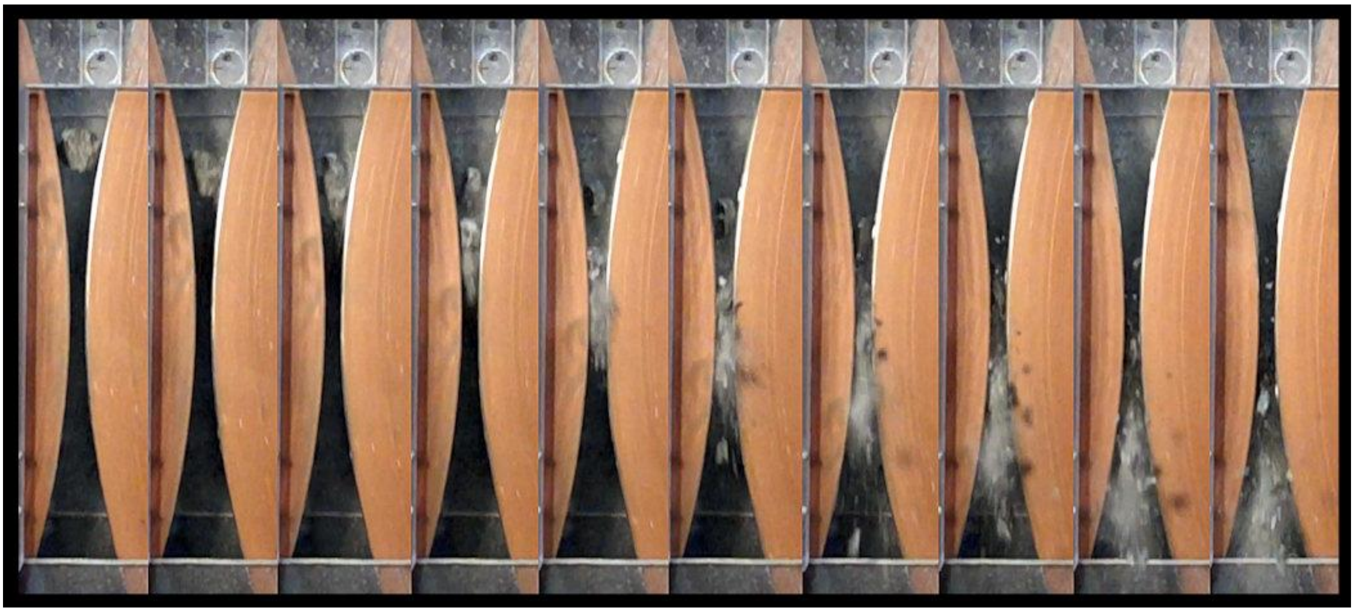


Figure 140. A series of high-speed photographs of a typical crushing event.
(Source: Matus *et al* 2020)

The resulting samples were sized and the t_{10} for each sample was calculated, and then the A*b impact breakage parameter was calculated (see Section 4.2) at the Oulu Mining School (done by Tabatha Chavez Matus as part of her thesis work). The measured t_{10} of each sample at the measured energy level of that sample was compared against a t_{10} curve predicted by a commercial Drop Weight Test on a control sample taken out of the same parent sample of ore for multiple mine sites. Figure 141 below shows how the Geopöyä device A*b measurement directly compares to a commercial Drop Weight Test result on parallel samples.

A further opportunity was explored by the Geopöyrä development team, where correlation with the Bond Ball Work Index (see Section 4.3) was examined. Using a series of parallel samples taken from each ore type (from serveal different mine sites), standard Bond ball mill tests were performed to determine the Bond ball mill work index (BBMWi at a closing size of 150μm) for each sample, to compare to existing Geopöyrä A*b measurements.

Figure 142 shows a comparison between the BBMWi measured using the standard procedure and the BBWMi estimated using the Geopyörä data. A satisfactory correlation can be observed with four out of the five results within a relative error range of ± 20% (dashed lines).

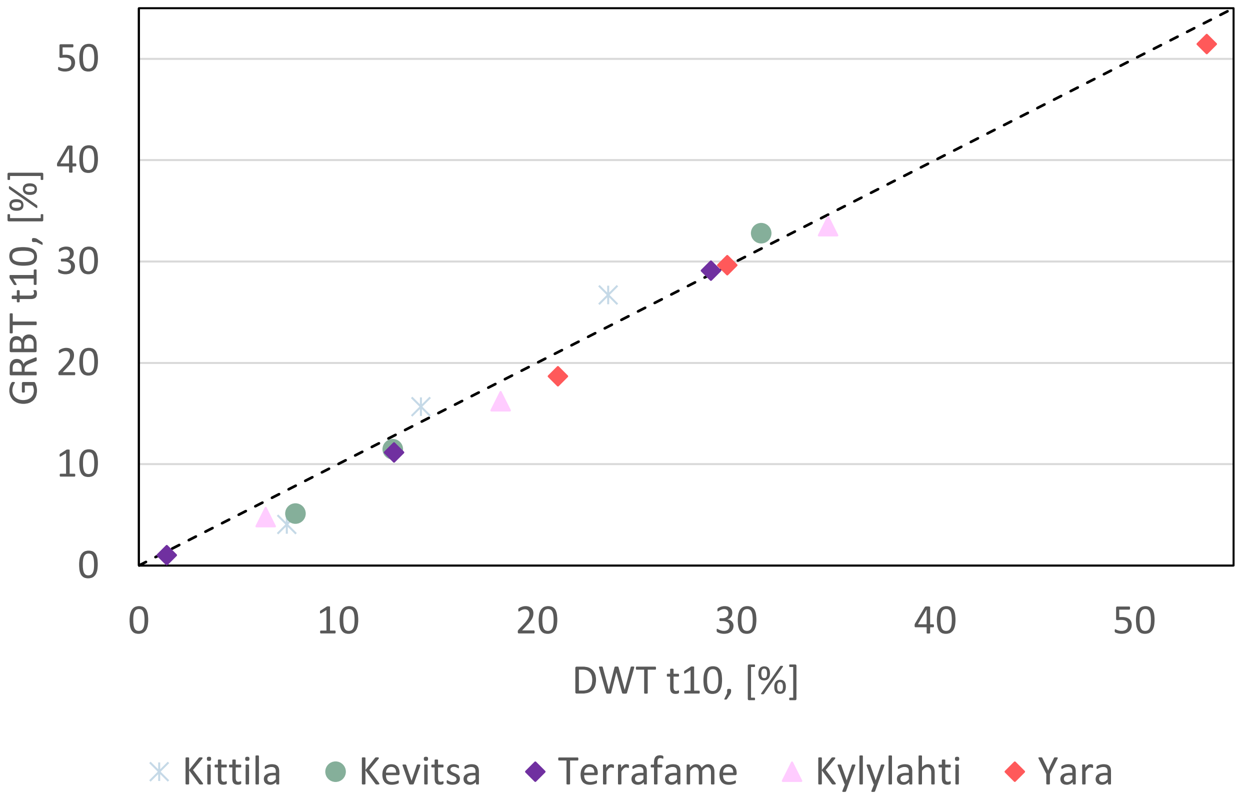


Figure 141. Geopöyrä Double Wheel Crusher versus DWT measured versus predicted t_{10} (Source: Matus *et al* 2020)

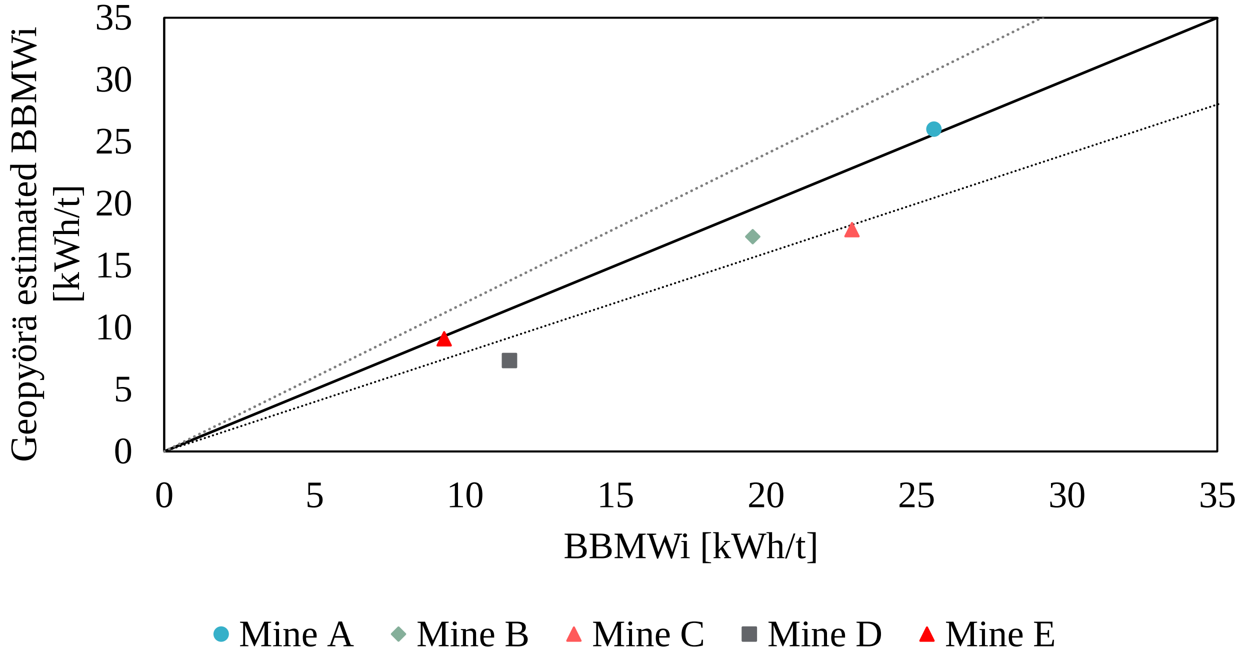


Figure 142. Comparison of BBMWi and the Geopyörä estimated BBMWi (Source: Matus *et al* 2020)

This represents a unique opportunity. The way this data would be used is not to design a ball mill, but to map variability of ore hardness. If the same test can measure directly the A*b impact breakage parameter, and approximately estimate the BMWi, and do so quickly and cheaply, then this test would have potential to be a useful geometallurgical tool.

What could be an interesting application would be the use of the Geopöyrä device installed in a process plant to measure A*b real time. This would require a periodic sample taken from the SAG mill feed, then that sample broken in the device, then the products sized quickly. If this was done by an optical sizing device, then the sample could be sized as it falls out of the Geopöyrä unit. The A*b parameter and the BMWi could be estimated. This data could be used to predict the SAG mill specific energy and the Ball mill product P_{80}. This information could be used to optimize the SAG mill and ball mill together in real time during operation. This concept is shown in Figure 143.

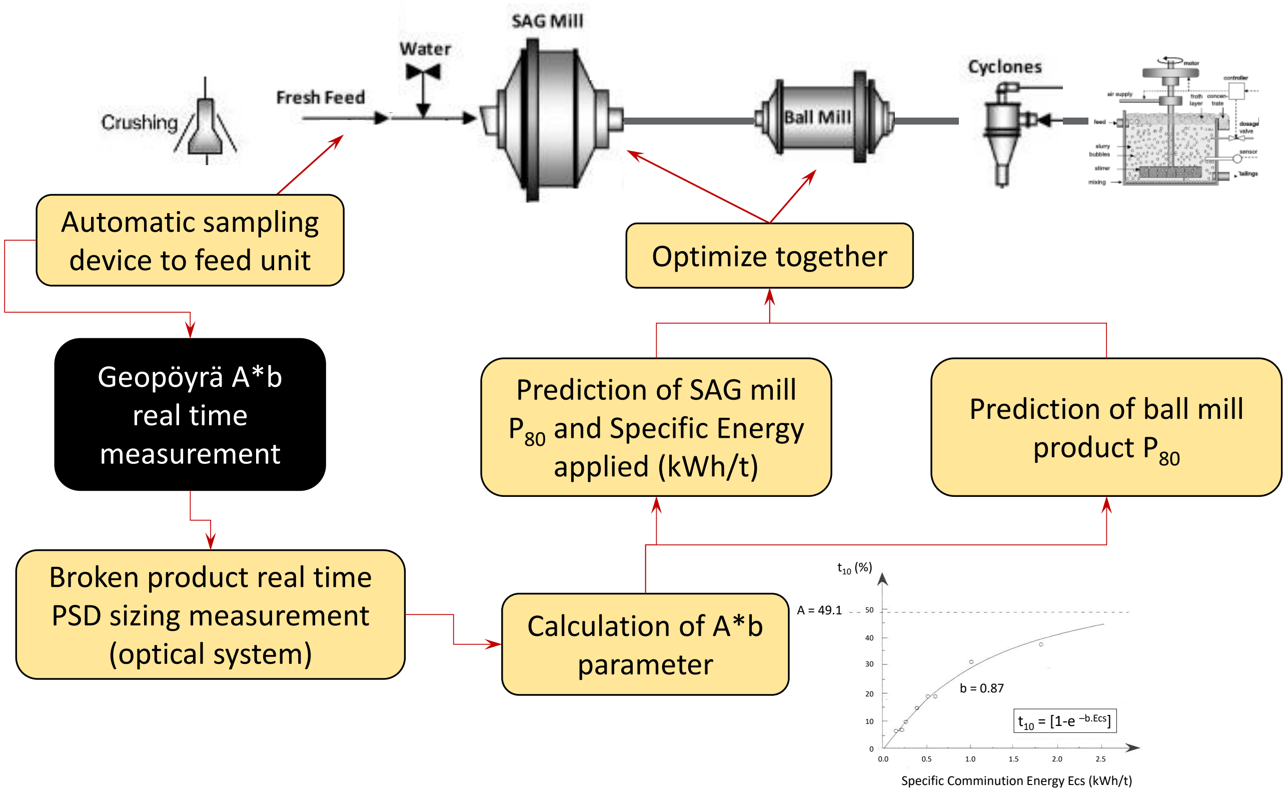


Figure 143. Possible use of Geopöyrä to measure A*b real time in an operating plant (Image: Simon Michaux)

The opportunity is to use this device to match ore signatures found in a geometallurgy study with real time operation. This concept has been discussed in Section 15.

5.11 Comparative Comminution Hardness Index Tester HIT

Another proxy test for the A*b impact breakage parameter (see Section 4.2) is the Hardness Index Tester (HIT), developed by SimSAGe Pty Ltd (https://simsage.com.au/). This is a relatively simple device (Figure 144) that is a low cost device for estimating the comminution parameter A*b and the Bond Ball Mill Work Index at any mine site, with potential applications on fragments up to 25 mm from drill core.

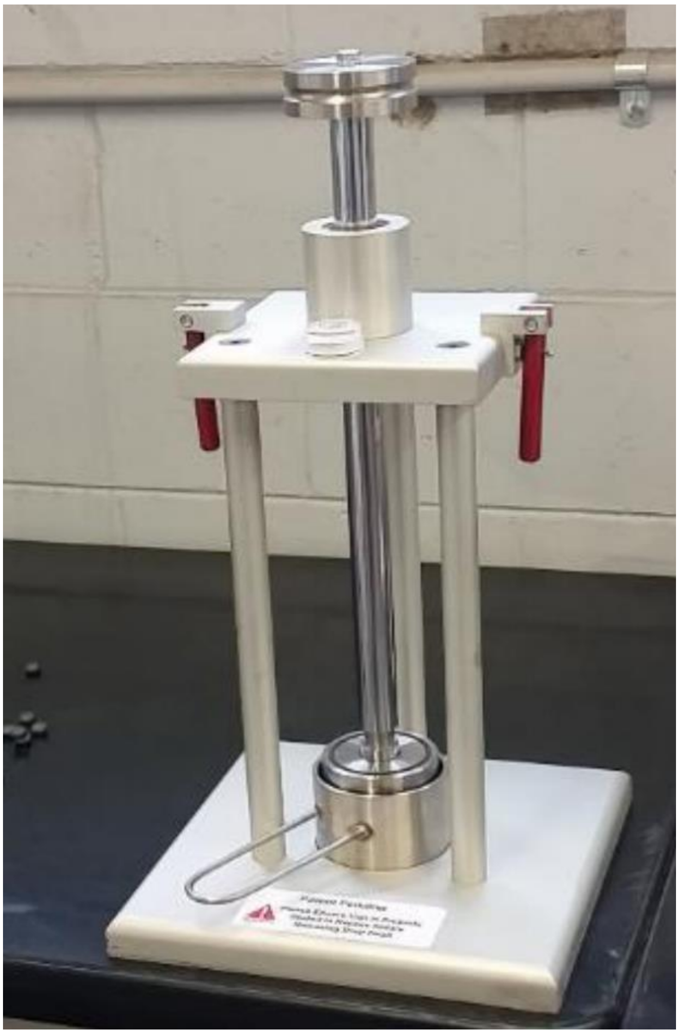


Figure 144. Image of the HIT device (Prototype V2; Patent Pending 62/241,852, PCT/I B2016/001591) (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

The application of the HIT was not to replace the standard comminution tests like the Drop Weight Test (see Section 4.2) or the Bond Mill Work Index (see Section 4.5), but rather to generate a high number of comminution and geomechanical parameters for the rock mass. Ultimately, the objective would be to determine the variability of the rock mass and, therefore, easily distinguish any potentially problematic zones for the mill.

The principle phenomena behind the HIT methodology is that A*b is the slope of the curve of 'zero' input energy (Napier-Munn 2005) of the ore being tested. Mathematically, this means the slope of the curve for that ore as it connects to the origin. This could be used as a fundamental rock breakage characteristic in some applications of geometallurgy modelling. The slope at a relatively low Ecs (~0.2 kWh/t) is a reasonably effective estimate of the slope at zero for that ore (i.e. the true A*b for the fitted curve).

So the HIT test will break particles one at a time at an applied energy of 0.2 kWh/t according to the machine settings and procedure. The HIT test requires a minimum of ten fragments in a narrow size fraction (for example -22.4 + 19 mm or -16 + 13.2 mm). Fragments may be sorted to ensure all fragments are within a set mass tolerance around the bulk sample mean (for example ±20 %). This will ensure the mean mass of any set of ten fragments will be within five per cent of the population mean. As such the process was in theory in line with the particle sorting adopted by the SMC Test® protocol (see Section 5.3). The specific energy (Ecs) calculated from input energy (Ei) and average particle mass (m) as follows:

$$Ecs = \frac{E_i}{m} = \frac{MgH}{m} \text{ (J/kg)} \qquad \text{Equation 16}$$

where:

M = a mass of the crusher assembly (crusher hammer, crusher shaft and crusher weight) (kg)
g = the gravitational constant (9.8 m/s^2)
H = a height of the crusher hammer (m)
m = a weight of the fragment of granular material in the sample cup
Ecs can be converted to the units of kWh/t by dividing by 3600

The broken product from all ten fragments is dry sized for one minute using a single sieve, representing the t_{10} size (for example, the t_{10} would be 2 mm for -22.4 + 19 mm size fraction fragments). The mass per cent of undersize is referred to as t_{10}. The slope, or t_{10}/Ecs, is the raw hardness index (HDI):

$$HDI = \frac{t_{10}}{Ecs}\left(\frac{\%}{kWh/t}\right) \qquad \text{Equation 17}$$

HDI needs to be corrected for the slope offset and particle size to estimate the JKDWT A*b parameter. This is done in consultation of the manufacturer (SimSAGe Pty Ltd). Calibration against the standard SMC or DWT parameters is possible, taking into account the offset for the slope and effect of particle size. Table 9 shows the sample preparation and experimental procedure to run this test.

Table 9. Standard Operating Procedure for HIT Testing (Source: Kojovic 2016, Copyright: Toni Kojovic)

SAFE OPERATING PROCEDURE	
HIT (Hardness Index Tester) – Axb UNSORTED FRAGMENTS	
Task Background:	Single particles are placed in a cup, and broken by drop mass from a fixed height; broken particles are collected, sized on one sieve, and the mass split across the sieve recorded.
Scope & Application:	This SOP details the safe method of using the HIT device
Hazards:	Noise, dust, physical strain, physical injury
PPE:	Standard site requirements, Hearing Protection, Eye Protection, Dust Mask
Tools/Equipment:	Brush, Tray, Scale, Sieves
Related documents:	HIT Data Entry (Axb UNSORTED).XLS
Other Requirements:	

Step	Explanation	Critical Comments
1. Sample prep	Obtain approximately 500g of sample Screen out the particles in the 22.4x19.0mm size range using 200mm diameter screen. Count and weigh all the particles. Select a minimum of 10 particles using a random selection method	
2. Test	Record the mass of the combined 10 particles Place the sample cup in position Please first particle into cup, and release drop mass by pulling on the pair of red levers simultaneously. Lift up the drop mass up until it automatically locks into an engaged position using the latch pins. Transfer the crushed fine particles to a tray by using the handle on the sample cup Repeat the procedure for the remaining 9 particles.	Ensure sample cup is always properly seated in recess. Brush out cup if required
3. Sizing	Record the mass of the combined total of 10 broken particles Place the broken material onto 2.00mm sieve, with a pan and lid. Shake or sift for 60 seconds. Record the mass of the screen oversize. The u/size (pan) mass is calculated from the initial mass of broken particles by difference. Store oversize and undersize material into pre-labelled sample bag(s).	Time required for screening will depend on mass of particles, their friability, and method (RoTap or manual). The sieve is selected to determine the t10 percentage.
4. Report	Raw results are saved in data entry template and a copy sent to SimSAGe for checking by Toni Kojovic.	
5. Calculation of A*b	The slope (T10/Ecs) is determined from the raw data for each sample. Conversion to an equivalent DWT basis A*b is provided by SimSAGe.	The conversion will be conducted by SimSAGe during the trial period.

Figure 145 shows a comparison between the HIT measured A*b and the Full JKDWT Drop Weight Test measured A*b on parallel samples. Figure 146 shows a comparison between the HIT measured A*b and the SMC test measured A*b, again on parallel samples. As can be shown, correlation between both conventional tests is reasonably good. It is to be remembered that the purpose of this test is not to replace the JKDWT in the characterization of ore to design a mill, but to domain map a deposit in context of comminution process response.

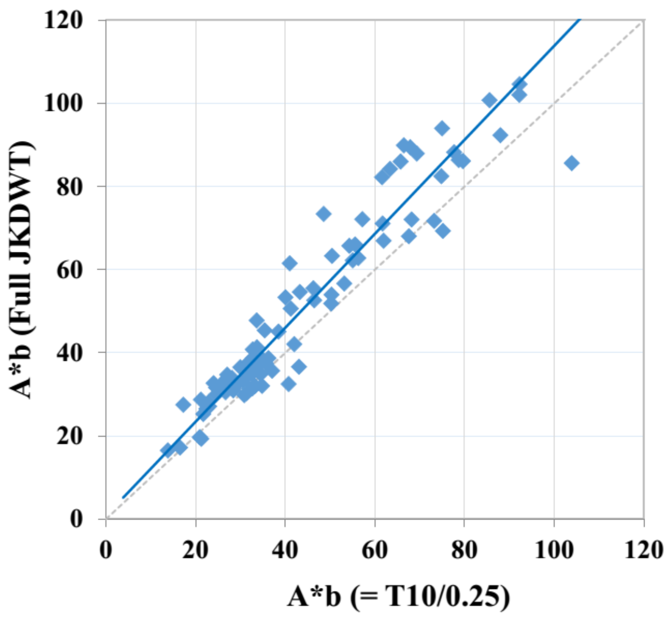


Figure 145. Comparison of JKDWT A*b Values and Corresponding A*b Estimates Using T_{10} for 31.5×26.5 mm Fraction (93 samples, 32 ore deposits) (Source: Bergeron et al 2017, Copyright: Toni Kojovic)

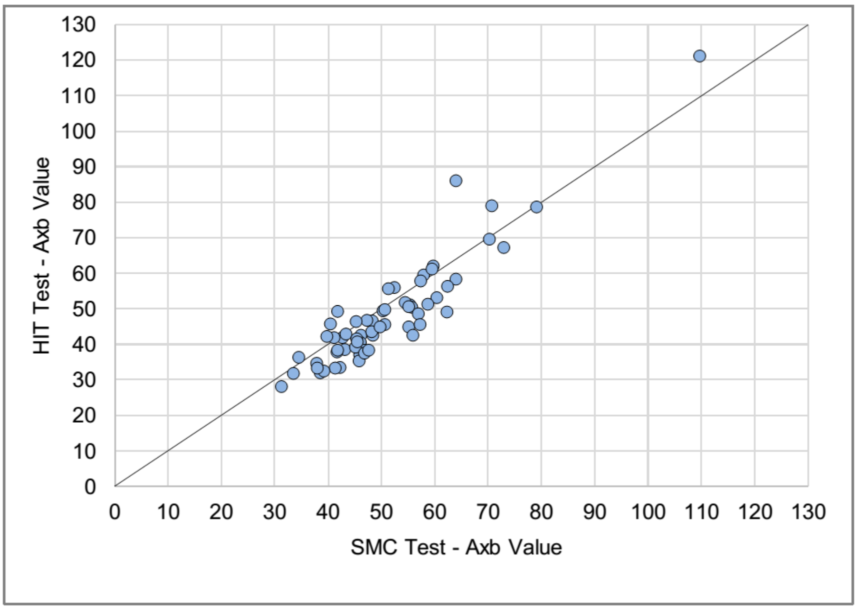


Figure 146. Comparison of SMC Axb Values and Corresponding HIT Axb Proxy Estimates (62 drill core samples) (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

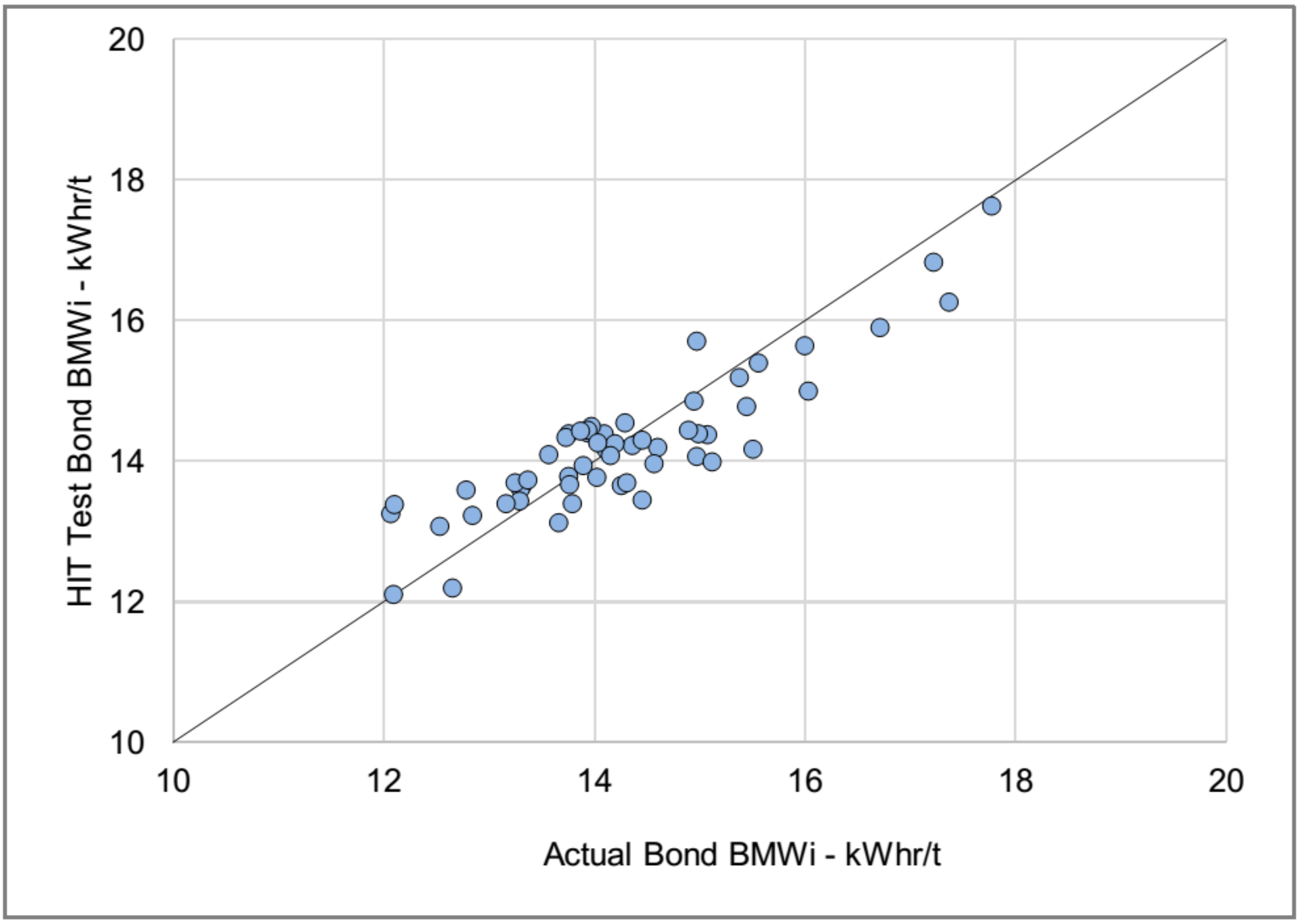


Figure 147. Comparison of Estimated and Measured Bond BMWi Results (62 drill core samples) (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

Figure 147 shows the HIT proxy estimated Bond Ball Work Index, BMWi (see Section 4.5) and the actual measured BMWi on parallel samples. This result is interesting. One test can estimate the comminution response of both impact breakage and grinding. So one test on one sample group (10 fragments) can be used to estimate and comparatively rank the comminution footprint of that ore.

The opportunity could be to predict the performance of the comminution circuit. Figure 148 and Figure 149 shows an example of this, compared against actual plant performance. This ore was processed in an established mining operation, where the SAG mill specific energy, and the Ball Mill product output was measured for a trail period of 53 days. The SAG mill specific energy and the Ball Mill P_{80} was calculated in simulation based on a HIT test. Figure 148 shows a comparison between simulated SAG specific energy and measured SAG mill specific energy. Figure 149 shows comparison between simulated and measured Ball Mill P_{80}.

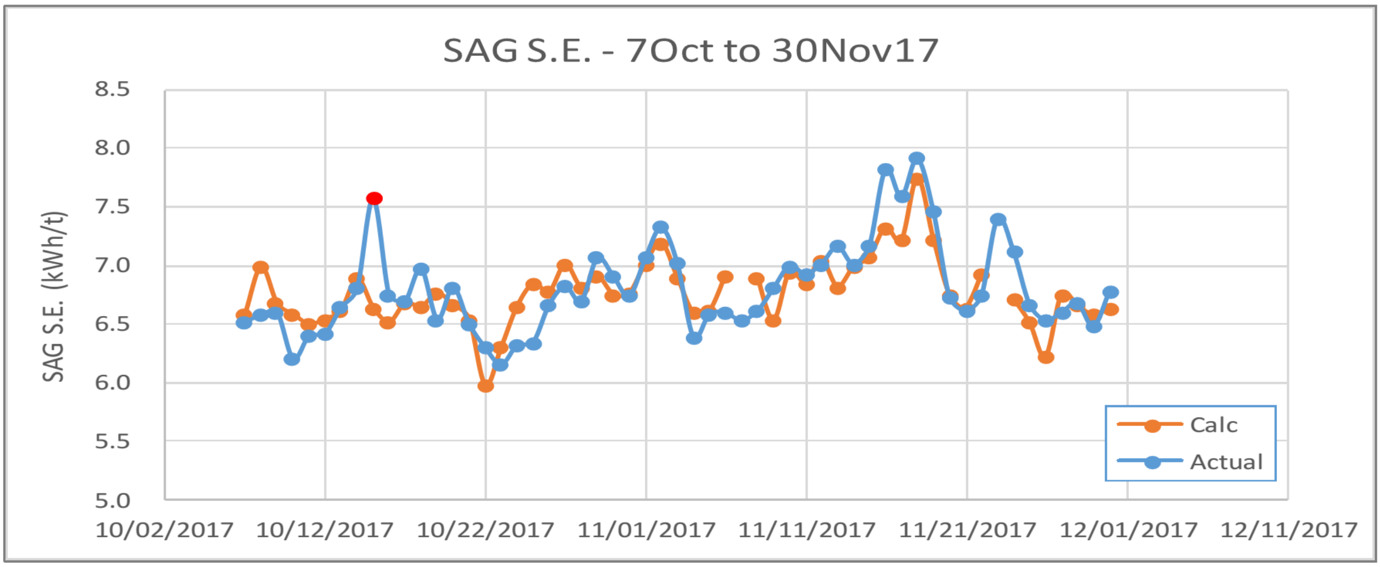


Figure 148. Trend Showing Actual and Calculated SAG Specific Energy for the Initial Trial Period (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

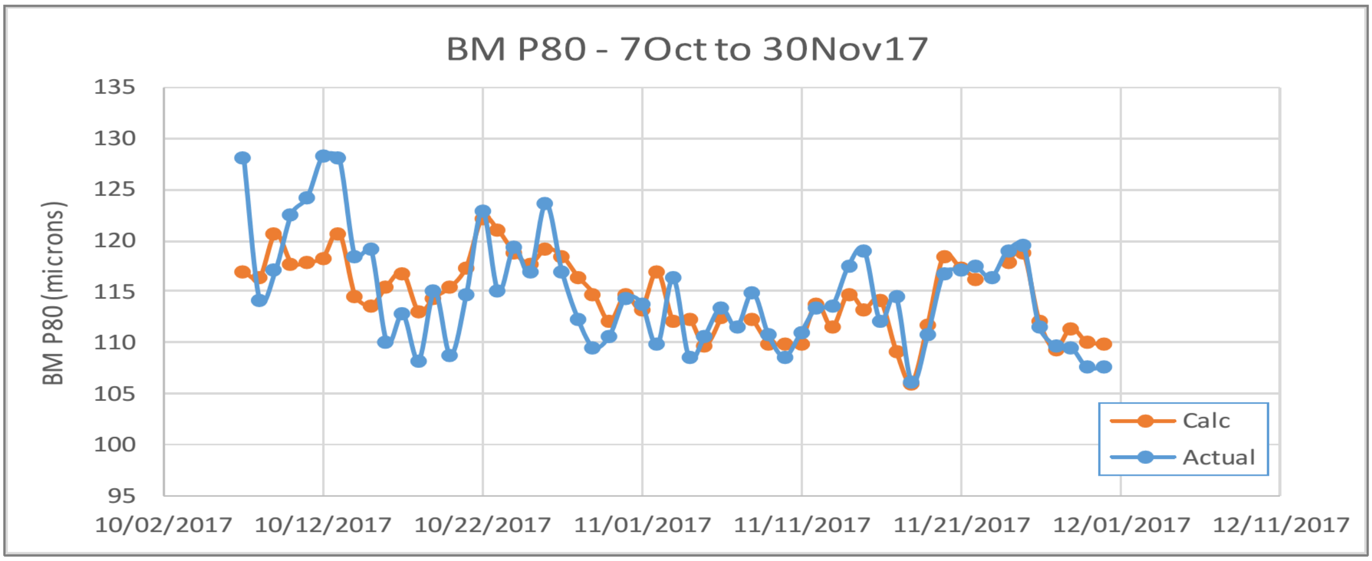


Figure 149. Trend Showing Actual and Calculated Ball Mill Cyclone Overflow P80 for the Initial Trial Period (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

The HIT evaluation program, running for 53 days was extended shown in Figure # and # (Kojovic et al 2019) was extended, generating a further 107 days of sampling. This additional period provided an ideal data set for model validation. To this end, the model developed from initial 53 days of monitoring was used to forecast performance for the remaining 107 days not used in model calibration. This approach was considered a textbook example of empirical model validation (Kojovic and Whiten, 1994). The calculated and actual SAG Specific energy and ball mill P_{80} trends over the periods covered in the regression analysis and validation are shown in Figure 150 and Figure 151.

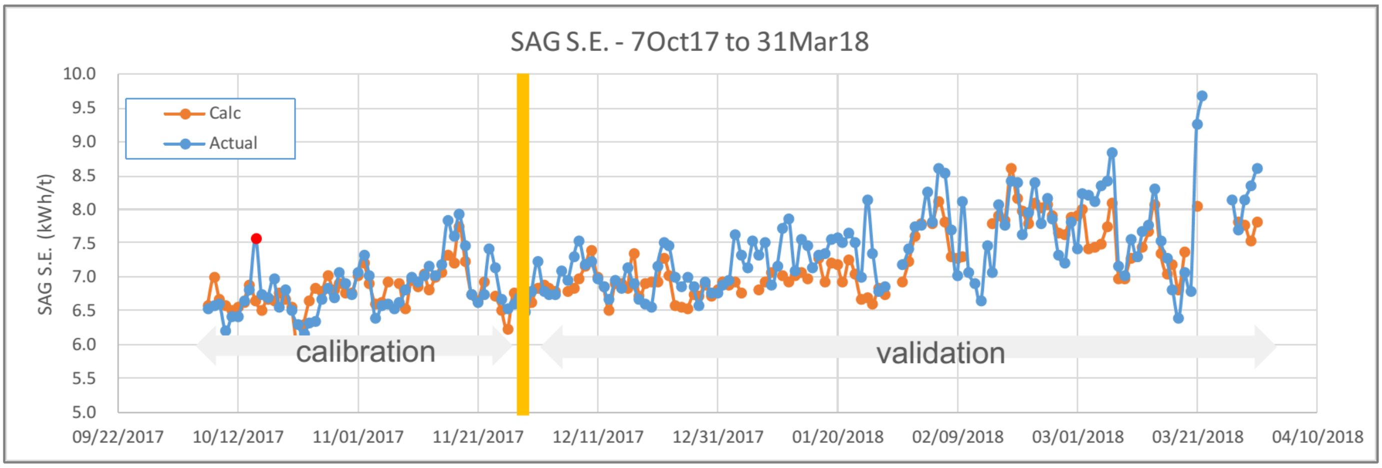


Figure 150. Trend showing Actual and Calculated SAG Specific Energy for the Initial Trial and Forecast Periods (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

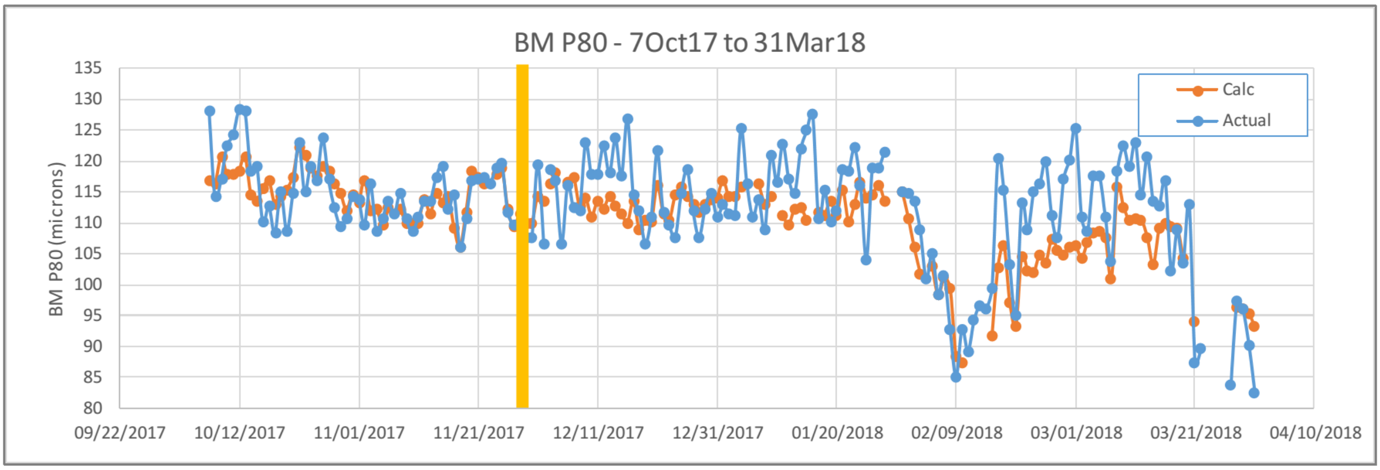


Figure 151. Trend showing Actual and Calculated BM Cyclone Overflow P_{80} for Initial Trial and Validation Periods (Source: Kojovic et al 2019, Copyright: Toni Kojovic)

This result shows the potential value of this basic approach. There are many ways this testing approach could be used, in geometallurgy and operational process prediction.

5.12 Comparative flotation test JKMSi

The JK mineral separation index (JKMSi) is a small-scale flotation test that requires feed material of 10 g of sample (Vos & Bradshaw 2014, and Morgan *et al* 2012). Figure 152 illustrates the comparison between the small-scale float test and a batch float response. Figure 153 is an example of a plant flotation response compared to the JKMSi.

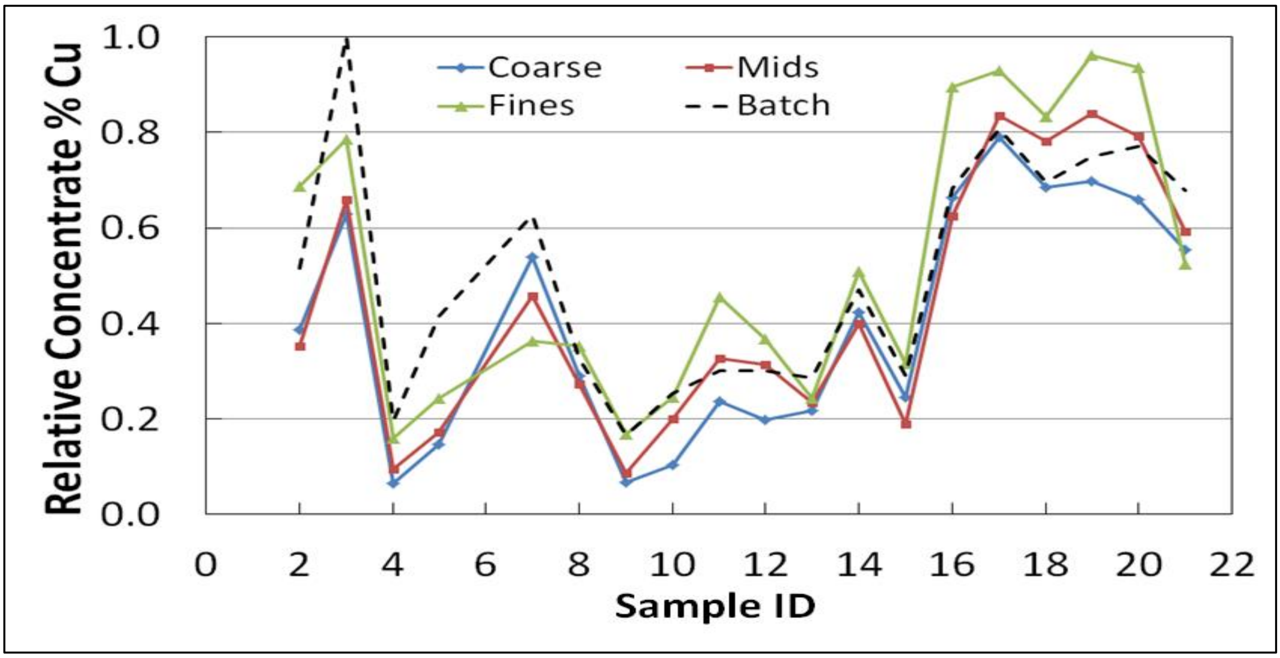


Figure 152. Comparative flotation response test JKMSi compared to conventional batch flotation tests (Source: Vos & Bradshaw 2014, Image: François Vos, Copyright: AusIMM)

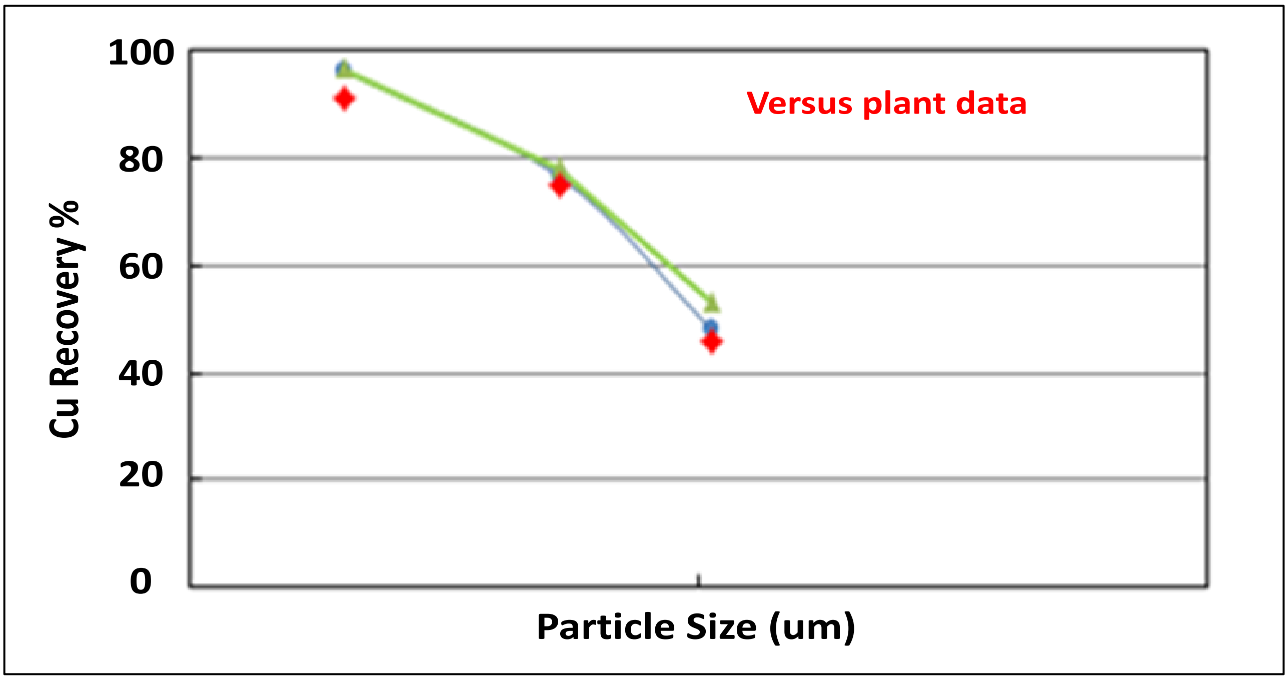


Figure 153. Comparative flotation response test JKMSi compared to production plant recovery (Source: Vos & Bradshaw 2014, Image: François Vos, Copyright: AusIMM)

5.13 CSIRO proxy diagnostic leaching test

The focus of the small-scale leach tests has been to determine the leach index (relative indication of leach performance), recovery, impurity deportment, reagent consumption, effect of size/degree of liberation and mineralogy of samples for their relative ranking (Figure 154). Predictive tests for heap leach hydrology (related to agglomerate characteristics) and slurry rheological behavior are also being developed (Kuhar *et al* 2011).

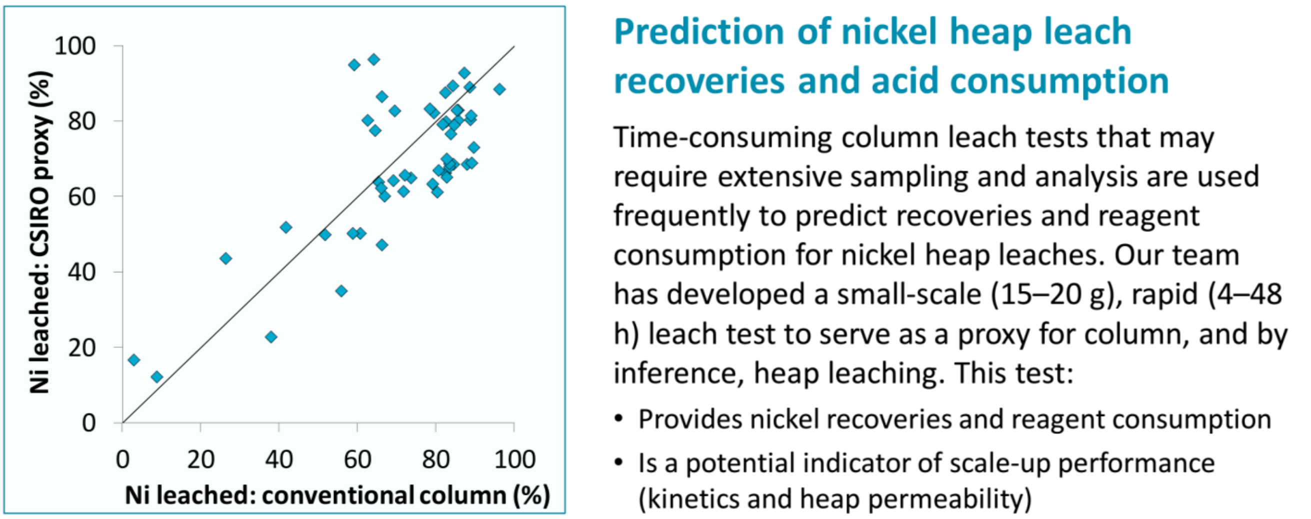


Figure 154. Correlation between nickel leached in the CSIRO proxy test and that from conventional column least tests (Source: Kuhar et al 2011, https://www.csiro.au/en/Research/MRF and https://people.csiro.au/K/L/Laura-Kuhar Image: Laura Kuhar, Copyright: CSIRO)

To use this test, a sample of 5 to 10 kg of each end member ore type is to be sent to CSIRO for characterization. This would result in the diagnostic leach test being refined to target minerals of interest in context of the geometallurgical campaign. Sample mass requirements for the diagnostic test itself is 15 to 20 grams but can be less depending on circumstances.

5.14 Establishing what comparative tests are appropriate for the target deposit

For each end member ore rock texture, conduct a bankable large-scale test (for example if the process behaviour of interest was leaching, do a column leach test), characterize the rock in multiple scales and forms (SEM mineralogy, optical mineralogy, QXRD, chemical assays, etc.), and conduct a series of small-scale comparative tests. Assemble the result together for all of the samples end member extreme rock textures. Develop a deposit specific relationship between one (or a combination of) comparatives tests and the bankable test. Develop understanding of why these samples might rank in context of mineralogy. Ensure sample representivity has been maintained across all tests. Design a propagation of analysis for all analytical outcomes.

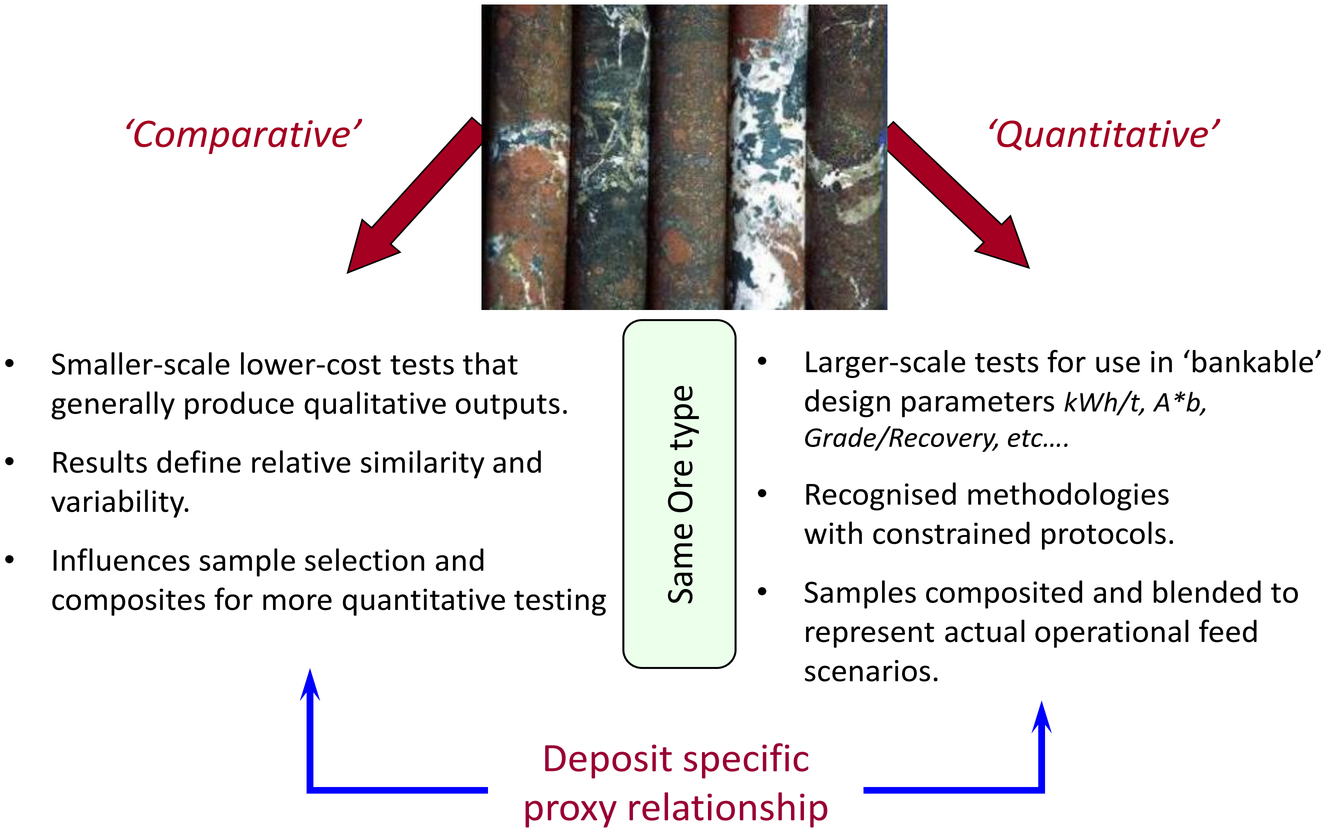


Figure 155. Development of the relationship between comparative proxy tests and larger scale bankable tests (Image: Simon Michaux)

Comparative tests are selected by past demonstrated usefulness and intuition-based selection of new technology. As each deposit is unique, the data outcomes will vary in usefulness. A good metric for usefulness is the quality of the relationship between the comparative test and the bankable test. Strength of correlation between tests is perhaps the most practical measure. Another good metric is the range of results returned. Does the full range of experimental results of the comparative tests exceed the set error bars of the same comparative test? The Orientation Study is an excellent time to test assumptions and question new ideas. The final Orientation Study conclusions need to be measurement based in a practical fashion that can demonstrate propagation of error, sample representivity and has a clear data QA/QC procedure.

6 ORIENTATION STUDY DESIGN

The tested sample interval is selected based on the number of tests (and what sample mass they require). If possible select the same intervals of chemical assay measurement, or a multiple. This is usually a 2m interval of core (sum several intervals together?). By the time the tests being selected for this part of the study is done, a great deal of work should be done to consider the context of these tests and what the data is needed for. It is simple to try and solve all problems in one experiment campaign. Budgets and time frame to delivery agreements rarely allow for this.

Figure 156 represents a starting point, from which a cut down test schedule, fit for purpose to the original geometallurgical objective, can be designed. A clear understanding of what the data is needed for and how it will be analyzed needs to be in place before this is done. Some tests are destructive, which disqualifies some kinds of tests after they are done (e.g. most metallurgical tests). Some do not (e.g. hyperspectral analysis or EQUOTip). As such, the experimental test work is to be planned together, then done in a planned order.

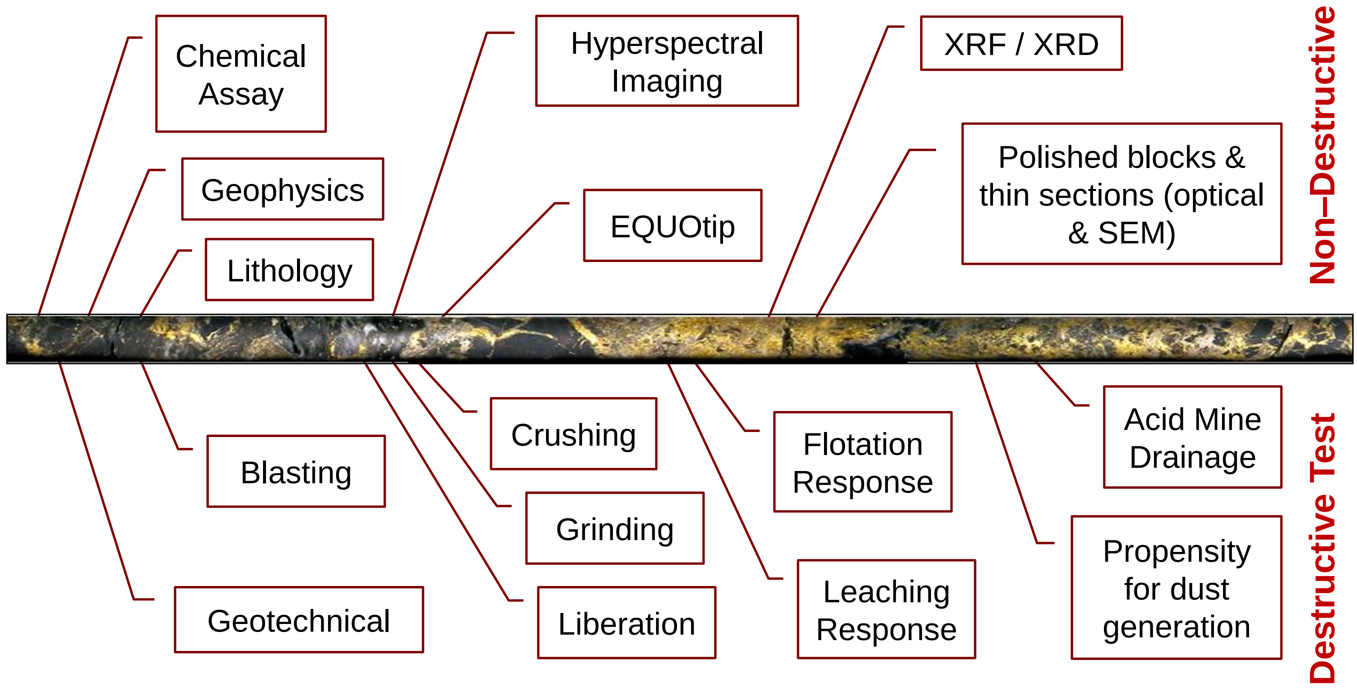


Figure. 156 Experimental test work done on each sample core length interval
(Image: Simon Michaux)

Below is a case study, where the objectives were to understand comminution and flotation process behavior. The questions the experimental design and analysis were developed to answer were:

- What is the mineralogy that controls impact breakage?
- What is the mineralogy that controls grinding?
- What is the mineralogy that controls flotation recovery?
- Can these phenomena be related to a small-scale proxy test?
- Is it possible to map these into the deposit?

Figure 157 shows the Orientation Study and Figure 158 shows the Hypothesis Sample study.

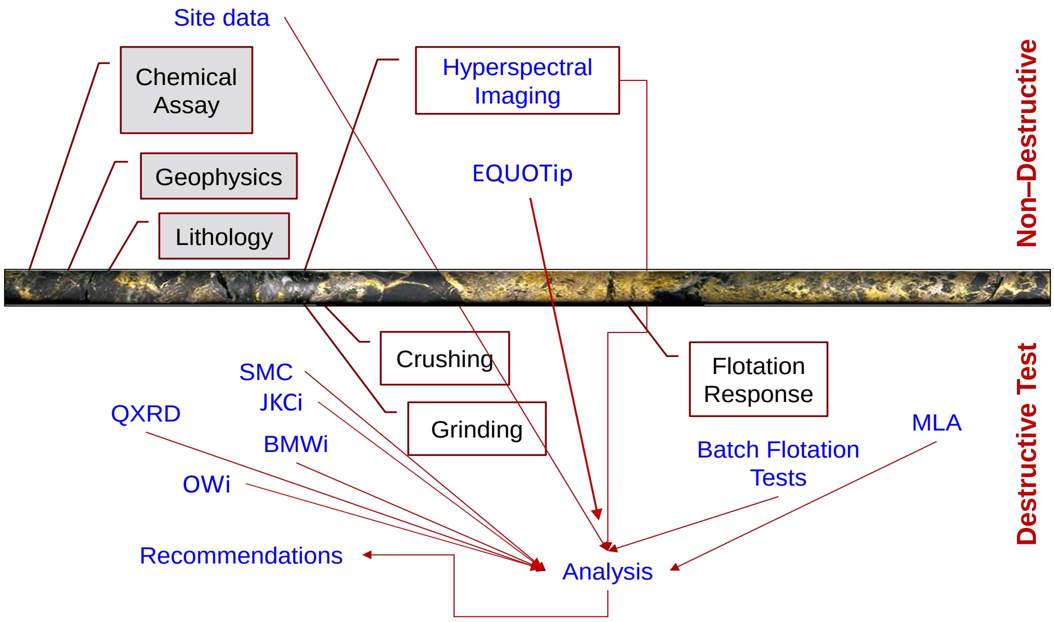


Figure 157. Case Study P, Orientation Study 16 samples
(Image: Simon Michaux)

At the end of the Orientation study, a working relationship between the JKCi, an operating work index (OWi) and the BMWi had been established. It was decided that the SMC test would be used to estimate the Axb impact breakage parameter. A small-scale comparative test for flotation was not able to be done in this study due to logistical constraints. A full batch flotation test was done for each sample in the Hypothesis set. An understanding of what mineralogy might control comminution and flotation was determined with the Hyperspectral imaging, QXRD and MLA results. EQUOTip was and Point Load Index was to be used as model inputs with both JKCi-CRU and JKCi-GRD outputs of the Comminution Index to model BMWi domains.

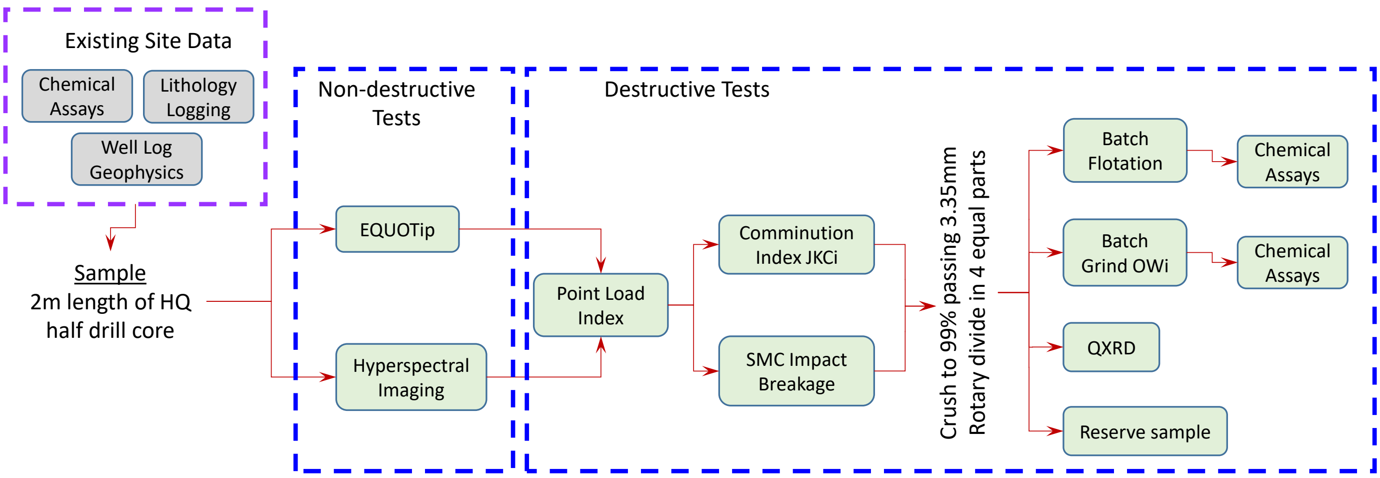


Figure 158. Order of test work in Hypothesis Sample set in Case Study P
(Image: Simon Michaux)

Figure 158 shows the established order of test work. Data QA/QC was based on a mass balance reconciliation between tests against the original sample mass. The Point Load Index breaks intact core. The Comminution Index and the SMC test can be run on the Point Load Index products. The products of the Comminution Index and the SMC tests were re-combined and then crushed to 99% passing 3.35mm and then was rotary divided into four equal sub samples. One sub-sample went into the freezer as a flotation feed sample (to reduce the risk of sample oxidization). One sub-sample was tested with the Batch Grind Operating Work Index (OWi) procedure. The products were sampled again at target size fractions, which were sent to chemical assays. One sub-sample was sent to QXRD analysis. One sub-sample was kept in reserve if needed for future work (in the freezer at -200C in case flotation work was needed). Figure 159 shows the planned Hypothesis Sample analysis and test work flow sheet.

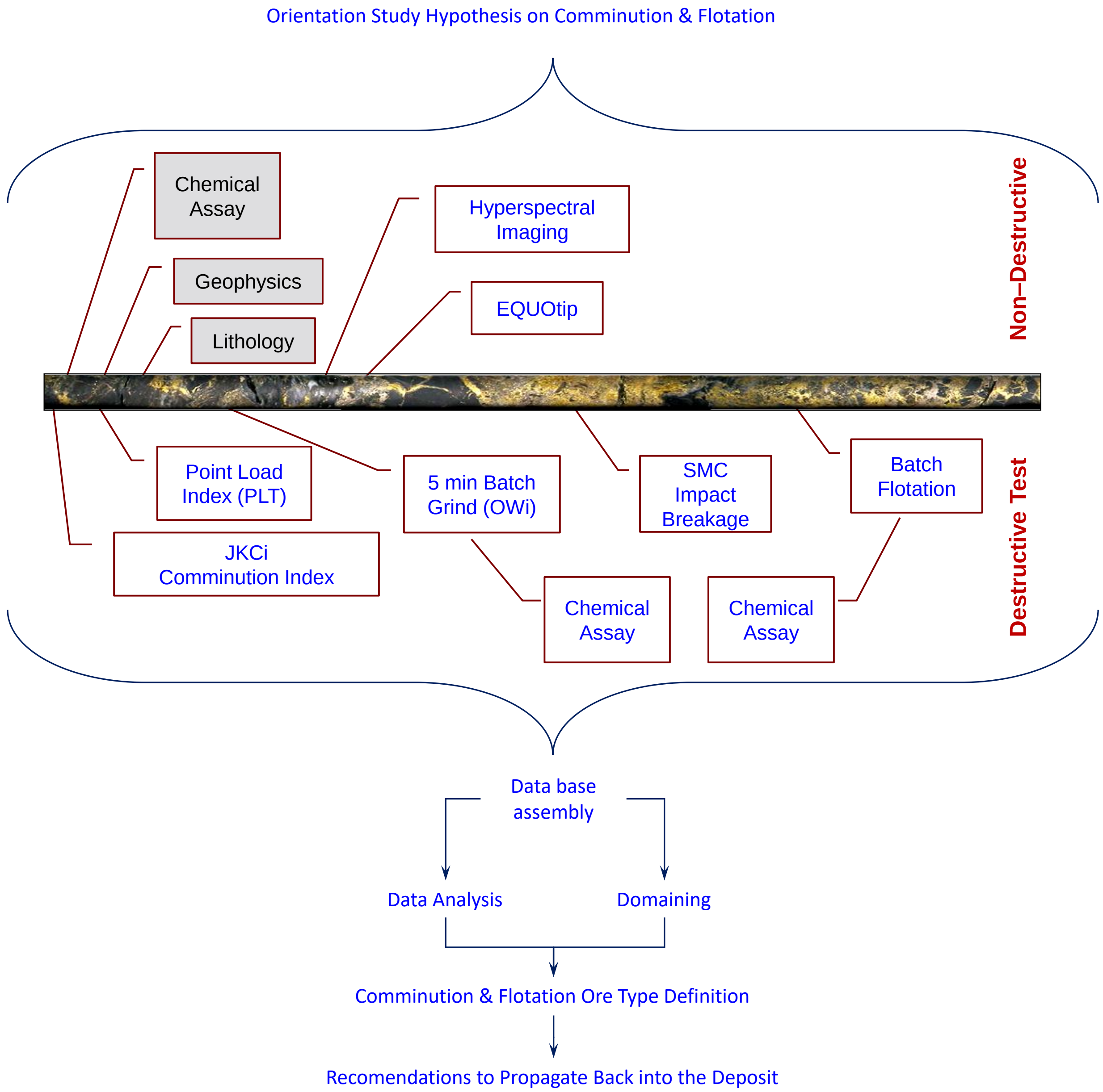


Figure 159. Case Study P, Hypothesis Sample set 218 samples
(Image: Simon Michaux)

7 ORIENTATION STUDY DATA ANALYSIS

Data analysis of orientation study samples needs to address the following questions achieve the following tasks:

- What tests showed the greatest spread in variability?
- What tests are useful here?
- Are some end member ore types very similar in process response space?
- Should they be merged into one ore type in this process?
- Do the tests interrelate?
- Is a good relationship between bankable tests and proxy tests viable for this deposit?
- What are mineralogical influences that can be seen as patterns across the samples?
- Set hypothesis spectrum for next experimental phase
- Define what tests are to be done in Hypothesis Sample Study and what QA/QC metrics they are subject to

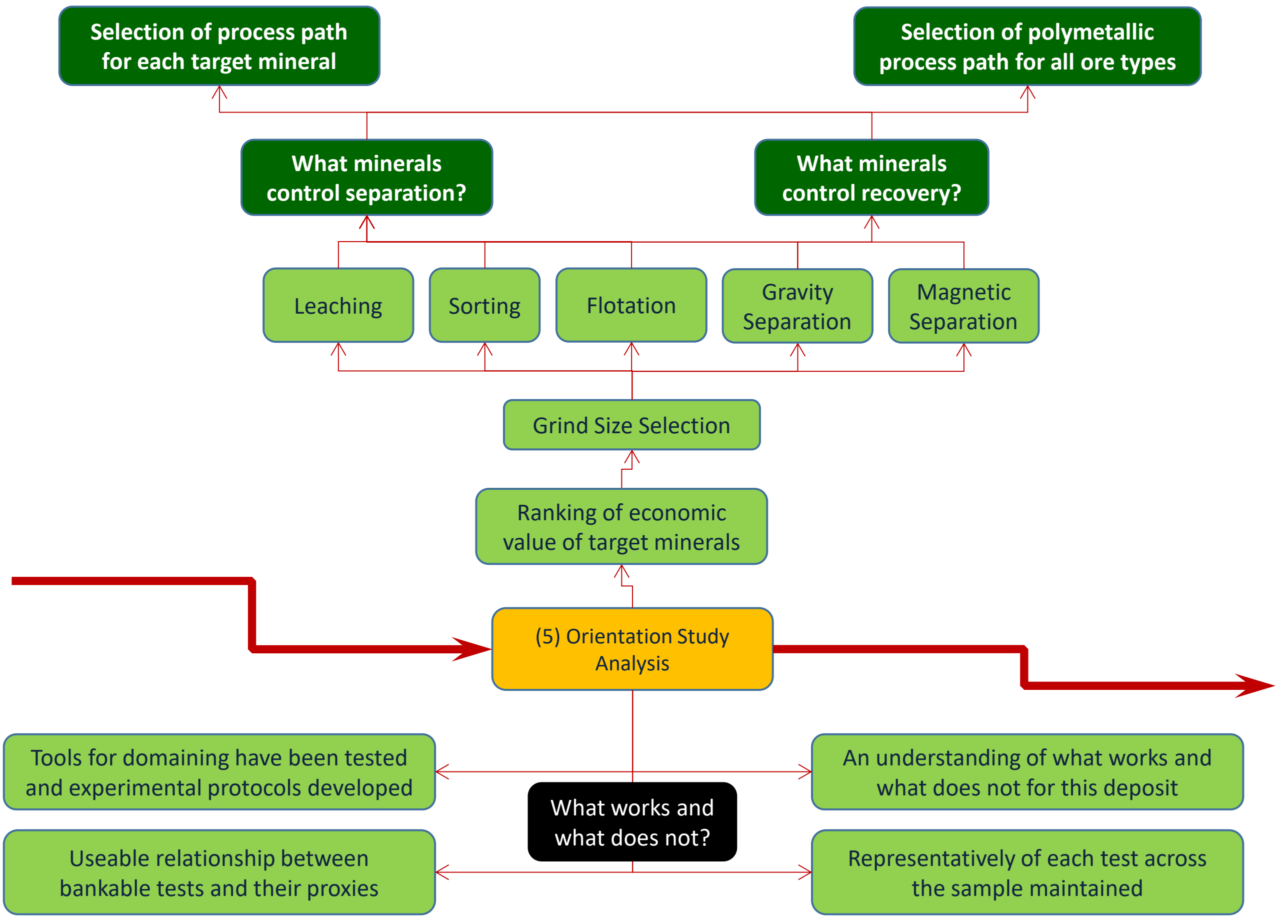


Figure 160. Orientation Study analysis
(Image: Simon Michaux)

7.1 Experimental design of the Orientation Study

The following is an example of an Orientation Study experimental design.

7.1.1 Step 1 – Sample selection.

Selection of all of the rock texture end members (in this case 10), together forming the samples for the Orientation Study. Each sample would be approximately 20-30kg. It is of paramount importance that the mineralogy and lithology rock texture is consistent throughout this sample. It is preferable to have less sample mass and more consistent mineral rock texture, than a larger sample mass. This sampling is to be done with the consultation of a geologist that has experience in working with this deposit. Usually, this takes the form of drill core, and/or can be coarse residue from chemical assay sampling campaigns.

Figure 161 and Table 10 shows an example from the literature. This was a case study from Nickel Rim South (NRS), which was a high grade copper–nickel–PGE deposit (Lotter *et al* 2011). In this case study,a total of six distinct geometallurgical populations were defined within the deposit based on their mineralogical characteristics. These include four ores within the primarily Cu bearing, PGM.

Footwall:

- Main (core).
- Upper.
- Fringe.
- Low Sulphur PGM.

Along with two ores from the Ni bearing, higher pyrrhotite contact zone:

- Sublayer Breccia.
- Footwall Breccia

Table 10 shows a summary of the key mineralogical features within each ore type. Figure 161 shows typical textures from each of the geometallurgical units, as imaged by QEMSCAN. The scale of each image is approximately 3.5 mm x 3.5 mm. Images show that the Main Footwall and Upper Footwall textures are coarsest. Sulphide textures in all other geometallurgical units are finer grained and more commonly associated with silicate gangue (Lotter *et al* 2011).

Table 10. Key mineralogical features within each ore type of the NHS case study (Source: Lotter *et al* 2011)

	Ore type	Stratigraphic position/host rock	Minerals and texture	Presence of pyrrhotite/Ni in pyrrhotite	Presence of PGM
Footwall	Main (core)	Granitic Footwall	Coarse MS veins of Cpy and Pn	Low levels (3–6 wt.%), Ni in solution is low (0.05 wt.% Ni)	Elevated levels dominated by michenerite and maslovite/ moncheite
	Upper zone	Footwall Breccia and Granitic Footwall	Coarse MS veins of Po, Cpy and Pn	20–25% Po, Ni in solution averages 0.4% Ni	Low PGM content
	Fringe	Along margins of main zone, in Granitic Footwall	Disseminated or stringers of Cpy, Bn, Mill, Pn	No pyrrhotite	Elevated and more varied species including michenerite, maslovite/monchiete, froodite, sperrylite, niggliite; finer than in main geomet unit
	Low Sulphur PGM	Lower extremites of Fringe in Granitic Footwall	Fine Disseminated Sulphides (~2%)	No pyrrhotite	Elevated PGM content
Contact	Sublayer Breccia	Hosted by Mafic Norite Breccia (elevated MgO) at top of stratigraphic sequence	Blebby to SMS; Po, Pn and minor Cp	20–25% pyrrhotite. High levels of Ni in solution (average 1.07%)	Very low PGM content
	Footwall Breccia	Hosted in Felsic to intermediate Footwall Breccia directly below Sublayer	Blebby, SMS and MS	20–25% pyrrhotite. High levels of Ni in solution (average 1.07%)	Very low PGM content

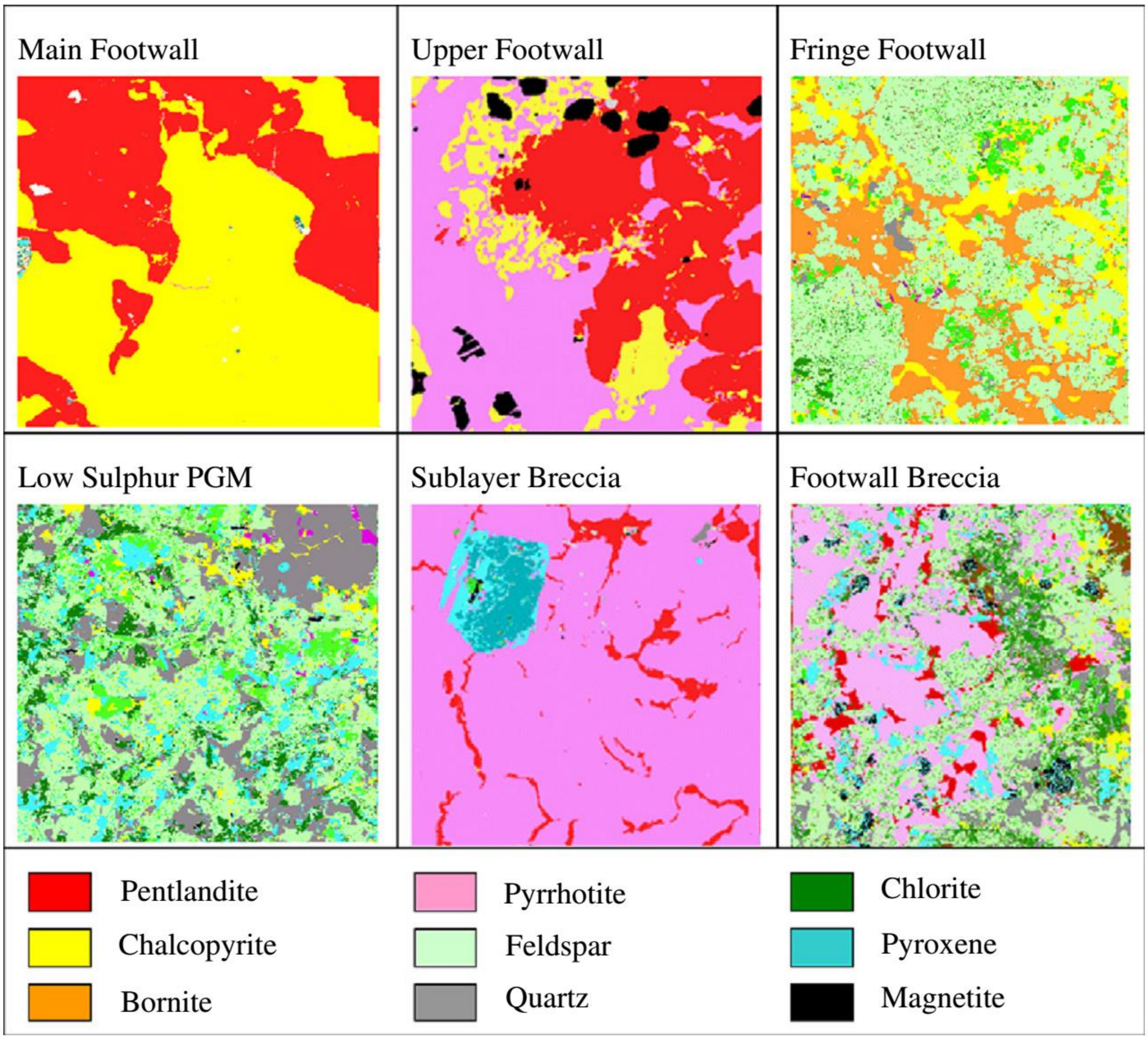


Figure 161. Geometallurgical unit textures as measured by QEMSCAN for the ore type of the NHS case study (Lotter et al 2011)

For the purpose of this report, Figure 161 will be broken up and used as a theoretical example of pictures of different samples for the orientation study. The QEMSCAN mapping has really highlighted the different mineral signatures with the use of color. This is shown in Figure 162.

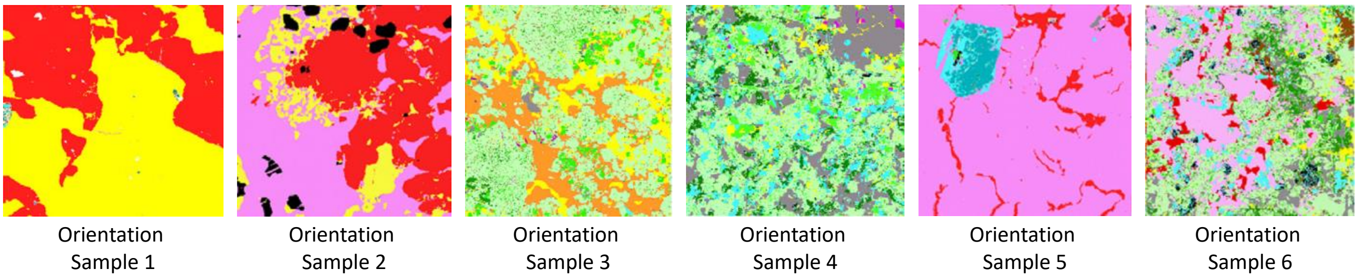


Figure 162. Theoretical example of six end member texture samples selected for the Orientation Study (Image: Lotter et al 2011)

7.1.2 Step 2 – Sample preparation.

A number of experimental tests are to be run of each sample, where a number of very different process separation methods are examined. This has to be done representatively, where all tests done represent particles of rock taken from all parts of the original sample mass. As a general rule of thumb, each Orientation Sample would be approximately 20 to 30kg in mass.

Crush the sample to a form where it can be rotary divided or riffled into subsections. If sorting response is being examined, the sample is to be crushed to a top size of approximately 10-12mm, and a sorting sample riffled out (in this case 1/6th). The remaining 5/6th of sample mass is then crushed to a top size of 3.35mm (99% of sample passing a 3.35mm sieve). The top size of 3.35mm is selected to ensure that there are enough particles to allow for particle statistics to be applied to each sub-sample.

The remaining 5/6th sample is rotary divided into 5 sub-products using a rotary divider. So of the six sub-products, one is -12mm crushed material (for sorting) and the other five are -3.35mm crushed material.

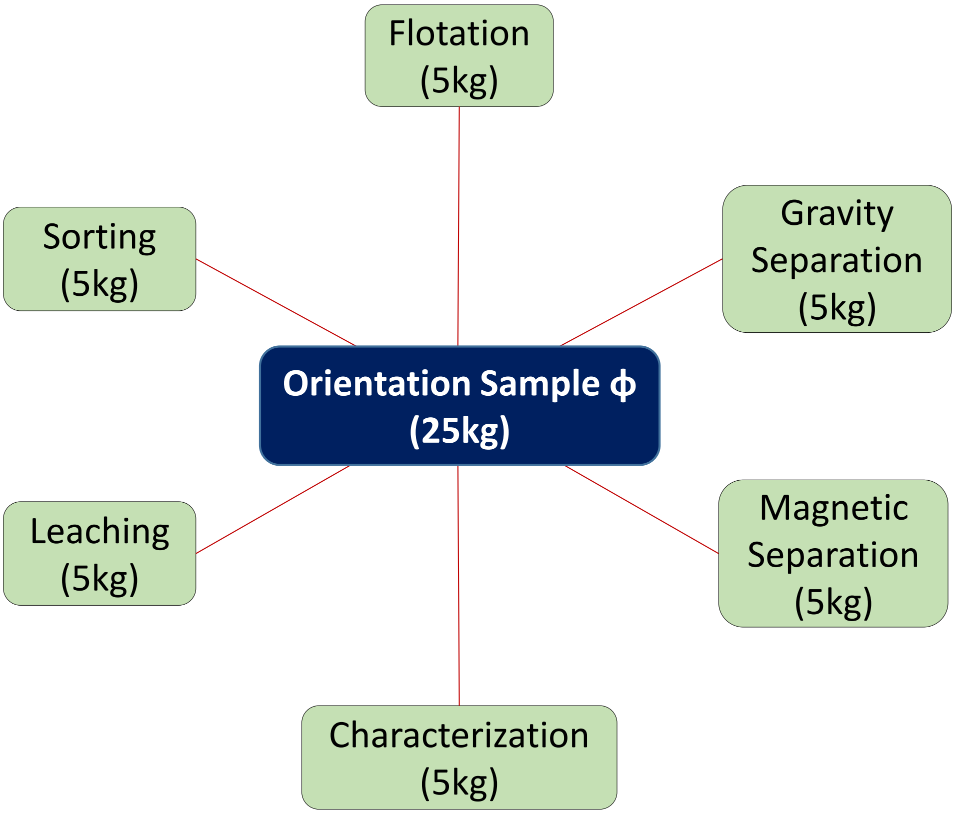


Figure 163. Primary crushing preparation for each Orientation Study sample (Image: Simon Michaux)

7.1.3 Step 3 – Characterization of each Orientation Study sample.

Before any other testwork (if possible), each of the Orientation Study samples need to be comprehensively characterized. A number of methods are used in parallel for a variety of reasons.

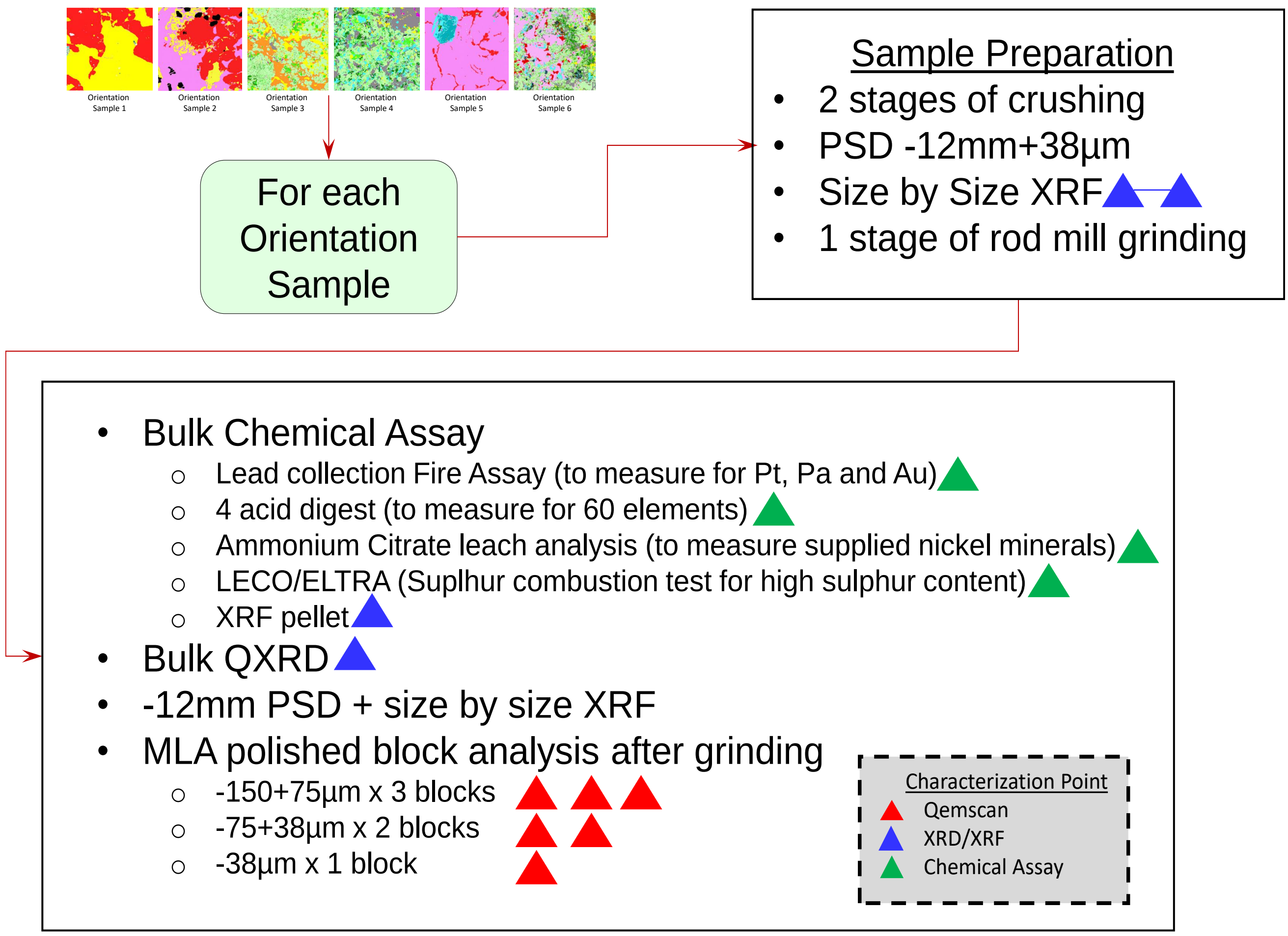


Figure 164. Primary sample characterization for each Orientation Study sample (Image: Simon Michaux)

Chemical assay measurement is the basic standard for the whole campaign and for further routine work, due to the comparatively low cost and small mass requirements. The following suite have shown to be useful:

- Lead collection Fire Assay (to measure for Pt, Pa and Au), (requires 50g sample)
- 4 acid digest (to measure for 60 elements)
- Ammonium Citrate leach analysis (to measure supplied nickel minerals)
- LECO/ELTRA (Suplhur combustion test for high sulphur content)

If there is the presence of precious metals like gold (Au), platinum (Pt) or palladium (Pd), then one of the fire assay tests are required. Due to the very fine grain size, and very low grades mostly encountered, the sample mass required is often much larger than other assay methods. A minimum of 50g is recommended.

While most mining operations will only assay for their target metal, process response is usually controlled by gangue mineralogy in some form. The 4 acid digest chemical assay measurement method is recommended.

Often, the target element is associated with some kind of sulfide mineral. These can be difficult to quantify with conventional assay measurement methods. The Ammonium Citrate leach analysis and the ELTRA (or LECO) tests are recommended.

In addition to this, further characterization can be done if the rock texture (and campaign objectives) has a signature that could influence process response. An XRF pellet (requires 10g sample) test will give the element distribution in the sample. A QXRD (requires 50g sample) test will give mineralogy present in a bulk context.

There are three systems of automated mineralogy. QEMSCAN, MLA and Ziess Mineralogic. All three have strengths and weaknesses. Selection of what method to use is a judgment call to be done by the geometallurgy analyst team.

The limitation of this kind of analysis is the cost of test work and the long turnaround time. As such, automated mineralogy is done at the start of the process (characterization of sample before process testwork) and at the very end of the best process path (as shown by chemical assays and XRD/XRF analysis). If this is done for each orientation study sample, this can create an expensive experimental program budget, which may need to be reviewed at each stage of planning for this reason.

- -250+150µm x 3 blocks
- -150+75µm x 2 blocks
- -75µm x 1 block

These three size fractions are recomended. The coarser fractions will need more poliched blocks measured to have a number of particles measured relevant in context of sampling particle statistics.

7.1.4 Step 4 – Selection of grind size for the whole campaign.

One of the critical decisions to make that has the capacity to make the entire geomet campaign fail if done incorrectly is the selection of sample particle grind size. This is the particle size (expressed as a P_{80}, or particle size that 80% of the sample is smaller than) to which all samples for process separation a milled down in size through grinding (See Section 4.7).

The process of comminution is to subject the sample to size reduction. The particle size required for the effective liberation of the target mineral is defined by the mineralogy. For each mineral in the sample, there would be a range of mineral grain size distribution. For process separation to economically effective, enough of those particle have to be liberated or partially liberated.

An unconventional method of defining what the size distribution of the mineral grains are inside an ore type, could be the use of electrodynamic fragmentation in a Selfrag unit (see Section 4.7.1). If done correctly, and if the mineralogy is susceptible to this methodology, all minerals could be liberated intact at their true particle size. This method has its advantages if the ore has more than one valuable mineral (possible polymetallic process path), where each target mineral would have its own unique mineral grain size distribution. For example, an ore with a Cu mineral with a P_{80} grain size of 100 micron and a Au mineral with a P_{80} grain size of 10 micron.

Usually this is defined through a sophisticated test procedure to determine the grade recovery curve for that mineral in that ore type. The theoretical grade-recovery curve for an ore is a definition of the maximum expected recovery by flotation of a mineral or element at a given grade. This is defined by the surface area liberation of the value minerals and is consequently directly related to the grind size utilized in the process. The theoretical grade-recovery can be readily used to quickly identify potential recovery increases that can be gained through optimization of flotation circuits and whether the process is running efficiently.

To establish the theoretical grade-recovery curve for a material a mineralogical liberation study should be undertaken using tools such as automated mineralogy (QEMSCAN or MLA).

Ideally this work should be done either before or in parallel to a geometallurgical study. Given enough automated mineralogy data, a theoretical grade recovery curve could be estimated based on mineral grain size across a family of ore types.

A decisions is to be made for what P_{80} each of the process test samples are to be reduced to through crushing then grinding. As a laboratory scale rod mill produces a particle size distribution that better resembles a production scale circuit compared to a laboratory scale ball mill, all sample preparation should be done with wet grinding in a rod mill with a standard rod load. The choice of mild steel or another metal alloy for the rods is a matter of discernment in context of the ore mineralogy.

To prevent oxidization of the sample, after rod milling, the wet sample is placed in a vacuum sealed bag (ideally) and then stored in a freezer at a lower temperature than -18^{0}C. This should be done for any sample that would or could be later be subject to flotation or leaching.

The ideal situation is the definition of closing samples grind size is taken from previous work done on the same ore types.

7.1.5 Step 5 – 1st pass process separation for each Orientation Sample

Once the samples have been characterized and then prepared to an appropriate grind size in a rod mill, the different process separation methods can be trialed. Any engineering process could be studying in this context. The well understood methods are:

- Flotation
- Hydrometallurgical leaching
- Gravity separation
- Electromagnetic separation
- Sorting technology

The first step is to trial each of these process methods on the sample directly (Figure 166). This gives a base recovery performance for flotation and leaching in each ore type. It also examines the effectiveness of what gravity separation, electromagnetic separation and sorting has in context of what minerals report to what process sub-product. The sub-products of each of these process tests are to be characterized appropriately. This could be termed as Orientation Step 1, where each of these methods could be considered a process path.

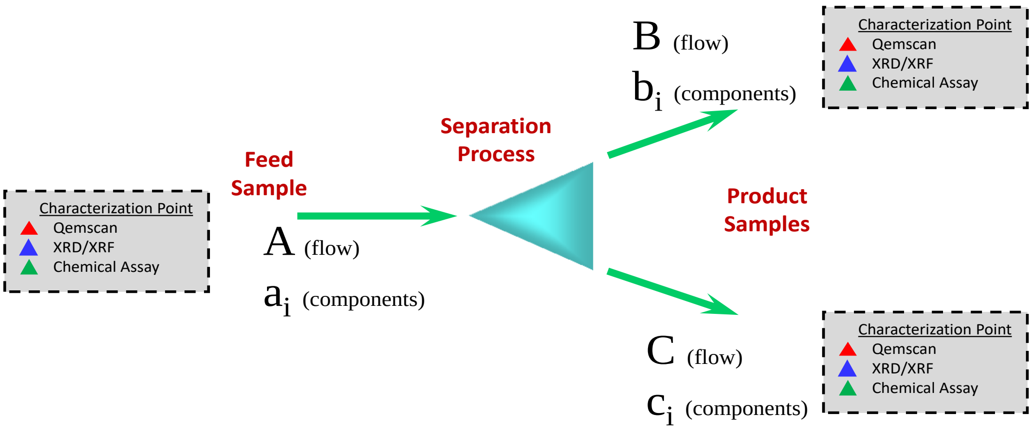


Figure 165. Mineral reconciliation for each process separation test done
(Image: Simon Michaux)

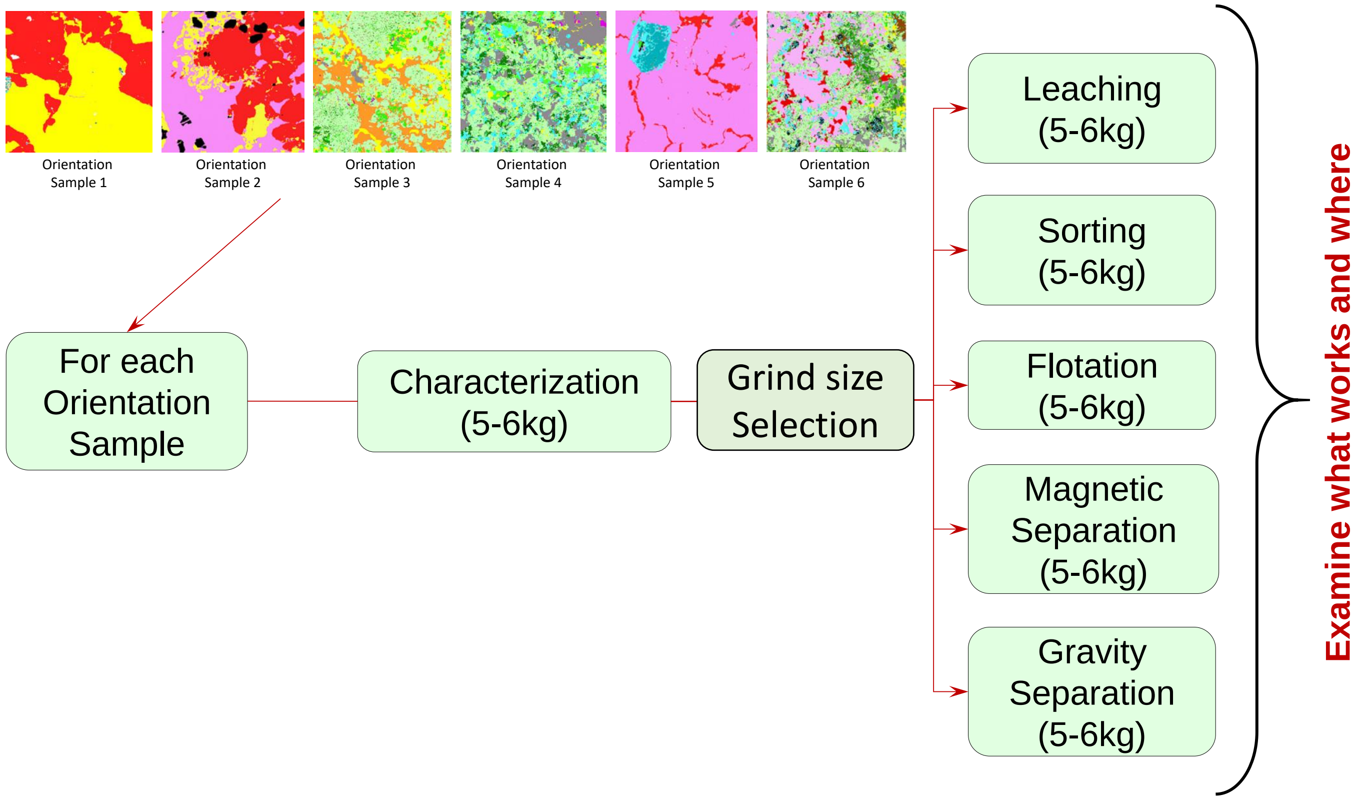


Figure 166. First pass process separation for each sample in Orientation Study Step 1 (Image: Simon Michaux)

Each process separation method should conducted in a way where the feed sample was characterized appropriately and the process products are also characterized in the same way. The separation process could be examined in context of what minerals where separated, by reconciling the mineral content of all the products in reference to the feed (Figure 165).

All data is collected for each orientation study sample and examined in context of what was successful in context of recovery and what possible pre recovery steps may be effective.

7.1.6 Step 6 – Data Analysis of the 1st pass process separation for each Orientation Sample

Each of the process separation tests done (4 or 5 different test methods for each Orientation Sample) would be examined in context of what minerals separated into what products. The characterized products would be compared to the characterized feed sample (Figure 136 and Section 3) to do a mineral mass balance reconciliation. The separation performance of valuable minerals as well as gangue minerals would be considered.

All separation methods are then considered together within each Orientation Sample and an understanding of what minerals were separated out into products and in what circumstances. Knowing what minerals are associated with the valuable target minerals is required. For example:

- Platinum PGE metals could be in grains 5µm in size, embedded into pyrrhotite and that pyrrhotite could be magnetic. Thus a magnetic separation has the capacity to isolate the valuable platinum group metals away from the sulfides.
- Gravity separation could extract out all the heavy minerals containing Au, PGE and heavy REE away from the sulfides.
- Gravity separation could extract out gangue mineralogy that may negatively influence flotation recovery.
- Leaching could be effective in extracting cobalt.
- Flotation response is poor and is being hampered by gangue minerals of a particular character

The products of each of these process separation tests could be subject to a proxy test like a diagnostic leach (done on small volume samples).

Consider how process combinations might work together. Using these insights, make recommendations for Step 7.

7.1.7 Step 7 – 2nd pass process separation for each Orientation Sample

Possible cleaning of the ore by removing a gangue mineral could be considered to improve later recovery. Assemble each test series into a proposed process path. This could be considered as Orientation Step 2. For example:

- Gravity separation could possibly remove quartz, clays and biotite
- Flotation could be used to remove sheet silicates and clays
- Magnetic separation could have practical difficulties to consider before flotation due to the required throughput volume in a production scale plant. That being stated, the possibility that gangue removal could improve recovery substantially could be considered for future developments.

So, for example gravity separation could be done and one of the products could be subject to a further process separation test (like flotation and/or leaching for example).

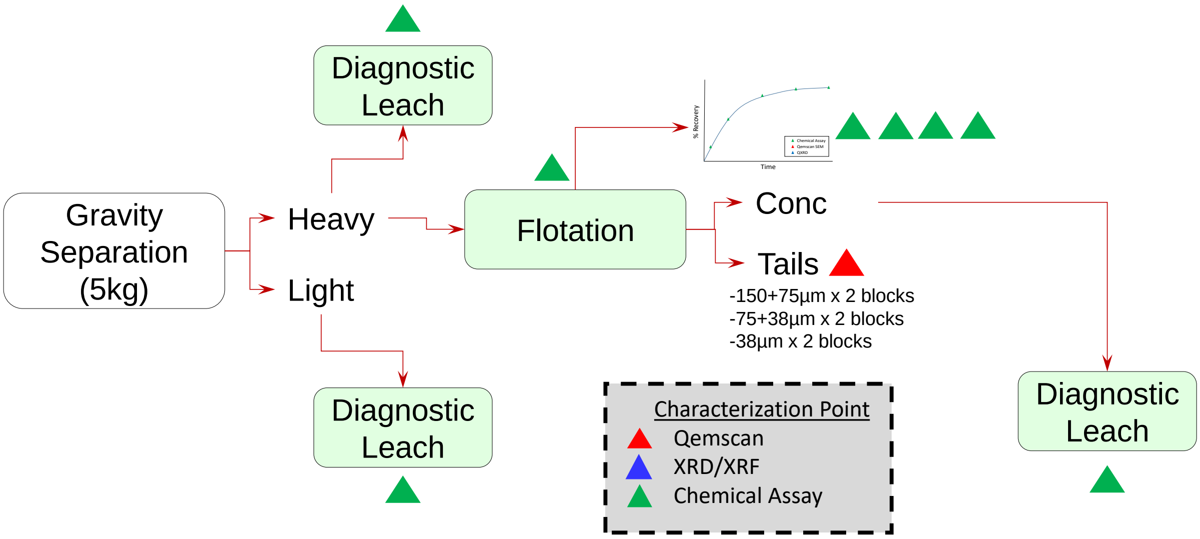


Figure 167. Flotation response after gravity separation
(Image: Simon Michaux)

The products of each of these process separation tests could be subject to a proxy test like a diagnostic leach (done on small volume samples).

7.1.8 Step 8 – Data Analysis of the 2nd pass process separation for each Orientation Sample

As in Step 6, a detailed examination at the performance of each separation method will be done by conducting a mineral reconciliation by comparing the characterized products against the characterized feed. What minerals are separated for each method and where do these minerals report to? Examples of this can be seen in Sections 7.3, 7.4, and 7.5.

A method of how to compare and rank the recoveries from flotation and leaching is shown in Section 7.2.

In the examination of what worked and why, metrics of how success is to be judged need to be established. Include several metrics of success

- Ultimate recovery
- Time taken to reach ultimate recovery
- Recovery compared to grade to show the actual metal produced
- Allow for economic metrics to be calculated from these results

Good recovery of the target mineral is the ultimate goal, but there are tradeoffs to be considered. For example, is the recovery of one mineral better than others (for example Au vs. Cu) where one is more valuable than the other? Consider how process combinations have worked together.

Is there a connection between recoveries and mineral content? The flotation recoveries of the base line response (1st pass) could be compared directly to the flotation recoveries from the same Orientation Sample that has been subject to gravity separation beforehand (2nd pass). Is there an improvement? If so, what difference in mineral quantities in the feed to flotation correlate with this improvement? This may be done by comparing all Orientation Samples together in the same fashion to give a reasonable number of data points.

7.1.9 Step 9 – Comparison process paths for each Orientation Sample

Compare the process paths developed in Steps 5 to 9 for each Orientation Sample. For each ore type, 8 to 10 different process paths can now be considered. The feed and products of each step within each of these process paths has been appropriately characterized, which means a mineral mass balance reconciliation can be conducted across each process path, and then process paths can be compared directly.

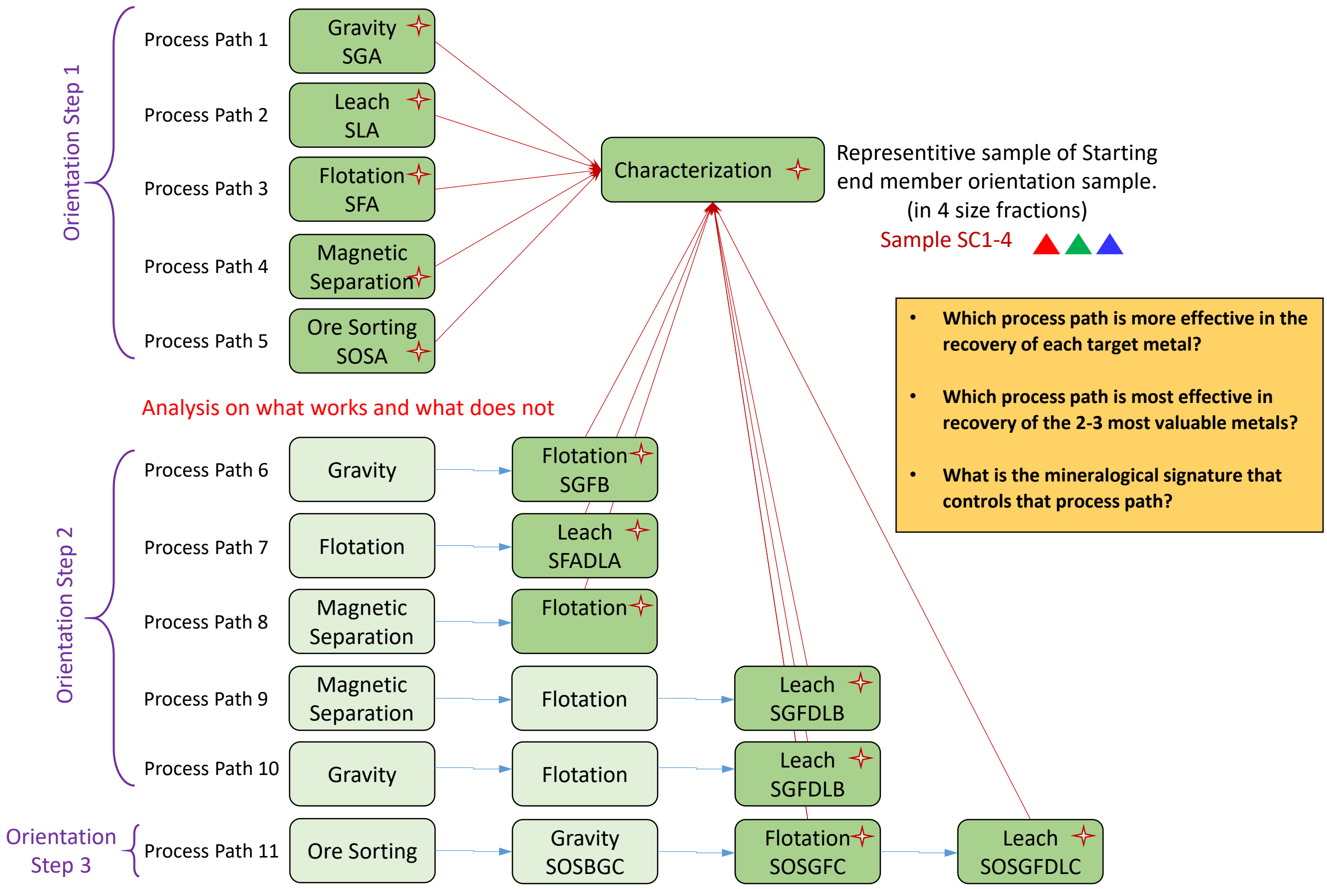


Figure 168 Comparison of different process paths for each Orientation Study sample (Image: Simon Michaux)

A method of how to compare and rank the recoveries from flotation and leaching is shown in Section 7.2. Questions to answer in this section could be (allow for economic metrics to be calculated from these results):

- What is the most effective process path for the recovery each target metal? (Au, PGE, Cu, Ni, Co, etc.)
  - Consider the tradeoff between grade or ore vs. the efficiency of recovery of that ore in a monometallic circuit
  - Rank the process paths in order of preference (to an agreed upon metric)
- What is the most effective process path for the recovery of several mineral targets in a polymetallic context?
  - Consider the tradeoff between grade or ore vs. the efficiency of recovery of that ore in a ploymetallic circuit
  - What are the tradeoff decisions (say PGE grade + recovery vs. Cu grade + recovery)?
  - Rank the process paths in order of preference (to an agreed upon metric)

- What is the mineral signatures (usually gangue minerals) that control each process path?
    - What separation steps are most useful in managing these minerals?

- What is the mineral signatures in context of what reports where for potential penalty elements in each process path?

7.1.10 Step 10 – Selection of the most effective process path for each Orientation Sample

For each Orientation Sample, select one process path in context of the original geometallurgical experimental aims and the requirements set by the client (owner of the deposit).

The most effective outcome will be a combination of the separation methods, done in a specific order. This order would be defined by the ore, where if the orientation samples have been selected appropriately, a range of recoveries and efficiencies would be observed.

Make the choice in a manner that is data supported and defendable in an audit.

7.1.11 Step 11 – Comparison of Process Paths Between Orientation Samples

Take the experimental results that have been analyzed in Step 9 for all Orientation Samples to compare them together. If the work has been done consistently, each process path can be directly compared at each point in the flow sheet between all Orientation Samples, where like is directly compared to like.

- Re-examine how the metrics of will define the best outcome in context of the original experimental aims of the geometallurgy campaign.

- The characterized products can be compared in context of effective separation, both in starting feed samples and in final products. Consider how separation process steps could work together in combination.

- For each end member ore type, examine the recoveries vs. grade for all the economic metals present and make a judgment call on how the valuable target minerals could be ranked in economic importance.
    - Does the high grade ore samples recover more effectively than the lower grade ores in some process paths compared to others?

    - For example Au may be worth more than Cu per unit mass, but if the Au is very low grade and very fine in grain size (e.g. Au in ore 0.02g/t at 5µm grain size) compared to Cu (e.g. Cu in ore 1% at 100µm), the cost of producing Cu will be much lower and may be more economic.

- Once the valuable minerals have been ranked, make a judgment call on what could be the overall most economically effective process path. Rank the process paths in this context.

- Gangue minerals and penalty elements can be examined in context of where they report to in a process path

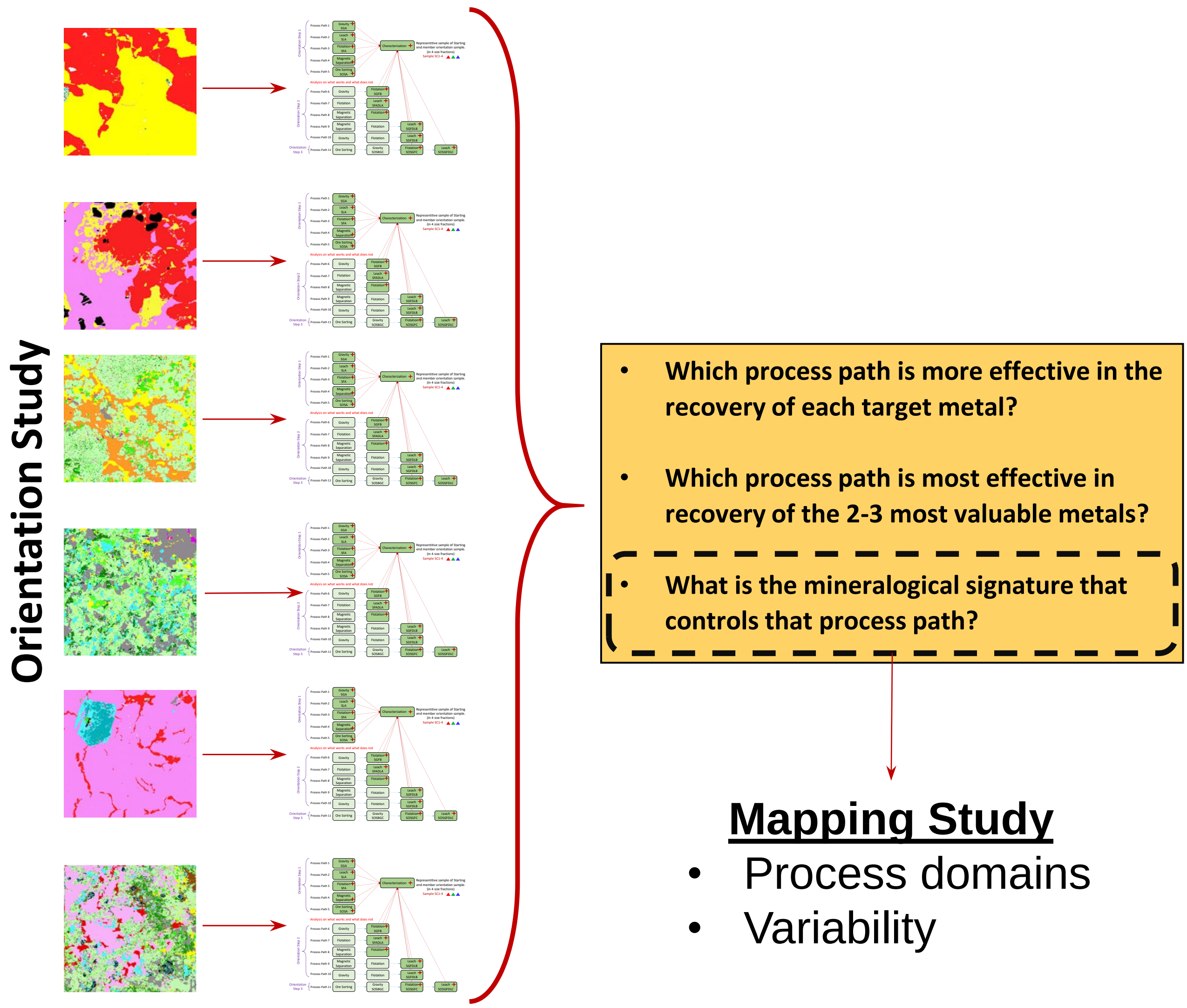


Figure 169. Comparison of different process paths for all Orientation Study samples (Image. Simon Michaux)

7.1.12 Step 12 – Trial of best Combination Process Separation for the Deposit

Based on what has been learned in Step 11, consider what might be the best overall combination of separation processes for the family of ore types in this deposit. If this process path has not been already tested, assemble the ideal process path and test all Orientation Samples. Once the experimental results have been completed, add the data to what has already been done, and repeat Step 11.

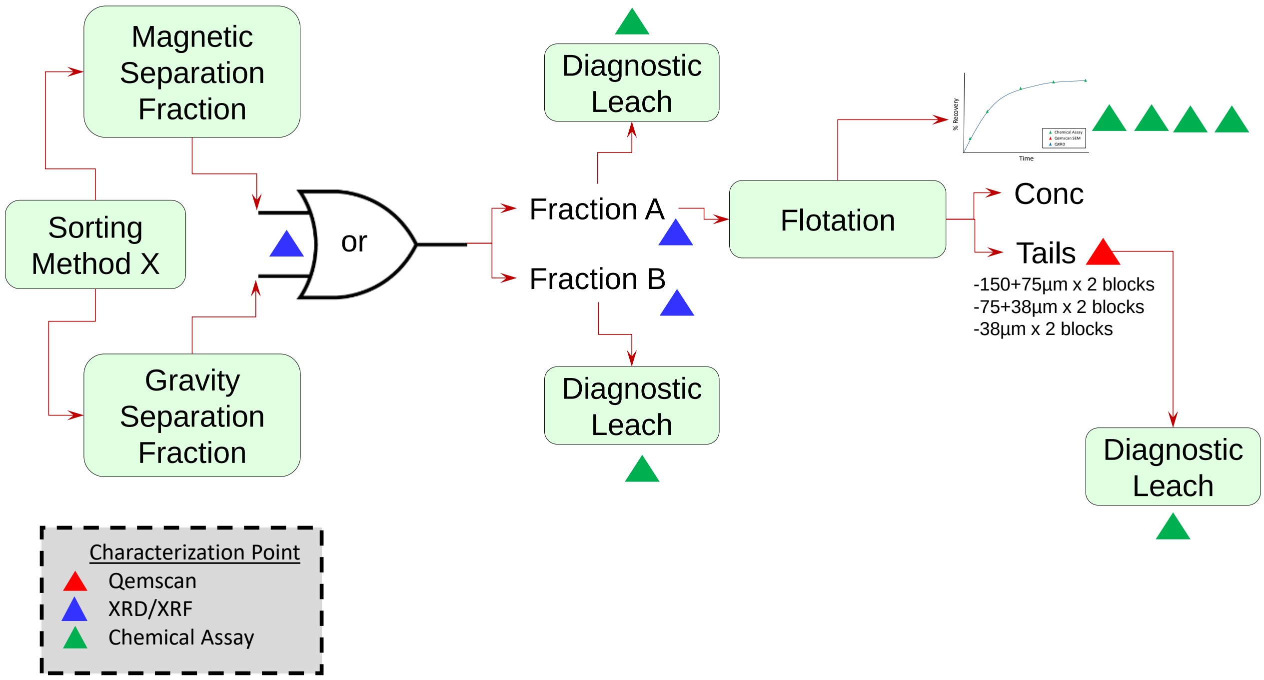


Figure 170. Development of the ideal process path based on previous testwork done and analyzed in Orientation Study (Image: Simon Michaux)

7.1.13 Step 13 – Selection of the most effective process path for the Deposit

Based on what was learned in Step 11, select for the whole deposit, one preferred process path in context of the original geometallurgical experimental aims and the requirements set by the client (owner of the deposit).

The most effective outcome will be a combination of the separation methods, done in a specific order. This order would be defined by the ore, where if the orientation samples have been selected appropriately, a range of recoveries and efficiencies would be observed.

- What is the most effective process path for the recovery of several mineral targets in a polymetallic context?
- What tradeoff decisions were made to make this the best choice?
- What is the mineral signatures (usually gangue minerals) that control each process path?
- What is the mineral signatures in context of what reports where for potential penalty elements in each process path?

Make the choice in a manner that is data supported and defendable in an audit.

7.1.14 Step 14 – Selection of the mineralogical mapping signatures of the most effective process path

Based on the choice made in Step 13 and what was learned in Step 11, assemble the recommendations of what mineralogy controls the chosen process path for each of its separation steps. Usually gangue mineralogy controls process behavior. Also, valuable minerals could be associated with other minerals.

These minerals will form the focus of analysis for the Geometallurgical Mapping Study (Section 12). Each mineral (estimated with chemical assay's if possible) would be examined in context of domains and variability along continuous sections of drill core.

This would form the list of minerals that the Mapping Study would examine.

7.2 Method to Mathematically rank and compare process recoveries

The purpose of the geometallurgical Orientation Study is the assessment of each of the end member rock textures, studied in context of multiple process paths, where the final absolute recoveries can be examined and ranked. In this section, a method of ranking recoveries is presented as an option to consider.

Consider the leaching recovery over time from four column leach tests shown in Figure 147. In terms of ultimate recovery, clearly Ore Type D is the most preferable and the four ore responses can be ranked as follows:

- Ore Type D
- Ore Type A
- Ore Type B
- Ore Type C

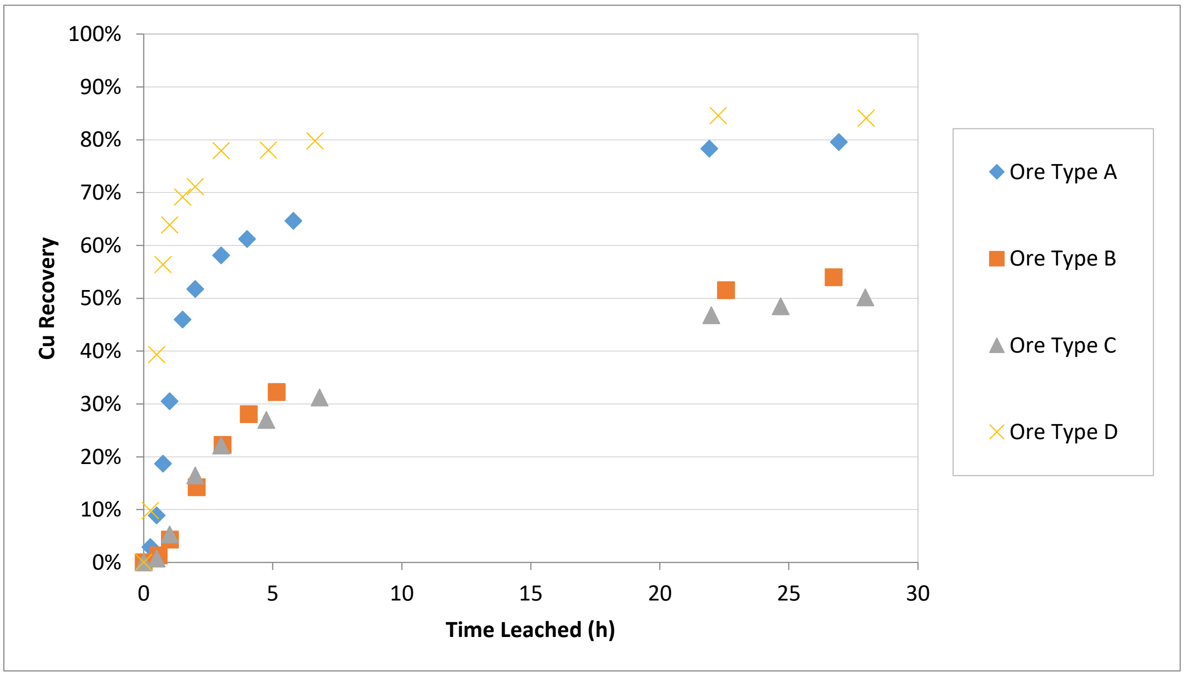


Figure 171. Leaching recovery of four different ore types in the same deposit
(Image: Simon Michaux)

How could this be done mathematically consistently in a form that could be graphically plotted against other parameters?

A model is fitted to this data using the same basic form as the A*b curve shown in Section 4.2 to rank and model comminution. This mathematical tool can be used in a similar fashion for leaching recovery. The equation below was fitted to the data in Figure 141, with the results shown in Table 9 and Figure 172.

$$Q = R[1 - e^{(-s.t)}] \qquad \text{Equation 18}$$

Where:

Q	=	Cu Output %
t	=	Time leached

Table 11. Fitted model parameters

Sample	R	s	R*s
Ore Type A	76.9%	0.45	34.6
Ore Type B	54.3%	0.16	8.7
Ore Type C	49.2%	0.17	8.2
Ore Type D	81.8%	1.21	98.9

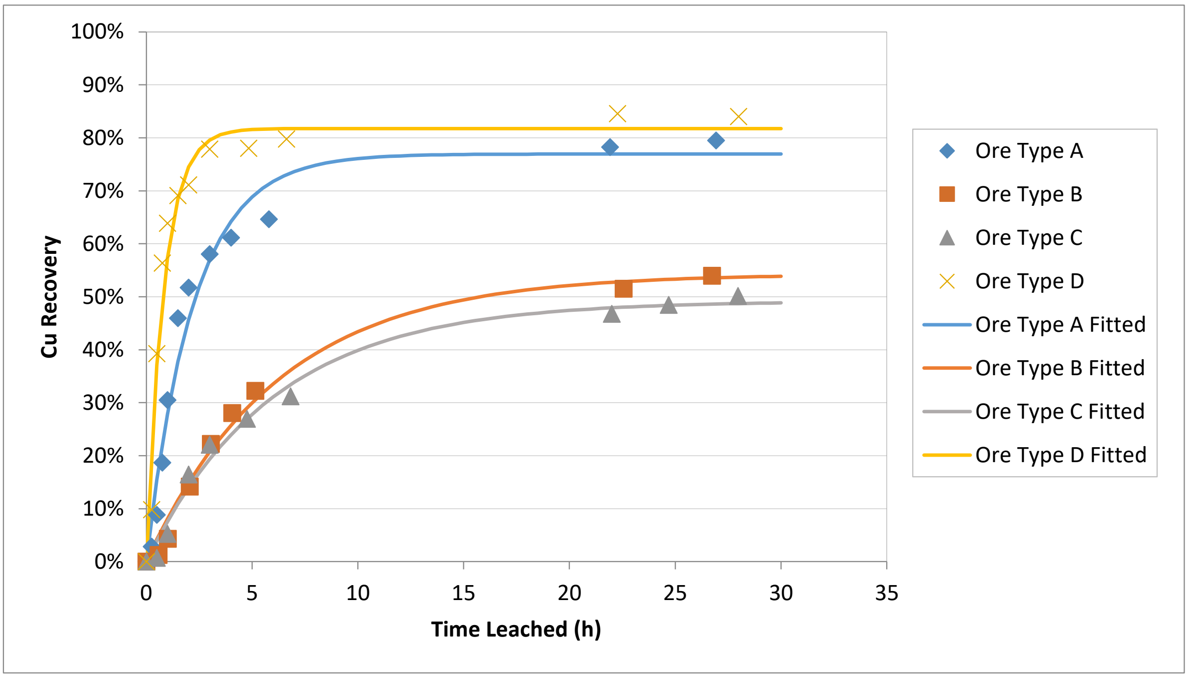


Figure 172. Fitted models to leaching recovery curves
(Image: Simon Michaux)

According to Table 11 the ultimate recovery could be ranked by using the 'R' parameter. The time taken to reach the ultimate recovery could be ranked by using the 's' parameter. R*s could be used as a ranking parameter accounting for both ultimate recovery and the time taken to reach that recovery (in a similar way to how the A*b parameter is used), resulting in:

- Ore Type D R*s = 98.9
- Ore Type A R*s = 34.6
- Ore Type B R*s = 8.7
- Ore Type C R*s = 8.2

The R*s recovery ranking parameter is dimensionless and can be used to mathematically correlate with other attributes or parameters. Figure 149 is an example where the data in Figure 148, and Table 9 is plotted against mica clay content for each sample.

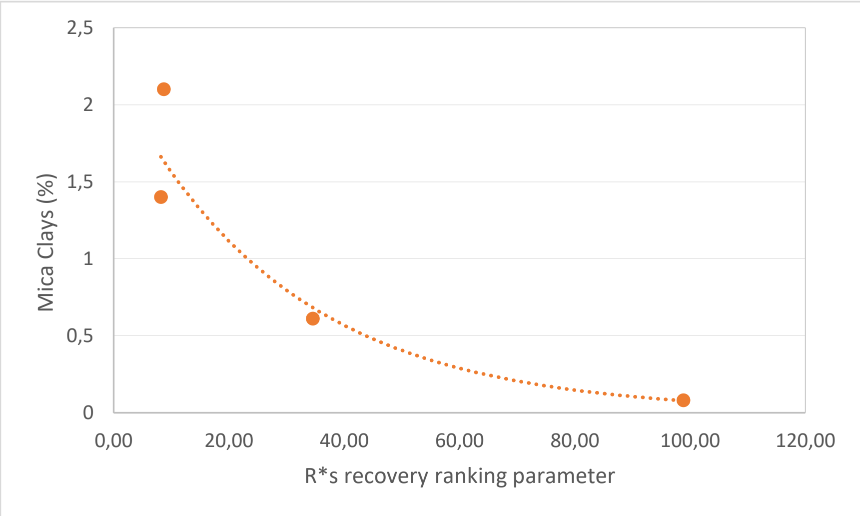


Figure 173. An example how R*s ranking parameter could
(Image: Simon Michaux)

The presence of clays or sheet silicates often negatively impact process performance and recovery. Figure 173 shows how the method of quantifying each sample by mathematically comparing in a consistent form, using the form described in Table 11.

7.3 Example A1 – What mineralogy controls grinding?

Figure 144 results show an example of assessing the variability of grinding as a function of end member rock type. Identical procedures has been run on all of these samples. A feed sample of 700cc in volume was prepared by crushing the sample where 99% of the sample mass passes the 3.35mm sieve screen (this is the same as the sample feed for the Bond Ball Index test). Each feed sample first had its size distribution measured on the √2 sieve series down to a size of 38µm. Then each sample was placed in a standard bond ball mill with a standard ball charge for the BMWi test. The mill was run (sample without water) for exactly 358 revolutions at a speed of 71.5rpm (5 minutes). The dry ground sample was collected, its mass measured and then wet screened on a 38µm screen. The resulting +38µm sample was dried overnight in an oven at

105^{0}C and its size distribution determined on the √2 sieve series down to a size of 38µm (1.18mm, 850µm, 600µm, 425µm, 300µm, 212µm, 150µm, 106µm, 75µm, 53µm, +38µm, -38µm and cyclosize).

Figure 174 shows the +38µm size distribution in the form of percent mass in each sieve (the calculation step before calculating the conventional cumulative percent passing size distribution). This method of data presentation is useful in that it returns peaks and troughs that are signatures of the grinding process and the mineralogy of the sample. The peak at 100µm is often seen in grinding tests and is thought to be related to the geometry of the spaces between the grinding balls when the empty mill is at rest. The peak at 2.36mm is also seen in grinding tests and is related to the size fraction that the mill has the most difficulty in grinding. The volume recirculating load in a standard bond ball index test is influenced by the volume in this size fraction. The harder the rock being ground, the more particles that are left at this coarse size fraction. The magnitude of these peaks is related to the number of particles left on the sieve at that size fraction (or the volume of the original sample at that particle size).

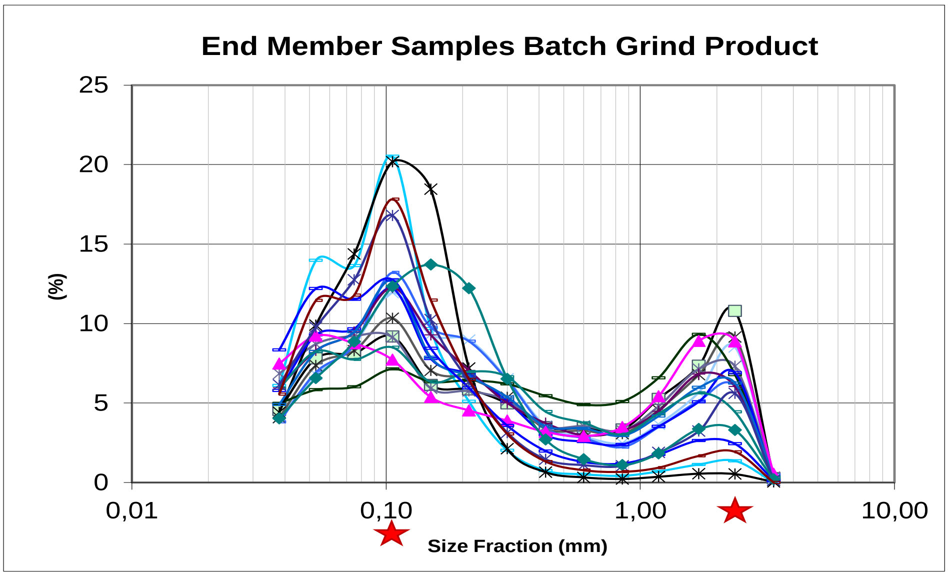


Figure 174. Grinding behavior of several tests from the same deposit
(Image: Simon Michaux)

It is noted that there is quite a range across both peaks. A high peak at 100µm corresponds to a low peak at 2.36mm; a sample that is soft to grind. Alternatively, a low peak at 100µm corresponds to a high peak at 2.36mm; a sample that is hard to grind.

The next step would be to examine and assess the mineralogy of the particles in both of those size fractions (-150+106µm and -3.35+2.36mm) by taking a sample of each and testing them. Gangue mineralogy usually controls grinding behavior (Figure 175). A useful approach could be to measure the QXRD of each peak size fraction for each of the end member rock textures, then compare these results to the mineral mapping done of the intact sample. The objective is to understand what mineralogy the rock breaks into and how what is observed in the peak size fractions might relate to the original rock.

An examination of how many particles (peak magnitude or % of original sample at that size fraction) in context of which minerals are dominant would provide an understanding of what minerals control grinding in the deposit. In this example, the dominant minerals found in the -3.35+2.36mm size fraction were potassium feldspar. The dominate minerals found at the -150+106µm size fraction were magnetite, carbonate and pyrite. If budget allows the size fractions could be characterized with automated mineralogy methods (which is often useful in studying other process behaviours like flotation or leaching).

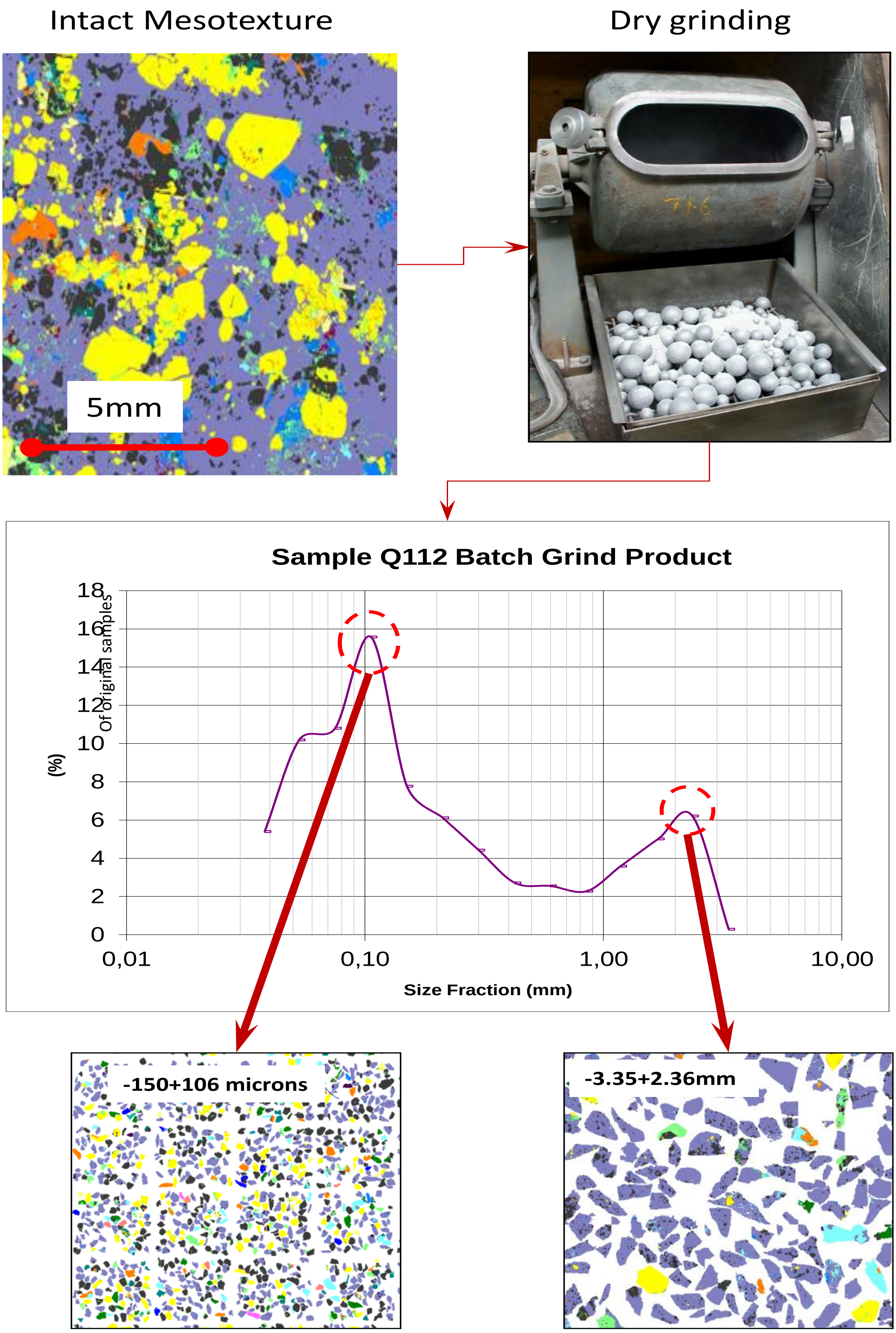


Figure 175. Study of mineralogy that dominates grinding
(Image: Simon Michaux)

7.4 Example A2 – What mineralogy controls comminution footprint

The comminution footprint refers to the deposit specific relationship between impact breakage (measured with the Axb parameter from the DWT) and grinding behaviour (measured with the Bond Ball Work index BMWi). An RBT test (to estimate the Axb Impact Breakage Parameter) and a Bond Ball Work Index (BMWi at a closing size of 150µm) was done for each end member sample. Care was taken to ensure these tests were representative samples from all parts of each end member sample. The curve shown in Figure 176 is referred to as the comminution footprint.

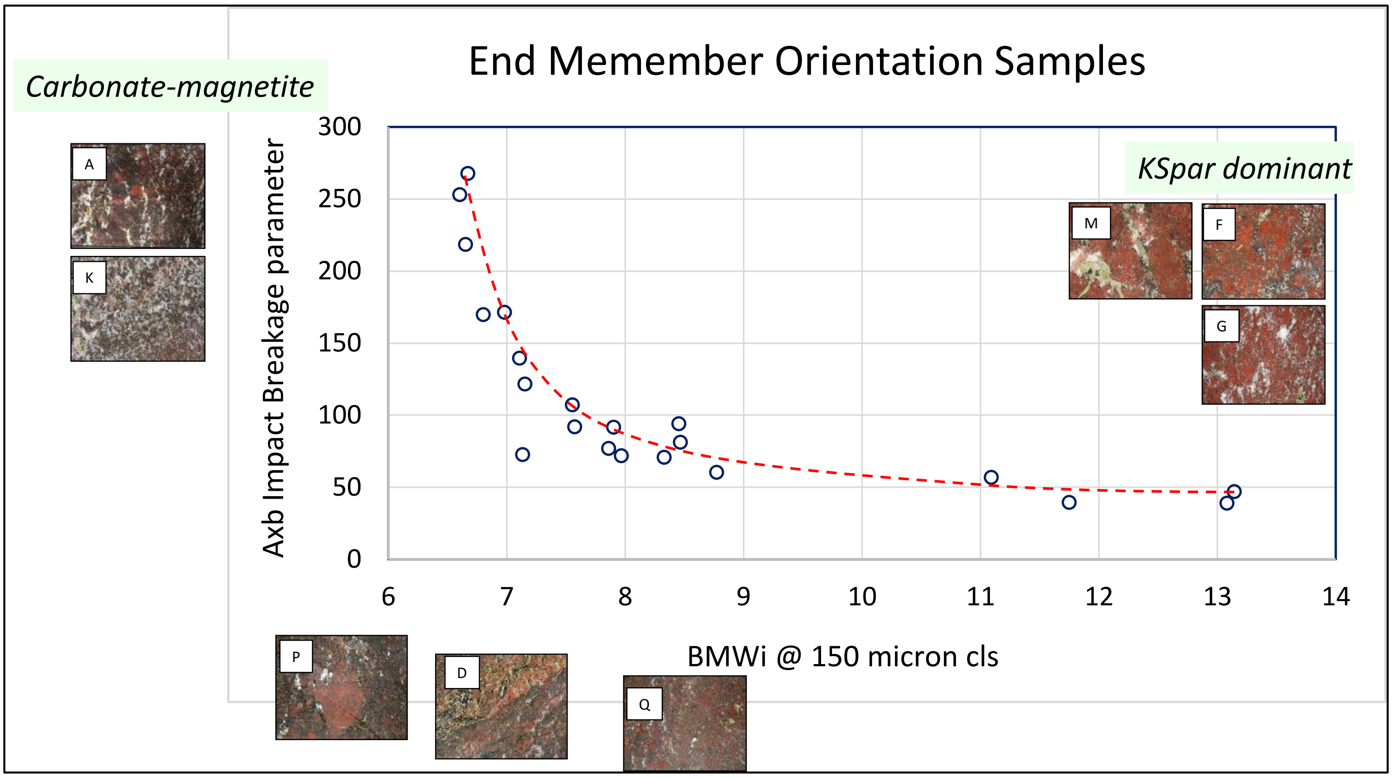


Figure 176. An example of a mineral hypothesis of the impact breakage vs. grinding behaviour curve (Image: Steve Walters, Copyright: AMIRA)

At the top left end of the curve (where the ore is softest), sample rock texture was dominated with carbonate and magnetite mineralogy. At the bottom right end of the curve, sample rock texture was potassium feldspar and quartz dominant.

Examples A1 and A2 were two different parts of the orientation study for the same deposit. The conclusions were that the minerals that controlled comminution in this deposit magnetite, carbonate, quartz and pyrite. This would form a hypothesis to guide the next step in the geometallurgical campaign where a smaller number of tests would be done on a much larger number of input samples. The outcomes of which would be propagated into the deposit for spatial domaining.

7.5 Example A3 – Process Circuit Performance Estimation Economic Modelling of each End Member Sample

The measured Axb and BMWi characterization parameters from Example A2 were then used to estimate the performance of a proposed comminution circuit (Figure 153). The objective would be to rank the ranked throughput and energy power draw consumption of each of the end members (if the feed was 100% one end member only). This is a crude estimate only to assist in the domaining of the deposit in context of process behavior. This is not a substitute for larger scale ore characterization and engineering mill design.

- For each end member sample, the Axb impact breakage parameter was measured with an SMC test and the BMWi was measured with a Bond Ball test. These results were then used as input into the procedure to estimate throughput and estimation of performance of a conceptual comminution process plant.

- For each end member sample, the comminution throughput, measured recovery (batch flotation study – not shown here) and the geological characteristics of the original rock texture were all used to estimate the economic signatures.

The mill throughput was estimated using a set of engineering equations developed from 30 operating AG/SAG mills (closed and open circuit), covering a range of diameters, lengths, speeds, ball loads, grate designs, ore characteristics and feed size distributions. These models are similar to the models developed by Morrell (2004) for scale-up, design and optimization. The equations have the form:

$$SP(kWh/t) = SAG(kW)/TPH = f'(F80,T80,SG,DWAB,BMWi,BL,CS,D,D/L) \qquad \text{Equation 19}$$

where:

- SP = AG/SAG specific power required for given set of operating conditions
- F80 = SAG mill feed 80 per cent passing size
- T80 = SAG Mill product 80 per cent passing size (or transfer size)
- SG = ore specific gravity
- DWAB = Drop Weight specific power required to break feed to transfer size, which is dependent on
- ore A and b parameters, feed size distribution and trommel size.
- BMWi = Bond ball mill grindability index
- BL = ball charge (per cent)
- CS = mill speed (per cent of critical)
- D = mill diameter
- D/L = aspect ratio = mill diameter/mill length (EGL)

The above equations are combined with the Bond ball mill design equation to complete the circuit throughput model, providing a link between the SAG and ball milling stages. That is, the throughput capacity is dependent on both the SAG and ball mill performance as they interact in practice.

$$TPH = \frac{BallMill(Kw)}{10 \times BMWi \times (EF_1 \times EF_2 \times EF_3 \ldots.) \times \left(\frac{1}{\sqrt{P80}} - \frac{1}{\sqrt{T80}}\right)}$$

Equation 20

where:

- TPH = expected feed capacity (tph)
- PBM = installed ball mill power (kW)
- EF1, EF2 … = Bond's efficiency factors
- T80 = SAG circuit transfer size
- P80 = final ball mill circuit product size.

The maximum throughput may be SAG or Ball Mill limited, depending on ore conditions. The sensitivity of the Excel model to A*b was fine-tuned using JKSimMet software (JKTech 1991, and Wiseman 1991) to better capture the effect of recycle load and pebble crushing on transfer size to the ball mill.

This modification provides a correction to the SAG transfer size T80 (from the baseline) when A*b changes at constant BMWi and increasing recycle fraction.

The effect of A*b and feed size F_{80} is included in the Drop Weight Axb (DWAB) term. However, ideally the complete feed size distribution is required to determine DWAB. The simplification presented here assumes the shape of the distribution remains constant when the F_{80} changes as part of a comparative geometallurgical mapping type approach.

The estimated throughput is based on the existing SABC circuit, comprising an open circuit SAG followed by two ball mills in closed-circuit with hydrocyclones (Figure 177).

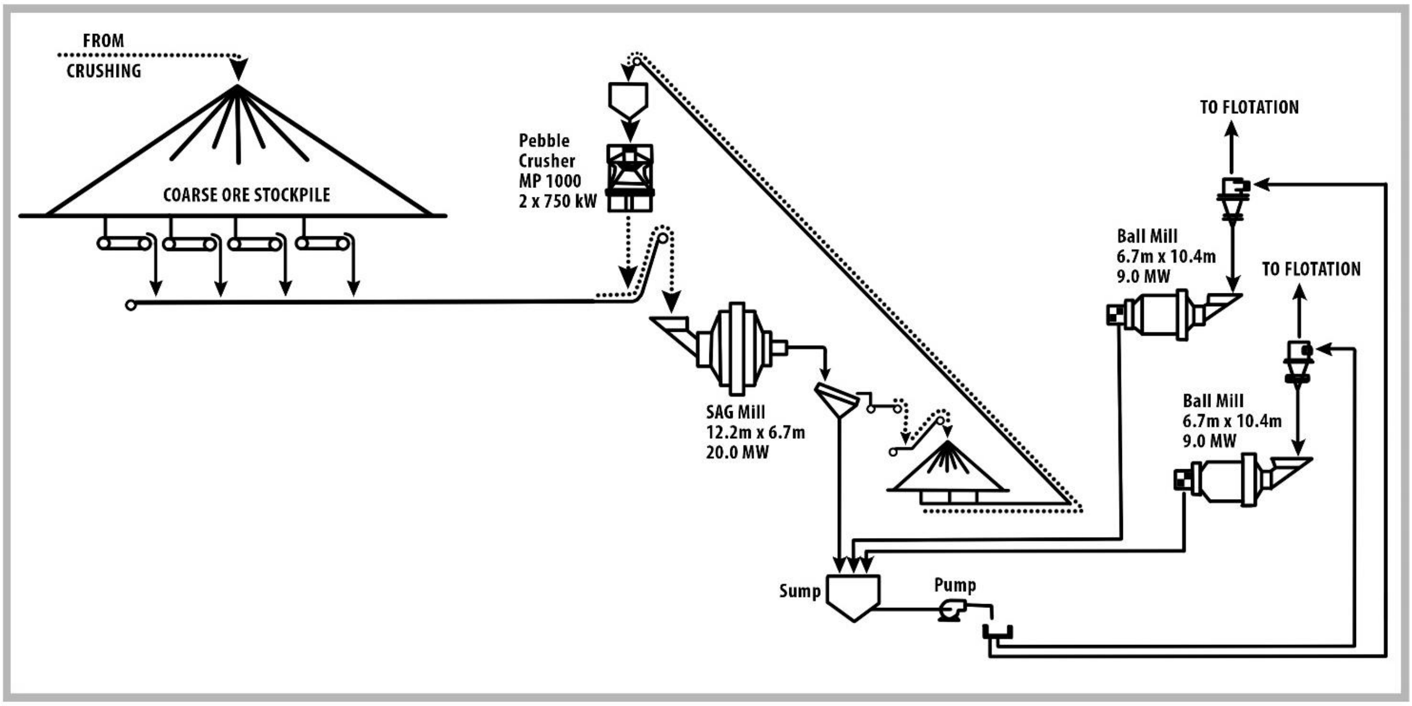


Figure 177. SABC flowsheet used in CI throughput modelling (Kojovic *et al* 2010)
(Image: Tania Michaux)

Figure 177 can be simulated more effectively using JKSimmet (JKTech 1991 and Wiseman 1991). The same approach can be achieved with flotation, hydrometallurgy and pyrometallurgy using the HSC thermochemical simulation software.

This approach was comminution based. A more comprehensive approach would be to include mineralogy, recovery estimation and possibly a penalty element estimate (Figure 178). In doing so, an estimate of throughtput, cost of production and metal recovered as a function of time unit (per hour) could developed for each rock type (or in this case orientation sample).

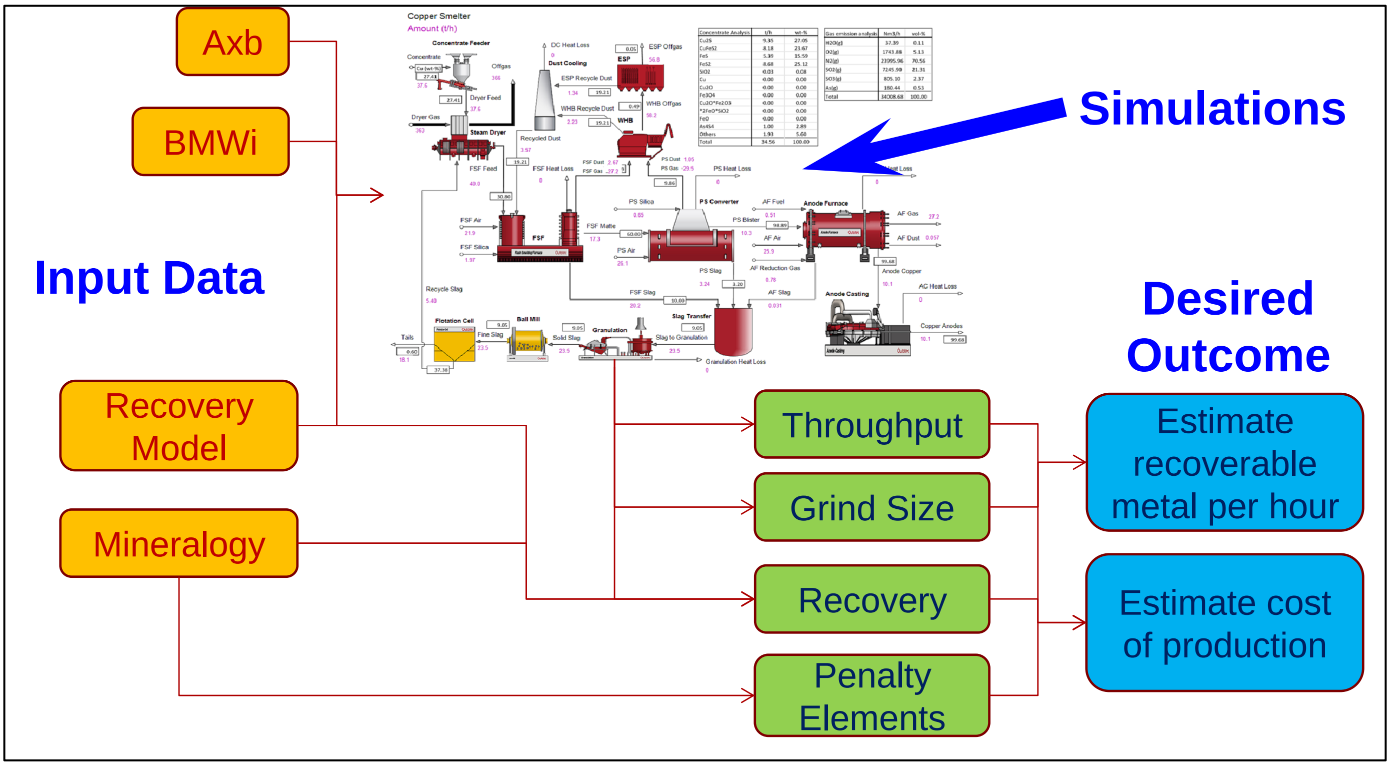


Figure 178. Process simulation for each end member sample
(Image: Simon Michaux)

7.6 Economic modelling and business model development

- Economic modelling and optimization
- Original objectives reassessed, but refitted in context of the business model
- Scope and mandate reassessed based on analytical outcomes
- Metrics of success agreed upon to optimize towards
- Links established in how this could be related to operation protocols
- Links to reconciliation protocols

For each end member (the rock texture extremes of the deposit being studied), an estimate of the metal production throughput and cost of production in a proposed process plant. In addition to this, an estimate of the economic net revenue value was determined. This was based on the formula shown in Figure 155.

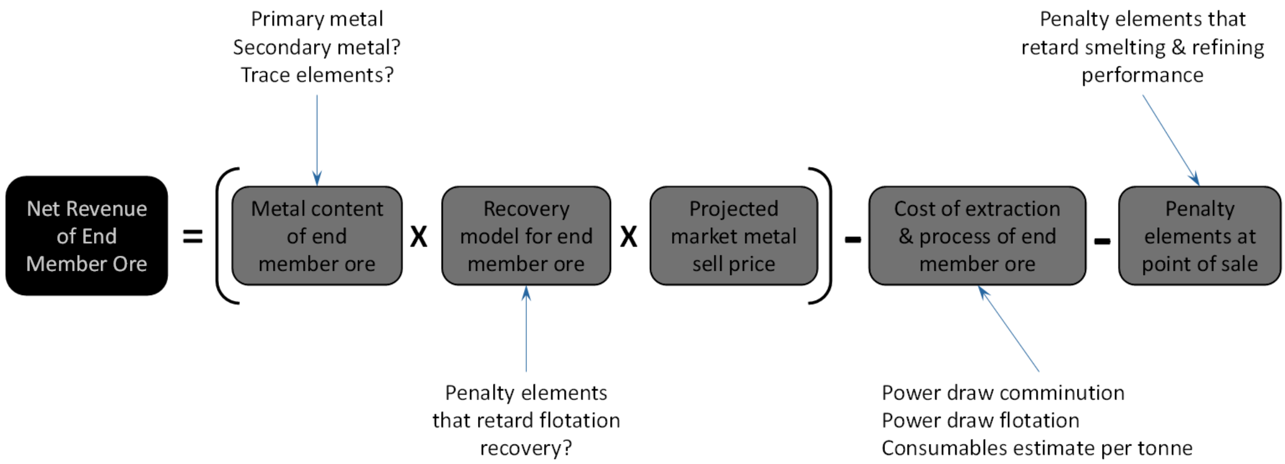


Figure 179. Economic Net Revenue value of a parcel of ore
(Image: Simon Michaux)

This was to provide the possible extremes of revenue generated by the operation. At this stage, the proposed circuit is a crude and simplistic model. This procedure was to be carried out on all of the samples planned in the geometallurgical campaign. If successful, the results would be useful in making ore blending recommendations.

7.7 Conclusions of orientation study

At the conclusion of the orientation study, the following should be understood:

- An understanding of what works and what does not for this deposit
- Useable relationship between bankable tests and their proxies
- A series of hypotheses of what characterizes the target process behaviour, to be used in the Hypothesis Sample set
- Experimental design for what tests are to be determined on the Hypothesis Sample set (the remaining samples of the 1500-2000m drill core)
- Representatively of each test across the sample maintained
- The order of tests to be done, where some of them will destroy the core
- Data QA/QC protocols established
- Data analysis & modelling planned out to milestone conclusion
- Sample size and drill core depth interval for each Hypothesis sample
- Tools for domaining have been tested and experimental protocols developed

8 BUILDING THE GEOMETALLURGY DATA SET

As each test in the Orientation Sample set is completed, the resulting data is to be assessed in context of QA/QC protocols then entered in the geometallurgy database (see Figure 180-184). This step is often misunderstood or not done very well. All analysis going forward will be using this data set in the assumption the data is reliable and in an appropriate form.

The nature of a geometallurgical study is that data is collected in various forms that is conventionally not put together for analysis (metallurgical test work in context of spatially defined meso texture mapping). Also, many of the tests done do not have the same precision and reliability of outcomes as the larger scale bankable tests.

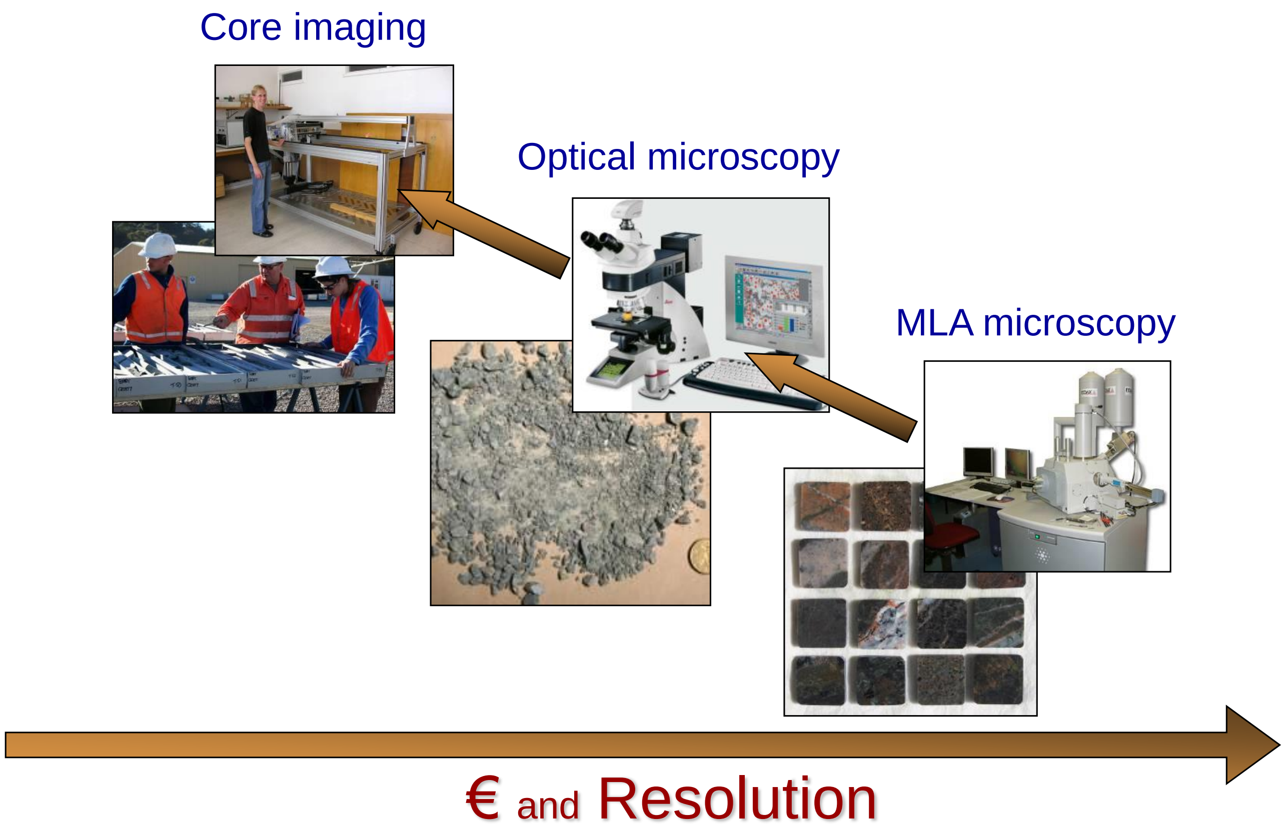


Figure 180. Different kinds of data results from different measurement platforms
(Image: Steve Walters, Copyright: AMIRA)

Much of the data collected in a geometallurgical campaign may come from different measurement platforms that are not traditionally used together. Some thought is required in how to appropriately integrate these concepts in a useful manner.

A very clear understanding of what each data outcome is and what form it takes needs to be developed, across the whole data set. This is important to understand and manage the analysis outcomes. The purpose of this analysis is to provide signatures for process behaviour that can be domained into the deposit.

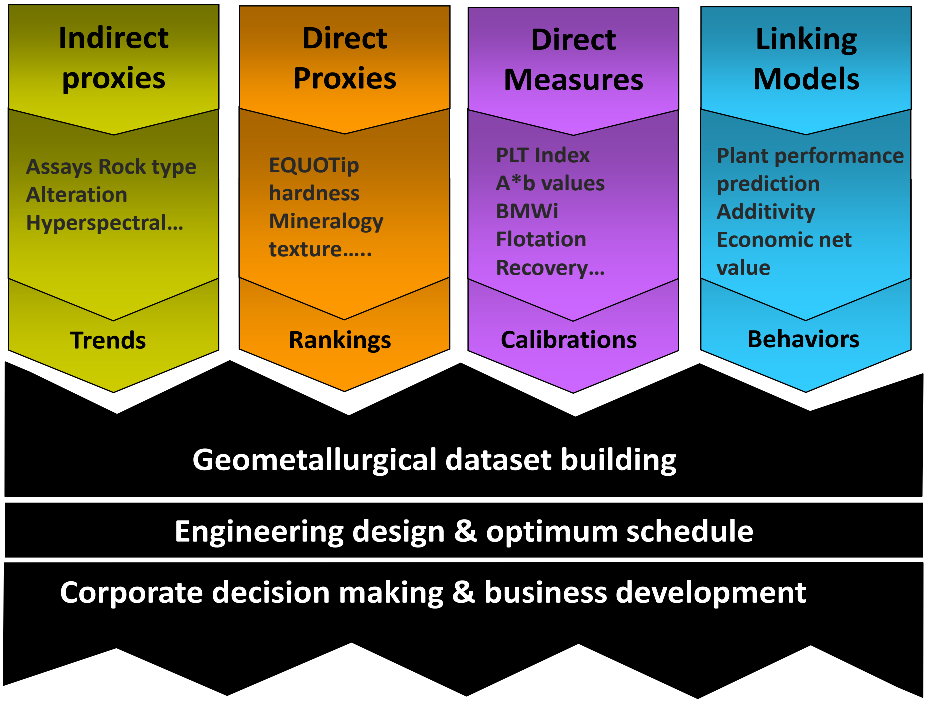


Figure 181. Integration of the different kinds of data
(Image: Simon Michaux)

The structure of the data set needs to be thought through in a fashion where all the relevant information can be collected as each test is done. Figure 182 shows some of the things to consider while attending to this step.

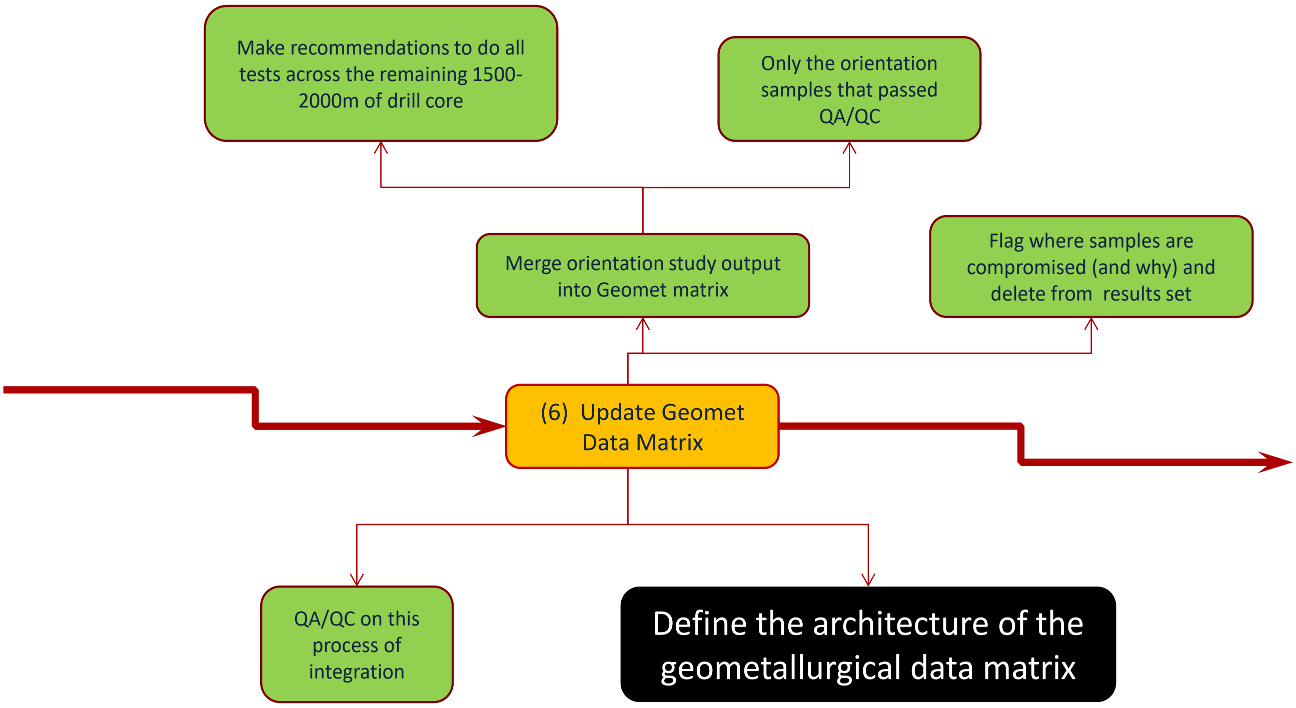


Figure 182. Update the geometallurgical database and develop its final architectural structure
(Image: Simon Michaux)

While establishing the structure of the data matrix with the planned tests, also consider how to link to documentation and procedures. Consider what level of error would be acceptable to allow a test result to be included into the matrix (for example, a maximum mass loss of 1%). The data from the Orientation Study should be included in this matrix.

Work completed previously at site will provide excellent data in terms of spatial domaining. The geometallurgy analytical team is to examine this work and collect the raw data used in these client outcomes. The lithology domains as defined by site geologists have good data backing their analytical decisions. The raw data behind the lithology domains should be incorporated into the geometallurgical data matrix if possible.

The final data set (Figure 183) is an integration of very diverse kinds of information, all linked back to a single sample identification (usually an alphanumeric related to drill hole number and sample depth section). Once the data set has been completed with test data that has been assessed in terms of its suitableness, the analysis steps can begin.

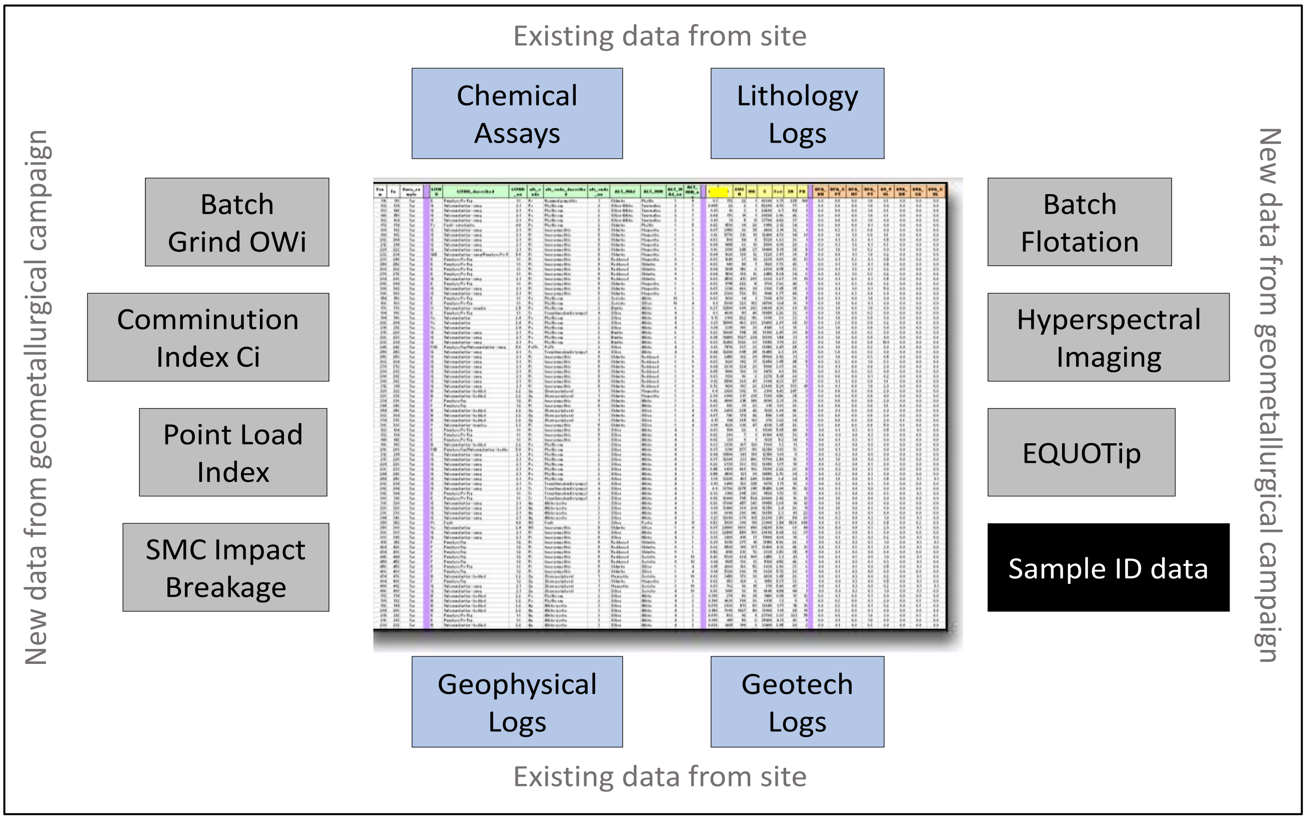


Figure 183. Case Study P: Building the geometallurgical dataset structure
(Image: Simon Michaux)

In the final matrix, each test entry should also be accompanied by each number entry used in its final calculation. This may seem excessive, but it will be easier to track QA/QC issues if this is done as standard practice from the beginning. The number of Hypothesis Set samples usually is between 150 and 350. The easiest way to manage this is to have each sample and everything associate with that sample on a single row. Thus, each sample has its own unique row in the data base with many columns. For example, in Case Study P, each sample studied had (not all tests were done for all samples). Sample identification (Sample ID, drill hole, core type, core depth, original mass, etc.) 6 columns

- Lithology 3 columns
- Chemical Assays 21 columns
- Geotechnical logging 7 columns
- Downhole Well Log Geophysics 25 columns
- EQUOTip measurements 5 columns
- Hyperspectral Imaging 17 columns
- Point Load Index test 8 columns
- Comminution Index (Ci) 16 columns
- XRD 23 columns
- SMC test 8 columns
- Comminution Batch Grind Operating Work Index (OWi) 36 columns
- Batch Flotation tests (including chemical assays) 38 columns

Only some of these columns are to be used in analysis. The rest are there to support data QA/QC. Figure 184 shows a starting point in the relationship between experimental data acquired, the selection of what data is selected for modelling and then the data outcomes, all in the data matrix structure. Also, links of documented procedures on how these data analyses was done also needs to be present and linked to the experimental design. The ultimate form of a geometallurgical matrix is a cross-correlation table populated with selected attributes integrated into defined sample intervals.

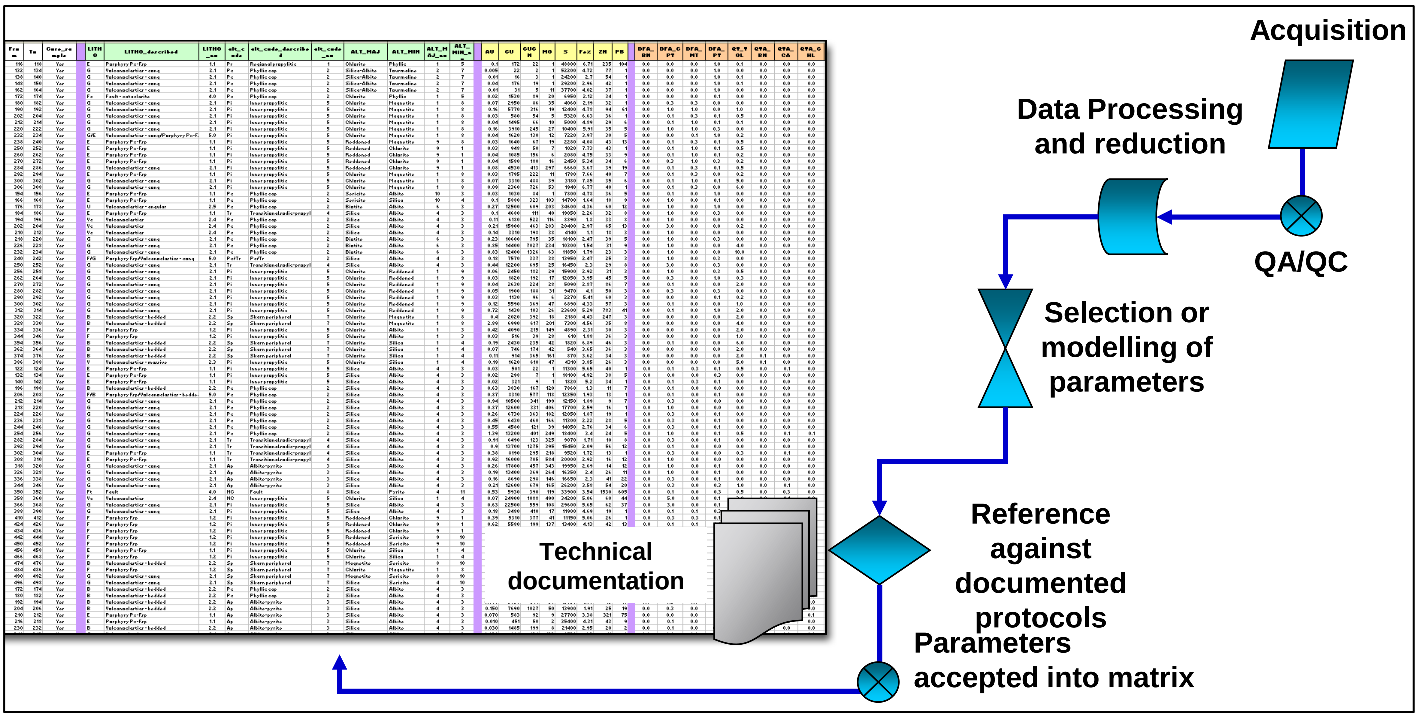


Figure 184. Data acquisition, QA/QC, documentation and building the data matrix (Image: Simon Michaux)

9 MULTIVARIATE ANALYSIS OF ORE DEPOSIT DATA

Once the experimental data has been collected and assessed for its QA/QC reliability, further analysis can be done. There are several stages.

- Statistical analysis of all parameters
- Cross correlation of all parameters
- Statistical data domain definition with Principle Component Analysis (PCA)
- Mineralogical signatures and patterns of each ore type understood in context of target process behaviour
- Ore type definition
- Propagation of engineering simulation (HSC) outcomes from Orientation Study (if possible)
- Propagation of error analysis across all stages of study
- Assessment of analytical outcomes in context of original geometallurgical objectives

Some of these procedures are substantial and complex works in their own right. As the data is processed, the outcomes need to be added to the geometallurgy data matrix. This is a step that also needs to be thought through and planned. What gets used and what for needs to be documented and a protocol developed before starting.

Each of the measured parameters and trends should be subject to statistical analysis. Each parameter should be subject to the following procedures (across the whole study, across domains, across ore types after they are defined, across individual tests with multiple readings):

- Mean
- Median
- Mode
- Range
- Maximum
- Minimum
- Number of data points

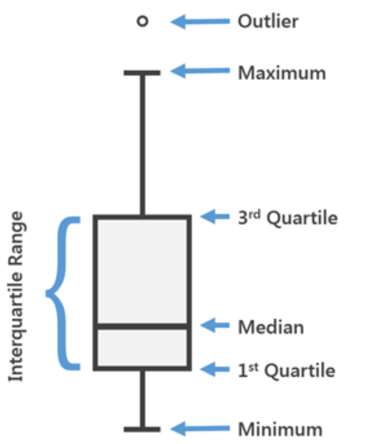


Figure 185. Box and Whisker Plot (Image: Simon Michaux)

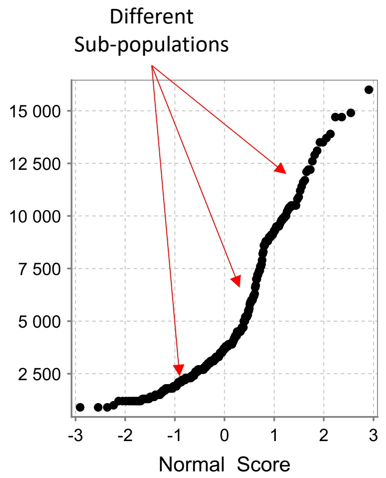


Figure 186. Probability Plot (Image: Simon Michaux)

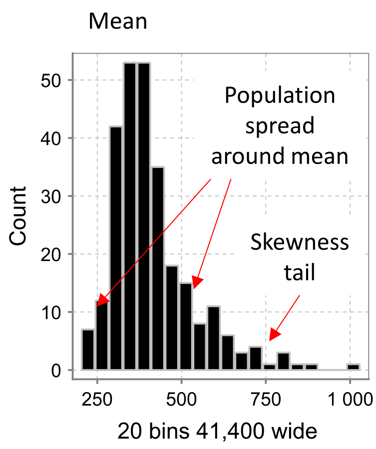


Figure 187. Histogram (Image: Simon Michaux)

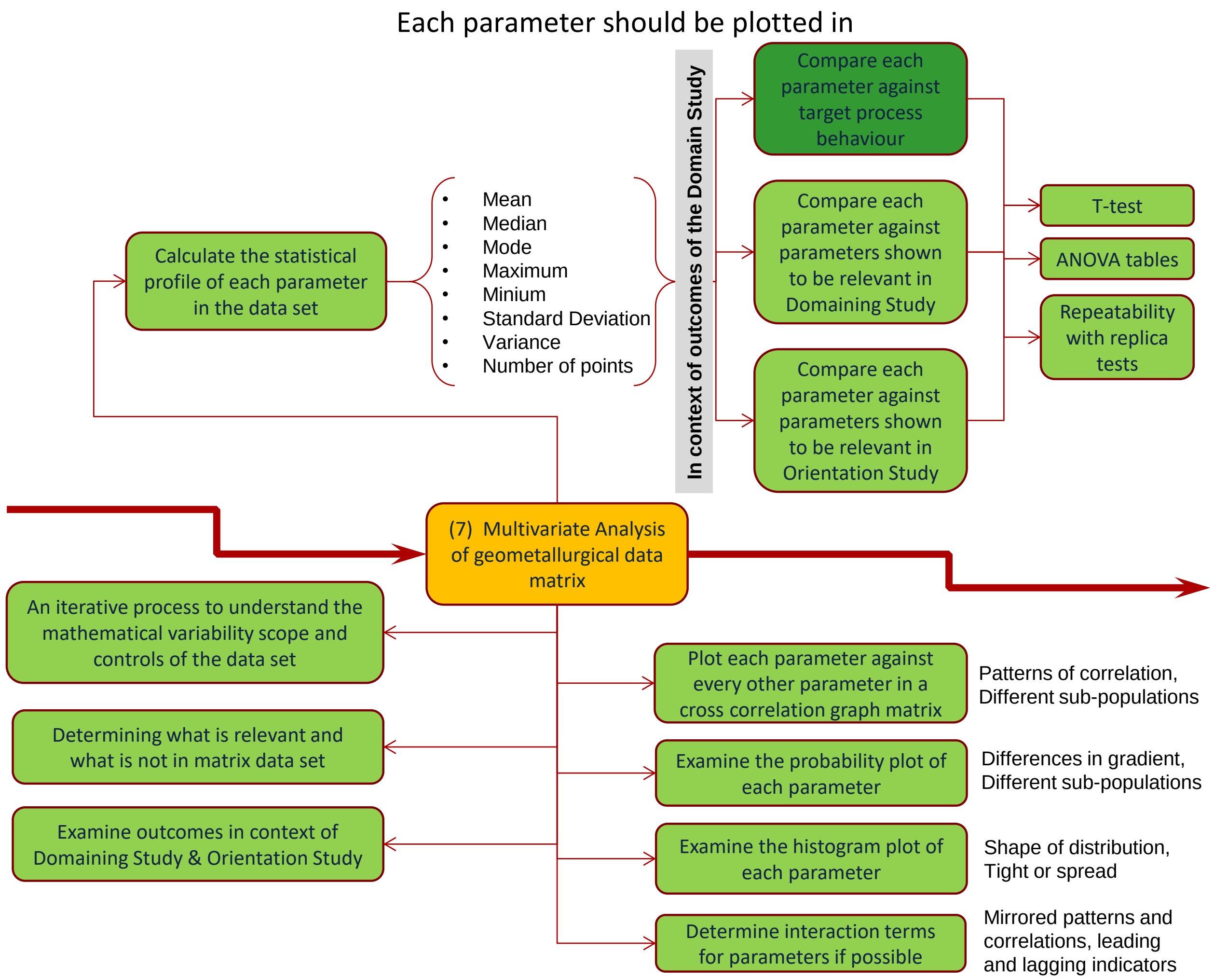


Figure 188. Multivariate Analysis of Geometallurgical Matrix data set
(Image: Simon Michaux)

Each parameter should be plotted in cumulative plots, box and whisker plots and histograms. This provides a strong foundation to base conclusions of the study. Figures 189 to 192 show data from Case Study P. In this case study, impact breakage (Axb) was estimated with SMC tests and grinding was measured with a Bond Ball work index (BMWi). In addition to this, the Comminution Index was measured on a much wider sample range. In Figure 189, all parameters were compared to all others, (where the two algorithm outputs of the Comminution Index were CRU.INDEX or Ci-CRU (related to crushing) and GRD.INDEX or Ci-GRD (related to grinding)). The percentage mass of the -4.75mm and the -106 µm fractions are data routinely measured as part of the Ci data QA/QC checks and are not used in the calculation of the Comminution Index. These were included in the data analysis as they are also appropriately measured crushing responses of the representative sample.

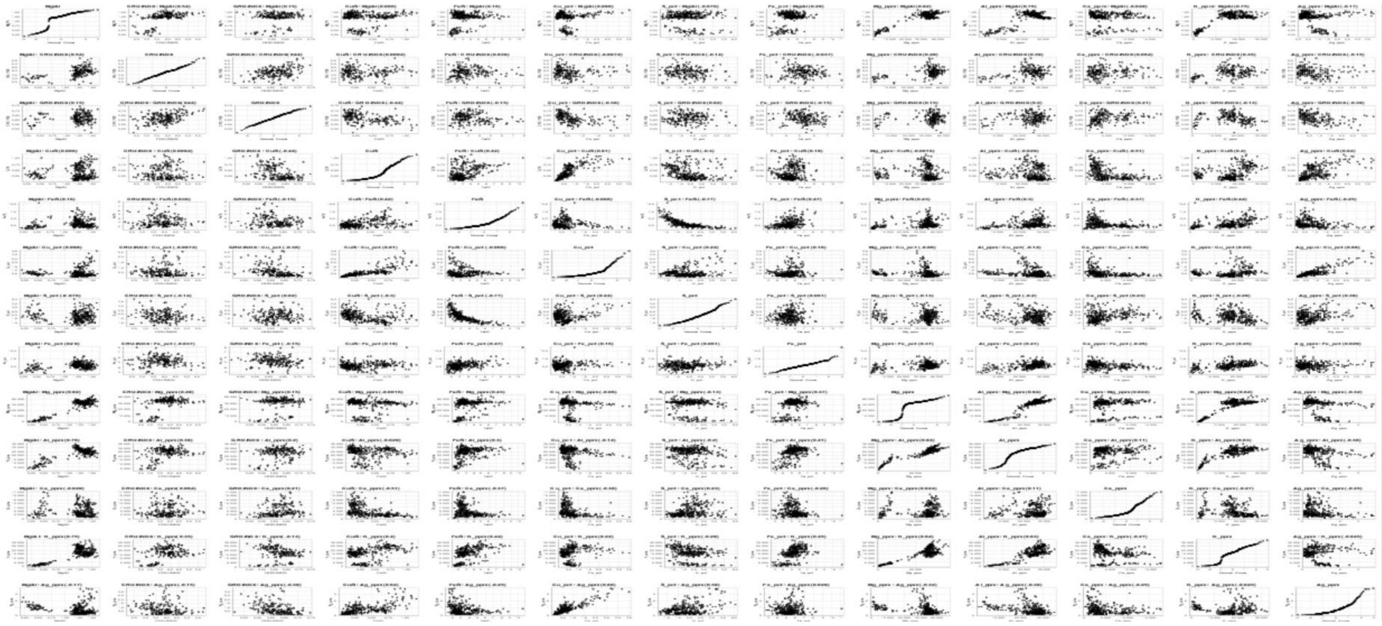


Figure 189. Compare all parameters against all others (Case Study P)
(Image: Simon Michaux)

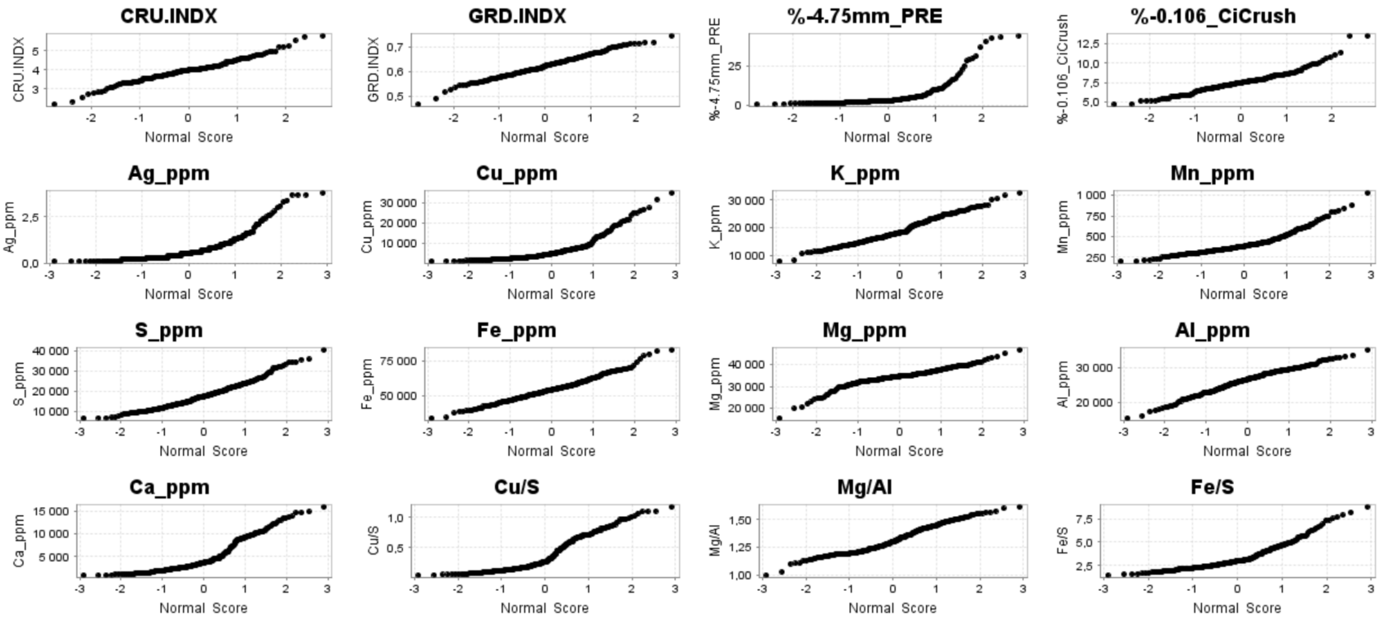


Figure 190. Probability plots of variables studied (Case Study P)
(Image: Simon Michaux)

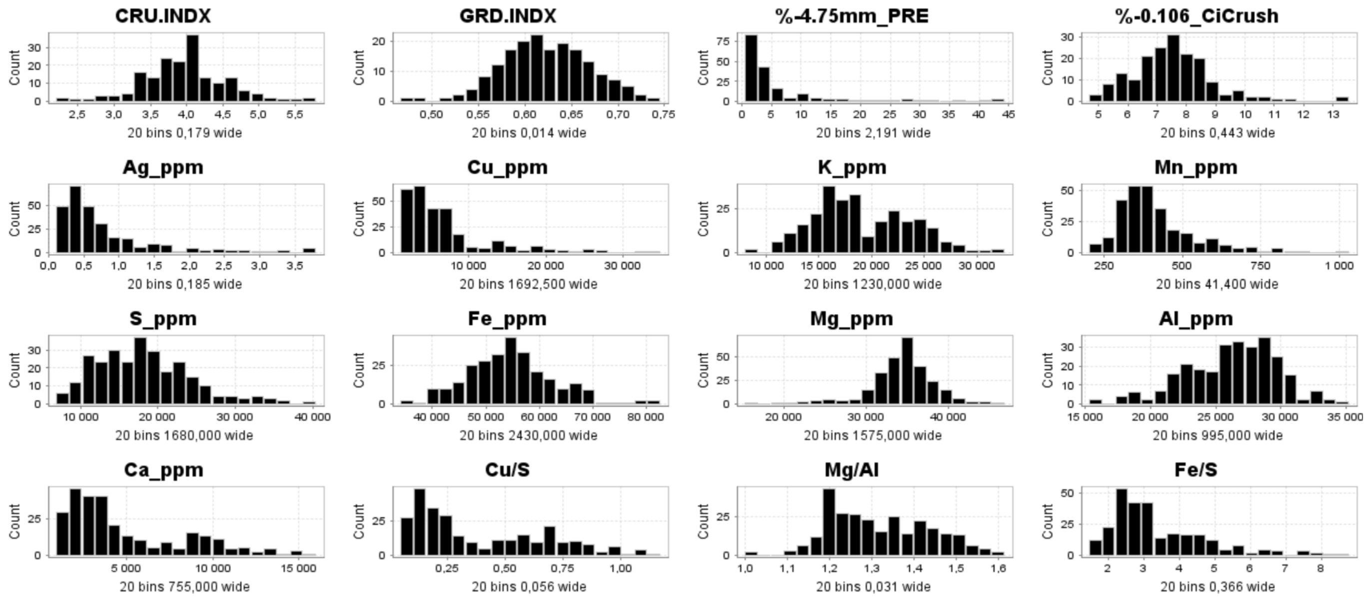


Figure 191. Histogram plots of variables studied (Case Study P)
(Image: Simon Michaux)

The outcomes of the Domain Study and the Orientation Study are to be understood when conducting this multivariate analysis. T-tests should be applied for each parameter between domains and ore types once they are defined. If the experimental design has been developed appropriately, parameters and trends can be placed in a series of ANOVA tables to test statistical significance and interaction terms. All of this is to be added to the geometallurgy data matrix in the same form that the experimental data was structured.

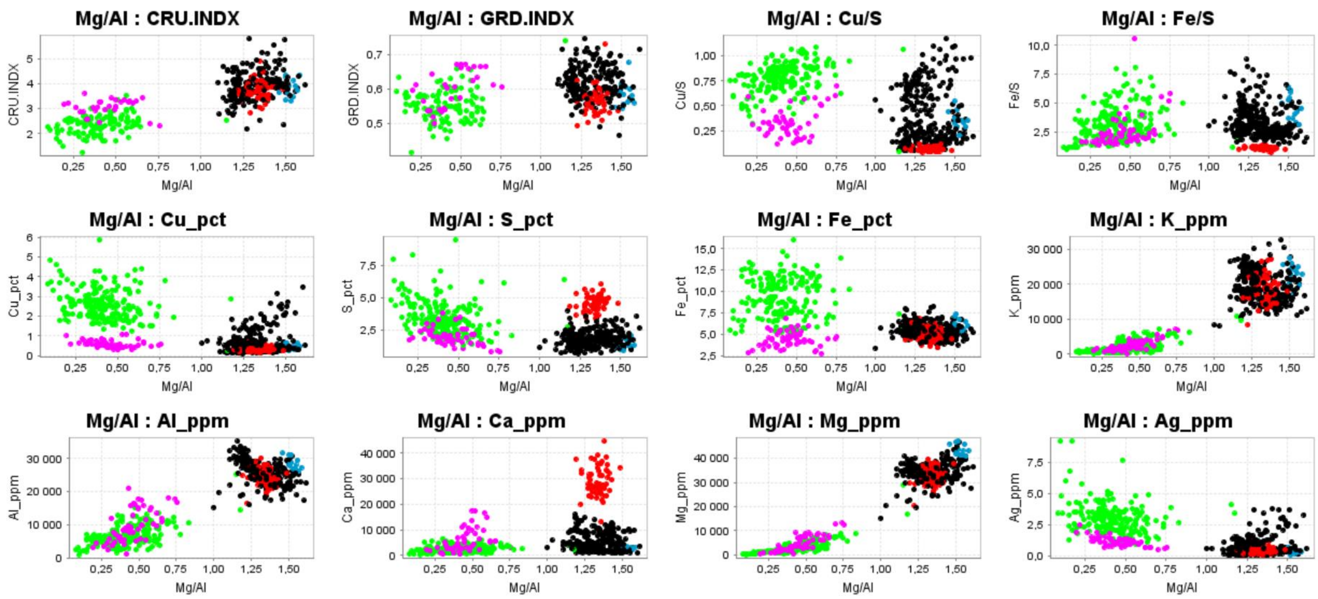


Figure 192. Correlation between parameters in the geometallurgy data matrix (Case Study P)
(Image: Simon Michaux)

9.1 Statistical data domain definition with Principle Component Analysis (PCA)

One of the methods used to partition a marge and varied data matrix into meaningful groups is the application of a statistical technique called Principle Component Analysis (PCA).

PCA is a mature statistical technique that is widely used for finding patterns in data of multiple dimensions. PCA finds a set of orthogonal dimensions, which account for all the variance in a particular dataset, by reducing the dimensionality of a complex system of correlations into a smaller number of dimensions. The first principal component (PC1) accounts for as much data variance as possible and each subsequent principal component accounts for remaining data variance (PC2, PC3, PC4…).

This approach is to spatially model classes created during principal component analysis and attempt to partition the data set into groups that describe the most variability (PC1, PC2, etc.). Each class has a given distribution for each parameter, where the target processing response and variability is examined in each block by association.

9.1.1 Principal component Method:

1. Subtract the mean from each of the data dimensions in order to create a data set that has a mean of zero.

2. Calculate the covariance matrix for the dataset. The covariance is a measure of how much each dimension varies from the mean with respect to each other (i.e. the relative spread of data in a dataset). The formula for the covariance is:

$$\operatorname{cov}(X,Y) = \frac{\sum_{i=1}^{n}(X_i - \overline{X})(Y_i - \overline{Y})}{(n-1)} \qquad \text{Equation 21}$$

The value obtained from the covariance calculation is not as important as if they sign is positive or negative.

- If the result is positive, it implies that a direct relationship exists between the two datasets.
- If the sign is negative: the data sets are inversely related to each other.
- If the result of the covariance calculation equals zero, the two data dimensions are independent of each other.

As there are multidimensional datasets, a covariance matric is used. The covariance matrix is defined as:

$$C^{n\times n} = \left(c_{i,j}, c_{i,j} = \operatorname{cov}\left(Dim_i, Dim_j\right)\right) \qquad \text{Equation 22}$$

This equation states that if you have an n-dimensional dataset, then the matrix has n rows and columns and each entry in the matrix is the result of calculating the covariance between two separating dimensions.

3. Calculate the eigenvectors and eigenvalues for the matrix:

Let A be a square matrix. A non-zero vector C is called an eigenvector of A if and only if there exists a number (real or complex) λ such that AC = λC. If such a number λ exists, it is called a eigenvalue of A. The vector C is called the eigenvector associated to the eigenvalue associated to the eigenvalue λ.

4. Select components and form a feature vector. A feature vector is an n-dimensional vector of numerical features representing an object.

5. Derive new datasets by condensing multidimensional data into orthogonal dimensions (i.e. Principal Components)

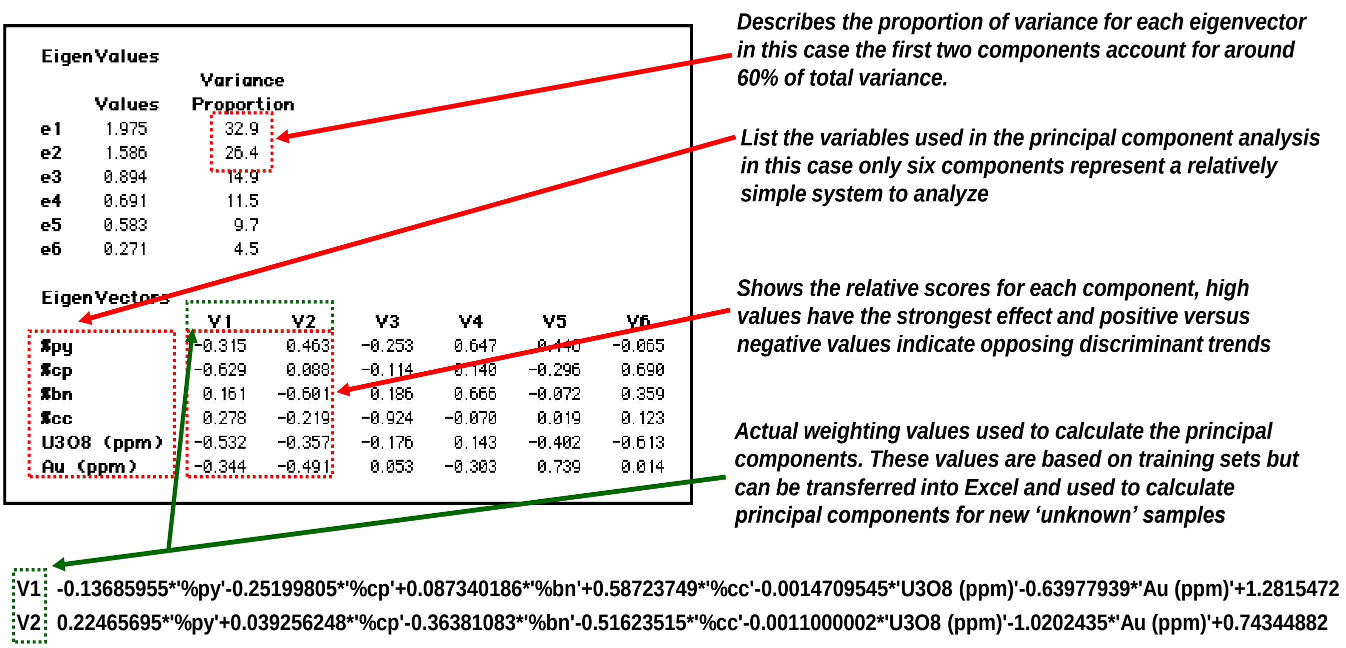
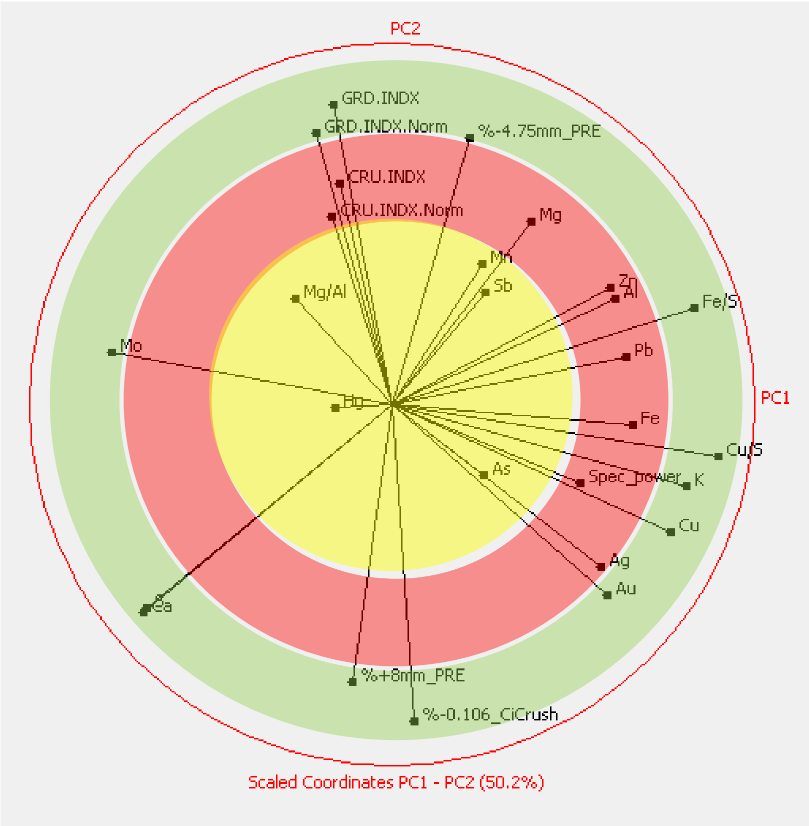


Figure 193. Principle component methodology

PCA expresses the data in such a way as to highlight similarities and differences within the data set. It is well suited to analysis of rock-based attributes using large datasets in geometallurgy.

It is to be remembered that this PCA based characterisation has been set up to group based on fundamental multivariate controls of the whole data set, not on constraining the processing response. This can potentially result in large variability within classes that may cover the entire distribution of processing performance results.

It is for this reason that the PCA procedure is not the final analytical goal of a geometallurgical study. The decision of what a Class (or process defined ore type) could or should be is to be guided by patterns found in the target process behavior data with a secondary consideration for what the PCA tool returned and what was learned in the domaining study. Figure 194 to 196 shows several PCA plot examples from Case Study P.



Primary Influence

- Ca, S, %-106, Cu/S
- Ci-GRD, Cu, K, Fe/S
- Au, Ci-GRD_Norm
- %-4.75, %+8mm

Secondary Influence

- Ag, Zn, Al, Fe, Pb
- Ci-CRU, Ci-CRU_Norm
- Spec_Power

Reduced Influence

- Mn, Sb, Mg/Al
- As, Hg

Figure 194. Principle Component Analysis of Geometallurgical Matrix data set
(Image: Simon Michaux)

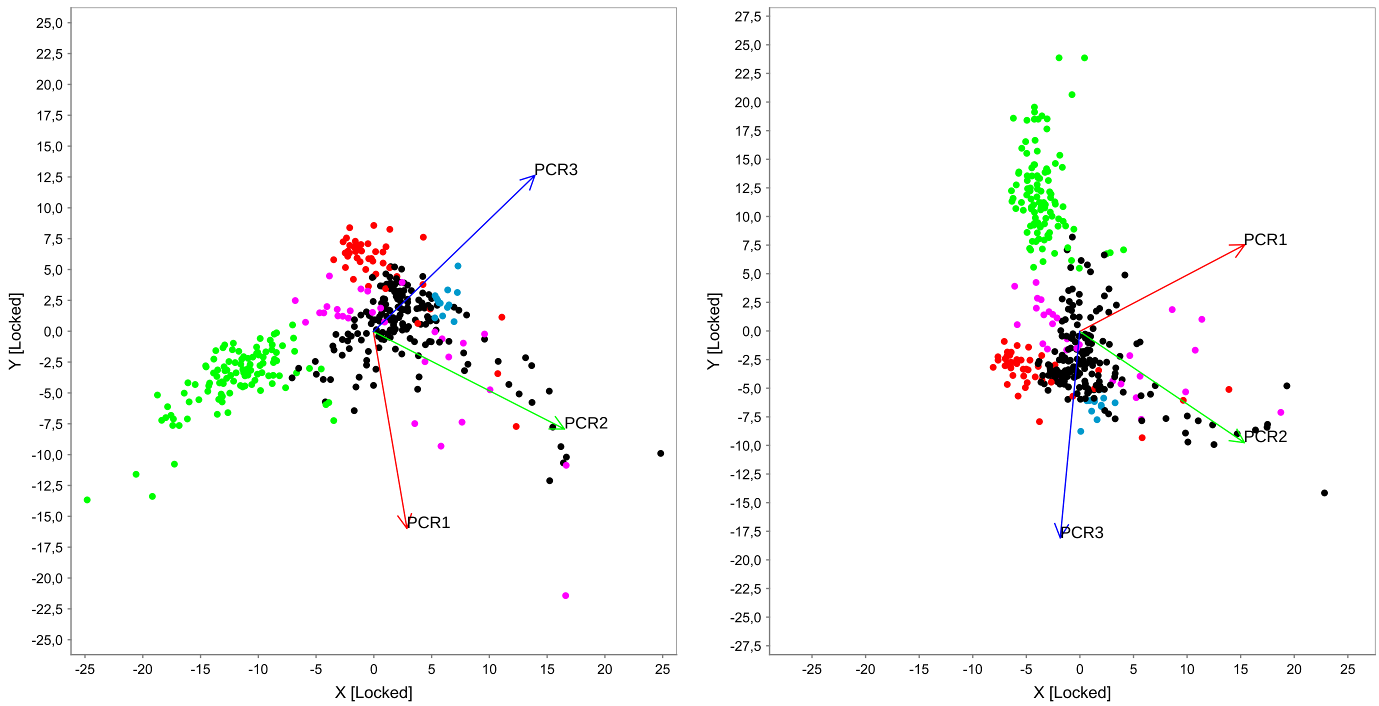


Figure 195. PCR1 vs. PCR2 vs. PCR3 (Case Study P)
(Image: Simon Michaux)

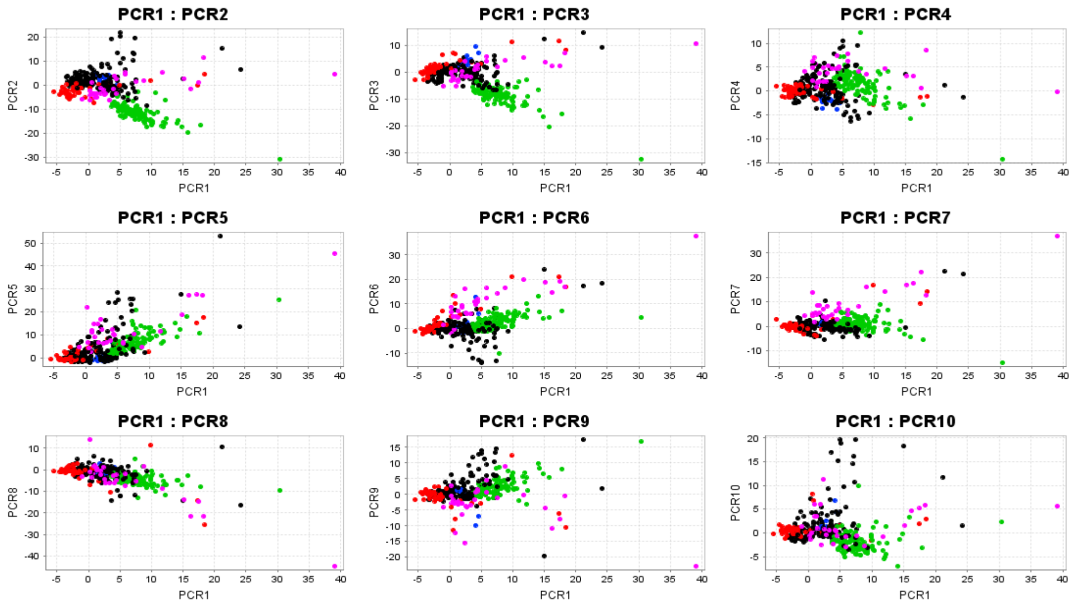


Figure 196. Principle Component Analysis (PCA) plots (Case Study P)
(Image: Simon Michaux)

10 PROCESS BEHAVIOR ORE TYPE CLASS DEFINITION

Geometallurgy is an iterative process where the experimental plan and the data analysis is constantly reexamined to assess to ensure that the overall campaign objectives are achieved.

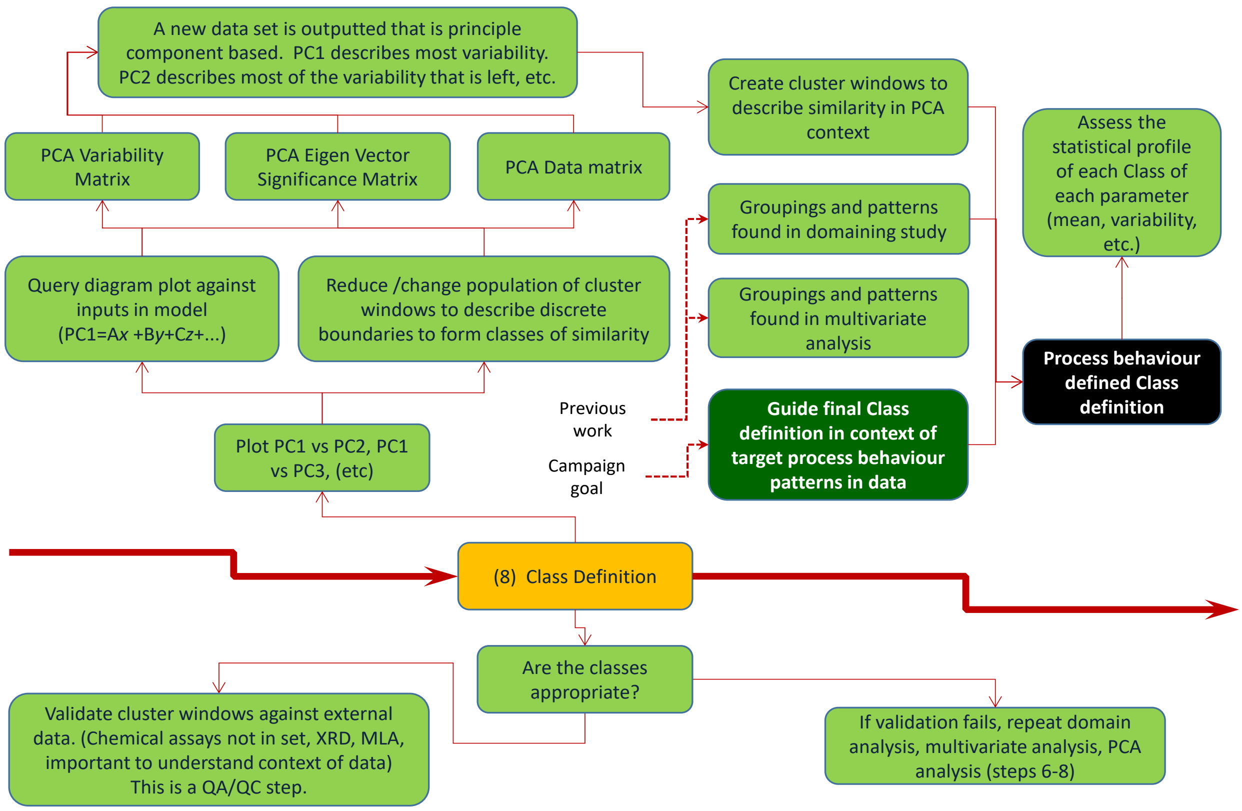


Figure 197. Class Definition of Geometallurgical Matrix data set
(Image: Simon Michaux)

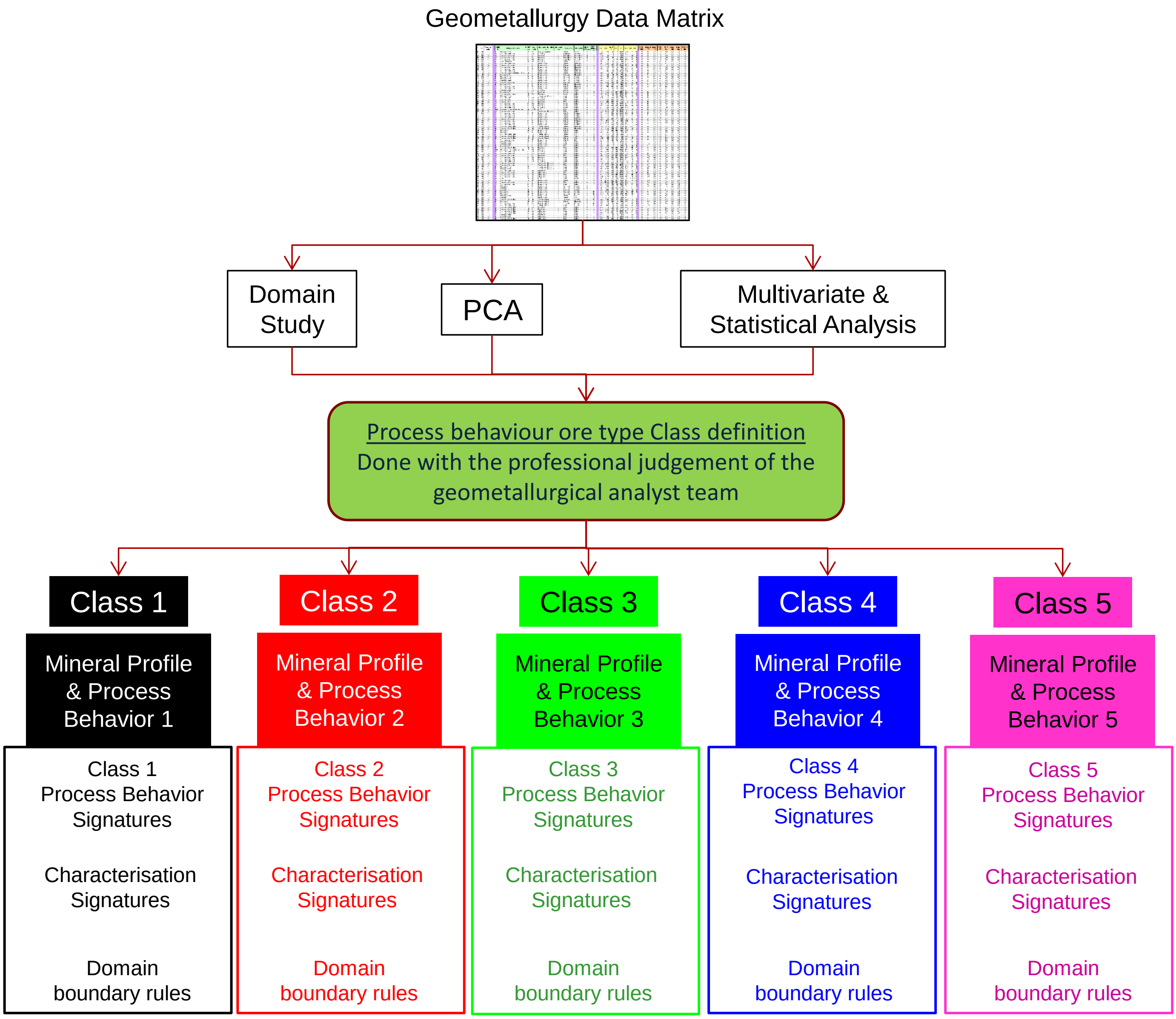


Figure 198. Process behavior guided ore type class definition
(Image: Simon Michaux)

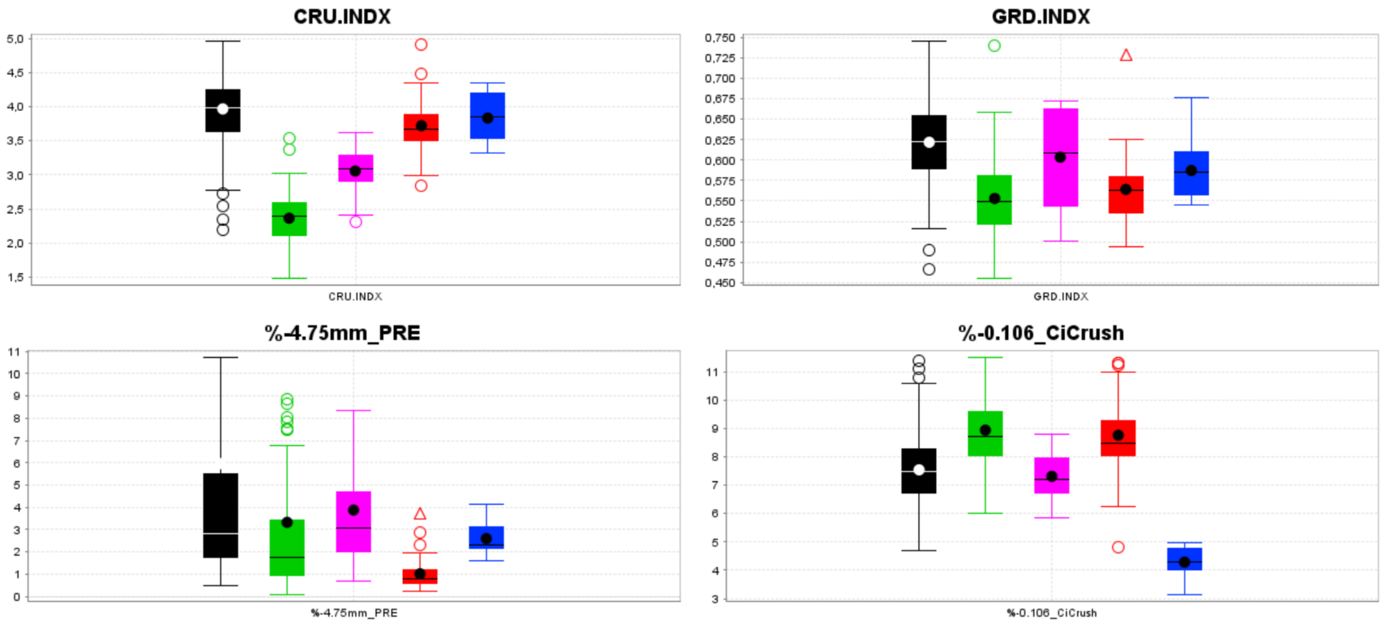


Figure 199. Box and whisker plots of the Comminution Index CRU (crushing), GRD (grinding), crushing QA/QC steps (Case Study P) (Image: Simon Michaux)

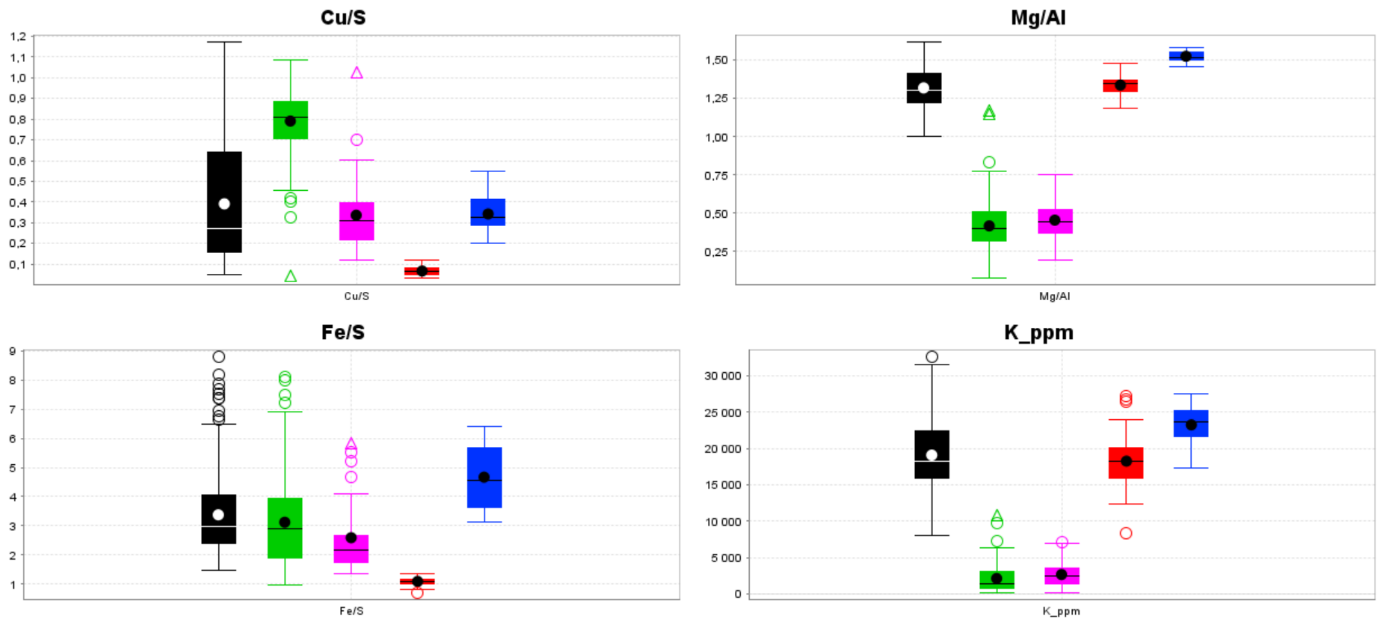


Figure 200. Box and whisker plots (Case Study P) (Image: Simon Michaux)

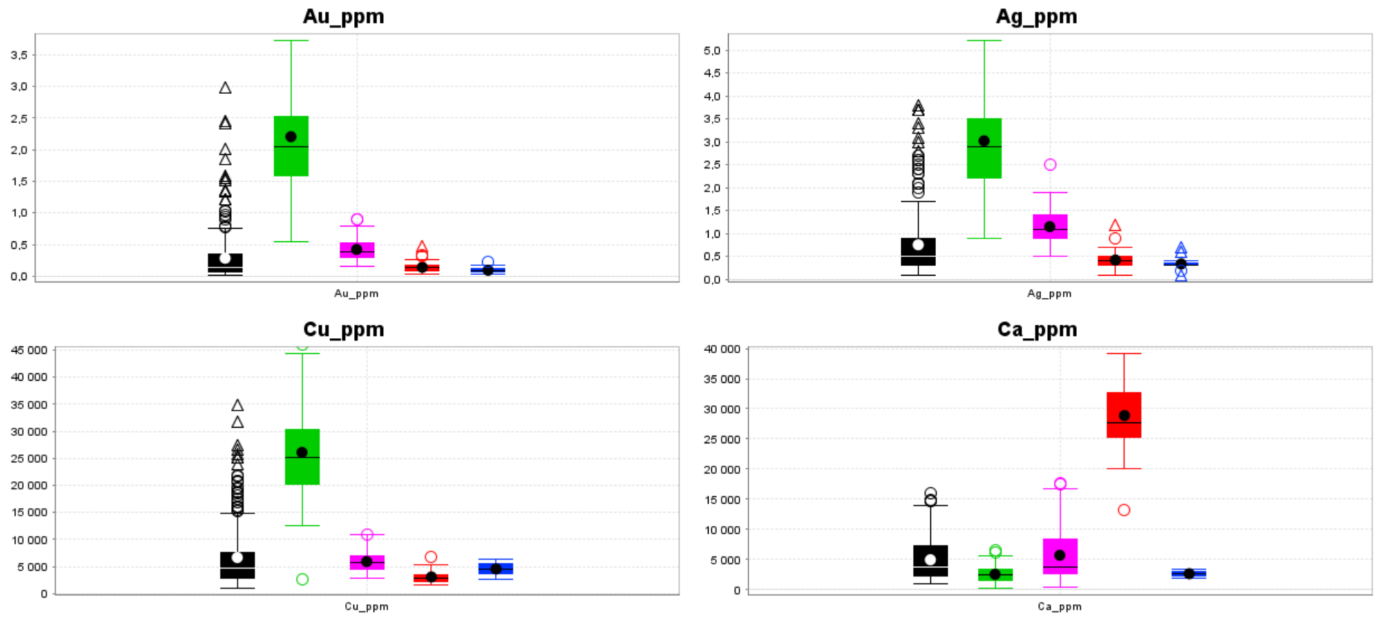


Figure 201. Box and whisker plots (Case Study P)
(Image: Simon Michaux)

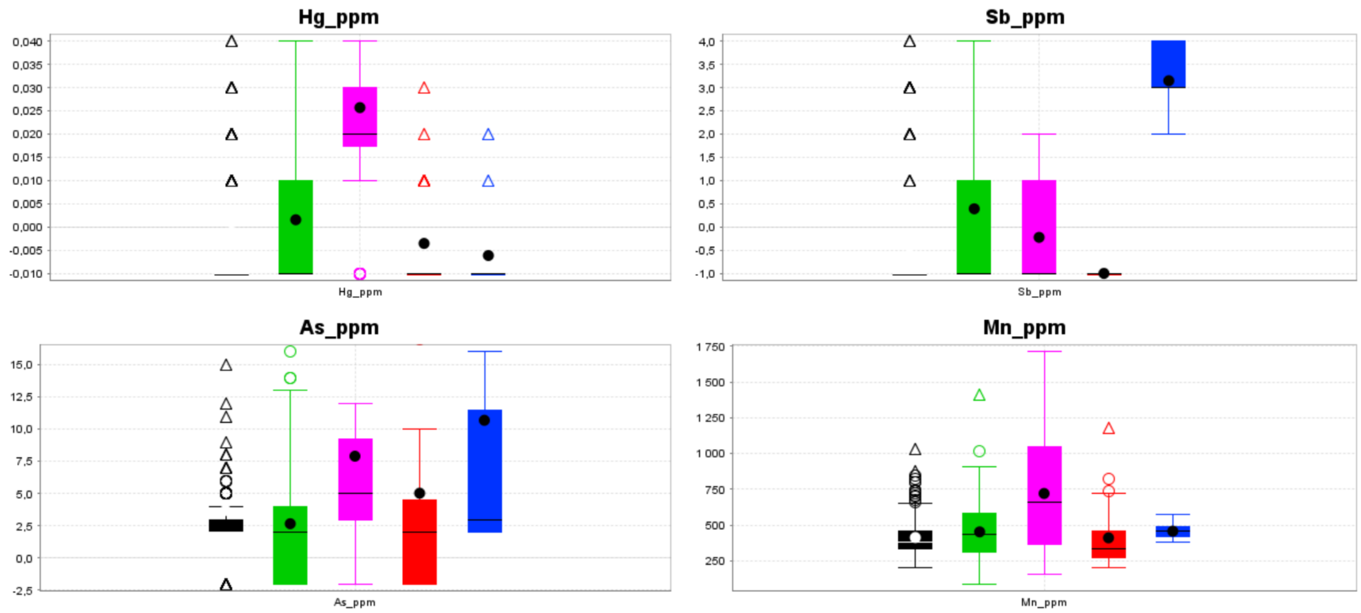


Figure 202. Box and whisker plots (Case Study P)
(Image: Simon Michaux)

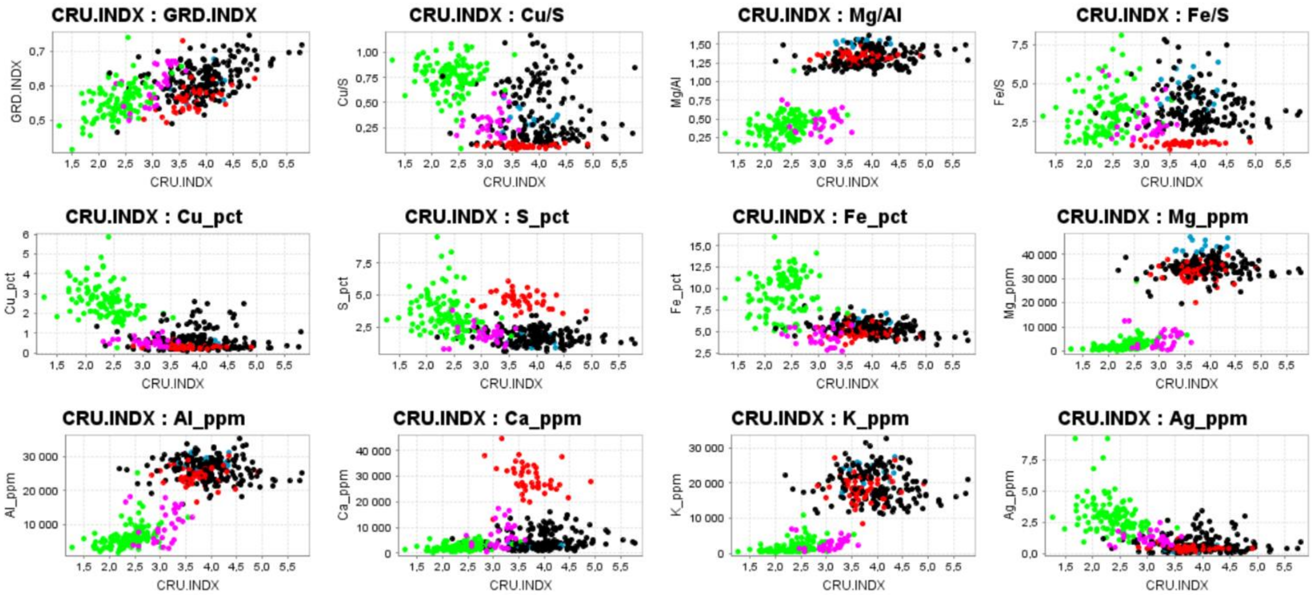


Figure 203. Scatter plots between parameters: Comminution Index Ci-CRU (Case Study P) (Image: Simon Michaux)

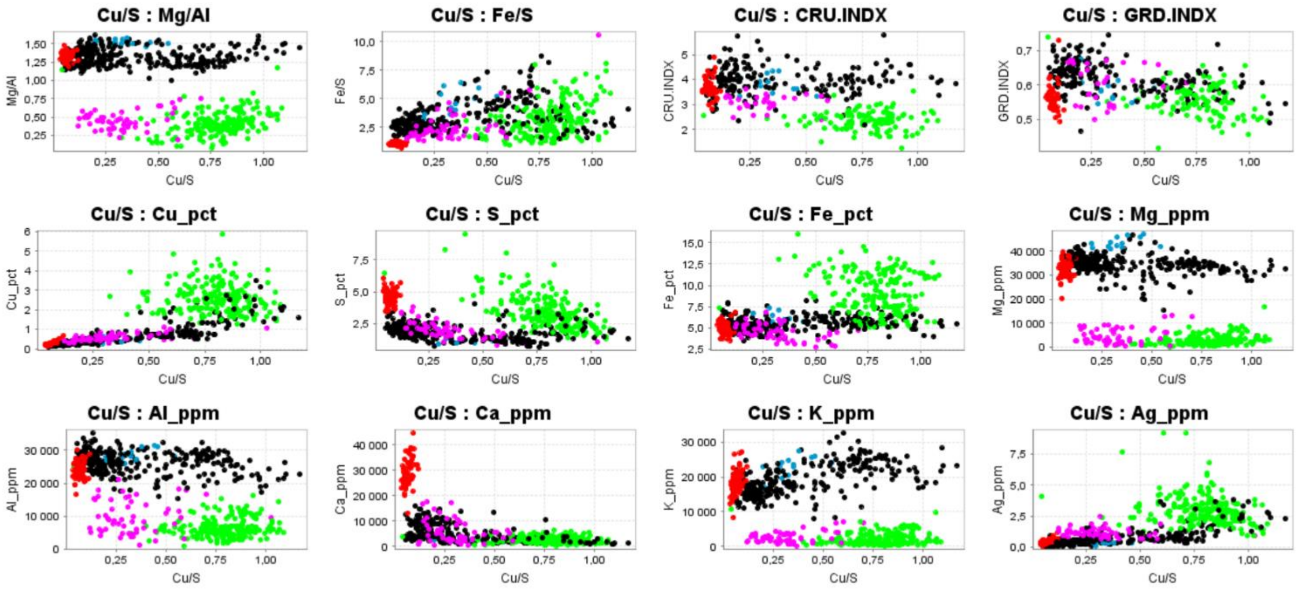


Figure 204. Scatter plots between parameters: Cu/S (Case Study P) (Image: Simon Michaux)

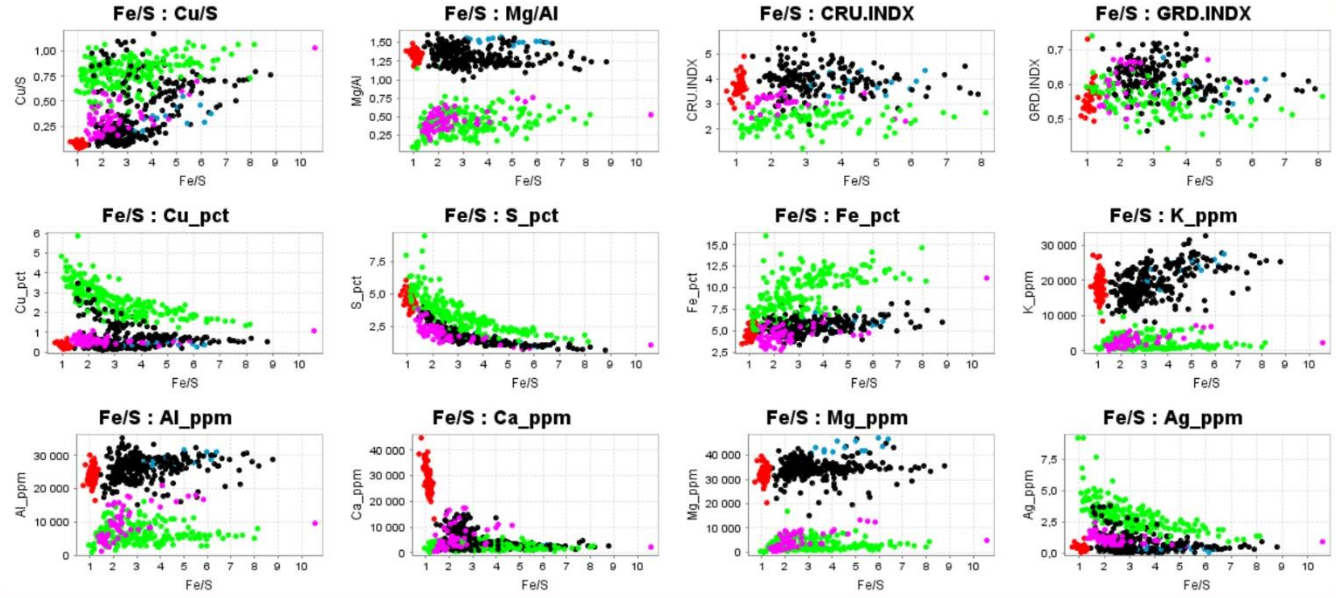


Figure 205. Scatter plots between parameters: Fe/S (Case Study P)
(Image: Simon Michaux)

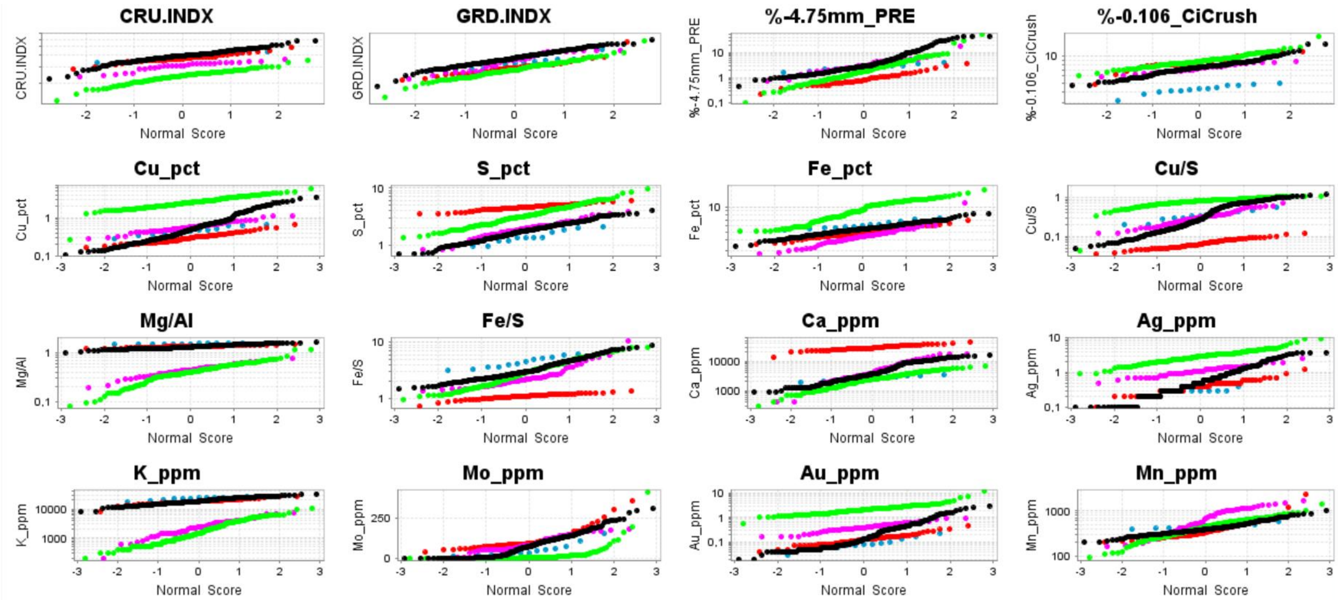


Figure 206. Probability plots for parameters (Case Study P)
(Image: Simon Michaux)

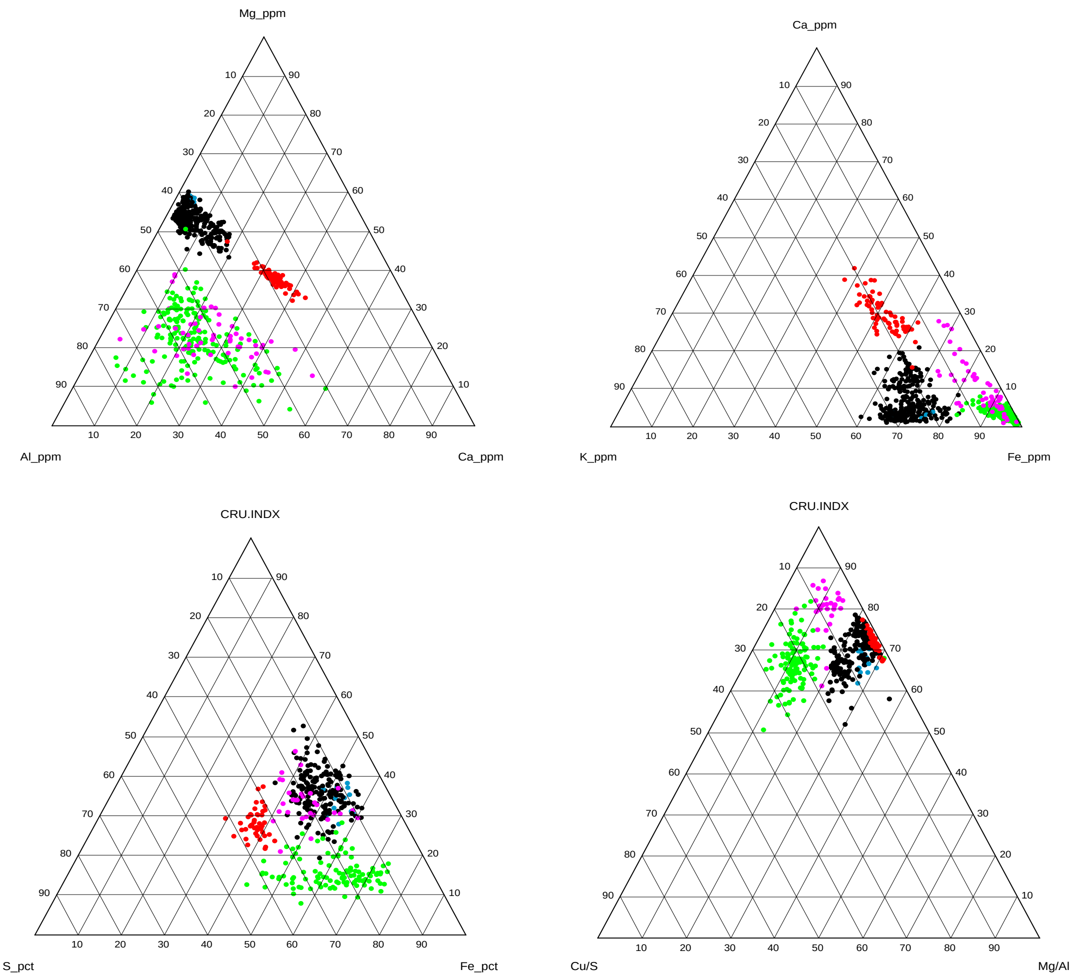


Figure 207. Ternary plots for parameters (Case Study P)
(Image: Simon Michaux)

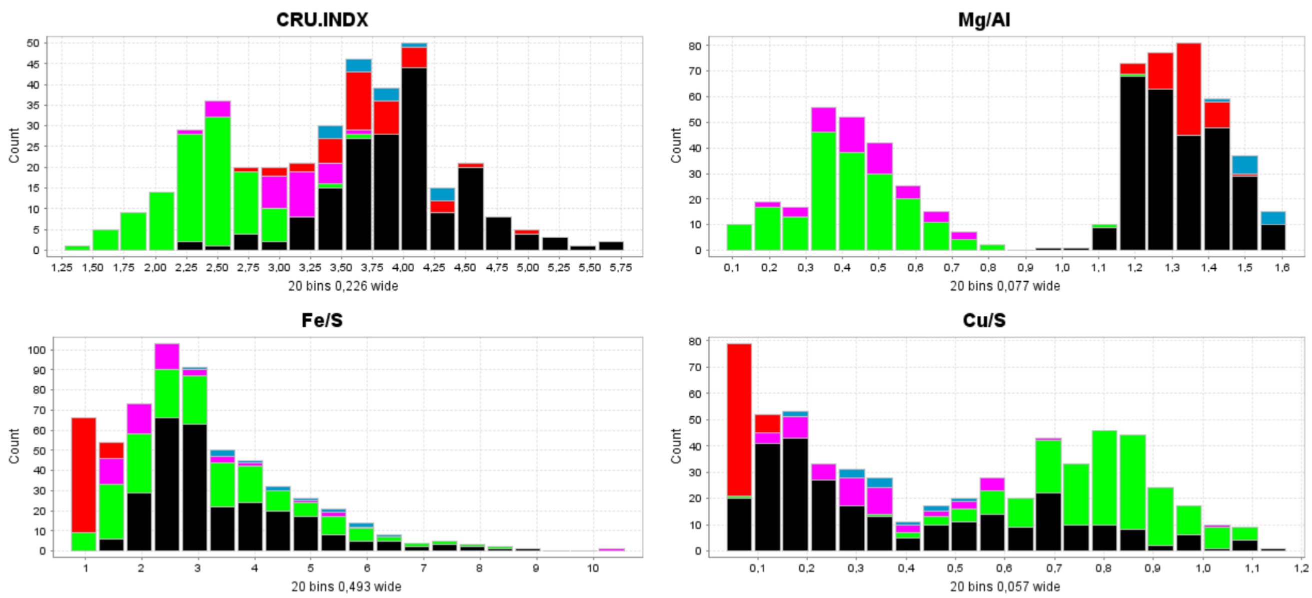


Figure 208. Histograms for parameters (Case Study P)
(Image: Simon Michaux)

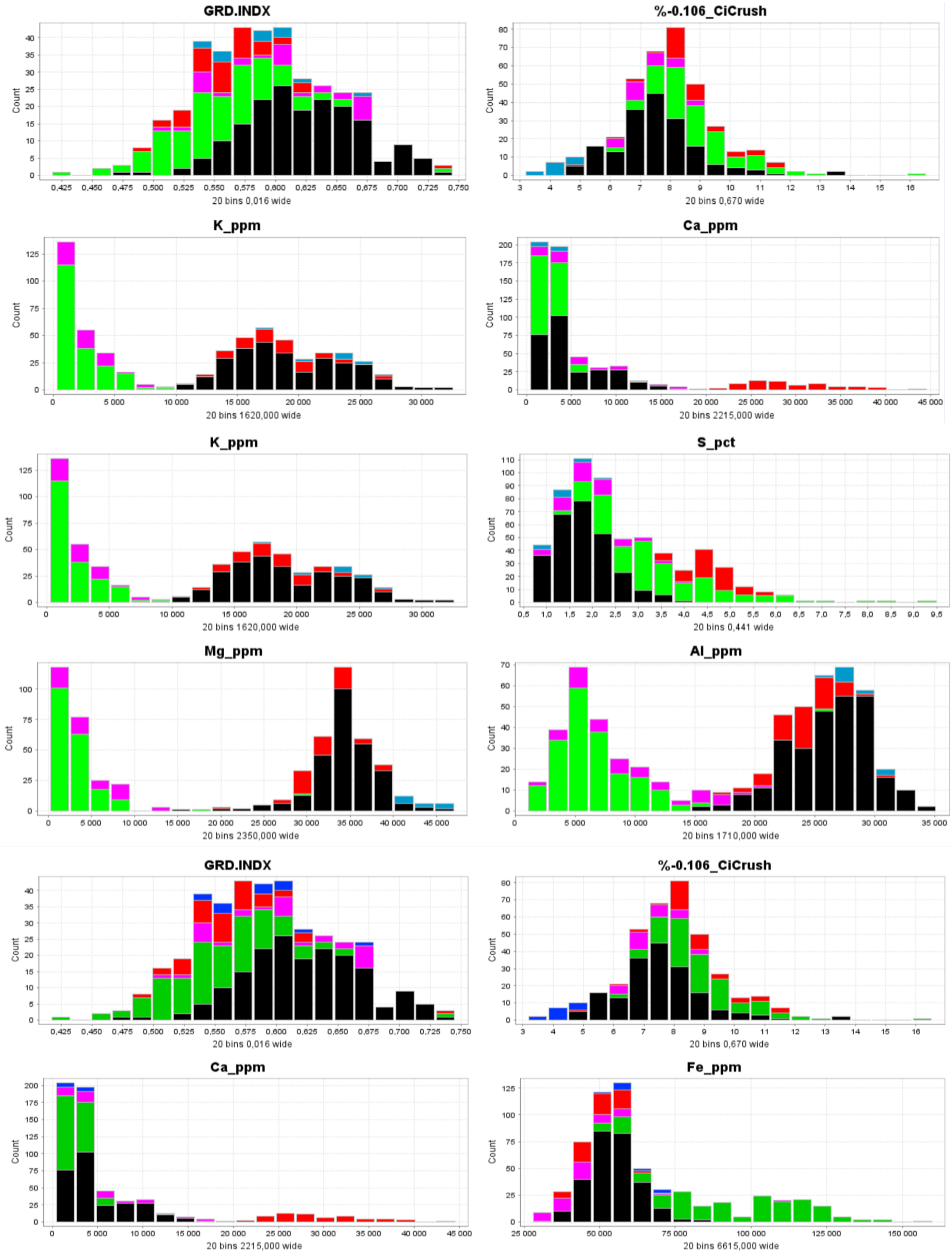


Figure 209. Histograms for parameters (Case Study P)
(Image: Simon Michaux)

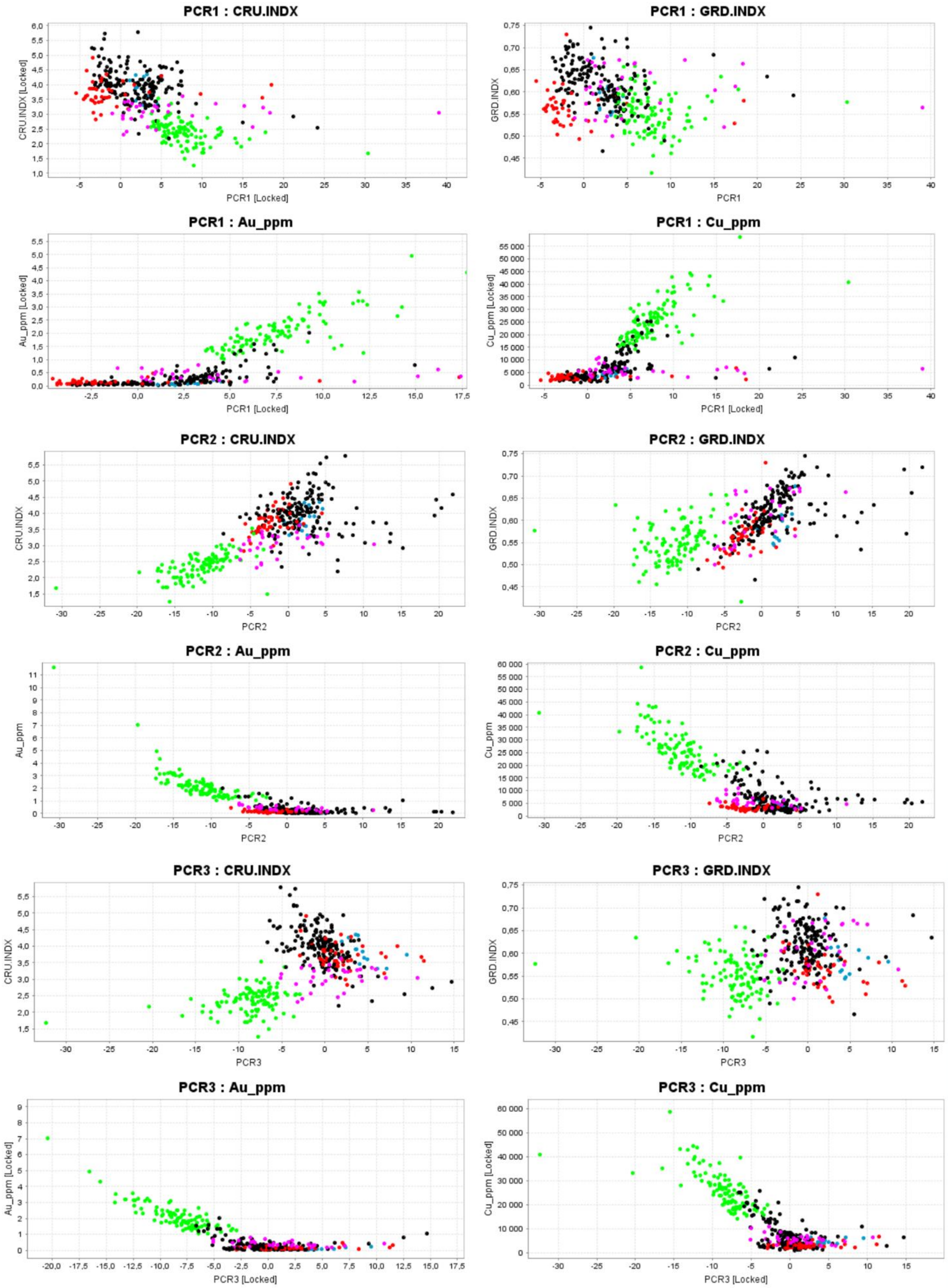


Figure 210. Principle Components (PCA) plotted against parameters (Case Study P)
(Image: Simon Michaux)

When the established classes are stable enough under scrutiny, the study can move to the next step. The outcomes should be cross examined again in context of what was learned in the Domaining Study. Figure 211 below shows an example of Class domaining down hole according the boundary conditions decided upon by the analyst team. This example only has two chemical assay profiles, when to assess the Class boundary domains, eight chemical assays are needed.

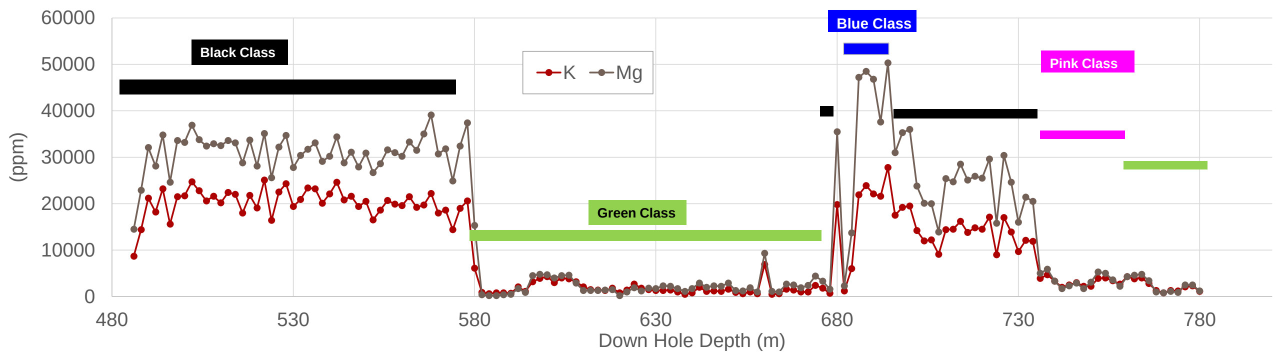


Figure 211. An example of Class domaining down hole in a single drill hole (Case Study P) (Image: Simon Michaux)

The process behaviour ore type definition step was done using chemical assays (in this Case Study P), not using the collected hyperspectral characterization data. The hyperspectral characterization data was used in the domaining study and also as a validation step in the Assessment of the Study step (Section 13).

10.1 Process modelling Class by Class

The comminution process behaviour of each class was studied, and a predictive model was developed. Figure 212 shows the starting point for the planning of this step.

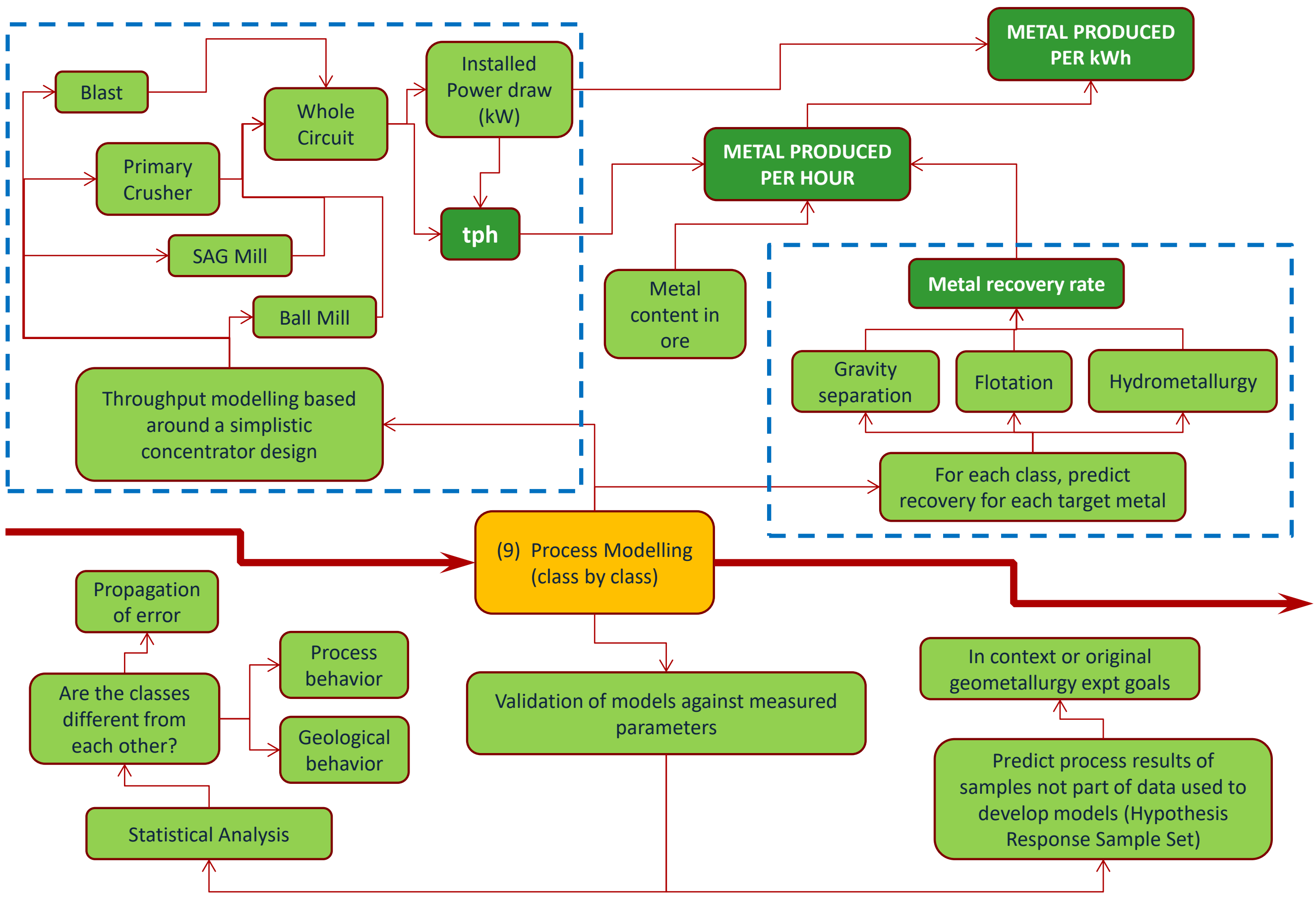


Figure 212. Step 9 - Process modelling class by class (Case Study P)
(Image: Simon Michaux)

For each defined mineral association class, a model is developed to predict the target process. Class-based modelling in is based on a combination of principal component analysis (PCA), exploratory data analysis and class-based multiple regression.

Develop a working relationship between the target measured parameter (bankable test) and a proxy estimates (proxy test measurement(s)), specific to that family of rock types in the target deposit. This would become a proxy process model for that mineral association class. To make a proxy model, enough data points are needed to provide data support for further analysis (Kojovic 1988).

- Approximately ratio of 4:1 data points to variables in model
- Acceptable level of R^2 defined by where you are in the mining process
- Acceptable average residual error defined by where you are in the mining process

Individual class-based regression is designed to produce predictive process models (for example impact breakage comminution). This is model would be based on at data parameter inputs considered from the geometallurgical database used in PCA definition of that class. It would be very useful if the final class based process model be based on multivariate mineralogy from assay. This would mean that acceptable models could be extrapolated to many tens of thousands of assay data points. The ability to have this type of predictive data at the same scale of assays is profound. This would be a major step-forward in supporting spatial modelling and process domaining linked to resource optimization.

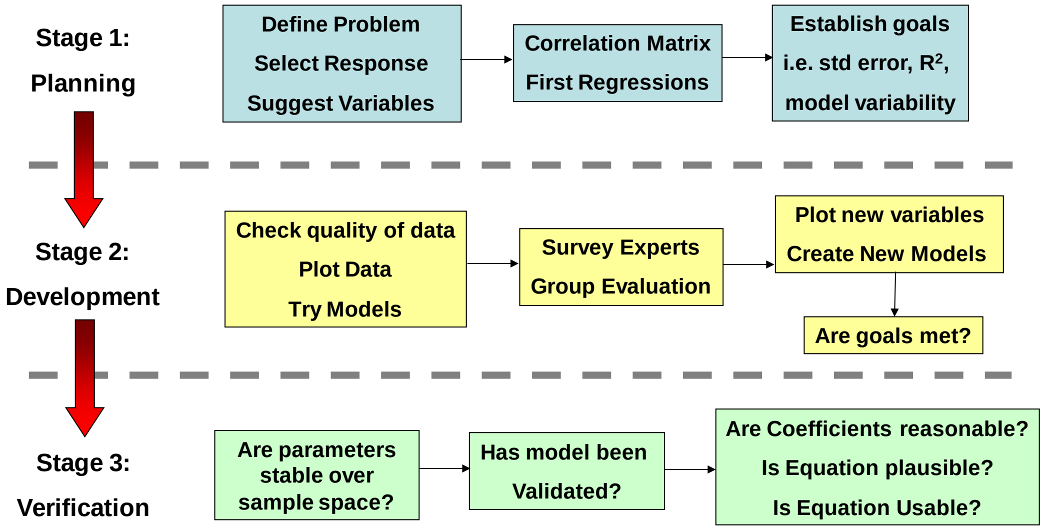


Figure 213. The Model Building Process
(Image: Simon Michaux)

Mathematical modelling is a process designed to achieve a method for predicting a response given a set of input variables. In the context of mineral processing this usually involves adjusting model parameters or coefficients to fit measured data.

The methods used in model building could be (but are not limited to) multiple linear regression analysis:

$$y = a_1x_1+a_2x_2+......a_nx_n \qquad \text{Equation 23}$$

Where: y is the dependent variable

$x_1,x_2,\ldots,x_n$ are the independent variables

$a_1,a_2,\ldots,a_n$ are the coefficients to be determine

Models have been fitted using nonlinear least squares method. A least squares solution (SSQ) is used to obtain the initial approximation of the coefficients for the linear model.

- Minimize over $a_1,\ldots,a_n$:
- This equation gives the maximum likelihood estimates of the fitted parameters if the measurement errors are independent and normally distributed with constant standard deviation and mean.
- The above quantity in equation 1 is referred to as X^2 (chi-squared).
- The parameter can be used to obtain a quantitative measure for the goodness of fit for a model.

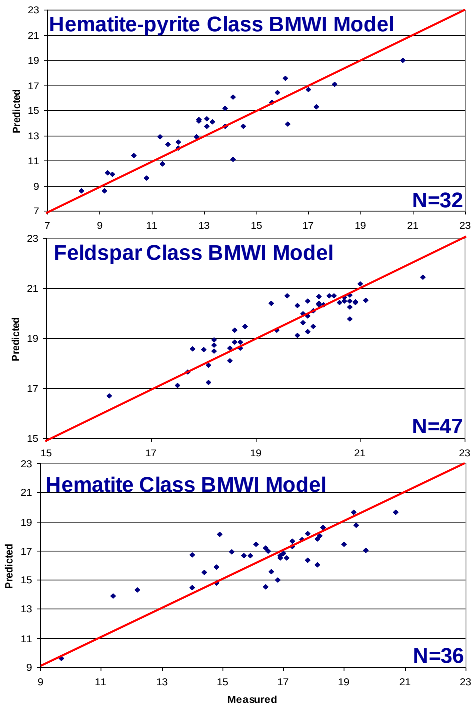
After a model has been determined which provides an acceptable fit to the data, before it is used some form of validity should be made:

- Model validation is primarily concerned with determining that the models output behaviour has the accuracy required for the models intended purpose.
- As a model is only an approximation of the real process, a difference between the model and system results (model error) must be anticipated.
- Validation process deals with the evaluation of the magnitude of this model error.
- Validation Techniques.
- Face validity, comparison with other models, traces of model terms, historical data validation, predictive validation, parameter variability, operational graphics.

A comment on model building:

> *"It is important to realize that model building is a creative process, where many factors such as the model builder's experience, preferences and ideas as well as the purpose for which the model is to be used enter into selection of the form of the model.*
>
> *Statistical procedures for assessing a model exist, but the model building process cannot ultimately be completely reduced to a recipe based on several statistical tests."*
>
> *Toni Kojovic, 1988 (Kojovic 1988)*

Shown below are examples of class based process models (crushing and grinding).



Model:

$BMWI = -66.65 + 1.24Flu^{0.61} - 0.27Hem - 1.49Py + 19.23SG + 8.39(Py/Sul)^{0.84} + 0.72QHard^{1.22} + 0.72(Chl/Sul)^{1.31}$

S.E.=0.90 R^2=0.84 Model prediction relative error =7.4%

Model:

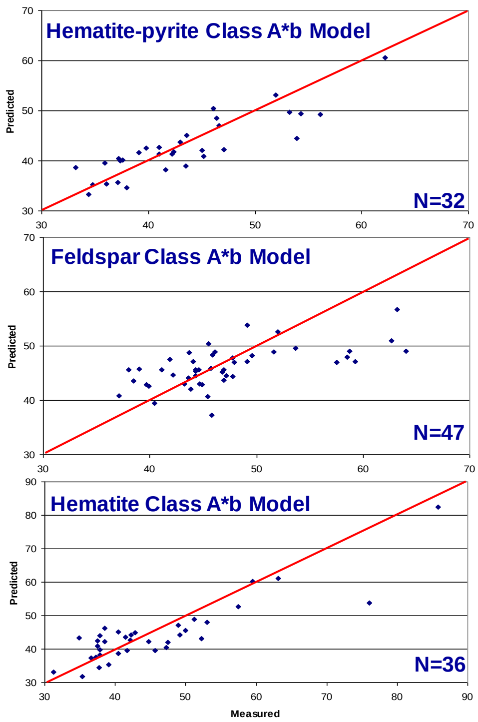
$BMWI = -104.4 - 0.17Ksp - 0.88Qtz - 0.12Sid - 0.47Flu - 0.43Sul - 0.54(Ser/Ksp) + 0.88(Sul/Hem) + 8.56QHard$

S.E.=0.41 R^2=0.83 Model prediction relative error =2.1%

Model:

$BMWI = -44.99 + 0.67Ser^{0.56} + 0.19Sid + 0.13Flu^{1.64} + 0.63Bar^{0.81} + 6.80SG + 1.90QHard$

S.E.=0.99 R^2=0.66 Model prediction relative error =6.2%



Model:

$A^*b = 177.1 + 0.5Ser - 1.05Sul + 0.87Hem + 0.05Py^{2.75} - 44.66SG - 2(Qtz/Sul) + 12.36(Py/Qtz)^{1.65} - 17.3(Py/Sul)^{1.55}$

S.E.=0.97 R^2=0.76 Model prediction relative error =6.5%

Model:

$A^*b = 24.99 + 1.29Qtz^{0.8} - 1.58Flu^{0.65}\ 15.36Sul^{0.49} + 4.06Hem^{0.61} + 2.74Cpy^{1.42} + 7.89Bn^{0.49}$

S.E.=1.22 R^2=0.38 Model prediction relative error =8.5%

Model:

$A^*b = -282 - 1.93Sid + 0.01Bar^{1.69} + 1.65Hem + 194.3(1/Ser) - 41.01SG + 383.1(Qtz/Hem)^{0.09} + 5124(Sul/Hem)^{3.62} + 41.75((Sid+Ser)/Qtz)^{0.82} - 82.97((Sul/Hem)/((Sid+Ser)/Qtz))$

S.E.=1.40 R^2=0.77 Model prediction relative error =9.0%

The viability of the proxy model has to the bankable test is then established. In an ideal practice, many different proxy models developed in parallel to predict each measured/bankable process target. Once the form and structure of the proxy model is established, the next step is to constrain it using the mill circuit to identify the sensitivity required on measurements like BMWi and A*b. From this, identify the critical parameters relating to that target process that should be collected for that ore deposit system.

The quality of the outcome has to be balance against the cost of tests required vs accuracy of model, available budget and time pressure to complete the study.

Once the most efficient proxy model is established, it can then be used in conjunction with other process target proxy models. Figure 190 below shows the outcome of two proxy models (Impact breakage for crushing and BMWi for grinding) to study the comminution footprint in context or ore characterization.

A useful data analysis to conduct is determining what mineralogy controls comminution footprint. It is understood that mineralogy controls most other mining processes. In comminution, it is the gangue mineralogy that dominates the outcome. As such QXRD is the preferred mineralogical measurement tool. Figure 191 below shows an example of this being done successfully.

10.1.1 Example Case Study P

Specifically, the impact breakage parameter Axb and grinding Bond Ball Work Index (BMWi). After the process behaviour ore type Classes were defined, the boundary conditions were used to select samples to be sent off for larger scale tests. The SMC test was selected to measure the Axb parameter. The Bond Ball Index test was selected at the defined closing size (defined by automated mineralogy assessing target metal mineral grain).

The drill core to source these samples was selected by the Class definitions boundary conditions. A series of composite samples were selected from characterized drill core, where different tests were taken from the same core intervals in a representative manner. The tests were completed and passed QA/QC test standards. The data for each Class was isolated and modelling was able to be conducted. The parameters looked at first were defined by the PCA and multivariate studies (similar to Figure 194). Influencers are ranked in Table 12.

Table 12. Known influences from the completed tests in Case Study P (Simon Michaux)

PRIMARY INFLUENCE	SECONDARY INFLUENCE
Potassium (K)	Calcium (Ca)
Comminution Index impact breakage (Ci-CRU	Sulfur (S)
Comminution Index impact breakage (Ci-GRD)	Fe/S ratio
Magnesium (Mg)	Lead (Pb)
Aluminium (Al)	%4.75_PRE CRUSH
Mg/Al ratio	Specific_power
Silver (Ag)	%-106_CiCRUSH
Gold (Au)	
Copper (Cu)	
Cu/S ratio	

All of these parameters were studied for correlation in a multivariate and non-linear regression specialist modelling software. Through an iterative process, the most robust model was selected for each Class to predict both Axb and BMWi.

Each Class was controlled and influenced by different parameters, or the same parameters but with different model inputs.

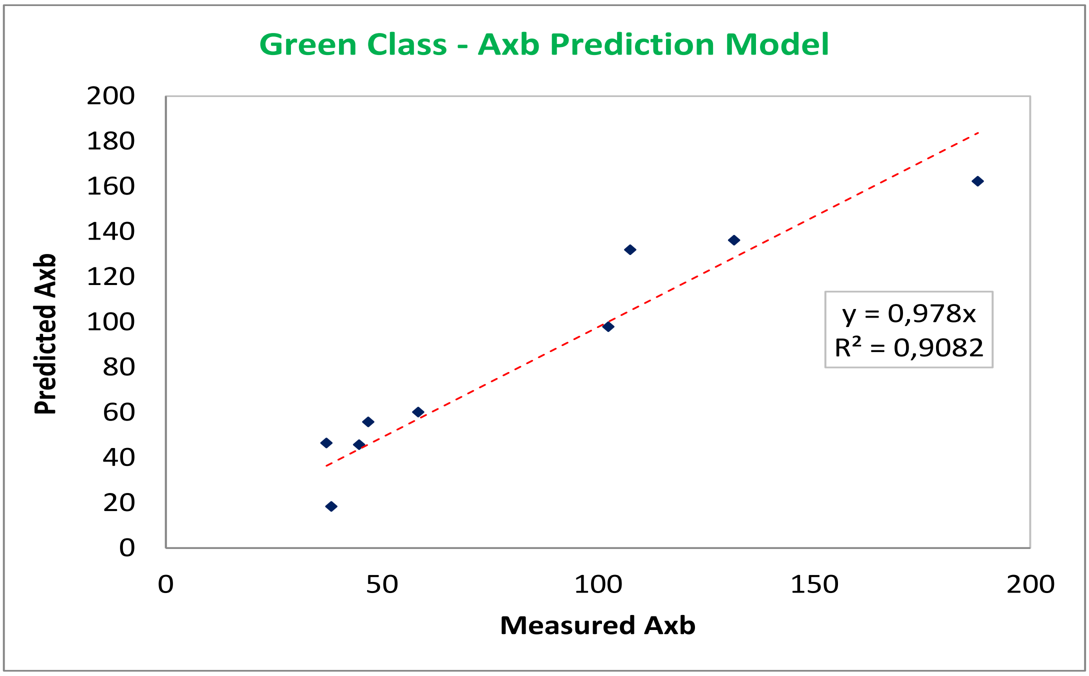


Figure 214. BMWi for Green class prediction, Case Study P (Simon Michaux)

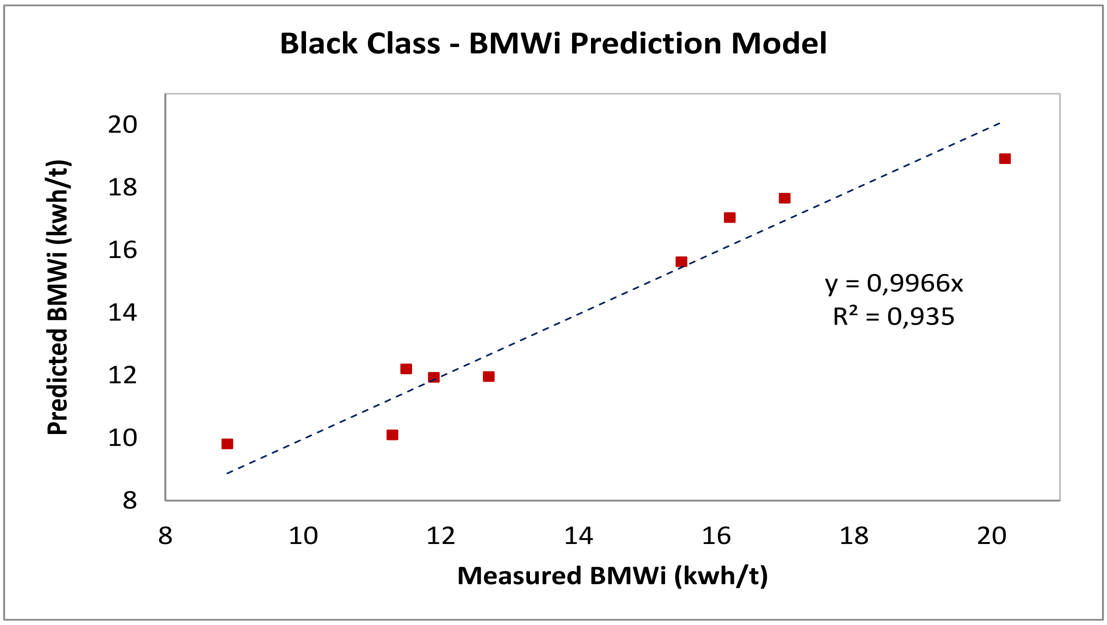


Figure 215. BMWi for Black class prediction, Case Study P (Simon Michaux)

Figures 214 and 215 show class based BMWi grinding prediction models. These examples were very successful with a high R^2 (the best examples for Axb and BMWi from across the five classes). The full outcome of this example is not shown in this report.

Normally, the correlation is not so strong. It is to be remembered that the number of parameters in the developed model has to be limited by the number of degrees of freedom.

10.2 Engineering Simulation

Once the relevant parameters have been measured or estimated, they should be used to estimate process performance. This can be done by using a simulation software (Figure 216 and 217) to model an existing plant, or develop a model of a conceptual plant. In doing so, plant throughput, recovery and power draw estimation can be modelled. This should be done for each Orientation sample if possible.

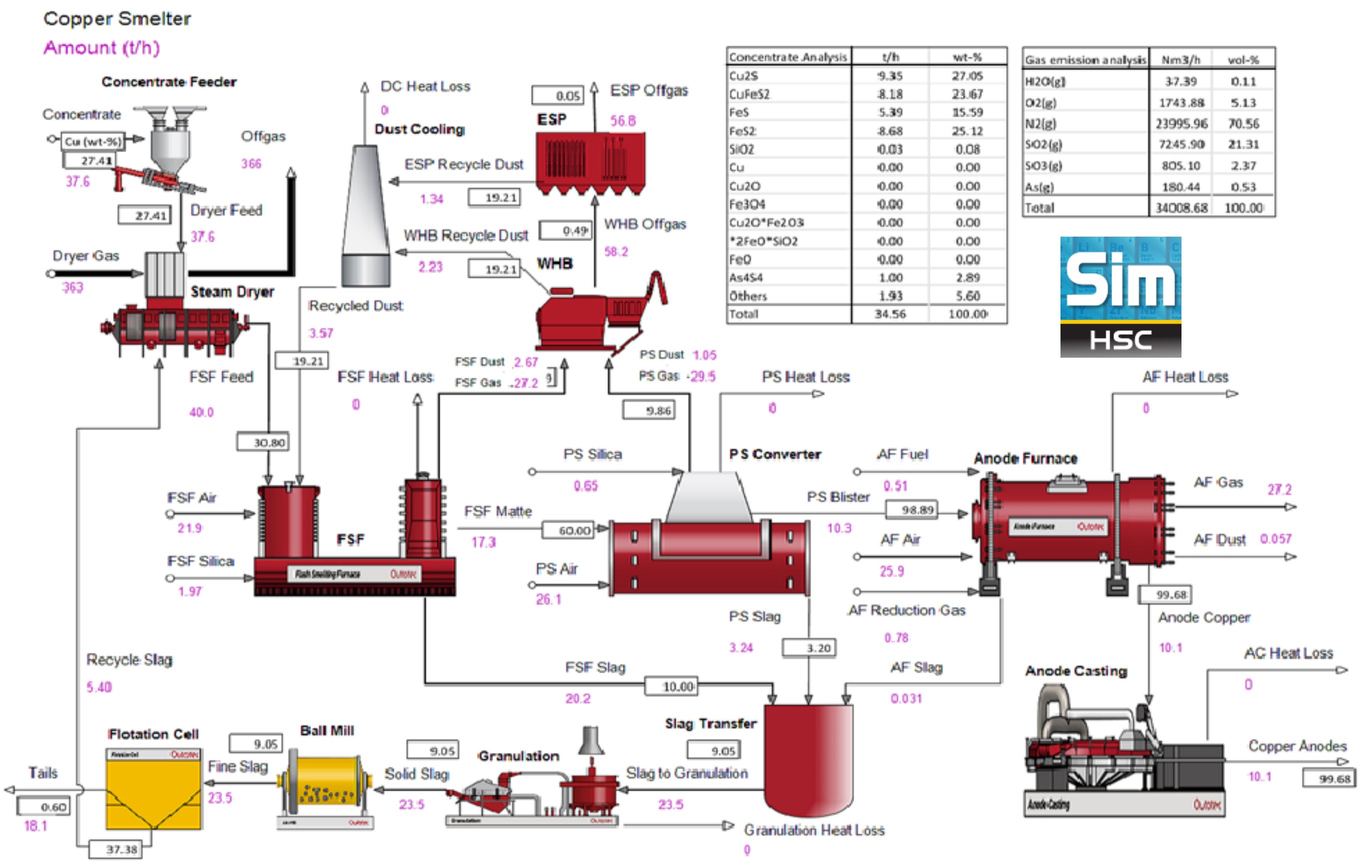


Figure 216. The HSC Thermochemical process simulation tool developed by Outotec (Image and copyright. Outotec)

Figure 216 shows a model developed in the Outotec HSC software. This software conducts thermochemical simulations of process behavior. It works well with flotation, hydrometallurgy and pyrometallurgy applications. It can be used to estimate outcomes from bench scale test work inputs.

Figure 217 shows a model developed in JKSimMet, which is a, general-purpose computer software package for the analysis and simulation of comminution and classification circuits in mineral processing operations. This software was developed from the research outcomes at JKMRC from University of Queensland. This software has been designed to work with a plant survey done on an existing process plant running at steady state. The prediction models of this software are empirical and have the capacity to optimize plant performance over a wide range of operation.

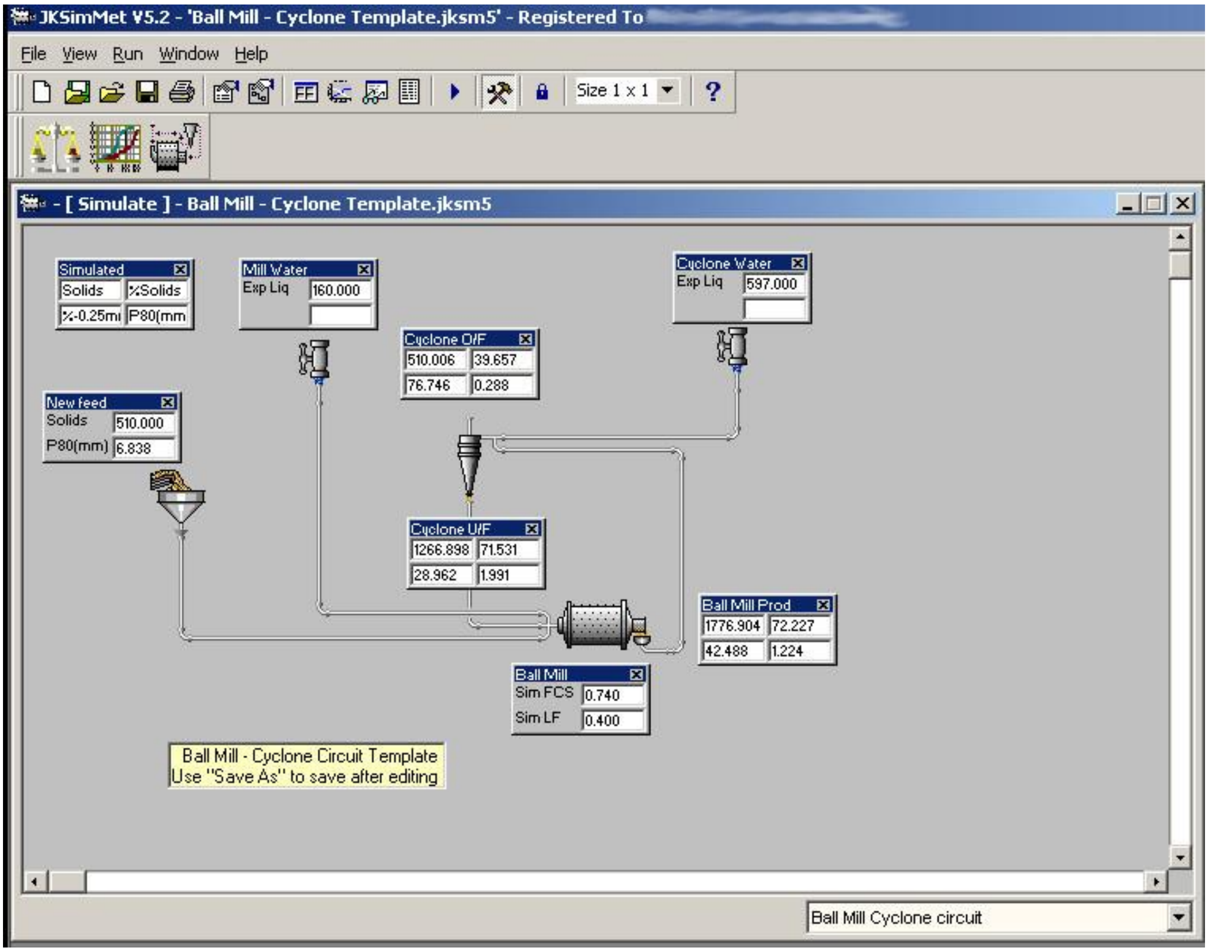


Figure 217. The JkSIMmet software
(Image and copyright. JKTech)

The outcomes of these kinds of simulations can be used to estimate the performance of the target process operation. If the model inputs of these simulations was based on the Orientation Samples discussed previously in this report, then it could be possible to predict the operational extremes the target operation may undergo.

In doing so, a series of process paths around each orientation sample could be developed and the outcomes ranked in context of effectiveness.

11 HYPOTHESIS SAMPLE STUDY

Once the Orientation Study is finished, a clear understanding of what small-scale tests, work in the target deposit is developed and planning of the experimental design of Hypothesis samples is able to be done. The Orientation Study examined a target process behavior in the 10 to 20 end member rock texture extreme samples (tightly constrained mineralogy across the whole sample of drill core). The purpose of the Mapping Hypothesis study is to map out the variability of the process behavior being studied.

A series of hypotheses can developed as a result of the work done previously. Where previous work was conducted in an exploratory fashion, this work would be done in a validation of hypothesis fashion. For example:

- This ore deposit responds best to a combination of leaching and flotation
- Gravity separation will remove gangue minerals X, Y and Z, thus improve recovery
- The prediction model where minerals A, B and C control comminution can be validated with testing samples of known compositions of A, B and C, where the comminution result is compared to the predicted value.

Further sampling and test work (Hypothesis response and validation samples) is a smaller sample set 500-1000m of drill core.

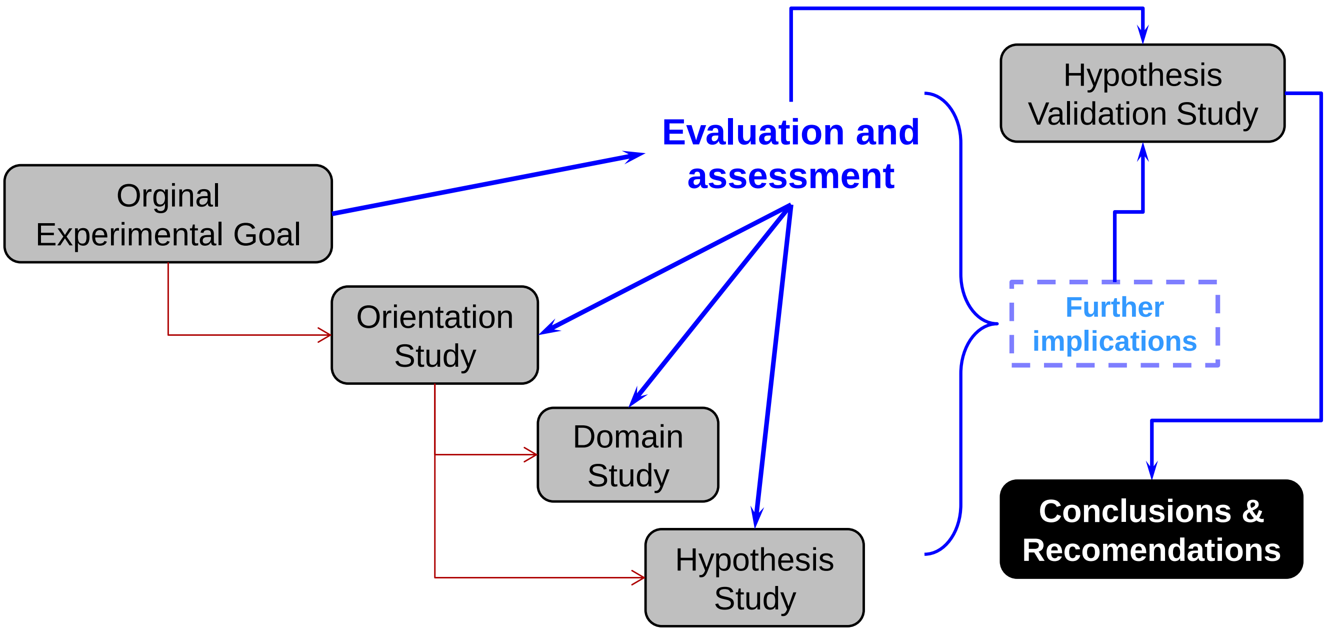


Figure 218. Evaluation of hypothesis response against orientation and mapping studies (Image: Simon Michaux)

11.1 Hypothesis Sample experimental test work

A reduced number of tests are done on each Mapping Hypothesis sample, based on what was learned in the Orientation Study. By the very nature of what is required in a geometallurgy mapping campaign, many data points (sample points) are needed.

The Hypothesis Sample set is taken from the original 1500-2000m of drill core (representing 4-5 continuous sections across the target deposit). Ideally, the Orientation Study sample set was also taken from these drill core sections. The Hypothesis Sample study should be thought through, where the test work has been statistically designed to allow the various stages of analysis the best possible chance for clear outcomes. The best way to ensure this, is to select the input samples appropriately and to manage the experimental test work to a high level of precision. This would allow for a lower value for propagation of error when examining the final results. Things to consider:

- For each sample, an intelligent selection of tests that interrelate with final objectives
- Order of tests understood (non-destructive then destructive)
- Competent technicians who understand methodology and objectives
- Sample preservation for future work

Once the analysis of these samples is completed, examine the analytical outcomes against previous work to determine if the geometallurgical campaign has been appropriately designed.

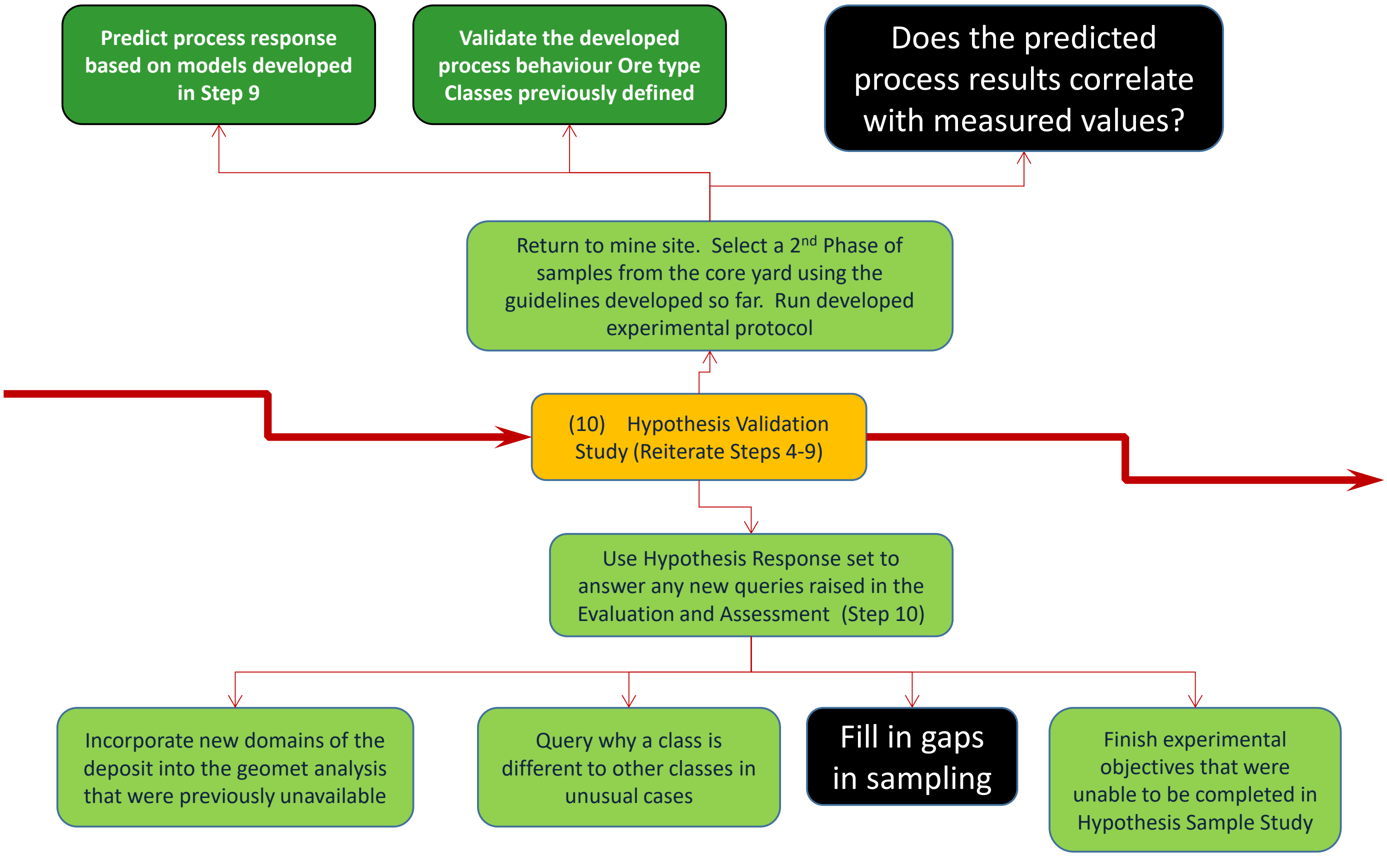


Figure 219. Hypothesis Validation of geometallurgical experimental aims (Image: Simon Michaux)

12 MAPPING STUDY PROCESS DOMAINING ACROSS DRILL CORE SECTIONS

One of the main objectives of a geometallurgy study is to determine zones of behavior for the target process in context of how it appears across the deposit (Liipo *et al* 2019). Ideally, the process signatures developed in Section 4 would then be projected onto continuous drill core is some form.

Once the Orientation Study is finished, a clear understanding of what small-scale tests, work in the target deposit is developed and planning of the experimental design of Mapping Study samples is able to be done.

The Orientation Study examined a target process behavior(s) in the 10 to 20 end member rock texture extreme samples (tightly constrained mineralogy across the whole sample of drill core). The purpose of the Mapping Study is to map out the variability of the process behavior being studied.

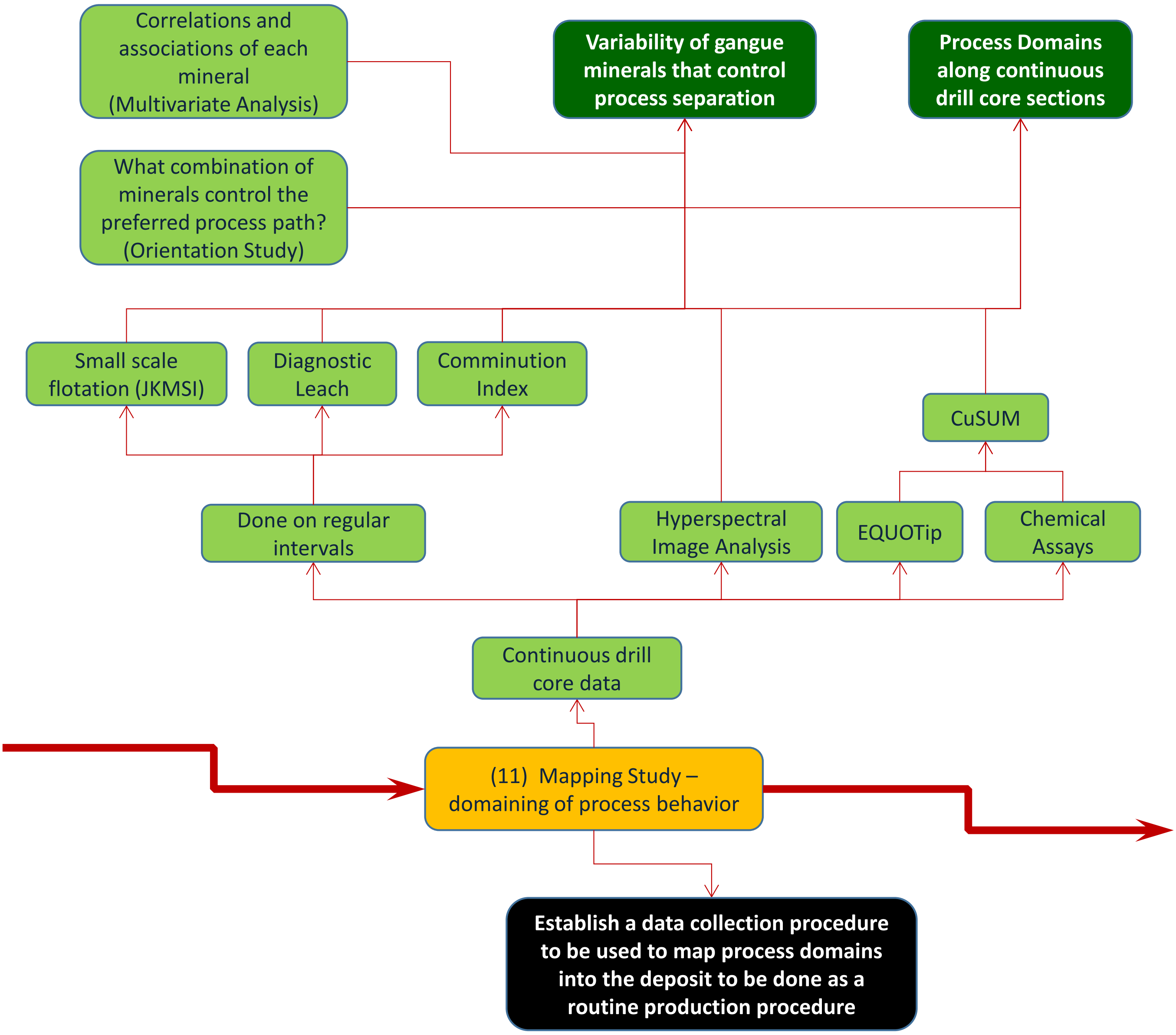


Figure 220. Mapping Study experimental design
(Image: Simon Michaux)

12.1 Geometallurgy Mapping Study Experimental Design

The basic steps to be considered for a Geometallurgy Mapping Study could be as follows:

1. Use the minerals that control and influence the chosen process path from the Orientation Study, as discussed in Section 7.

2. Examine each of the minerals in the chemical assay data base using statistical analysis. Determine the correlations and associations of each mineral. Use Principle Component Analysis.

3. Examine domains and variability of each of the minerals along sections of continuous drill core.

4. If needed, conduct extra experimental tests using the proxy tests discussed in Section 5.

5. Examine the combination of minerals that control the preferred process path together in context of domains and variability.

6. Establish a data collection procedure to be used to map process domains into the deposit to be done as a routine production procedure.

12.2 Mineral signatures that control process response

A successful outcome of the Orientation Study is an understanding of the mineral signatures that control a process response. All orientation samples can be examined to determine what mineralogy controls a given process response. For all orientation samples, the very good recoveries of flotation (for example) ranging to the very poor flotation recoveries can be ranked. Within this ranking, trends of what minerals (or combination of minerals) also trend. In doing so the mineralogy that controls flotation can be determined.

As such, a combination of minerals (usually both penalty and gangue) can now be used to guide the mapping study. In some cases, this has been done quite successfully, by using chemical assays done on drill core. It can even be possible that the client is already collecting the relevant data, which could then be used in a different context.

So the relationship between process path and minerals present could be mapped and modelled using chemical assays. This is a case by case possibility of viability. As rock texture often controls recovery, this method would not work in all situations.

12.3 Proxy test approach to mapping process domains

Establish proxy signatures that can be mapped with cheaper tests. Geometallurgy is the mapping of process behaviour into the deposit in a spatially defined domain fashion. The Orientation Study was to establish what mineral signatures controlled process behavior (Section 5). Often to achieve this, complex and expensive test work on a small number of samples is involved.

Firstly, establish which minerals control the preferred process path. This should be an outcome of the Orientation Study. Then establish if that process behavior can be mapped by a proxy test, or a proxy test in conjunction with something else.

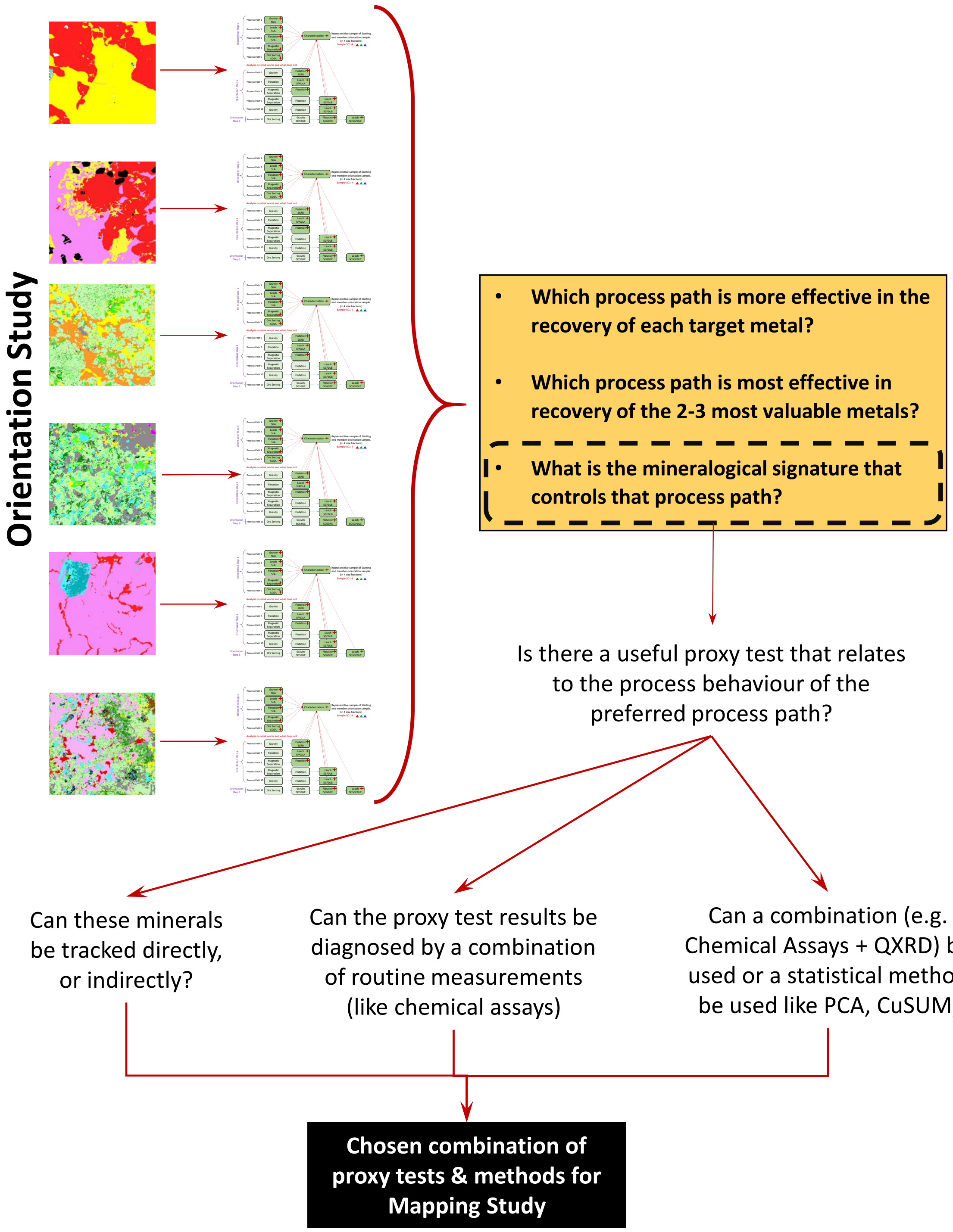


Figure 221. Choice of which small scale proxy tests and methods to be used in the Mapping Study (Image: Simon Michaux)

Another approach is to develop a working relationship between an expensive and complex test and a proxy test combination. In doing so, the signatures found in the more complex characterization tests could be mapped in greater numbers, enough to conduct a domaining study.

To develop for example, a relationship between the rock texture mineralogical signatures observed in the automated mineralogy tests and the less precise tests (e.g. QXRD and chemical assays) (Figure 222). In doing so a larger number of cheaper and less complex tests can be applied in the Mapping Study.

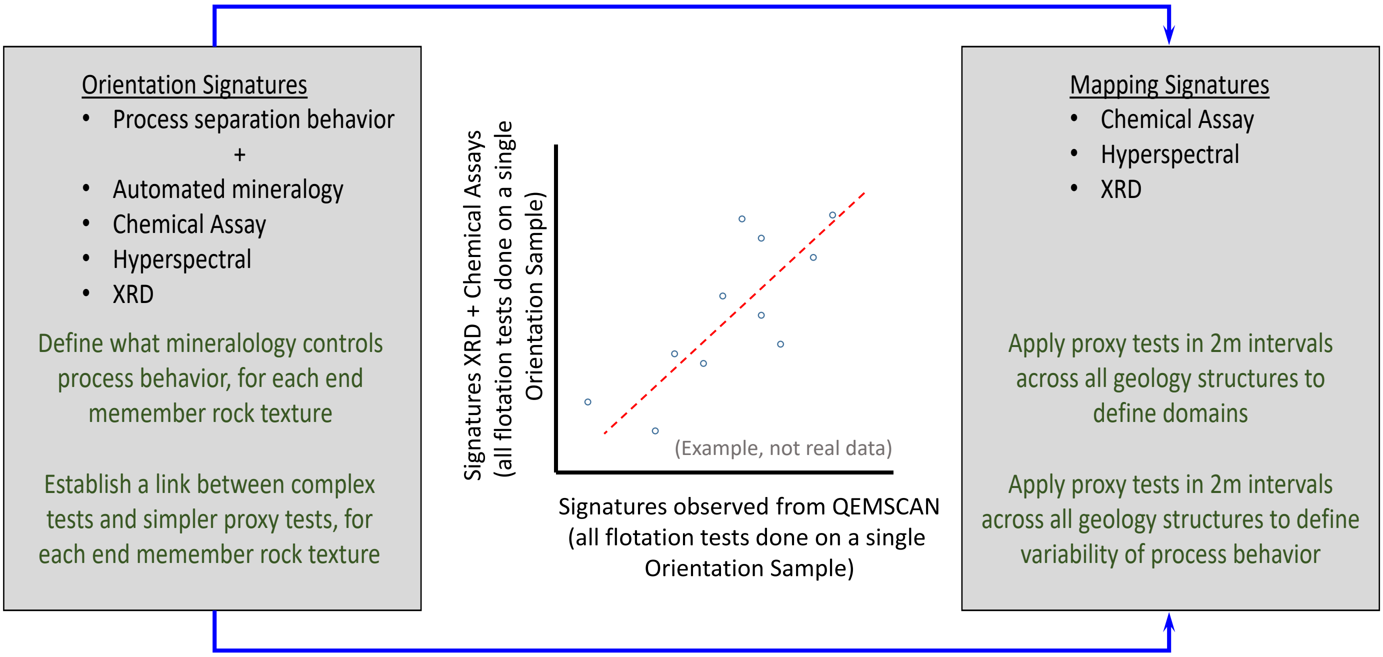


Figure 222. Develop a relationship between the mineralogical signatures observed in the expensive and time consuming tests and the less precise tests that could be used as a proxy for this ore type family (Image: Simon Michaux)

This method may be required if metal recovery is heavily influenced by the rock texture, and that texture is complex. It is to be considered if the process response can be modelled in this fashion at all. If the rock texture defining process response can only be measured by automated mineralogy, then the concept of a mapping study may not be viable.

The viability of the proxy model has to the bankable test is then established. In an ideal practice, many different proxy models developed in parallel to predict each measured/bankable process target. Once the form and structure of the proxy model is established, the next step is to constrain it using the mill circuit to identify the sensitivity required on measurements like BMWi and A*b. From this, identify the critical parameters relating to that target process that should be collected for that ore deposit system.

The quality of the outcome has to be balance against the cost of tests required vs accuracy of model, available budget and time pressure to complete the study.

12.4 Use of Proxy Test Outcomes to Map Process Domains

Once the most efficient proxy model is established, it can then be used in conjunction with other process target proxy models. Once those signatures have been measured and mapped in large enough numbers, domains of process behavior can be examined. What is to be attempted is the grouping of different process behaviors to a resolution that would change process engineering design, coupled with mineral character content.

Figure 223 below shows the outcome of two proxy models (Impact breakage for crushing and BMWi for grinding) to study the comminution footprint in context or ore characterization.

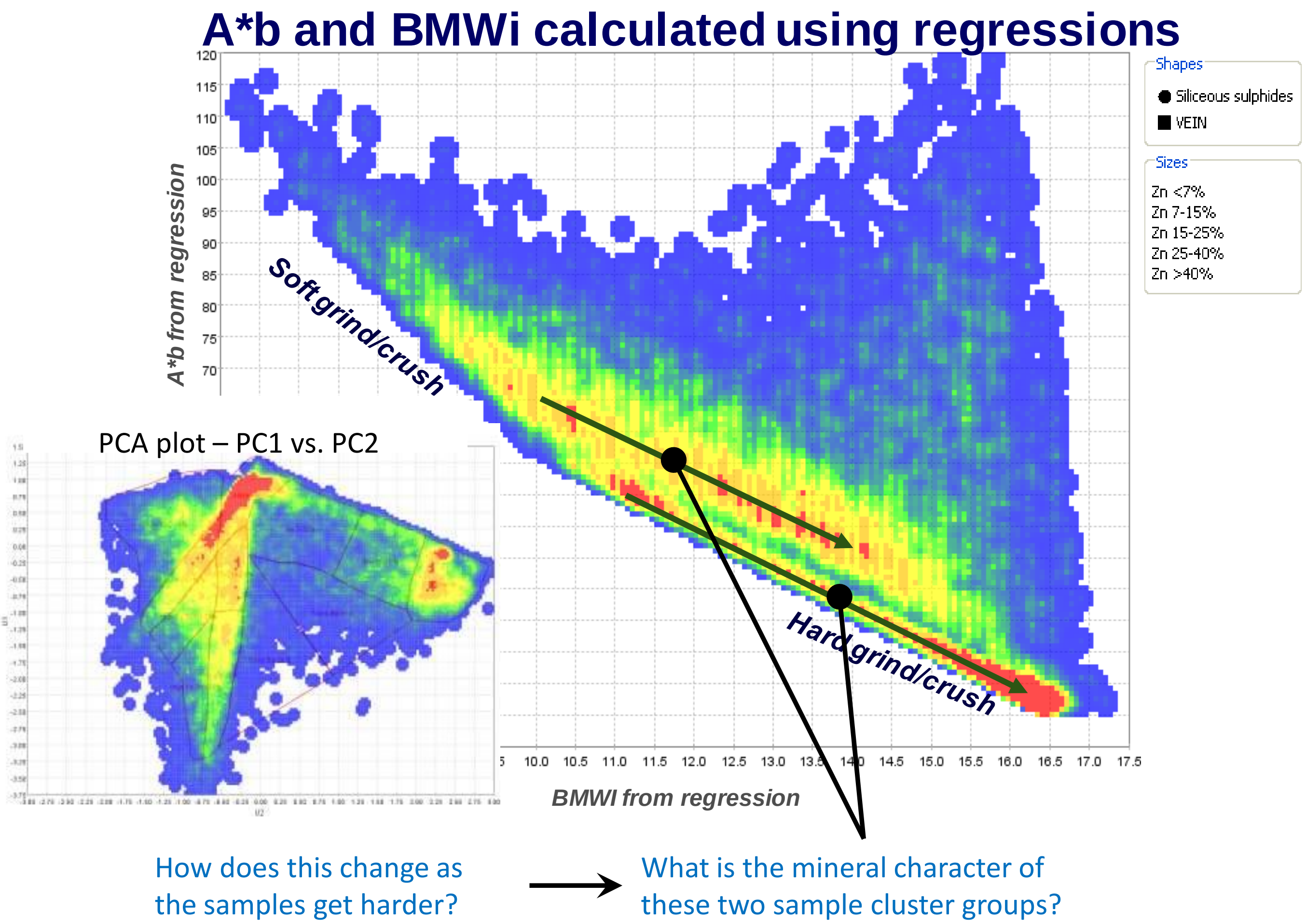


Figure 223. Class by class linked to comminution footprint (Aqqaluk ore deposit) (Source: AMIRA P843 2009, Copyright: AMIRA)

A useful data analysis to conduct is determining what mineralogy controls comminution footprint. It is understood that mineralogy controls most other mining processes. In comminution, it is the gangue mineralogy that dominates the outcome. As such QXRD is the preferred bulk mineralogical measurement tool. Figure 224 below shows an example of this being done successfully. The data can then be domained into a comminution footprint (two separate examples in Figure 225 and 226).

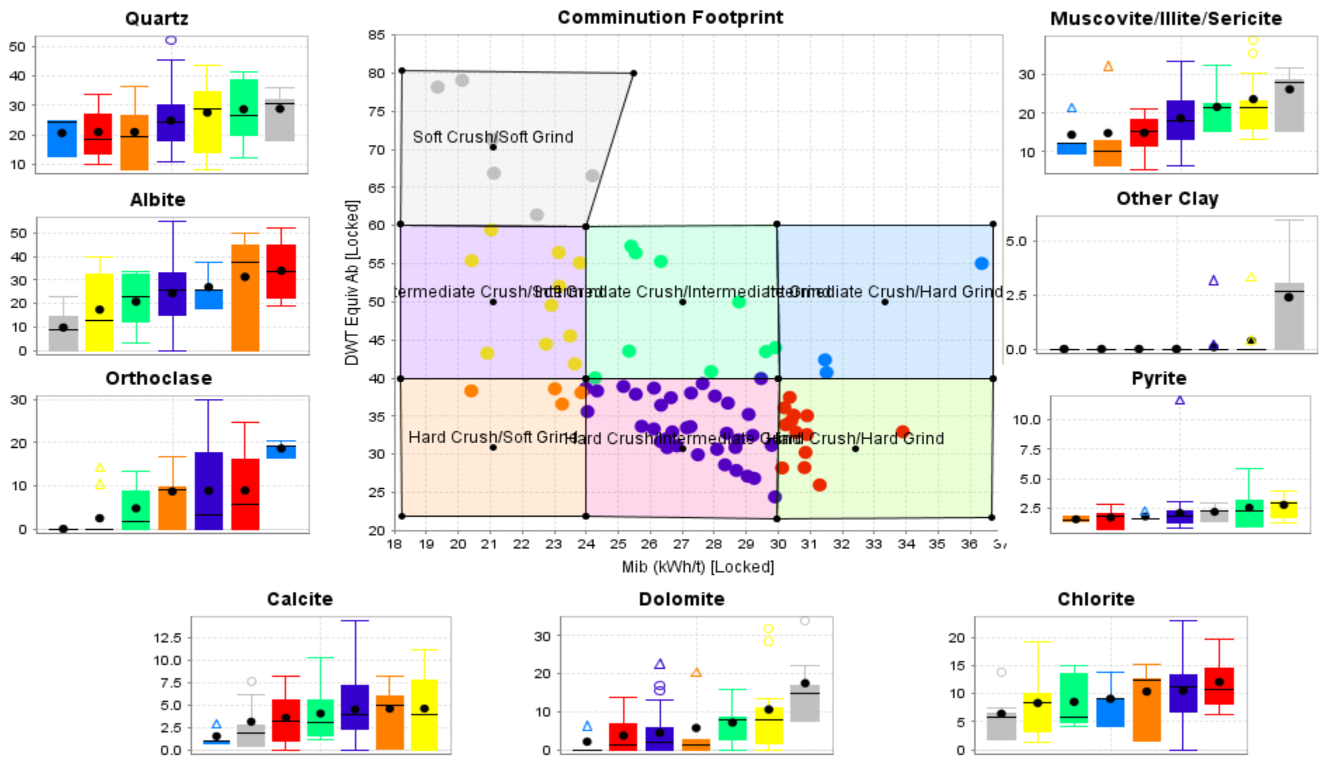


Figure 224. QXRD Mineralogy Linked to Comminution Performance
(Source: AMIRA P843 2009, Copyright: AMIRA)

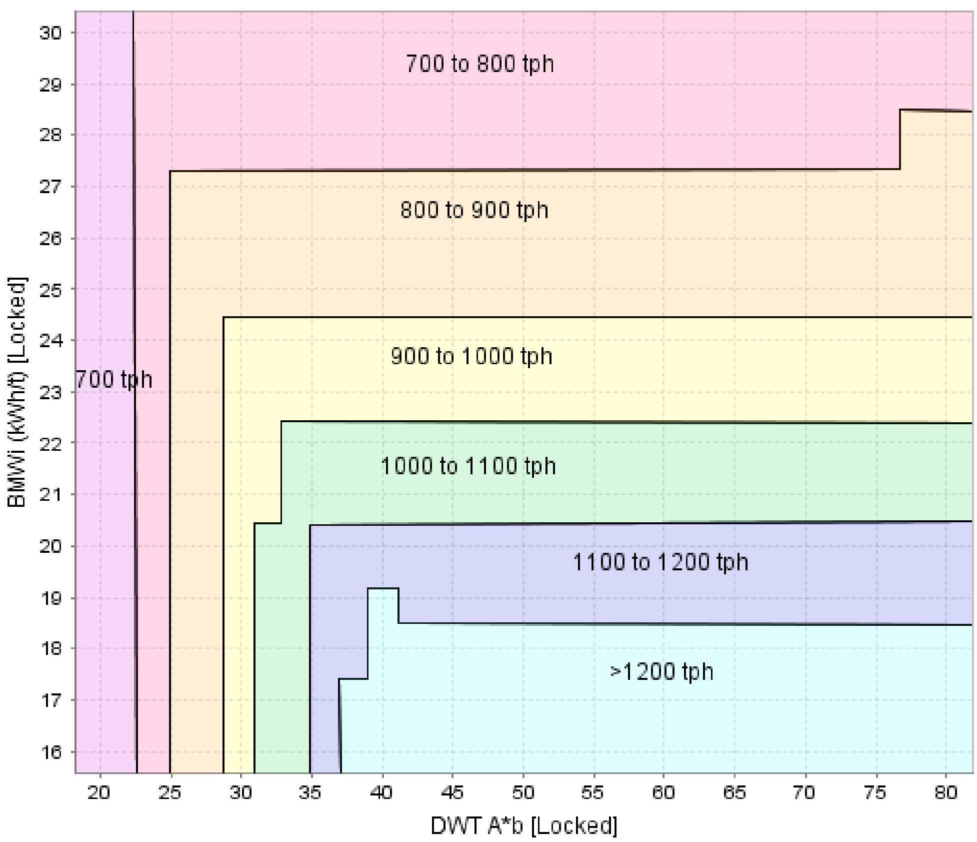


Figure 225. Comminution footprint in throughput space at a mine site in the Southern Hemisphere
(Source: AMIRA P843 2011, Copyright: AMIRA)

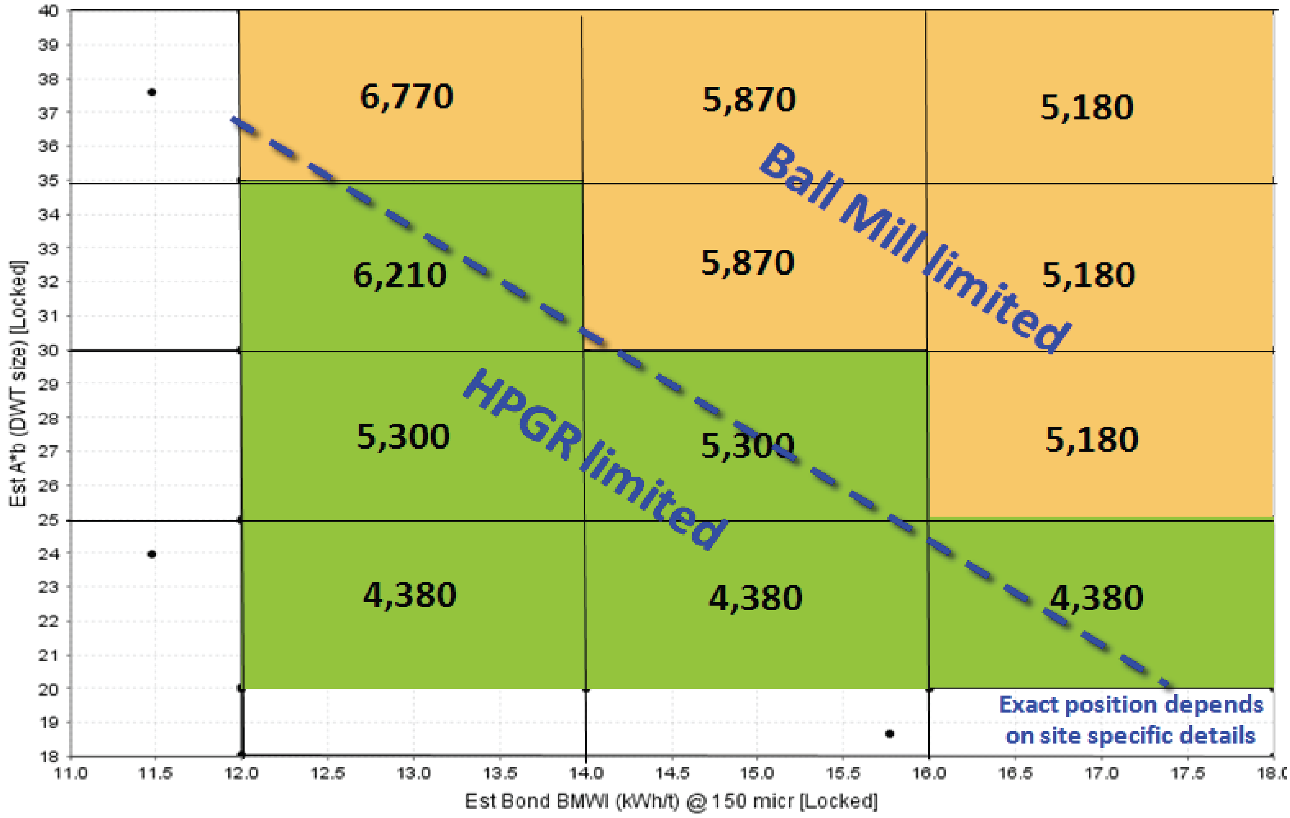


Figure 226. Comminution footprint domaining in conjunction with ore hardness derived throughput bottlenecks for a mine site in the Northern Hemisphere (Source: AMIRA P843 2011, Copyright: AMIRA)

12.5 Domaining behaviour and quantifying variability

Domaining behaviour and quantifying variability is now examined. If the 1500-2000m of core that form the source of Hypothesis Set were sampled correctly, not only the extremes should be mapped but several continuous sections across all major structures should appear in this data matrix.

The sorts of parameters that can be used in continuous measure that is useful in domaining is:

- Core imaging to group similar rock textures
- Chemical Assays done as a routine by the client already
    - valuable metals
    - gangue mineralogy
    - penalty elements
- Chemical Assays done as a new request as outcomes of the Orientation Study
    - to track minerals that control process response
- Lithology domains and rock alteration domains from site geologist
- CuSUM statistical method applied to chemical assays and other parameters
- Imaging of core (if possible, and if can be analyzed by the client)
- QXRD to give mineralogical distribution
- XRF pellet to give elemental distribution
- Hyperspectral drill core characterization
- EQUOTip
- SMC (if done in enough samples)
- Small scale flotation test
- Small scale diagnostic leaching tesst

Once a decision has been made in how to approach the mapping in context of useful proxy tests, develop a smaller scope testing regime to be done consistently on each Mapping Study sample. Run this set of small scale cheaper tests on the rest of the samples (of the order of 100 to 500 measurements) to map variability and domains. This is often called the Mapping Study (Figure 227).

Another critical decision to be made after careful consideration is the core length interval to be used for each sample. Each Mapping Study test regime will require a sample mass to be effective. Ideally this should be kept as small as possible to allow the targeting of genuine variability. Chemical assays are often done at 2m intervals (sometimes 4m). Usually the mass in these intervals is more than enough.

The mapping study can be approached in context of continuous measurement. That is, all chemical assay intervals over a continuous section of core. Ideally this section of core would transverse major geological boundaries and/or geological structures. Selection of these drill core sections should be done in context of collaboration with a geologist that has worked with the deposit and has excellent knowledge of the deposits unique characteristics.

It is also possible to have a selection of samples that are not continuous but are consistent in spatial positions. For example 1 chemical assay interval in 10 intervals are sampled. This is a decision to be made by the geometallurgical team in context of campaign objectives and project budget.

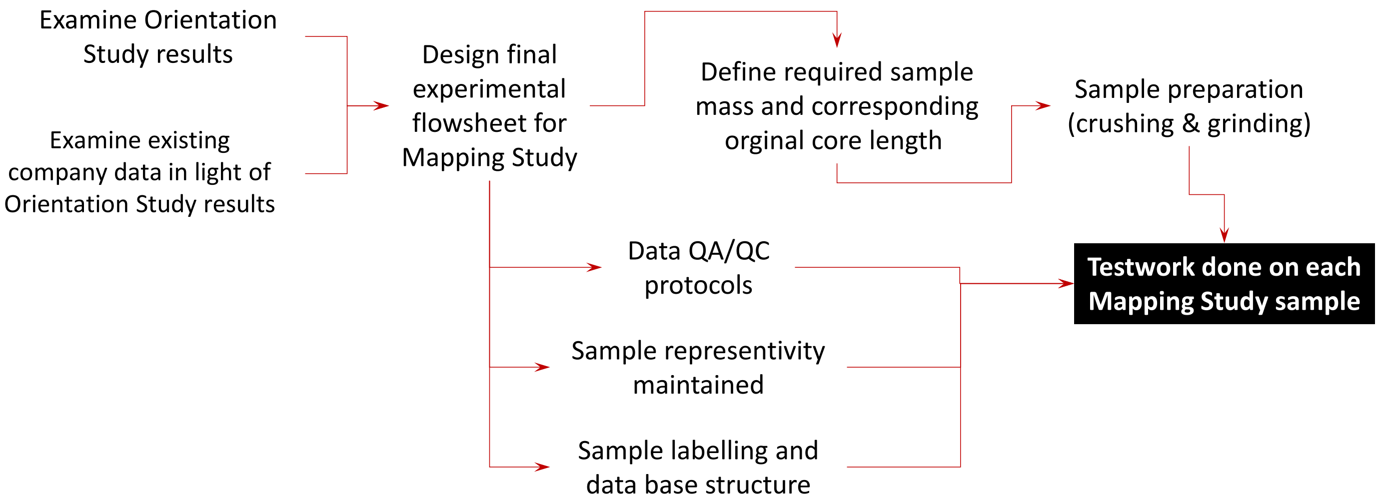


Figure 227. How the Orientation Study relates to the Mapping Study
(Image: Simon Michaux)

Collect the data from the Mapping Study. Put the results with the context of, where each test is defined spatially (drill hole + depth). Assemble all data in a downhole context for each sample. Enter the data into the developed geometallurgical matrix (see Section 7 & 8).

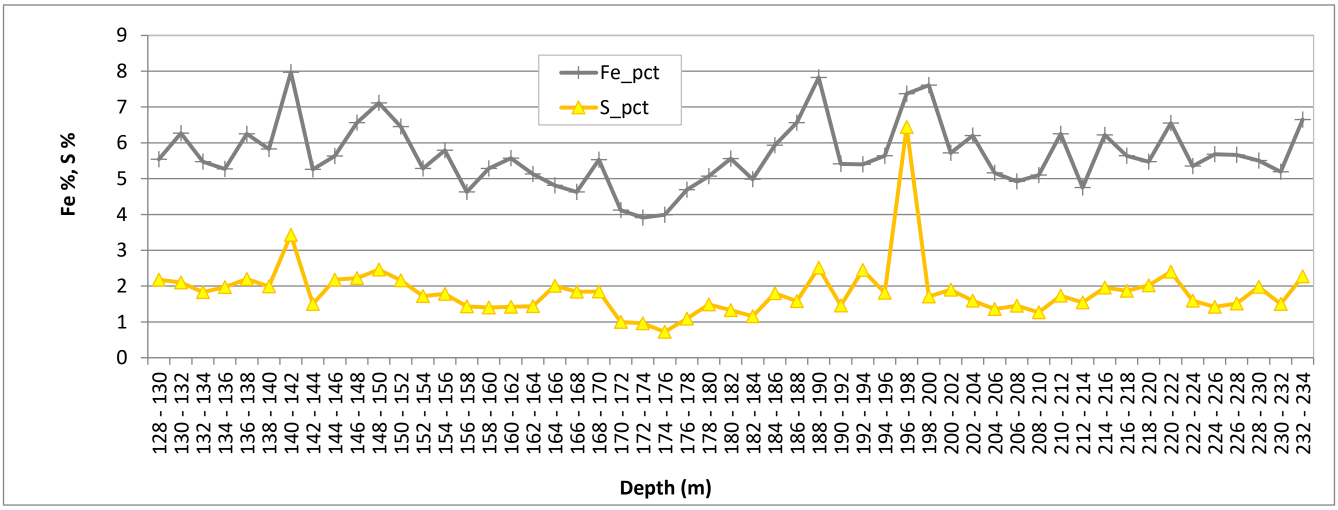
Use cross correlation of different data types to define domains of process behavior. In doing so, process defined domains can be diagnosed and process response variability can be quantified.

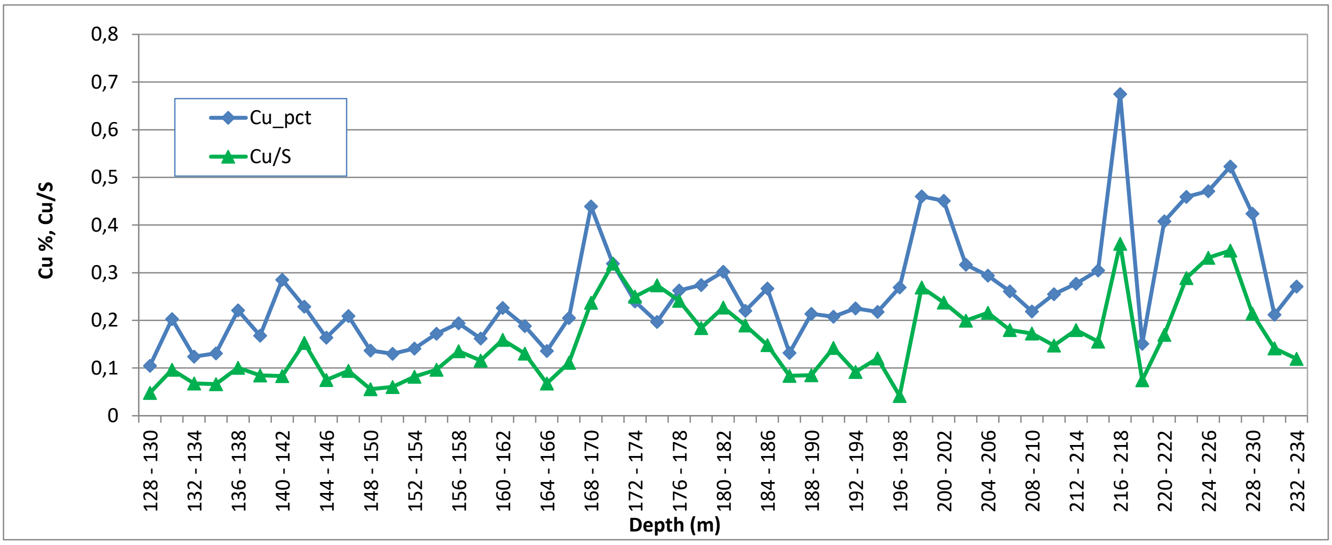
The objective here is to put all available data in a continuous downhole fashion in a cross-correlation form. In doing so, domains of behaviour can be examined, although in this case, the target context for domaining is the process behaviours. Comminution and flotation often have very different domains, because one being physical in nature (comminution is driven by stress and strain applied to meso texture scale rock) and the other is chemical in nature (flotation is driven by surface chemistry of micro scale particles). Thus, it is usual to domain these process behaviors separately and accept that they may have different spatial signatures.

12.6 Chemical Assays and CuSUM statistical method

In terms of examining continuous downhole data, chemical assays are an excellent start as this data is collected as routine across the entire deposit and will continue to be the case during operation (AMIRA P843 2009, AMIRA P843 2011). It is to be remembered though that chemical assays are not the same as minerology, but a proxy for mineral content. Figure 228 shows an example of a continuous section of drill core from Case Study P with several assays in a downhole format.

Figure 229 shows the same data as Figure 228 but with the CuSUM statistical tool applied to the same chemical assays. As can be seen, the CuSUM tool is excellent and highlighting different domains from data that seemingly has no structure. It is to be remembered, it is the gradient of the CuSUM line, where other parts of the curve of the same gradient are very similar are considered to be the same statistical domain for that parameter. This would be a parameter to study in context of the target process behavior. Different minerals (and their proxy chemical assays) will influence different process behaviors. Comminution for example is influenced by gangue mineralogy, where magnesium, aluminum, potassium and calcium are all of note. Flotation is generally influenced by different minerals and at trace element concentrations.





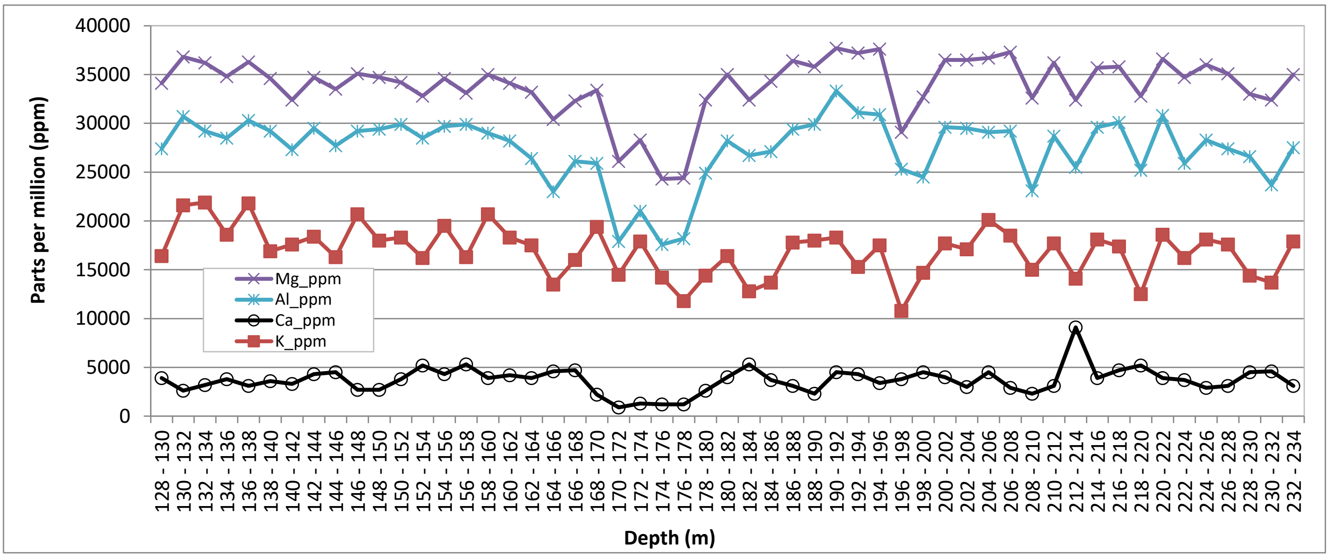


Figure 228. Chemical Assay's (Case Study P) (Simon Michaux)

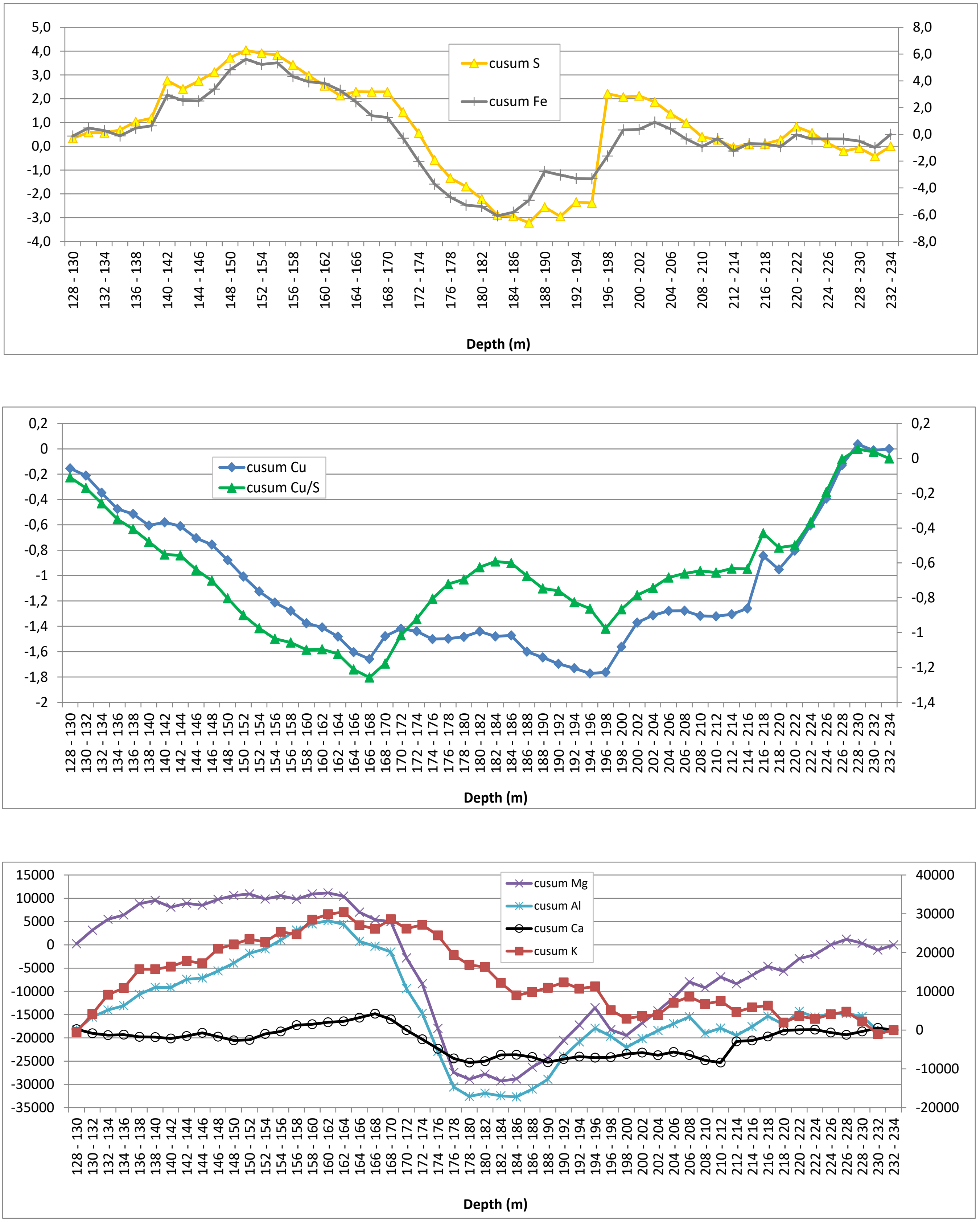


Figure 229. CuSum on Chemical Assay's (Case Study P) (Simon Michaux)

The objective here is to put all available data in a continuous downhole fashion in a cross-correlation form (Figure 230). In doing so, domains of behavior can be examined, although in this case, the target context for domaining is the process behaviors. Comminution and flotation often have very different domains, because one being physical in nature (comminution is driven by stress and strain applied to meso texture scale rock) and the other is chemical in nature (flotation is driven by surface chemistry of micro scale particles). Thus, it is usual to domain these process behaviors separately and accept that they may have different spatial signatures.

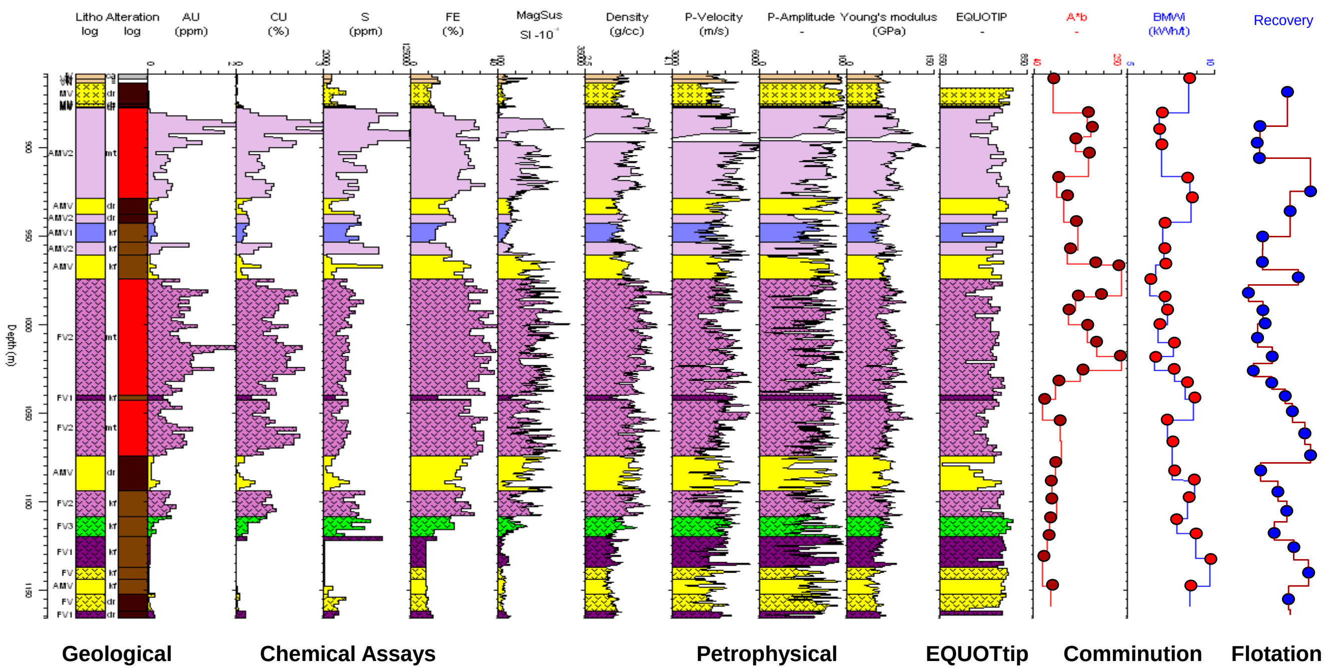


Figure 230. Cross correlation of different data types to define domains of behavior (Source: Amira 2008b, Copyright: AMIRA)

13 ASSESSMENT OF GEOMETALLURGY CAMPAIGN

"A good reality check can not only save your career, it can elevate it in the face of adversity"

S. Michaux – some time ago

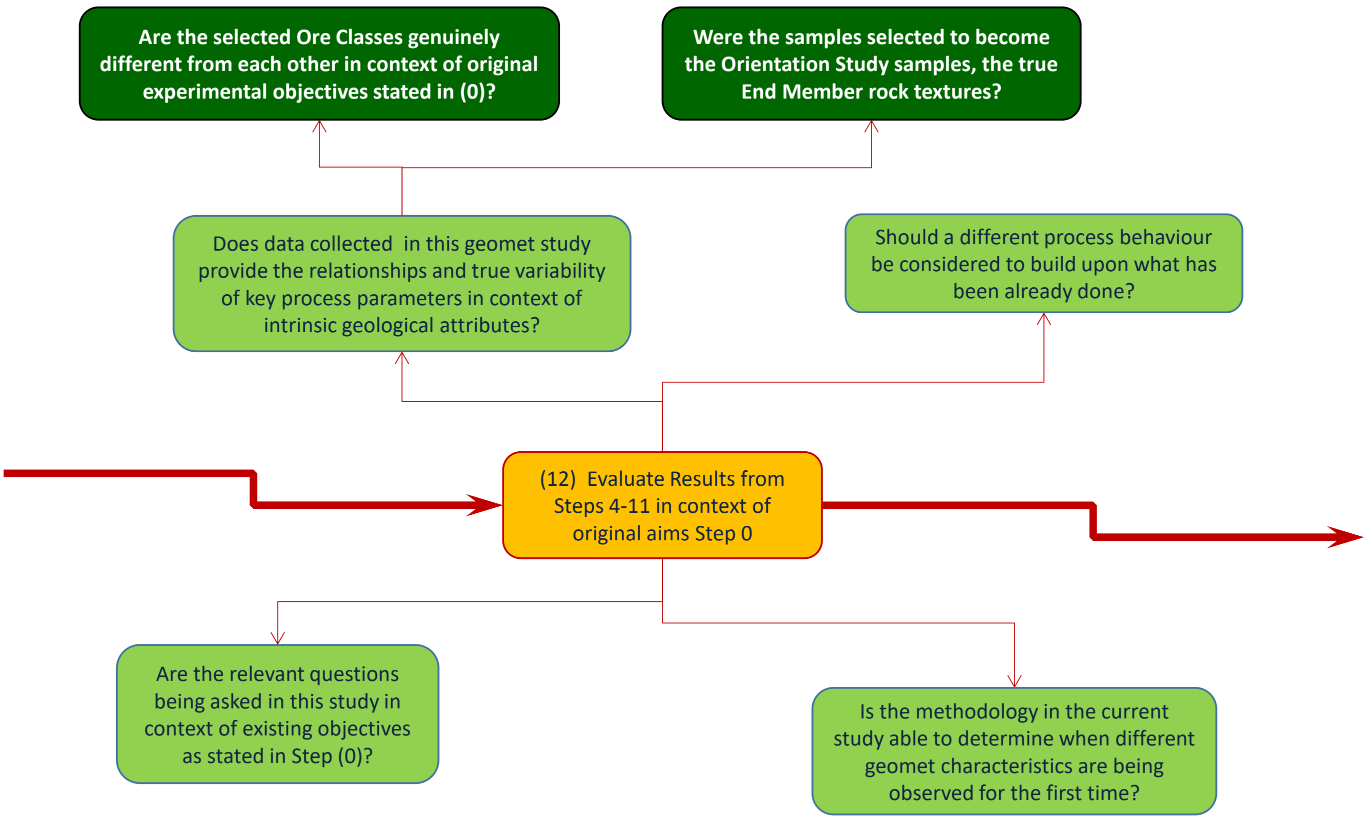


Figure 231. Assessment of study

(Image: Simon Michaux)

13.1 Assessment for potential sampling gaps in the study

Once the domaining Mapping Study has matured to the point where statistical structures in the deposit are understood spatially, examine whether the process test samples are truly representative. The multivariate statistical analysis, the PCA analysis and the domaining study should give a good understanding of the data set. The Orientation Study also should show what mineralogy is important in controlling mineral separation through the preferred process path.

This combination of data and knowledge has the capacity to assess if:

- Whether the samples taken for process separation test work are representative in the range for the relevant minerals.
- It is at this point where it can be asked whether the original end member texture Orientation Samples selected, genuinely were the end member textures of the deposit.

Figures 232 and 233 show an example of this with Case Study P. As can be seen, samples collected for the comminution test SMC do not have a representative range in the gangue mineralogy that later was shown to influence breakage. The Case Study P sampling campaign was based on the requirement to collect samples across the full range of valuable target metal (in this case Au and Cu). Gold at a grade in the range of parts per million will have little influence on the breakage of meso-scale particles. In Case Study P, it was later shown that the calcium content and the Fe/S ratio were important in defining different ore classes.

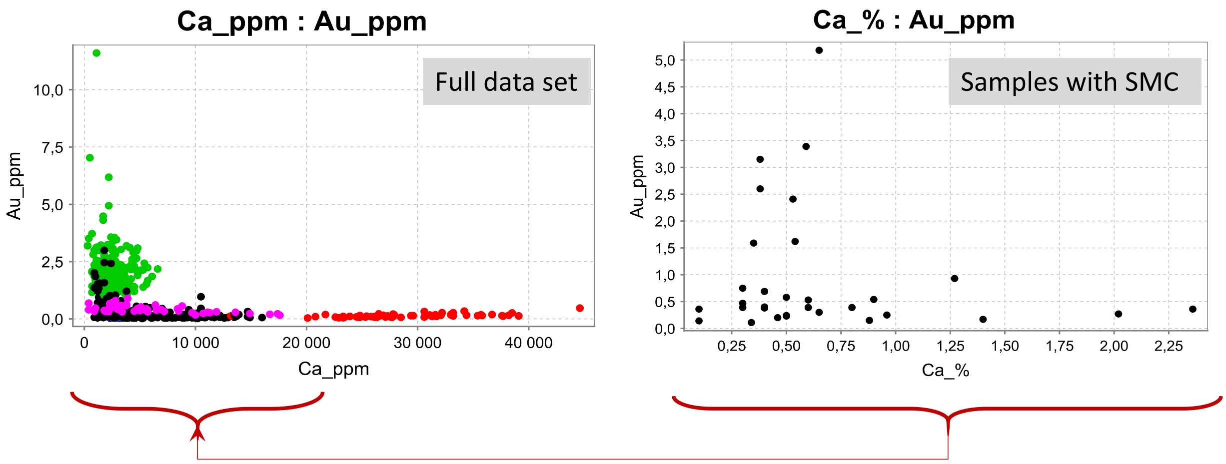


Figure 232. Gaps in sampling for comminution SMC tests in the calcium range (Case Study P) (Image: Simon Michaux)

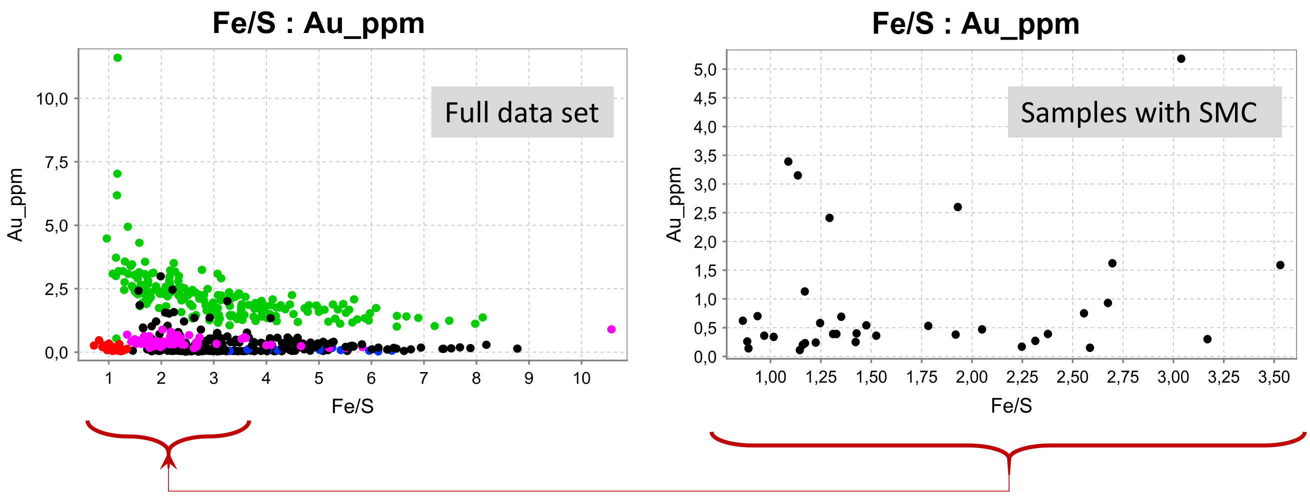


Figure 233. Gaps in sampling for comminution SMC tests in the Fe/S (Case Study P) (Image: Simon Michaux)

If there are gaps in the sampling (only look at the controlling minerals that control/influence the preferred process path), then draft up a mineralogical signature profile of what characteristics of the missing samples might have. Then go back to the client and together with their geological staff, use this signature profile to find a section of drill core and collect new samples. Make the judgment call whether only a small amount of testwork is required, or would the new samples require the full spectrum of tests as done on the Orientation Study.

13.2 Assessment on whether the study addressed the original experimental aims and hypotheses

At regular points during the geometallurgy campaign, do a reality check on what the data outcomes are reporting in context of the original campaign objectives (see Section 2). The approach of honour what the mineralogy is telling you is very important. A geomet study is often a trail blazing exercise into the unknown.

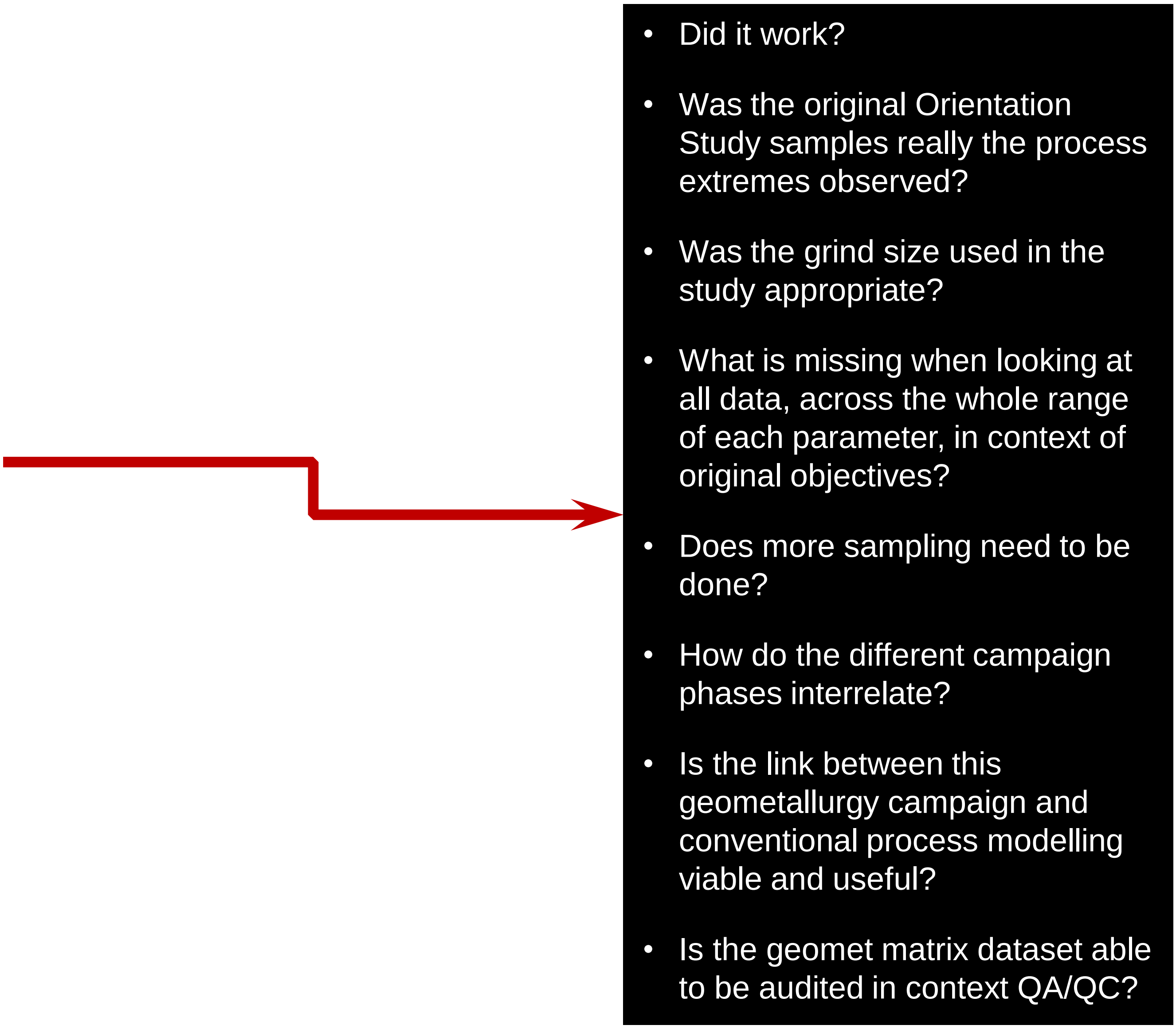


Figure 234. Finish of the geometallurgical campaign
(Image: Simon Michaux)

14 POSSIBLE USES OF THE GEOMETALLURGY OUTCOMES

A geometallurgical campaign can be expensive and take time. Once this is done, however, the data and knowledge collected can be used as support for further tasks with great effect.

14.1 Use of Geometallurgical Analytical Outcomes to Define Sampling Strategies for Full Scale Laboratory Testing for Process Design

The conventional approach to ore characterization was to collect a number of samples and send them off for metallurgical testing. These samples were sometimes collected using the advice from the site geologists and sometimes not. Most of the time, little or no mineralogy characterization was done on those exact samples. Knowledge of how those samples can be related back to the ore deposit was often vague. Often samples were a blend of several ore types to make up the required mass.

Part of the geometallurgy paradigm approach is the identification and selection of what is termed end member orientation samples, which are to represent the rock texture extremes of the deposit in context of a process response behaviour. If this is done correctly, they should represent the extremes to be observed for a target process test. If the orientation study is done successfully, the mineralogy that controls process performance would be understood.

Also part of the geometallurgy paradigm approach is the domaining of process behaviour. This, if done successfully should show what parts of the deposit will respond to a given process separation or liberation.

The combination of these two concepts could be used to select larger scale samples for conventional bankable tests (tests that are accepted by the mining industry in a due diligence context).

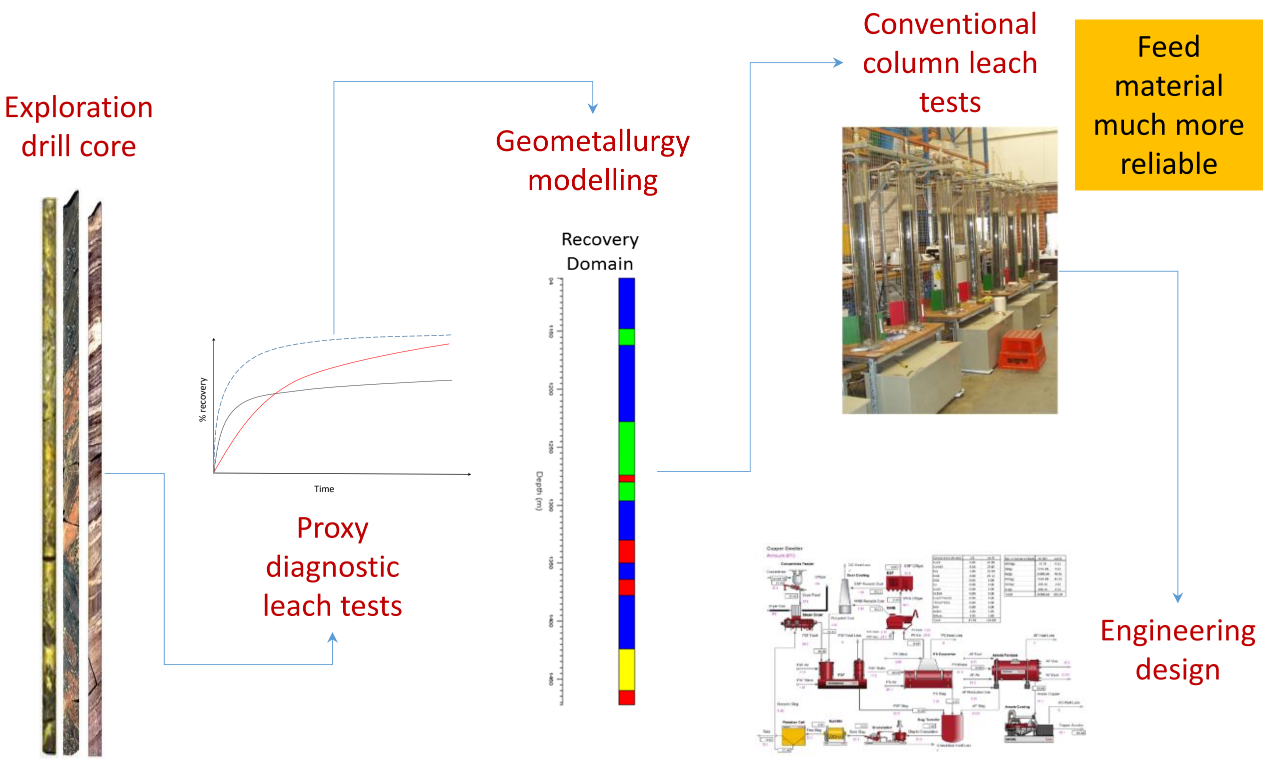


Figure 235. Using a geometallurgy study to select feed samples for a bankable larger scale test campaign (Image: Simon Michaux)

The mineral signatures that control the process test in question could be used as a guide in sample selection to ensure that the ore collect is of the same and only the same rock type texture. Sample dilution could introduce variability without being quantified. In doing so, samples for bankable tests could be meaningfully selected to establish what the true range of process performance would be.

The same ore type signatures could be used to create blends of samples. In doing so, all the test extremes could be quantified as well as a series of blends could be tested in a controlled fashion.

14.2 Mine and Process Plant Design

Mining and extraction of ore deposits is becoming more complex with higher risk over time. The choice of what process path is not as straight forward as it has been in the past. In some cases, there is multiple open pits, and multiple process paths in the same operation, resulting in much higher CAPEX and OPEX. How to manage the extraction of ore in a planned fashion for a complex operation like this requires a more sophisticated feasibility study process.

To design and then operate such an operation, disciplined and sophisticated ore characterization is needed. Much more has to be put into initial ore body knowledge and risk mitigation in design due to scale and complexity. Measurement based efficiency of each individual process in context of a family of ore types is required to support decision like which process path is more appropriate. Table 13 and Figure 236 show a theoretical example in how this could be used to design parallel process paths and decide what parts of the deposit to send to each plant. This is could be required in future, as opposed to a vague one recovery target for the whole process circuit for all ore types that is rarely observed in practice.

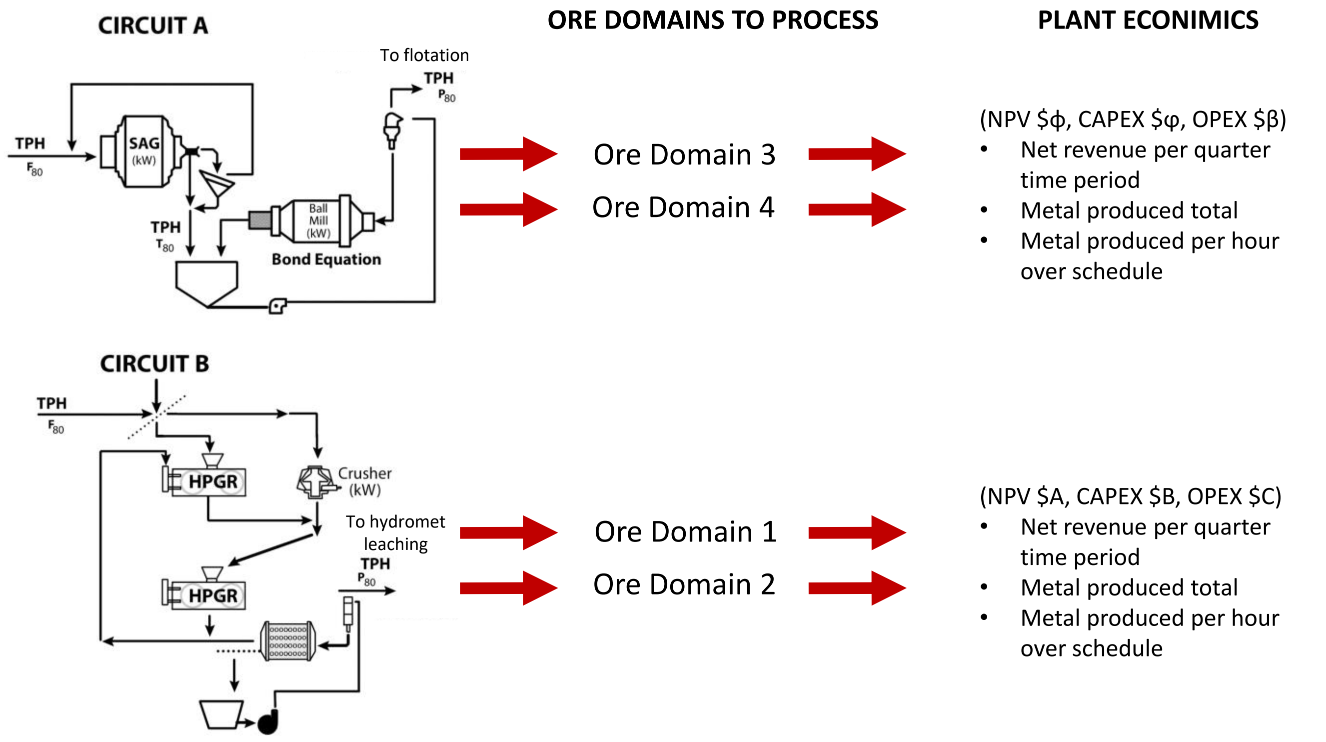


Figure 236. The choice of possible process path for different parts of the deposit – theoretical example linked to Table 11. (Image: Simon Michaux)

Table 13. A geometallurgy study can domain the deposit in context of process performance

Process Attribute Ore Value	Ore Domain 1	Ore Domain 2	Ore Domain 3	Ore Domain 4
Valuable metal 1 (Au) grade	0.06g/t			1.1g/t
Valuable metal 2 (Cu) grade	1.20%			0.02%
Valuable metal 3 (Ag) grade		2.2g/t	1.3g/t	
Valuable metal 4 (Mo) grade		0.96%	0.47%	
Valuable metal X (?) grade				
Penalty elements	Yes (High As content)	No	No	Yes (Low F content)

Process Attribute Ore Characterisation	Ore Domain 1	Ore Domain 2	Ore Domain 3	Ore Domain 4
Mineral Liberation Size	75 micron	5 micron	150 micron	212 micron
Clay and sheet silicate content	High	Medium	Low	Low
Ore sorting feasible?	Yes	No	No	No
Ore conditioning feasible?	No	No	No	Yes
Impact breakage (Axb) ranking	21	19	45	75
Low energy abrasion energy (ta)	0.35	0.41	1.6	2.1
Bed breakage propensity (outside SAG comfort zone)	Yes	Yes	No	No
Grinding energy (BMWi) ranking	22	20	13	11
Fine grinding needed?	No	Yes	No	No

Process Attribute Separation Process	Ore Domain 1	Ore Domain 2	Ore Domain 3	Ore Domain 4
Flotation recovery	Poor	Average	Good	Good
Flotation kinetics	Poor	Average	Good	Good
Leaching recovery	Average	Poor	Good	Average
Leaching kinetics	Average	Poor	Good	Average
Gravity Au recovery	Yes			No
Pressure oxidisation required for Au recovery	No			Yes

Process Attribute Engineering Design Request	Ore Domain 1	Ore Domain 2	Ore Domain 3	Ore Domain 4
SAG Mill			Yes	Yes
HPGR	Yes	Yes		
Ball Mill	Yes	Yes	Yes	Yes
Fine Grinding Mill		Yes		
Flotation Circuit			Yes	Yes
Leach Dump	Yes			
Leach Heap				
Leach Tank	Yes			
CiL Carbon Leaching circuit		Yes		Yes

Process Attribute Environmental impact	Ore Domain 1	Ore Domain 2	Ore Domain 3	Ore Domain 4
Acid Mine Drainage Potential	High	Medium	Low	High
Dust Generation Potential	Low	High	Low	Medium
Water Pollution Potential	High	Medium	Medium	High
Impact of Flora & Fauna Potential	Low	High	Low	Low
Mine Site Rehabilitation Requirements	High	Medium	Low	High

Choices like what are shown in Table 13 and Figure 236 can be used for design. There are several options. Ore can be mined into multiple stockpiles, each feeding an optimized process path with a unique comminution closing size and cutoff grade (Figure 237).

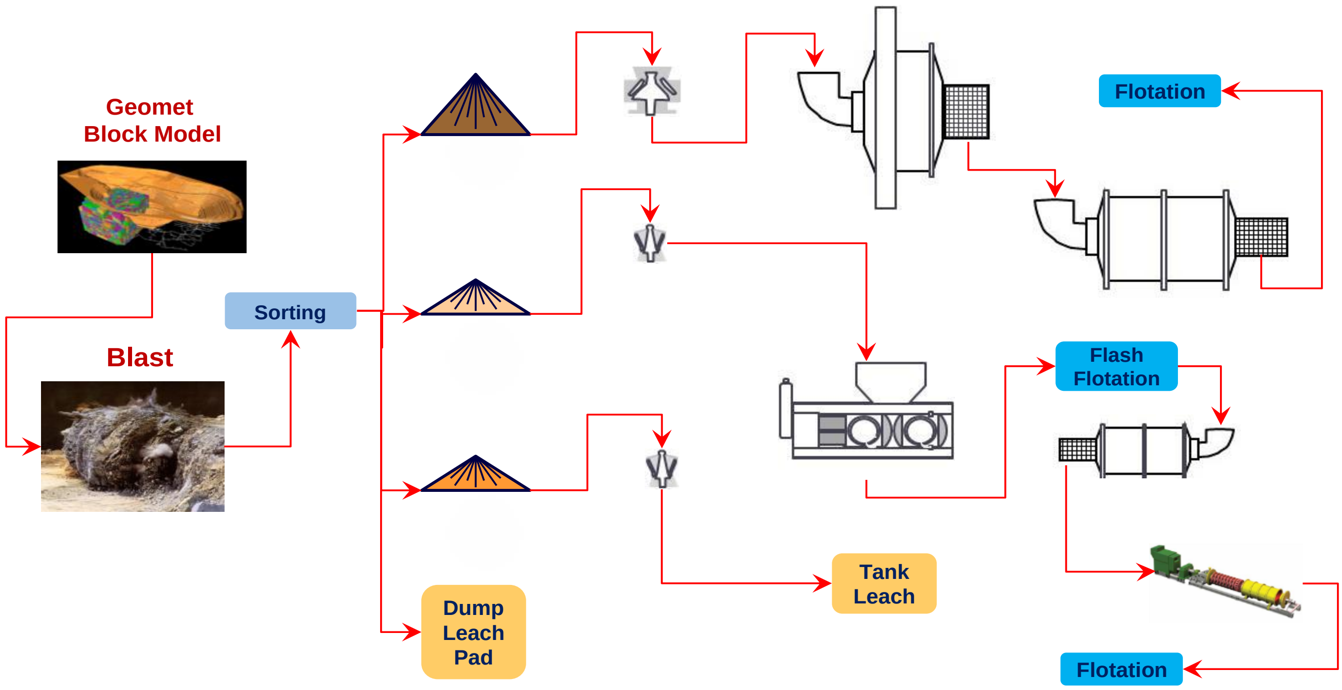


Figure 237. Multiple stock piles, feeding multiple process plants
(Image: Simon Michaux)

Conventional process plants are designed to operate at a steady state throughput to a predicted average recovery. Technology has advanced greatly in the last few decades, which now allows a more sophisticated complexity in engineering. It is now conceptually possible to engineer a process path that is designed to operate to a dynamic throughput conditions. Variable throughput that has the capacity to keep ore for a longer residence time in the process plant, or for a shorter time (high grade softer ore should be kept in the process plant for as long as possible to increase recovery, low grade harder ore should be pushed through as fast as possible). This is only possible with more sophisticated ore knowledge, where the process response of different ores is known. What is also required to be known is what ore is entering the plant from the stockpile (using ore tracking technology). Operational protocol needs to be developed accordingly.

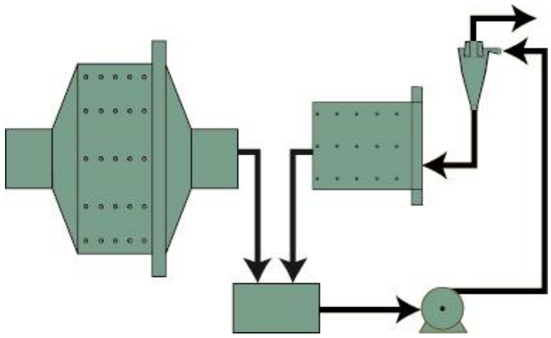


Figure 238. One process path that is heavily instrumented and optimized for a dynamic feed
(Image Copyright: JKMRC)

Sorting technology can be engineered but is difficult to justify economically. A technological breakthrough is considered to be likely. If this does happen, then sorting could be used to greatly reduce what volume of ore needs to be sent to grinding (the highest mining cost), for a similar metal production to all the ore being processed (Figure 239). This will only be effective if ore sorting response is understood.

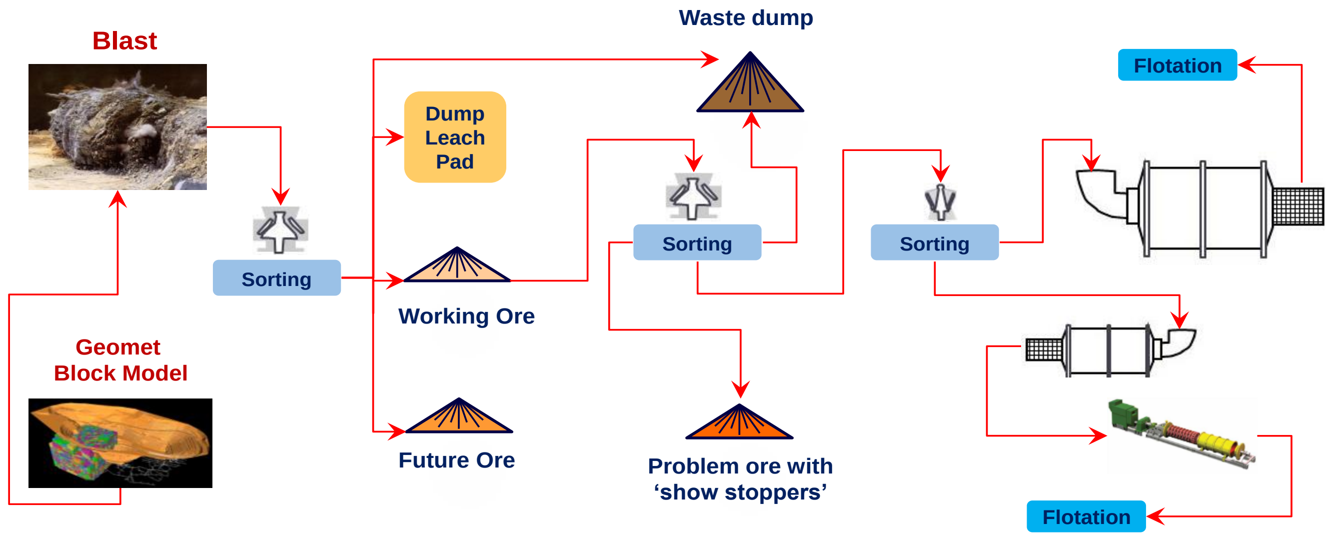


Figure 239. Sorting technology being integrated into process design
(Image: Simon Michaux)

14.3 Geometallurgical Block Model Generation

The development of a block model with geometallurgical attributes has been the ultimate objective to many geometallurgical research and developments. Some of the outcomes of the Geometallurgical Mapping Study could be incorporated into a block model (Figure 240).

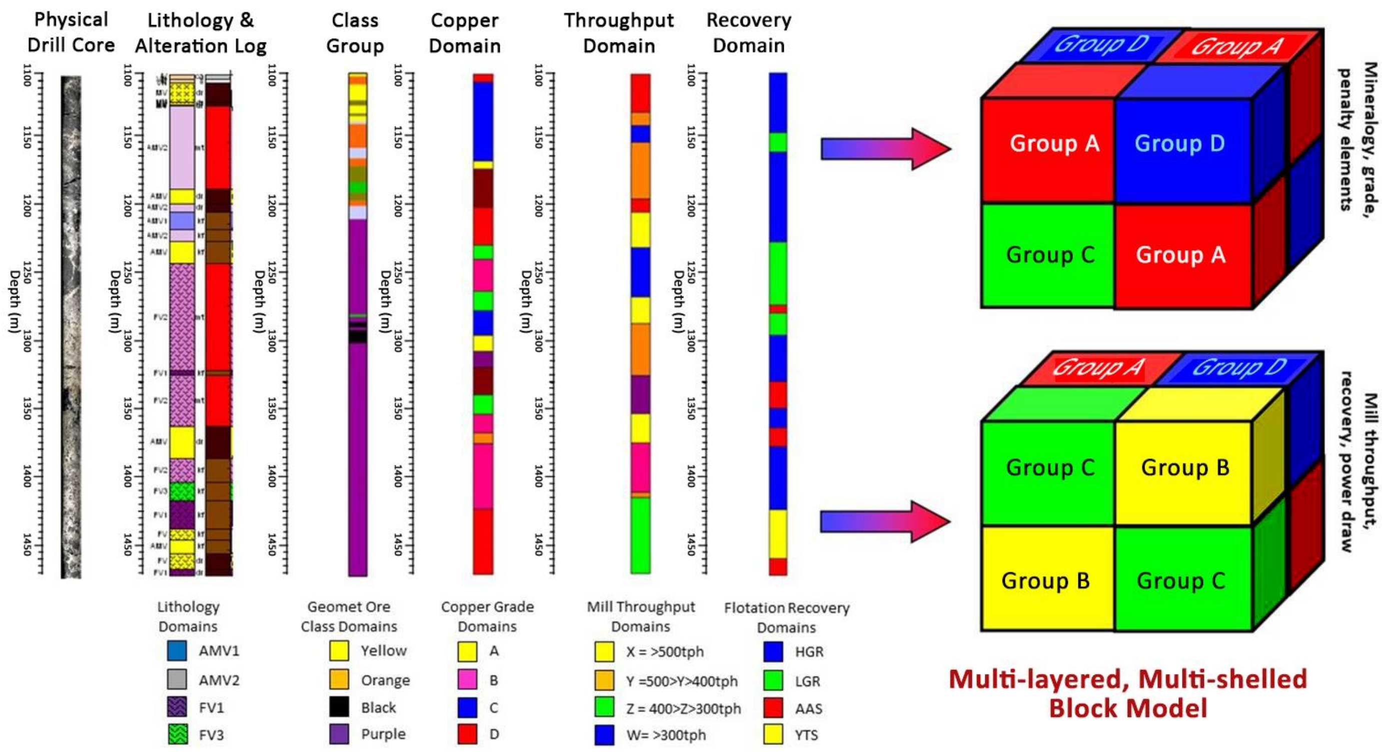


Figure 240. Classes and domains integrated into a block model
(Image: Tania Michaux)

There is a very difficult challenge to this idea however. For the geostatistical kriging to be successful, the attributes modelled are required to be 'additive'. This means that experimental measurements of an attribute could be averaged in a spatially appropriate fashion to predict what that attribute might be at a theoretical point between them.

Most metallurgical attributes are not additive, due to how they are calculated. For example the BMWi and the A*b impact breakage parameter have been shown to be probably not additive. Figure 241 shows an example where the Comminution Index done continuously on a 161m section of drill core. This example is done where the measured particle size fractions are summed together for several interval sections (2m, 4m and 8m).

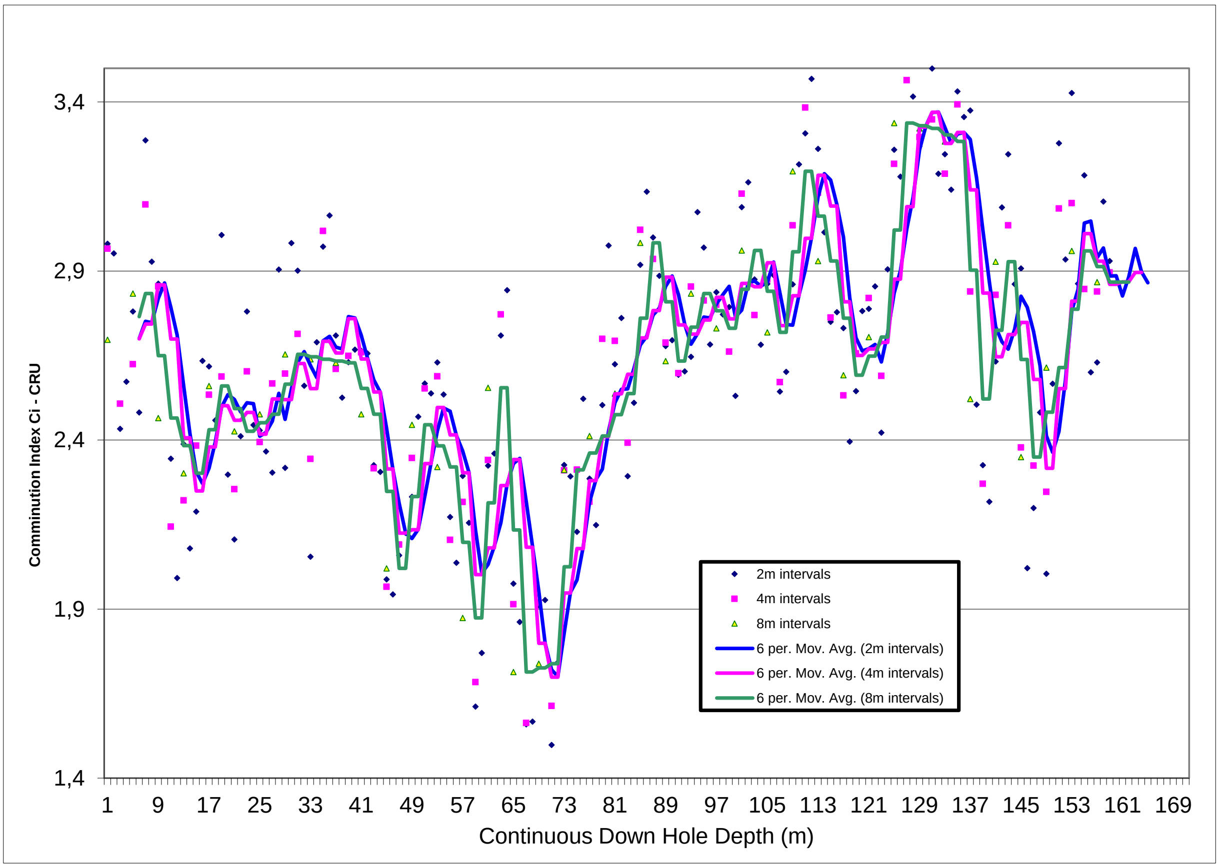


Figure 241. Comminution Index done continuously on a drill core section, calculated on several interval lengths, from an IOCG style Au and Cu deposit in the Southern Hemisphere
(Image: Simon Michaux)

This suggests data used to calculate the Comminution Index may well be additive when using size fraction data summed together. This implies that while process test results like BMWi and A*b are not additive, the supporting size fractions could be.

Current state of knowledge of how to build a genuinely additive block model with process attributes is not developed enough to be considered reliable enough to use. Future research and developed may resolve these difficulties.

14.4 Economic Modelling and Business Model Development

The true value of this kind of integrated data is how to develop the business model. To do this, the integration of these diverse attributes into more effective mine planning and optimization models is embodied in the cross-discipline area of geometallurgy. Figure 242 shows the tasks that are needed to be done to develop an economically viable operation.

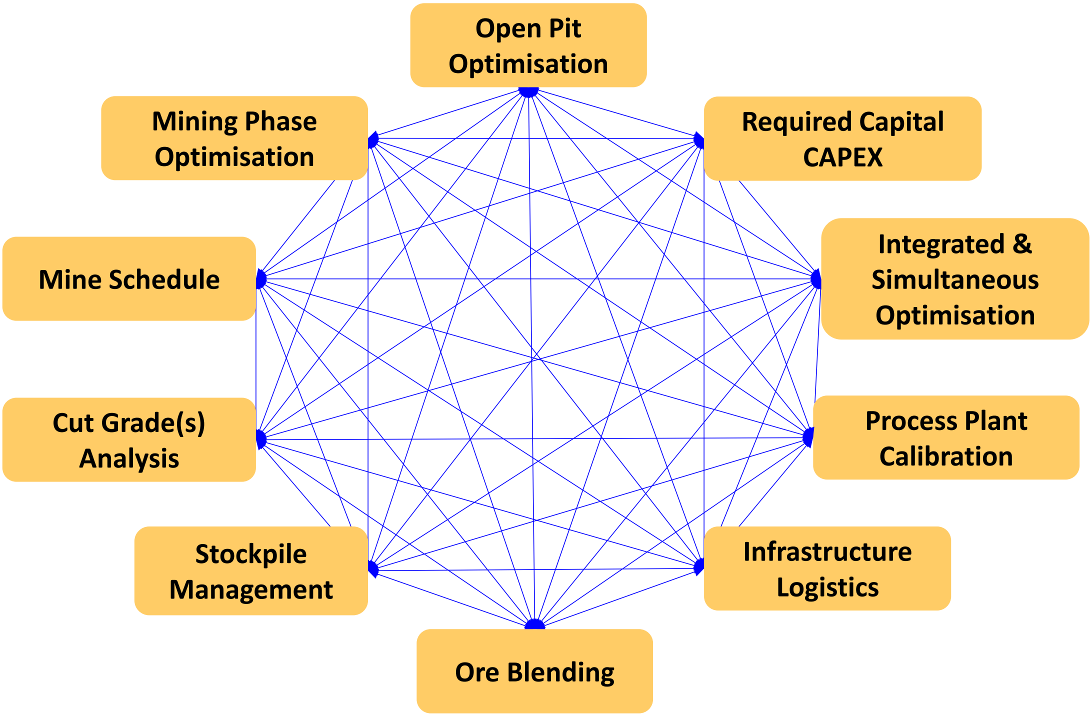


Figure 242. Enterprise Optimization (Whittle Consulting)
(Image: Simon Michaux)

In the past these tasks were done separately and insolation. As technology has improved, it has become possible to optimize all these tasks together at the same time (Whittle Consulting https://www.whittleconsulting.com.au/). The tasks in Figure 225 are all able to happen with data inputs. A geometallurgy campaign would be able to support each task with more sophisticated information. Just so, the operation planning would be more sophisticated, where the plan would resemble reality more closely (avoiding some of the problems discussed in Section 1).

The difference between a conventional assessment and a geometallurgical styled assessment can be seen in Figure 226. As can be seen, the optimized case has variability embedded in it as a function of time. This means that tasks like plant expansions, pit cutbacks and maintenance can be planned, with the foresight of when performance could be poor and when it could be productive.

14.5 Corporate problem solving with the Theory of Constraints

There are operational opportunities using geomet data. With the capability to map this ore type process behavior into the deposit (here previously, only grade and geology domains were mapped). As extraction progresses through the deposit, it is possible to examine operational challenges. Data can be used to support the study of production bottlenecks across the operation. These bottlenecks will move along the logistics chain over the life of the mine. The methodology to do this would be to use the options analysis and theory of operational constraints (TOC) (Rochman 2003).

The Theory of Constraints is a methodology for identifying the most important limiting factor (i.e. constraint) that stands in the way of achieving a goal (usually generating maximum revenue) and then systematically improving that constraint until it is no longer the limiting factor (Goldratt & Cox 1993). The goal for mining may be maximum metal produced, or maximum recovery.

In manufacturing, the constraint is often referred to as a bottleneck in production parts. In a mining, the constraint could be something like ball mill throughput, or rate of tonnes produced ROM. The Theory of Constraints (TOC) takes a scientific approach to improvement. It hypothesizes that every complex system, including manufacturing processes, consists of multiple linked activities, one of which acts as a constraint upon the entire system (i.e. the constraint activity is the "weakest link in the chain").

The Theory of Constraints provides a specific methodology for identifying and eliminating constraints, referred to as the Five Focusing Steps. As shown in the following in Figure 243 and Table 14, it is considered a cyclical process.

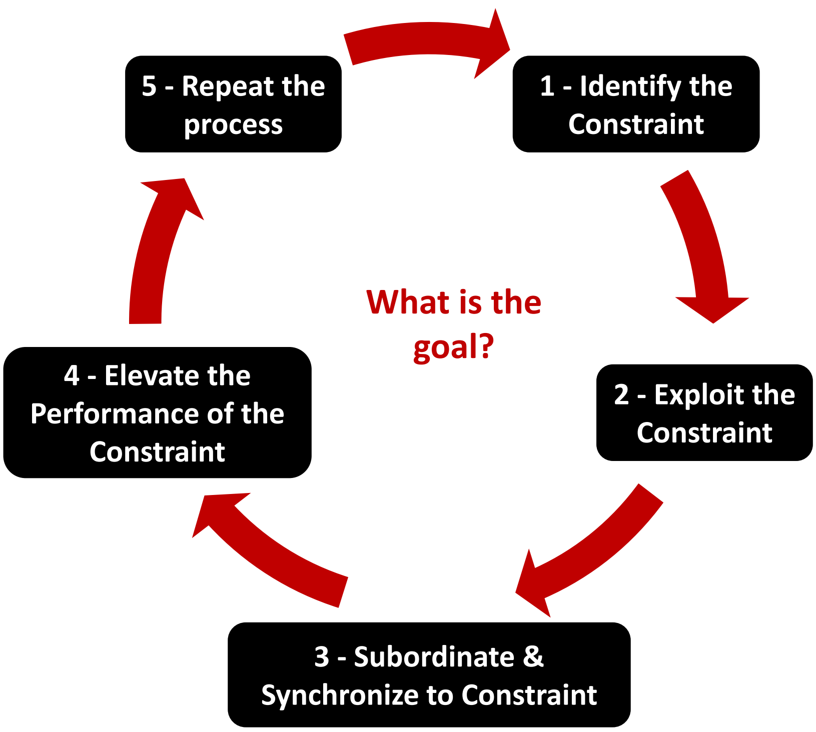


Figure 243. The Five Focusing Steps (a methodology for identifying and eliminating constraints) (Goldratt & Cox 1993) (Image: Simon Michaux)

Table 14: The Five Focusing Steps (a methodology for identifying and eliminating constraints) (Goldratt & Cox 1993)

Step	Objective
1 Identify	Identify the current constraint (the single part of the process that limits the rate at which the goal is achieved).
2 Exploit	Make quick improvements to the throughput of the constraint using existing resources (i.e. make the most of what you have).
3 Subordinate	Review all other activities in the process to ensure that they are aligned with and truly support the needs of the constraint.
4 Elevate	If the constraint still exists (i.e. it has not moved), consider what further actions can be taken to eliminate it from being the constraint. Normally, actions are continued at this step until the constraint has been “broken” (until it has moved somewhere else). In some cases, capital investment may be required.
5 Repeat	The Five Focusing Steps are a continuous improvement cycle. Therefore, once a constraint is resolved the next constraint should immediately be addressed. This step is a reminder to never become complacent – aggressively improve the current constraint…and then immediately move on to the next constraint. If, in the previous steps, a constraint has been broken, go back to Step 1. But do not allow inertia to cause a system constraint

The TOC methodology (Goldratt & Cox 1993) introduced a staged logical thinking process to be used in conjunction with the five focusing steps. The thinking process assists with working through the change process by identifying the following:

- What to change
- What to change to, and
- How to effect that change

The thinking processes consist of logic tools used to identify problems, then the development and implementation of solutions. These tools include effect-cause-effect (ECE) diagramming (also known as Cause and Effect Diagrams, Fishbone Diagrams, Ishikawa Diagrams, Herringbone Diagrams, and Fishikawa Diagrams) and its components:

- negative branch reservations
- the current reality tree
- the future reality tree
- the prerequisite tree
- the transition tree
- the evaporating cloud
- the negative branch reservation, and
- the ECE audit process.

These tools allow any organization to analyze and to verbalize cause and effect around any issue. In doing so, a complex and/or serious problem can be unravelled and understood at its fundamentals. This methodology assists in the search is for the possible cause / causes or basic reason with regard to a specific process, outcome, problem or situation, and makes relationships and interactions between different causes visible with regard to specific process. It also allows for the process of analyzing existing problems with the aim of initiating corrective measures and identifying areas where additional activities (improvement processes) can be started.

To conduct this procedure, the target industrial operation needs to be mapped out in a systems context, where all inputs, throughputs, outputs, actions and decisions are accounted for to produce a saleable product. Usually, a combination of different systems architecture are used and merged, similar to what is shown in Figure 244.

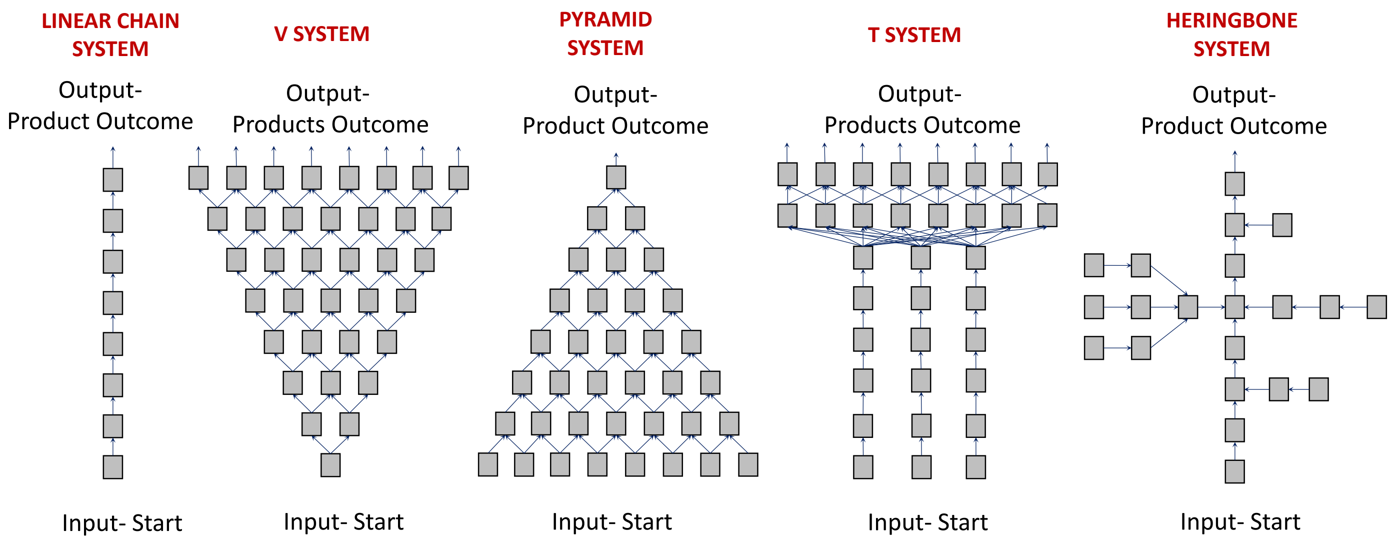


Figure 244. Different kinds of systems architecture
(Image: Simon Michaux)

This will help diagnose bottlenecks and strategically important parts of the process. Then each bottle neck can be examined where different choices can be made. Each different choice will have a different outcome in context of the goal (whether it be revenue, or metal produced). The Net Present Value (NPV) tool can be used to account for the time needed to make this change.

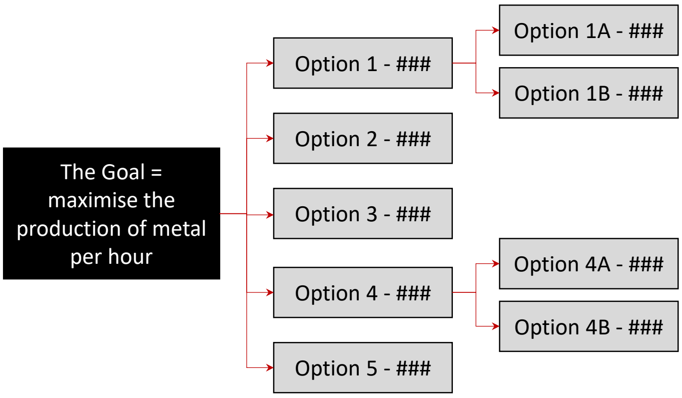
Net Present Value is the difference amount between discounted sums: cash inflows and cash outflows. It compares the present value of money today to the present value of money in the future at a discounted rate to account for inflation and returns. This is a fundamental tool in Discounted Cash Flow analysis (DCF). Formula 24 below shows how NPV is calculated.

$$NPV = \sum_{t=0}^{T} \frac{E\{CF_t\}}{(1+R)^t} - CAPEX \qquad \text{Equation 24}$$

The NPV calculation will increasingly discount the value contribution of costs/incomes with each addition time period from the present. This means that the NPV calculation will only account for a time period of 5-8 years, after which cash flows are disregarded. Formula 25 below shows a general rule of thumb in how to quickly estimate NPV (only to be used appropriately).

$$NPV = ({}^{1}/_{5}\ CAPEX) + OPEX \text{ for 1 year} \qquad \text{Equation 25}$$

All changes to the system can then be ranked in terms of effectiveness. A theoretical example is shown in Figure 245.



TOC Option	Cash Flow: Required Upfront new CAPEX	Cash Flow: Year 1 Revenue	Operation NPV @ 20%	Rank
Option 1A	104	180	46M	1
Option 1B	150	160	43M	2
Option 2	24	45	13M	5
Option 3	11	29	6.6M	5
Option 4A	35	50	14M	4
Option 4B	12	45	15M	3
Option 5	19	33	6.5M	7

Figure 245. Options analysis & theory of constraints decision tree – theoretical example (Image: Simon Michaux)

An example of TOC could be the choice for a mine operation to transport concentrate in trucks or build a pipeline to the port. The development of decision making could resemble:

- For the Life of Mine (LOM):
  - Pipe is cheaper but is a bottle neck
  - Trucking is more costly, may not be a bottle neck
  - Pipeline and trucking is an option
- Trucks are very flexible (an optimization value add) allowing cash to be moved up-front, improving NPV
- Optimize whole project for 3 options: pipe, truck, pipe and truck
- Choose the option with the best NPV, progress outcome to next study phase

Ore mineral characteristics data would form only part of the input data required to do a TOC analysis. Traditionally, a very simplistic input was used in conjunction with data inputs from other parts of the operation like power consumption, transport time and handling logistical metrics.

The value of the concentrate (market price * grade * recovery – cost of production – penalty element) has the capacity to change this sequence. This value would be highly variable and change for different parts of the deposit. A geometallurgy data set used effectively in TOC, could change the outcome of the analysis.

14.6 Corporate decision making for the executive board

The executive board often make decisively important decisions with poor data quality available. Have to rely on past experience and intuition. There is now a need to make defendable decisions now that the risk is so much higher and the time frame for things to happen is much longer. The market is now much more volatile and unforgiving (in 2019). Also, due to the time frame projects now have, there are multiple generations of executive board members, all of which need better data support at handover. Having quality data in place to do this requires the work to be done ahead of time. Questions an executive board may need to answer and then later defend their decisions could be:

- Should we authorize the start of this project?
- Or should we delay it until market conditions are better?
- And considering debt obligations of company
- Can we afford site rehabilitation?
- When will mine start producing a saleable product?
- Should this project have an open pit or should it be an underground block cave?
- Should this project use:
  - a HPGR or a SAG mill?
  - Should recovery be flotation circuit or a leaching circuit, or a hybrid of both?
  - Should final metal production use pyrometallurgy or hydrometallurgy?
- When should we do the plant expansions and pit cutbacks?

Risk could be mitigated by:

- At the feasibility stage(s), CAPEX closely resembles the final commissioning reality.
- Once operating, the deposit is comprehensively characterized which allows flexible changes in the mine schedule and process response can be accurately predicted.
- Where decisions on process plant expansion, open pit cutback expansions and maintenance shutdowns can all be planned in context of risk uncertainty with more precision.

14.7 Geometallurgy to support due diligence

The financing of mining operations is evolving quickly. The CAPEX for a typical copper mine is now of the order of 4 to 5 billion $USD. Shareholders and venture capitalist often put up approximately 30% of the required capital. The remaining 70% often is supplied by a banking hedge fund. Once a mining corporation proposes a project, due diligence is required to audit the feasibility study before capital is available. The same due diligence procedures would be needed state governments to authorize a mining license. Geometallurgy can facilitate more sophisticated Due Diligence methodology

- What is the real NPV?
- What is the real CAPEX?
- How long will commissioning take?
- When will CAPEX be paid off?
- What is the real OPEX?
- What is the real rate of metal production?
- What is the real IRR for revenue flow?
- What are the macro scale steps of the mine Life of Mine?
- What is the probably mine operation waste plume?
- What is the mine operation environmental impact?
- What are the requirements for mine site environmental rehabilitation?
- Should the investment hedge fund support this project?

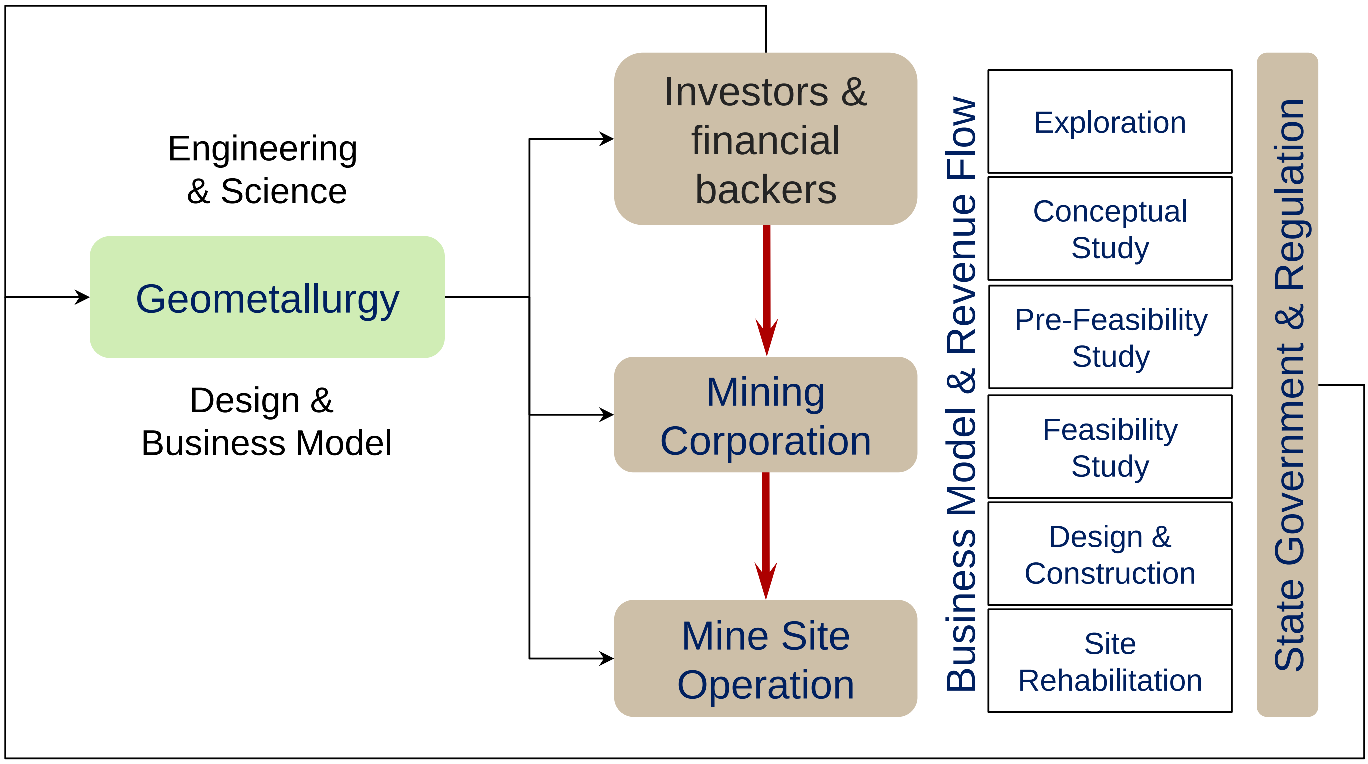


Figure 246. Geometallurgy supported due diligence
(Image: Simon Michaux)

15 DEVELOPMENT OF PRODUCTION PROTOCOL

Traditionally, once a mine operation has been designed, constructed and then commissioned, there is a handover to the operating staff. Often, very little information on why choices in engineering design were made is transferred to the operating staff. This handover point is often the subject of corporate amnesia. This means that the implications of the process engineering design in the plant interacting with the ore deposit have to be discovered the hard way.

Advances in engineering (information technology, instrumentation and automation in particular) offer a unique opportunity to resolve this. The comprehensive information and knowledge a geometallurgical campaign that has been effectively used in design can be used in an operational context.

As ore is excavated, it can be tracked (using ore tracking technology) as it progresses to a stock pile, then later into the process concentrator plant.

For example, a mine site will need to operate multiple working faces, each of which will have logistical challenges on a day to day basis. If multiple stockpiles have been developed as part of the mine plan, then different ore types can be put into appropriate stockpiles.

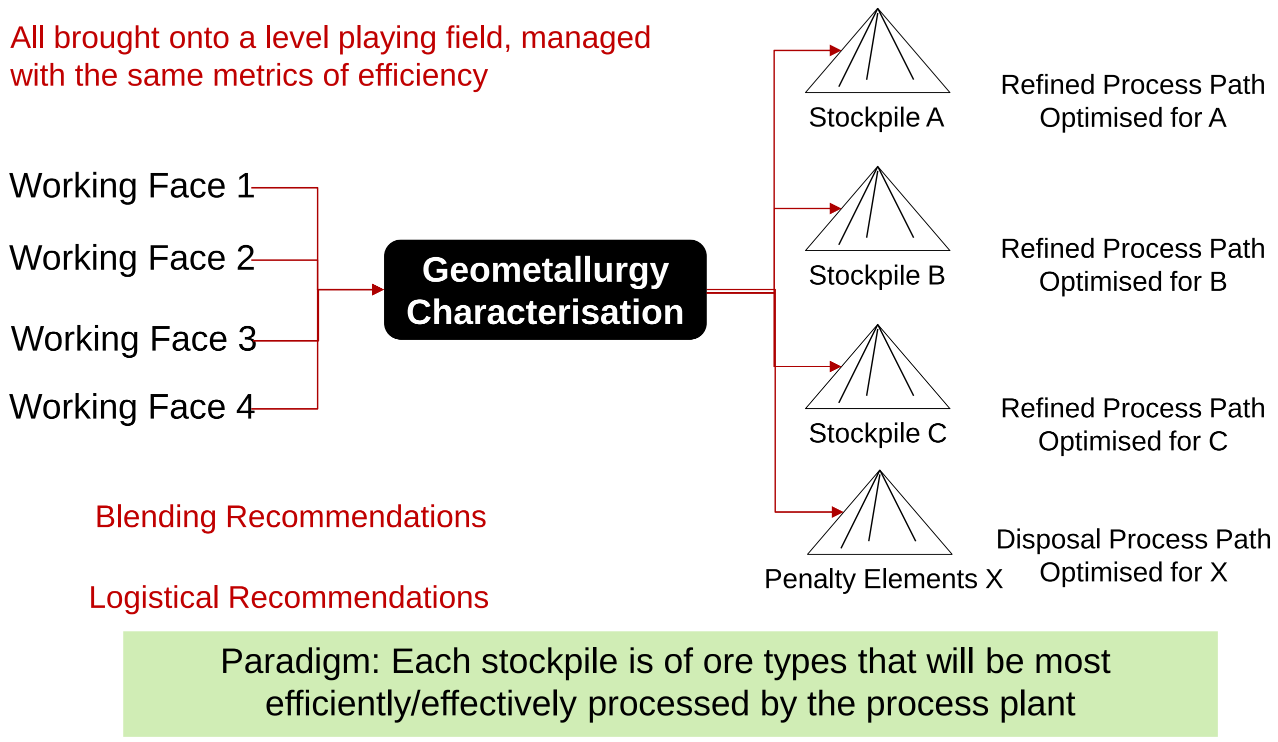


Figure 247. Multiple stockpiles
(Image: Simon Michaux)

For this to work, a coordinated and heavily interconnected optimized mining and processing operation has to be designed and then operated, where ore body knowledge and a real time operations center all need to be integrated. This can only work if ore body knowledge is at a degree of sophistication, where each parcel of ore is known in context of its process performance and true value.

The outcomes of a geometallurgy study can be used to develop a routine protocol at a production site. The Orientation Study, if done well, will show process behavior for the rock texture extremes of the deposit being mined. Ideally the process path being used in the operating mine is similar to one of the process paths examined in the Orientation Study, and the mineralogy that controls the effectiveness of that process path has been diagnosed.

Those mineralogy groupings that control process behavior across all the Orientation Samples will have measurement signatures that could be captured with online instrumentation in the process plant. Something as simple as an online XRF measurement tool or a more sophisticated Raman Spectroscopy instrument placed at the right place in the plant has the capacity to diagnose which combinations of the rock textures so far examined is passing into the process circuit. This in turn could predict ahead of time what the process response could be, through understanding the outcomes of the geometallurgy Orientation Study.

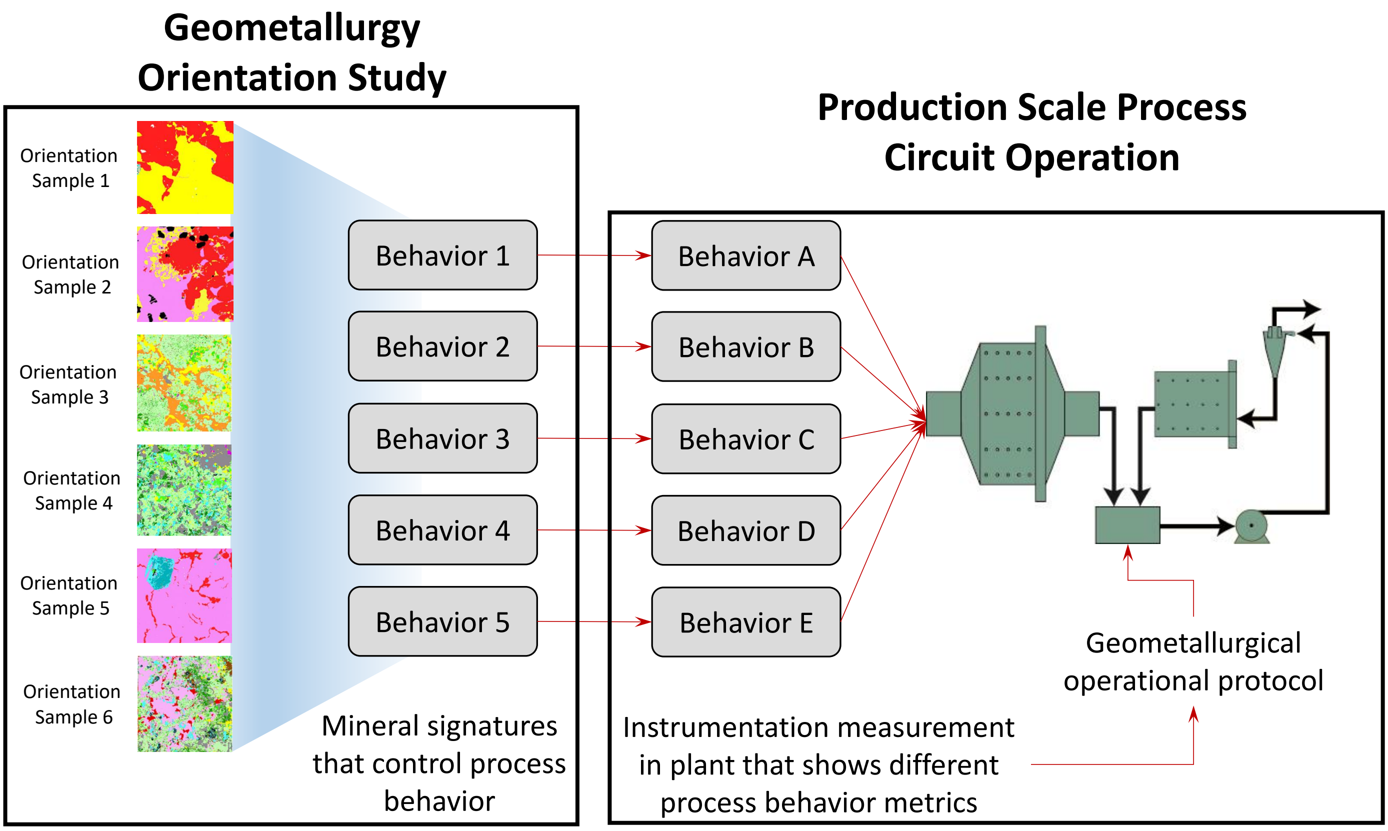


Figure 248. Geometallurgy during process operation

(Image: Simon Michaux)

To do this, a link between the process behavior of the Orientation Samples and some kind of online mineral measurement in the process plant to diagnose which combination of rock textures are being processed, needs to be established. This would be part of a geometallurgical operational protocol that is required to be developed, based on the outcomes of work done.

Regardless of the level of development a mine site has, a routine operating protocol is needed to guide data collection and reconciliation. Based on what has been learned from the Orientation Study and the Mapping Study, the following could be considered:

- What mineral signatures control process behavior?
- What data should be collected routinely (that are practical in a production environment)?
- How is that data to be used by site personnel?
- How should the process plant operation evolve over time to keep pace with the mine schedule?

16 CONCLUSION AND SUMMARY

The geometallurgy process can be complex but it has 6 broad brush steps. A more complex demonstration of this can be seen in Figures 249-251.

1. The geologically informed selection of a number of ore samples
2. Laboratory-scale test work to determine the ore's response to mineral processing unit operations
3. Determination of what mineralogy parameters control each process path, then a judgment on what is the most effective process path
4. Domaining and mapping of variability of the chosen process path
5. The propagation and distribution of these parameters throughout the orebody using an accepted geostatistical technique (or in more practical terms a chemical assay signature).
6. The application of a mining sequence plan and mineral processing models to generate a prediction of the process plant behavior.

The purpose of this report was to show the paradigm behind geometallurgy. In practice, only a part of this procedure would be done, as defined by the original goals of the campaign.

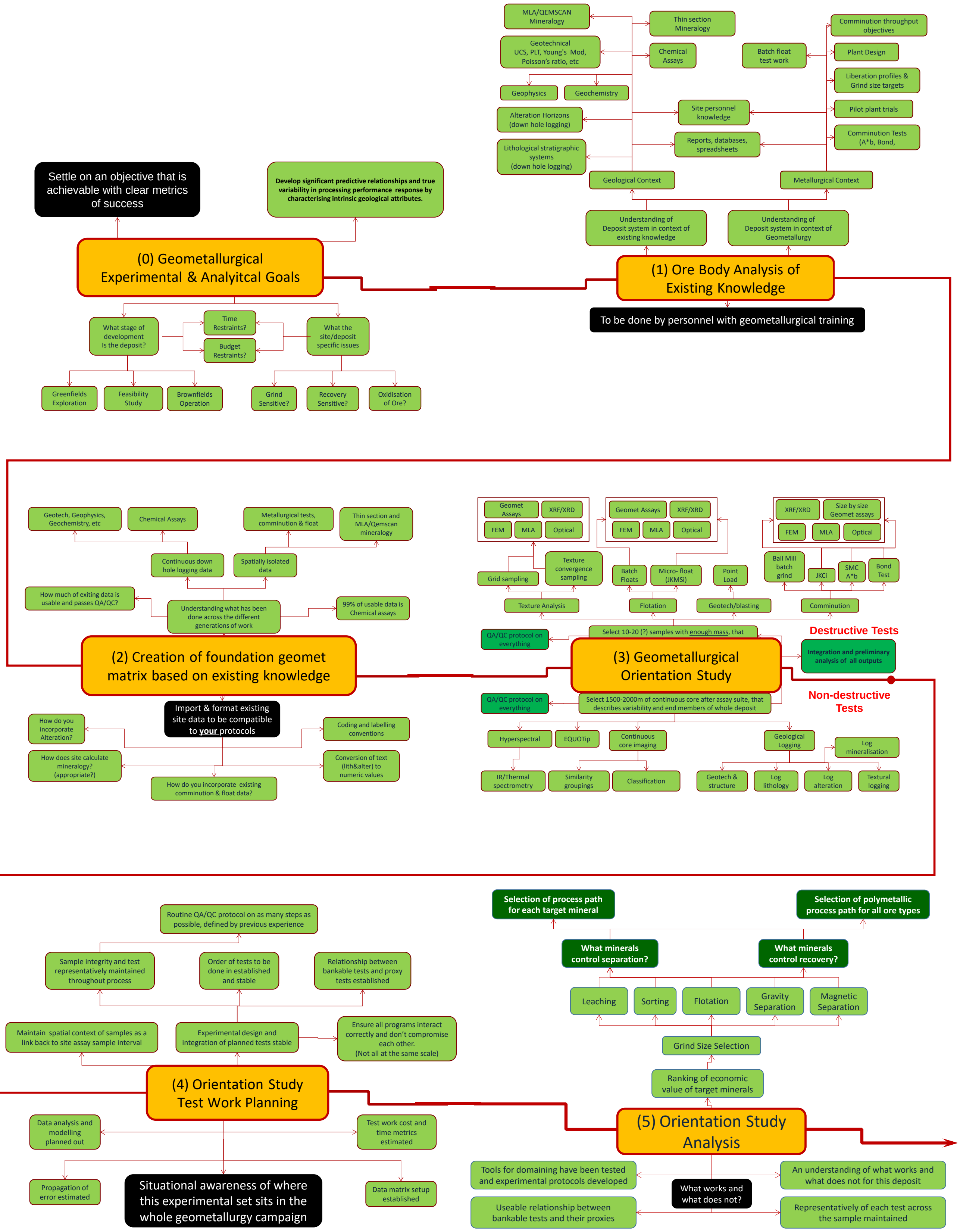


Figure 249. The geometallurgy steps 1 of 3
(Image: Simon Michaux)

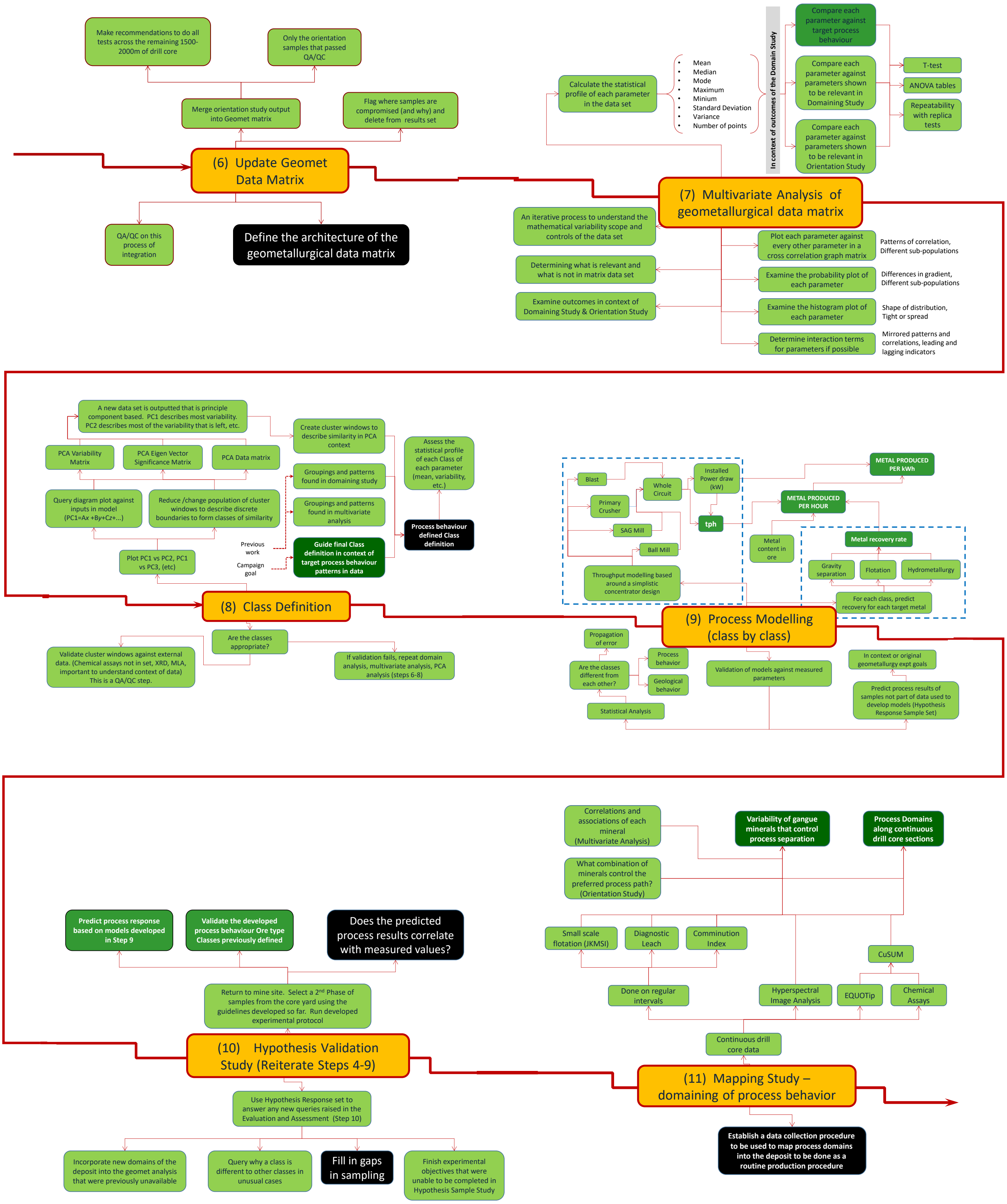


Figure 250. The geometallurgy steps 2 of 3
(Image: Simon Michaux)

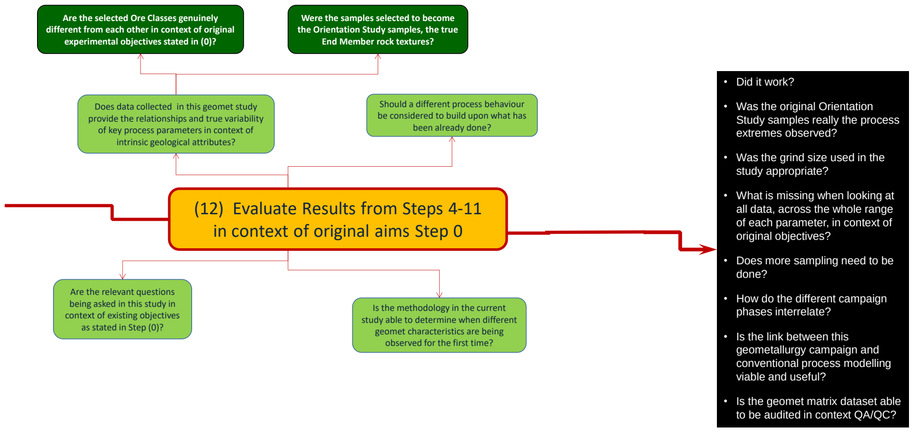


Figure 251. The geometallurgy steps 3 of 3
(Image: Simon Michaux)

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