Processing methods for recovery of indicator minerals from sediment and bedrock

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INTRODUCTION

The application of indicator mineral methods to mineral exploration has grown and developed significantly over the past two decades and they are now used around the world to explore for a broad spectrum of commodities. Heavy mineral suites have been identified for detecting a variety of ore deposit types including diamond, gold, Ni-Cu, PGE, metamorphosed VMS, porphyry Cu, uranium, tin, and tungsten. Indicator minerals, including ore, accessory, and alteration minerals, are usually sparsely distributed in their host rocks. They may be sparser in surficial sediments, therefore sediment samples must be concentrated in order to recover and examine them. Many indicator minerals have a moderate to high (>2.9 g/cm^3) specific gravity (SG), thus most processing techniques concentrate indicator minerals using some type of density separation, often in combination with sizing and/or magnetic separations. As few as one sand-sized grain of a particular indicator mineral in a 10 kg sample may be significant. To recover such potentially small quantities (equivalent to ppb) of indicator minerals, samples are processed to reduce the volume of material that must be examined. In reducing the volume of material, processing techniques must be able to retain the indicator mineral(s) without contaminating the sample and do so at a reasonable cost.

Indicator minerals can be recovered from a variety of sample media, including stream, alluvial, glacial, or eolian sediments and residual soils. They can also recovered from both weathered and fresh bedrock, as well as from mineralized float. The combinations of processing techniques used by exploration companies or government agencies for recovering indicator minerals are quite varied (e.g. Gregory & White 1989; Peuraniemi 1990; Davison 1993; Towie & Seet 1995; Chernet *et al.* 1999; McClenaghan *et al.* 1999). This paper describes some of the common processing methods used to reduce sample weight, concentrate heavy minerals, and recover indicator minerals (Fig. 1), including those for deposits of diamond, precious and base metals, and uranium. The methods used will depend on the commodities being sought, as well as cost per sample. As most oxide and silicate indicator minerals (e.g. kimberlite, Ni-Cu-PGE, and metamorphosed massive sulphide indicator minerals) are medium to coarse sand sized (0.25-2.0 mm; Averill 2001), concentration techniques that recover the sand-sized heavy minerals are required. In contrast, approximately 90% of gold grains, platinum group minerals (PGMs), and sulphide minerals are silt sized (<0.063 mm), thus concentration of these indicators requires a preconcentration technique that includes recovery of the silt- as well as the sand-sized fractions.

SAMPLE WEIGHT

The weights of material collected for indicator mineral studies will depend on the type of surficial sediment collected, the
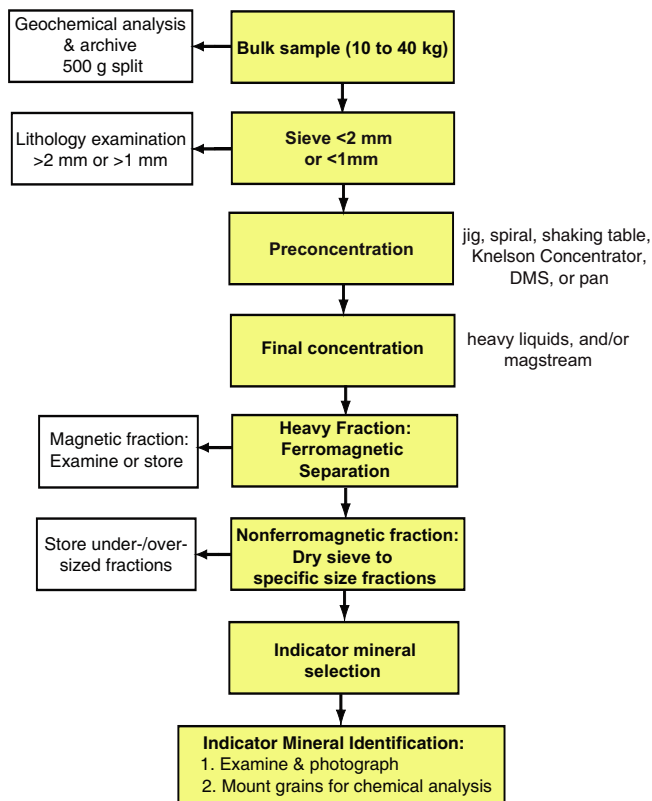

Fig. 1. Generalized flow sheet showing steps in sample processing used to reduce sample weight, concentrate heavy minerals, and recover indicator minerals.

grain-size characteristics of the sample material, the commodity being sought, and shipping costs (Table 1). For example, in glaciated terrain, clay-rich till samples may be 20 to 50 kg (or more) in order to recover a sufficient weight of sand-sized heavy minerals (Table 2, sample 5). Coarse-grained silty sand till, typical of shield terrain, requires smaller (10 to 15 kg) samples because it contains more sand-sized material in the matrix (Table 2, samples 1 to 4). Sediment samples collected for recovery of porphyry Cu indicator minerals (PCIM) need only be ~0.5 kg because porphyry Cu alteration systems are large and rich in indicator minerals (Averill 2007). Bedrock and float samples usually vary from 1 to 10 kg.

BEDROCK PREPARATION

Bedrock and float samples often need to be crushed prior to processing to recover heavy minerals. Crushing or disaggregating reduces rock fragment size to about 2 mm, or the average size of mineral grains in the sample. Since rock crushers can be difficult to clean between samples, barren quartz can be crushed as a blank between each bedrock sample to reduce cross contamination. To monitor heavy mineral carryover, the

Table 1. Examples of the variation in sample weight and processing procedures with sample and target type at Overburden Drilling Management Ltd.'s heavy mineral processing lab (Averill & Huneault 2006).

Target	Typical Sample Weight (kg)	Required Separations				
		Table	Micropan	Heavy Liquid (Specific Gravity)	Ferro-Magnetic	Para-magnetic
A. Sediment Samples						
Gold	10	Single	Yes	3.3	Yes	No
Kimberlite	10-30	Double	No	3.2	Yes	Yes
Massive sulphides (Ni-Cu-PGE, BHT, VMS, IOCG, MVT, skarn)	10	Single	Yes (PGE only)	3.2	Yes	Yes
Porphyry Cu	0.5	No	No	2.8, 3.2	Yes	Two
Uranium	10	Single	Yes	3.3	Yes	No
Heavy mineral sands (grade evaluation)	20	Triple	No	3.3	Yes	Optional
Tampering (investigation)	Variable	Optional	Yes	3.3	Yes	Optional
B. Rock Samples						
Gold, PGE, base metals	1	Optional	Yes	3.3	Yes	Optional
Kimberlite	1-10	Optional	No	3.2	Yes	Yes
Tampering (investigation)	1	No	Yes	3.3	Yes	Optional

blanks can then be processed and examined for heavy minerals along with the actual sample concentrates.

PRECONCENTRATION METHODS

If sample shipping costs are an issue, samples may be partly processed in the field to reduce the weight of material shipped to the lab. Samples may be sieved to remove the coarse (>1 or >2 mm) fraction, which may reduce weights from a few % to 30% (e.g. Table 2, columns B-C). Preconcentrating, using a pan, jig, sluice box, or Knelson Concentrator, also may be carried out in the field to further reduce the weight of material to be shipped. Preconcentrates may be examined in the field, significantly, reducing the time to obtain results for follow up. However, preconcentrating in the field can itself be expensive as well as time consuming, and the available methods may not provide optimal recovery of the indicator minerals of interest.

Whether sieved off in the field or in the lab, the coarse >2 mm fraction may be examined (pebble counts) to provide additional information about sample provenance and transport distance. The <2 (or <1) mm fraction is preconcentrated most commonly using sieving and/or density methods (e.g. jig, shaking table, spiral, dense media separator, pan, or Knelson Concentrator) to reduce the weight of material to be examined without losing indicator minerals. Some of the more common preconcentration techniques are described below.

Panning

Panning is the oldest method used to recover indicator minerals, primarily for gold and PGMs. Sediment is placed in a pan

Fig. 2. Panning is one of the oldest methods used to reduce sample volume and prepare a heavy mineral preconcentrate (photo from W. Spirito, Geological Survey of Canada).

and shaken sideways in a circular motion while being held just under water, heavy minerals sink to the pan bottom and light minerals rise and spill out over the top (e.g. Silva 1986; English *et al.* 1987; Ballantyne & Harris 1997). Pans have varying shapes (flat bottomed or conical) and sizes (Fig. 2), and can be made out of plastic, metal, or wood. The advantages of this technique are that it can be a field or lab-based operation, is inexpensive in terms of equipment costs, and, if used in the field, it reduces sample shipping weight and thus cost. Panning is

Table 2. Weight of fractions generated by a combination of tabling and heavy liquid separation (MI) to reduce till sample weight, concentrate heavy minerals, and recover indicator minerals. **A)** initial sample weight; **B)** sieving of <2 mm; **C)** & **D)** tabling; **E)** heavy liquid separation; **F)** ferro-magnetic separation; **G)** final heavy mineral concentrate weight. Till samples are from 1) the South Pit of the Thompson Ni Mine, Thompson, Manitoba; 2) Broken Hammer Cu-PGE occurrence, Sudbury, Ontario; 3) Pamour Au Mine, Timmins, Ontario; 4) Triple B kimberlite, Lake Timiskaming field, Ontario; and 5) Buffalo Head Hills, northern Alberta.

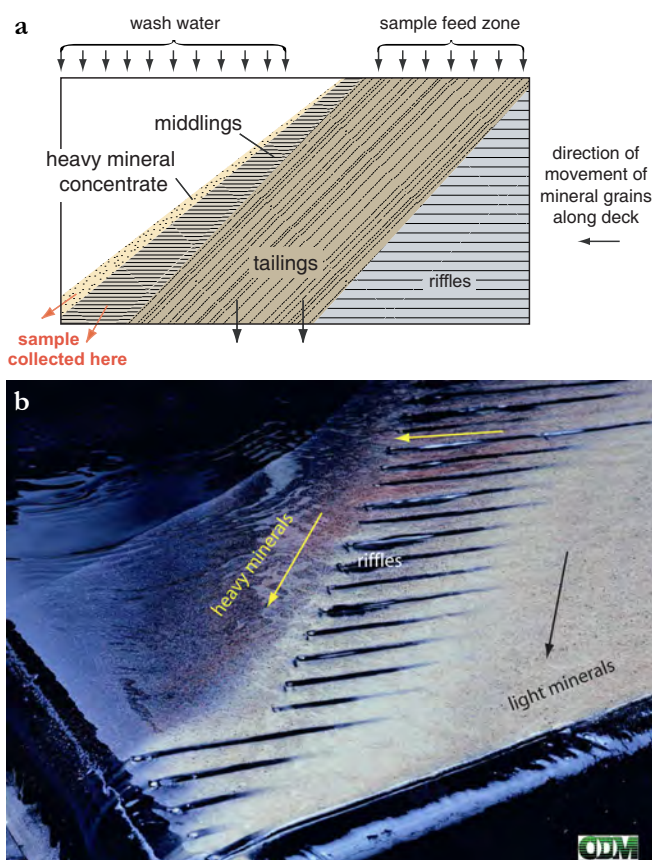| Location	Texture	A: Total sample weight (kg)	B: Weight of >2 mm clasts (kg)	C: Weight of sample put across the shaking table (kg)	D: Weight of concentrate produced from shaking table (g)	E: Weight of heavy liquid light fraction (g)	F: Weight of the magnetic fraction (g)	G: Weight of non-magnetic heavy mineral concentrate (g) 0.25-2.0 mm
1. Thompson Ni Belt	silty sand till	15.0	3.0	12.0	1015.9	104.5	36.4	47.9
2. Sudbury	very sandy till	15.0	5.6	9.4	1125.1	402.6	13.0	18.9
3. Timmins Gold camp	silty sand till	11.8	2.3	9.5	353.1	319.8	5.2	28.1
4. Triple B kimberlite	silty sand till	9.8	1.2	8.6	438.7	377.0	22.0	35.8
5. Northern Alberta	clay till	67.4	2.4	65.0	1,307.0	1,235.2	5.6	11.5

Fig. 3. a) Schematic plan view of a shaking table (Silva 1986); **b)** close up of heavy minerals separating from light minerals on a shaking table (photo from Overburden Drilling Management Ltd.).

often used in combination with other preconcentration methods to recover the silt-sized precious metal grains (e.g. Grant *et al*. 1991; Leake *et al*. 1991, 1998; Ballantyne & Harris 1997; Wierchowiec 2002). The disadvantages of this method are that is slow, is highly dependent on the experience and skill of the operator, and therefore requires consistent personnel to perform the panning.

Tabling

Preconcentration using a shaking (Wilfley) table is a commonly used method for separating minerals on the basis of density (Silva 1986). It recovers silt to coarse sand-sized heavy minerals for a broad spectrum of commodities, including diamonds, precious and base metals, and uranium (Averill & Huneault 2006). Silva (1986) describes this method consisting of a table with the deck covered with up to 1 cm high riffles along half of the surface (Fig. 3a). A motor mounted to the side drives a small arm that shakes the table along its length. A slurry of <2.0 mm sample material and water is fed along the top of the table, perpendicular to the direction of the table motion. The table is shaken sideways using a slow forward stroke and a fast return strike that causes the grains to crawl along the deck parallel to the direction of motion. The shaking motion, combined with the water wash, moves grains diagonally across the deck from the feed end and separates them on the table according to size and density (Fig. 3b). If kimberlite indicators are targeted, the sample may be tabled twice to ensure higher recovery of the key lower density minerals, such as Cr-diopside and forsteritic olivine, as well as the coarsest grains. The advan-

tages of this method are its moderate cost, ability to recover indicator minerals for a broad spectrum of commodities, and ability to recover silt- as well as sand-sized indicators. It is a well established method for the recovery of precious and metallic metal mineral grains, as well as kimberlite indicator minerals (e.g. English *et al*. 1987; McClenaghan *et al*. 1998; 2002). The disadvantages of this method include the loss of some heavy minerals during tabling, the longer time required to process each sample, and that the tabling procedure is dependent on the skill of the operator.

Dense media separator

A gravity method commonly used to preconcentrate kimberlite indicator minerals is the micro-scale dense media separator (DMS) (Fig. 4). As described by Baumgartner (2006), heavy mineral concentration is carried out using a gravity-fed high-pressure cyclone. The <1 mm fraction of a sample is mixed with ferrosilicon (FeSi) to produce a slurry that has a controlled density. The slurry is fed into a cyclone where the grains travel radially and helically, forcing the heavier particles toward the wall of the cyclone and the lighter particles toward the centre. The lighter and heavier particles exit the cyclone through different holes, with the light fraction discarded and the heavy fraction collected on a 0.25 or 0.3 mm screen. The >0.25/0.3 mm heavy fraction is then dried and screened to remove residual FeSi. A Tromp curve is used to define the efficiency and precision of the DMS separation. The cut-point, or threshold, spans a density range of 0.2 g/cm^3 at approximately 3.1 g/cm^3 and is calibrated to recover the common kimberlite

Fig. 4. Micro dense media separator used at Mineral Services Canada to separate kimberlite indicator minerals (photo from Mineral Services Canada).

Fig. 5. Lab-based Knelson concentrator used to prepare a preconcentrate for recovery of kimberlite indicator minerals from till by the Geological Survey of Finland (from Marmo *et al.* 2008).

indicator minerals that have a specific gravity of >3.1 g/cm³: pyrope garnet, chrome-spinel, Mg-ilmenite, Cr-diopside, forsteritic olivine, and diamond. The required cut point is established using the computerized control panel on the DMS and the Tromp curve is then tested using synthetic density tracers before proceeding with processing the samples. The density settings and cut points are checked once per day to maintain accurate specific-gravity thresholds. The advantages of the micro DMS system are that it is fast, less susceptible to sample contamination than other heavy mineral concentrating techniques, and not operator dependent. The method, however, is more expensive than other methods described here and it does not recover silt-sized precious and base metal indicator minerals.

Knelson concentrator

The Knelson concentrator is a fluidized centrifugal separator (Fig. 5) that was originally designed for concentrating gold and platinum from placer and bedrock samples. However, in recent years, it has also been used to recover kimberlite indicator minerals from sediment samples (e.g. Chernet *et al.* 1999; Lehtonen *et al.* 2005). The concentrator can handle particle sizes from >10 microns up to a maximum of 6 mm. The general processing procedure, from the Knelson Concentrator website: http://www.knelsongravitysolutions.com/, states that water is introduced into a concentrate cone through a series of holes in rings on the side of the cone. The sample slurry is then introduced into the concentrate cone from a tube at the top. When the slurry reaches the bottom of the cone, it is forced outward and up the cone wall by centrifugal force from the spinning cone. The slurry fills each ring on the inside of the cone wall to capacity to create a concentrating bed. Particles with high specific gravity are captured in the rings and retained in the concentrating cone. At the end of the concentrate cycle, concentrates are flushed from the cone into the sample collector. Chernet *et al.*'s (1999) procedure for kimberlite indicator minerals takes 5 to 11 minutes per sample. The advantages of the Knelson concentrator are that it is fast, inexpensive, and can be used in a lab or mobilized to the field to reduce the weight of material to be shipped to the lab. However, recovery of kimberlite indicator minerals from silt-poor material, such as esker sand or stream sediments, is difficult due to the absence of

fine-grained material to keep the slurry in suspension (Chernet *et al.* 1999).

Rotary spiral concentrator

Heavy minerals can be recovered using a rotary spiral concentrator, which consists of a flat circular stainless steel bowl with rubber ribs that spiral inward (Fig. 6), a detailed description of which is reported by Silva (1986). A spiral concentrator is mounted on a frame so it can be tilted and has a water wash bar extending laterally from one side of the bowl to the centre. As the bowl spins, water is sprayed from the bar and heavy mineral grains move up and inward along the spirals to the central opening, where they are collected in a container behind the bowl. Water washes light minerals down to the bottom bowl. The heaviest minerals are recovered first. The advantages of the spiral concentrator are that it can be field based and thus reduce sample weight to be shipped, it is inexpensive to acquire and operate, it is fast if the material is sandy, and it recovers indicator minerals across a broad size range, from silt- to sand-size grains. The method, however, is dependent on the experience and skill of the operator, the lower density threshold is variable, there is some loss of heavy minerals, and the method is slow if the sample is clay-rich. It is used mainly for gold recovery (e.g. Maurice & Mercier 1986; Silva 1986) but in the past 10 years it also has been used for the recovery of kimberlite indicator minerals (e.g. Sarala & Peuraniemi 2007).

Magnetic separation and sieving

Indicator minerals may also be preconcentrated using magnetic separation in combination with sieving, such as the lab procedures described by Le Couteur & McLeod (2006). Their specific methods include wet screening, typically at 0.86 mm and 0.25 mm. The 0.25 to 0.86 mm fraction is dried, weighed, and then treated to magnetic separation using a permanent Fe-Nd dry-belt magnetic separator operating at 2.1 Tesla. The magnet divides the sample into three fractions: 1) non-magnetic / diamagnetic, 2) weakly paramagnetic, and 3) strongly paramagnetic. The weak (2) and strong (3) paramagnetic fractions are combined in one "magnetic concentrate" that is then processed through heavy liquids. The advantages of this procedure are that it is fast and inexpensive. Their methods, however, do not allow for the recovery of silt-sized precious and base metal grains, and do not recover coarse (>1 mm) indicator minerals.

Fig. 6. Lab-based rotary spiral concentrator used by the Geological Survey of Finland (photo from P. Sarala, Geological Survey of Finland).

Fig. 7. Separation of heavy and light minerals is carried out using a specific heavy liquid of known density in separatory funnels. Heavy minerals sink and light minerals float to the top (photo from Overburden Drilling Management Ltd.).

FINAL CONCENTRATION METHODS

A preconcentrate is usually further refined using heavy liquids of a precise density to further reduce the size of the concentrate prior to heavy mineral selection (Table 2, column E). Heavy liquid separation (Fig. 7) provides a sharp separation between heavy (sink) and light minerals (float) at an exact known density. It is slow and expensive and therefore not economical for large volumes of sample material, hence the preconcentration procedures that come before this step. The most common heavy liquids used include methylene iodide (MI) with a SG of 3.3 g/cm^3 and tetrabromoethane (TBE) with a SG of 2.96 g/cm^3. The density threshold will depend on the indicator minerals being sought. Some labs use a combination of both heavy liquids, separating first using TBE to reduce the volume of material to be further separated using MI (e.g. Le Couteur & McLeod 2006). The recovery of kimberlite and magmatic Ni-Cu-PGE indicator minerals requires heavy liquid separation at SG 3.2 g/cm^3 (using dilute methylene iodide) to include the lowest density indicators Cr-diopside and forsteritic olivine. Recovery of porphyry Cu indicator minerals requires separation at a specific gravity of 2.8 to 3.2 g/cm^3 in order to recover the mid-density indicators tourmaline (dravite), alunite, and turquoise (Averill 2007).

Ferromagnetic minerals may comprise a considerable portion of the post heavy liquid concentrate (e.g. Table 2, column F) and thus removing them decreases concentrate size. The ferromagnetic minerals and any steel contaminants from sampling or processing are removed using a magnetic separator and commonly set aside unless magnetite, pyrrhotite, or magnetic ilmenite are important indicator minerals.

The non-ferromagnetic fraction is commonly sieved into two or three size fractions (e.g. 0.25-0.5 mm, 0.5-1.0 mm, and 1.0-2.0 mm) for picking of indicator minerals; however the final size range will depend on the commodity sought. For example, kimberlite indicator minerals are most abundant in the 0.25-0.5 mm fraction (McClenaghan & Kjarsgaard 2007) and thus to maximize recovery and minimize counting time and cost, the finest size fraction is most commonly picked.

A specific size fraction of the non-ferromagnetic heavy mineral fraction may be further separated electromagnetically into fractions with different paramagnetic characteristics to help reduce the volume of material to be examined for indicator minerals (Averill & Huneault 2006). Paramagnetic minerals are not magnetic, but are weakly attracted into a magnetic field. Paramagnetic minerals with different degrees of paramagnetism can be separated from one another by varying the strength of the magnetic field produced by an electromagnet. Minerals such as diamond are nonparamagnetic; pyrope garnet, eclogitic garnet, Cr-diopside, and forsteritic olivine are nonparamagnetic to weakly paramagnetic; and Cr-spinel and Mg-ilmenite are moderately to strongly paramagnetic (see Table 1 in McClenaghan & Kjarsgaard 2007). If the non- or paramagnetic portion of the concentrate contains a significant amount of almandine garnet, it may be processed through a magstream separator to remove the orange almandine from similar looking eclogitic or pyrope garnets. Magstream separation divides the concentrate into (1) a fraction containing most of the silicates (e.g. pyrope and eclogitic garnet) and no almandine, and (2) a fraction containing ilmenite, chromite, and other moderately magnetic minerals, such as almandine (Baumgartner 2006).

INDICATOR MINERAL SELECTION AND EXAMINATION

Indicator minerals are selected from non-ferromagnetic heavy mineral concentrates during a visual scan, in most cases, of the finer size fractions (e.g. 0.25-0.5 mm, or 0.3-0.5 mm, 0.25-0.86 mm) using a binocular microscope. The grains are counted and a selection of grains is removed from the sample for analysis using an electron microprobe to confirm their identification. Methods for examining a sample for counting/picking vary from rolling conveyor belts to dishes/paper marked with lines or grids. If a concentrate is unusually large, then a split is examined and the indicator mineral counts are normalized to the total weight of the concentrate. If a split is picked, the weight of the split and the total weight should both be recorded. Not all grains counted in a sample will be removed for microprobe analyses. If this is the case, the total number of grains counted and the number of grains removed should both be recorded.

Indicator minerals are visually identified in concentrates on the basis of colour, crystal habit, and surface textures, which may include such features such as kelyphite rims and orange-peel textures on kimberlitic garnets (Garvie 2003; McClenaghan & Kjarsgaard 2007). Scheelite and zircon in a concentrate may be identified and counted under short-wave ultraviolet light because they fluoresce. Gold and PGM grains may be panned from preconcentrates that were prepared in such a way that the silt-sized fraction has been retained (e.g. tabling). The grains may be counted and classified with the aid of an optical or scanning electron microscope. Commonly, gold grains are classified according to their shape/degree of wear (e.g. DiLabio 1990; Wierchowiec 2002), as this can provide information about relative transport distances.

INDICATOR MINERAL CHEMISTRY

Mineral chemical analysis by electron microprobe, scanning electron microprobe (SEM), laser ablation ICP-MS, or secondary ion mass spectrometry (SIMS) may be carried out to determine major, minor, and trace element contents of specific indicator minerals (Jackson 2009) because mineral chemistry is used to confirm identity, establish mineral paragenesis, and in some cases deposit grade (e.g. Ramsden *et al.* 1999; Belousova

et al. 2002; Averill 2007; Scott & Radford 2007; Spry *et al.* 2009). For example, kimberlite indicator minerals are characterized by a specific range of compositions that reflect their mantle source and diamond grade (e.g. Fipke *et al.* 1995; Schulze 1997; Grütter *et al.* 2004; Wyatt *et al.* 2004; Grütter 2007). Gold, PGM, and sulphide grains may be analyzed to determine their trace element chemistry (e.g. Grant *et al.* 1991; Podlipsky *et al.* 2007) or isotopic compositions (e.g. Hattori & Cabri 1992).

QUALITY CONTROL

Project geologists should use a combination of blank samples (no indicator minerals), spiked samples (containing a known quantity of specific indicator mineral species), and field duplicates, as well as repicking of 10% of the heavy mineral concentrates to monitor a heavy mineral processing lab's potential for sample contamination and quality of mineral grain recovery and selection. In addition, heavy mineral processing and selection labs should be asked to report their own quality control monitoring procedures and test results.

SUMMARY

This paper has described some of more common procedures available in commercial labs for processing surficial media and rocks to recover indicator minerals for mineral exploration. The processing method used will depend on sample media, commodities being sought, budget, bedrock and surficial geology of the survey area, and processing methods used for previous batches. When reporting indicator mineral results in company assessment files, government reports, or scientific papers, it is helpful to report the lab name, processing methods used, and sample weights. Monitoring of quality control is essential at each stage in the processing, picking, and analytical procedures, and should be monitored both by the processing labs and clients. Geologists are encouraged to visit processing labs so that they have a clear understanding of the procedures being used and can discuss customizations needed for specific sample batches.

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REFERENCES

AVERILL, S.A. 2001. The application of heavy indicator minerals in mineral exploration. In: McCLENAGHAN, M.B., BOBROWSKY, P.T., HALL, G.E.M & COOK, S. (eds.) *Drift Exploration in Glaciated Terrain.* Geological Society of London Special Volume, **1985**, 69-82.

AVERILL, S.A. 2007. Recent advances in base metal indicator mineralogy: An update from Overburden Drilling Management Limited. *EXPLORE, Newsletter of the Association of Applied Geochemists*, **134**, 2-6.

AVERILL, S.A. & Huneault. R. 2006. Overburden Drilling Management Ltd: Exploring heavy minerals. *EXPLORE, Newsletter of the Association of Applied Geochemists*, **133**, 1-5.

BALLANTYNE, S.B. & HARRIS, D.C. 1997. Alluvial platinum-group minerals and gold in Alberta: Results from the "Orientation Studies Project" and their significance to exploration. In: MACQUEEN, R.W. (ed.) *Exploring for Minerals in Alberta.* Geological Survey of Canada Geoscience Contributions, Canada-Alberta Agreement on Mineral Development (1992-1995), Geological Survey of Canada Bulletin, **500**, 279-329.

BAUMGARTNER, M. 2006. Industry leading laboratory services for diamond explorers. *EXPLORE, Newsletter of the Association of Applied Geochemists*, **133**, 5-10.

BELOUSOVA, E.A., GRIFFIN, W.L., O'REILLY, S. & FISHER, N.I. 2002. Apatite as an indicator mineral for mineral exploration: Trace element compositions and their relationship to host rock type. *Journal of Geochemical Exploration*, **76**, 45-69.

CHERNET, T. MARMO, J. & NISSINEN, A. 1999. Technical note. Significantly improved recovery of slightly heavy minerals from Quaternary samples using GTK modified 3" Knelson preconcentrator. *Minerals Engineering*, **12**, 1521-1526.

DAVISON, J.G. 1993. Diamond exploration samples: Laboratory processing. In: SHEAHAN P. & CHATER, A. (chairmen) *Diamond: Exploration, Sampling and Evaluation.* Prospectors and Developers Association of Canada, Short Course Proceedings, Toronto, 27 March 1993, 315-341.

DiLABIO, R.N.W. 1990. Classification and interpretation of the shapes and surface textures of gold grains from till on the Canadian Shield. In: *Current Research, Part C.* Geological Survey of Canada Paper, **90-1C**, 323-329.

ENGLISH, B.L., DESBOROUGH, G.A. & RAYMOND, W.H. 1987. A mechanical panning technique for separation of fine-grained gold and other heavy minerals. United States Geological Survey Open File Report, **97-0364**.

FIPKE, C.E., GURNEY, J.J. & MOORE, R.O. 1995. Diamond exploration techniques emphasizing indicator mineral geochemistry and Canadian examples. Geological Survey of Canada Bulletin, **423**.

GARVIE, O.G. 2003. Surface textures found on kimberlite indicator minerals. In: Indicator Mineral Methods in Mineral Exploration. Prospectors and Developers Association of Canada, Short Course 2, 8 March 2003.

GRANT, A.H., LAVIN, O.P. & NICHOL, I. 1991. The morphology and chemistry of transported gold grains as an exploration tool. *Journal of Geochemical Exploration*, **40**, 73-94.

GREGORY, G.P. & WHITE, D.R. 1989. Collection and treatment of diamond exploration samples. In: ROSS, J. (ed.) *Kimberlites and Related Rocks Volume 2 Their Crust/Mantle Setting, Diamonds and Diamond Exploration.* Geological Society of Australia Special Publication, **14**, Blackwell Scientific Publications, Oxford, 1123-1134.

GRÜTTER, H.S. 2007. Application of new-age clinopyroxene and garnet thermobarometry techniques in Arctic diamond exploration. In: *Indicator Minerals in Mineral Exploration, Workshop No. 3.* Exploration '07, Toronto, 27-32.

GRÜTTER, H.S., GURNEY, J.J., MENZIES, A.H. & WINTER, F. 2004. An updated classification scheme for mantle-derived garnet, for use by diamond explorers. *Lithos*, 77, 841-857.

HATTORI, K. & CABRI, L.J. 1992. Origin of platinum group mineral nuggets inferred from osmium-isotope study. *Canadian Mineralogist*, **30**, 289-301.

Jackson, S. 2009. Mineral chemistry: Modern techniques and applications to exploration. In: *Indicator Mineral Methods in mineral Exploration, Workshop B.* 24th International Applied Geochemistry Symposium, 21-25.

LEAKE, R.C., BLAND, D.J., STYLES, M.T. & CAMERON, D.G. 1991. Internal structure of Au-Pd-Pt grains from south Devon, England, in relation to low temperature transport and deposition. *Transactions of the Institute of Mining and Metallurgy, Section B, Applied Earth Sciences*, **100**, B159-B178.

LEAKE, R.C., CHAPMAN, R.J., BLAND, D.J., STONE, P., CAMERON, D.G. & STYLES, M.T. 1998. The origin of alluvial gold in the Leadhills areas of Scotland: Evidence from interpretation of internal characteristics. *Journal of Geochemical Exploration*, **63**, 7-36.

LEHTONEN, M.L., MARMO, J.S., NISSINEN, A.J., JOHANSON, B.S. & PAKKANEN, L.K. 2005. Glacial dispersal studies using indicator minerals and till geochemistry around two eastern Finland kimberlites. *Journal of Geochemical Exploration*, **87,** 19-43.

LE COUTEUR, P.C. & McLEOD, J.A. 2006. Heavy mineral processing at Vancouver Indicator Processors Inc./Teck Cominco Global Discovery Lab. *EXPLORE Newsletter of the Association of Applied Geochemists*, **133**, 15-18.

MARMO, J., O'BRIEN, H.. & LEHTONEN, M. 2008. Diamond Exploration in Finland. *EXPLORE Newsletter of the Association of Applied Geochemists*, **138**, 8-11.

MAURICE, Y.T. & MERCIER, M.M. 1986. A new approach to sampling heavy minerals for regional geochemical exploration. In: *Current Research Part A.* Geological Survey of Canada Paper, **86-1A**, 301-305.

McCLENAGHAN, M.B. & KJARSGAARD, B.A. 2007. Indicator mineral and surficial geochemical exploration methods for kimberlite in glaciated terrain, examples from Canada. In: GOODFELLOW, W.D. (ed.) *Mineral Resources of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces and Exploration Methods.* Geological Association of Canada Special Publication, **4,** 983-1006.

McCLENAGHAN, M.B., PAULEN R.C., AYER, J.A., TROWELL, N.F. & BAUKE, S. 1998. Regional till and humus geochemistry of the Timmins-Kamiskotia (NTS 42A/11, 12, 13, 14) area, northeastern Ontario. Geological Survey of Canada Open File, **3675**.

McCLENAGHAN, M.B., KJARSGAARD B.A., KJARSGAARD I.M. & PAULEN, R.C. 1999. Mineralogy and geochemistry of the Peddie kimberlite and associated glacial sediments, Lake Timiskaming, Ontario. Geological Survey of Canada Open File, **3775**.

McCLENAGHAN, M.B., WARD, B.C., KJARSGAARD, I.M., KJARSGAARD, B.A., KERR, D.E. & DREDGE, L.A. 2002. Indicator mineral and till geochemical dispersal patterns associated with the Ranch Lake kimberlite, Lac de Gras region, NWT, Canada. *Geochemistry: Exploration, Environment, Analysis*, **2**, 299-320.

PEURANIEMI, V. 1990. Heavy minerals in glacial sediment. Chapter 10 In: KUJANSUU, R. & SAARNISTO, M. (eds.) *Glacial Indicator Tracing*. A.A. Balkema, Rotterdam, 165-185.

PODLIPSKY, M. YU., NESTERENKO, G.V. & KRIVENKO, A.P. 2007. Isoferroplatinum mineral assemblage in the northwestern Salair gold placer. *Russian Geology and Geophysics*, **48,** 291-298.

RAMSDEN, A.R., GRIFFIN, W.L. & FRENCH, D.H. 1999. Application of resistate indicator mineralogy to metalliferous exploration: Tourmaline. In: *Research Review*. CSIRO Exploration and Mining, 49-52.

SARALA, P. & PEURANIEMI, V. 2007. Exploration using till geochemistry and heavy minerals in the ribbed moraine area of southern Finnish Lapland. *Geochemistry: Exploration, Environment, Analysis*, **7**, 195-205.

SCHULZE, D.J. 1997. The significance of eclogite and Cr-poor megacryst garnets in diamond exploration. *Exploration and Mining Geology*, **6**, 349-366.

SCOTT, K.M. & RADFORD, N.W. 2007. Rutile compositions at the Big Bell Au deposit as a guide for exploration. *Geochemistry: Exploration, Environment, Analysis*. **7**, 353-361.

SILVA, M. 1986. Placer gold recovery methods. California Department of Conservation, Division of Mines and Geology Special Publication, **87**.

SPRY, P.G. & TEALE, G.S. 2009. Gahnite composition as a guide in the search for metamorphosed massive sulphide deposits. In: *Indicator Mineral Methods in Mineral Exploration, Workshop B*. 24th International Applied Geochemistry Symposium, 31 May 2009, Fredericton, New Brunswick, Canada, 27-34..

TOWIE, N.J. & SEET, L.H. 1995. Diamond laboratory techniques. *Journal of Geochemical Exploration*, **53**, 205-212.

WIERCHOWIEC, J. 2002. Morphology and chemistry of placer gold grains – indicators of the origin of the placers: An example from the East Sudetic Foreland, Poland. *Acta Geologica Polonica*, **52**, 563-576.

WYATT, B.A., Baumgartner, M., Anckar, E. & Grütter, H. 2004. Compositional classification of kimberlitic and non-kimberlitic ilmenite. *Lithos*, **77**, 819-840.