Indium and the Future of Critical Metals in Australia

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For Abby

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Declaration

I hereby declare that this thesis contains no material which has been accepted for the award of any other degree or diploma at any university or equivalent institution and that, to the best of my knowledge and belief, this thesis contains no material previously published or written by another person, except where due reference is made in the text of the thesis. This thesis includes three original papers published in peer reviewed journals. The core theme of the thesis is the assessment of critical metal resources and anthropogenic metal flows. The ideas, development and writing up of all the papers in the thesis were the principal responsibility of myself, the student, working within the Department of Civil Engineering under the primary supervision of Dr. Gavin Mudd. In the case of Chapters 3 to 5, my contribution to the work involved the following:

Thesis Chapter	**Title**	**Status**	**Nature and % of contribution**	**Co-author name(s) Nature and % of Co-author's contribution***	**Co-author(s), Monash student Y/N***
3	Indium: Key issues in assessing mineral resources and long-term supply from wastes	Published	70%, Concept + data collection + manuscript	Gavin Mudd: Concept + input into manuscript 20% Simon Jowitt: input into manuscript 10%	N N
4	The world's by-product and critical metal resources part II: A method for quantifying the resources of poorly reported metals	Published	70%, Concept + data collection + analysis + manuscript	Gavin Mudd: Concept + input into manuscript 20% Simon Jowitt: Concept + input into manuscript 10%	N N
5	The world's by-product and critical metal resources part III: A global assessment of indium	In press	70%, Concept + data collection + analysis + manuscript	Gavin Mudd: Concept + input into manuscript 20% Simon Jowitt: Concept + input into manuscript 10%	N N

I have not renumbered sections of submitted or published papers within the thesis. The undersigned hereby certify that the above declaration correctly reflects the nature and extent of the student's and co-authors' contributions to this work. In instances where I am not the responsible author I have consulted with the responsible author to agree on the respective contributions of the authors.

Student signature: [redacted] **Date: 11 February 2017**

Main Supervisor signature: [redacted] **Date: 11 February 2017**

Publications during enrolment

This PhD thesis consists of the following works published during candidature as the lead author:

1. WERNER, T. T., MUDD, G. M. & JOWITT, S. M. 2015. Indium: key issues in assessing mineral resources and long-term supply from recycling. *Appl. Earth Sci. (Trans. Inst. Min. Metall. B),* 124**,** 213-226.

   (This paper received the Mann Redmayne Medal 2015, jointly awarded by the Australasian Institute of Mining and Metallurgy and the Institute of Materials, Minerals and Mining.)

2. WERNER, T. T., MUDD, G. M. & JOWITT, S. M. 2017. The world's by-product and critical metal resources part II: A method for quantifying the resources of rarely reported metals. Ore Geology Reviews, 80, 658-675.

3. WERNER, T. T., MUDD, G. M. & JOWITT, S. M. 2017. The world's by-product and critical metal resources part III: A global assessment of indium. Ore Geology Reviews. In Press. http://dx.doi.org/10.1016/j.oregeorev.2017.01.015

These works were additionally published during the period of candidature as a co-author, but are not directly included in this thesis:

1. MUDD, G. M., JOWITT, S. M. & WERNER, T. T. The world's by-product and critical metal resources part I: Uncertainties, current reporting practices, implications and grounds for optimism. Ore Geology Reviews. In Press. http://dx.doi.org/10.1016/j.oregeorev.2016.08.008

2. GOLEV, A., WERNER, T., ZHU, X. & MATSUBAE, K. 2016. Product flow analysis using trade statistics and consumer survey data: a case study of mobile phones in Australia. Journal of Cleaner Production, 133, 262-271.

3. CIACCI, L., NUSS, P., RECK, B., WERNER, T. & GRAEDEL, T. 2016. Metal Criticality Determination for Australia, the US, and the Planet—Comparing 2008 and 2012 Results. Resources, 5, 29.

4. YELLISHETTY, M., HUSTON, D., GRAEDEL, T., WERNER, T., RECK, B. K. & MUDD, G. M. 2016. Quantifying the Potential for Recoverable Resources of Gallium, Germanium and Antimony as Companion Metals in Australia. Ore Geology Reviews. http://dx.doi.org/10.1016/j.oregeorev.2016.11.020

5. MUDD, G. M., JOWITT, S. M. & WERNER, T. T. 2017. The World's Lead-Zinc Mineral Resources: Scarcity, Data, Issues and Opportunities. Ore Geology Reviews, 80, 1160-1190.

6. ZHU, X., LANE, R., WERNER, T. T. 2017. Modelling In-Use Stocks and Spatial Distributions of Household Electronic Devices and Their Contained Metals Based on Household Survey Data. Resources, Conservation & Recycling. In Press. http://dx.doi.org/10.1016/j.resconrec.2017.01.002

These papers have been submitted as a co-author at the time of initial thesis submission:

1. NORTHEY, S. A., MUDD, G. M., WERNER, T. T., JOWITT, S. M., HAQUE, N., YELLISHETTY, M., WENG, Z. 2017. The Exposure of Global Base Metal Resources to Water Criticality, Scarcity and Climate Change. Global Environmental Change. Submitted Oct 2016.

2. NORTHEY, S. A., MUDD, G. M, WERNER, T. T. 2017. Natural Resource Depletion: How should we address the limitations? Natural Resources Research. Submitted Feb 2017.

The following peer-reviewed conference presentations were made during the period of candidature:

1. WERNER, T. T.; MUDD, G. M.; JOWITT, S. M., Mineral resources of indium and long-term supply form wastes, World Resources Forum: Asia-Pacific, Sydney, Australia, 1-2 Jun, 2015.

2. WERNER, T. T.; MUDD, G. M.; CIACCI, L. Critical metals in an export focused economy: The story of indium in Australia, International Society for Industrial Ecology Socio-economic Metabolism Asia-Pacific Conference (ISIE SEM AP), Nagoya, Japan, 28-30 Sep, 2016.

3. WERNER, T. T., MUDD, G. M. A detailed assessment of indium supply potential from primary production, Ecobalance 2016, Kyoto, Japan, 3-6 Oct, 2016.

Abstract

Many of the technologies upon which we depend require specialty metals to function. Indium is one such metal, used in technologies like solar panels and LCD displays, however it is considered to be at risk of future supply disruptions. In order to determine how we might use indium more efficiently in future, there needs to be a good understanding of its use in society, and its remaining resources.

In this thesis, the resources, mining, use and wastage of indium is explored in the Australian and global contexts. Chapter 3 reviews the literature on indium, showing that there have been large discrepancies in past estimates of indium resources. It also shows that significant amounts of indium can accumulate in mine wastes at the deposit level. In Chapter 4, a method for estimating the global resources of by-product/critical metals is developed, which addresses many of the uncertainties associated with such metals not being reported in the mining industry. It provides a much stronger theoretical basis for future assessments of quantities of by-products present in mineral deposits. Chapter 5 follows this by showing the full application of this method in an assessment of global indium resources. It shows that at least 356,000 tonnes of indium are present in over 1,500 deposits globally, many of which are in countries not currently producing refined indium. The supply chain for this metal can therefore be considered quite adaptable. In Chapter 6, a retrospective, product-level dynamic material flow analysis of indium in Australia is presented. It shows that significant quantities of indium are likely to have accumulated in tailings and slags in this country. Additionally, it shows that the quantities of in-use are orders of magnitude lower than what is present in mineral deposits or mine wastes. This strongly suggests that recycling is a less effective means to securing future indium supply. It is hoped that this research as a whole will facilitate a more nuanced discussion about how critical metals may be sourced sustainably in future.

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1. INTRODUCTION

We live in a period where the demand for materials is greater than ever before. Certainly populations have grown, however as individuals we have come to consume more and to depend more on technologies which require a greater variety of materials. The computer on which this thesis was written requires dozens of specialised metals to function, including indium for an LCD display, palladium for printed circuit boards and lithium for batteries. Many such metals were discovered in the 19th century, but have only recently seen widespread use in society. Today, we maintain close contact with specialty metals, although they are often hidden in everyday technologies, making them largely unfamiliar to the lay person. Yet recognisable or not, our future already appears inherently bound to their continued supply. Indeed, in order to meet many of the targets specified under the United Nations' Sustainability Development Goals, specialty metals like indium, gallium and the rare earths will be needed (e.g. Murakami et al. 2015).

This dependency is thought to be problematic, as many of the metals we have designed into important technologies are among the most geologically scarce. This has fostered much concern over their long-term availability (Moyer, 2010). Often these metals also have complicated supply chains as they are commonly produced as by-products, or are sourced from relatively few countries (Elshkaki and Graedel, 2015, Mudd et al., 2014, Nassar et al., 2015). This means that the supply of these metals can be inflexible to demand, and that some countries can access these metals more easily than others, creating a kind of geopolitical risk. This type of risk was clearly demonstrated in 2010 when China restricted its neodymium exports, leading to a notable spike in global neodymium prices and a rush to substitute neodymium for other metals in its major applications (Sprecher et al., 2015). Some governments and institutions have responded to supply concerns by considering the 'criticality' of metals (BGS, 2015, Jowitt, 2015, Skirrow et al., 2013, USDoE, 2011). This is a means of determining which metals are of greatest strategic importance, and involves the quantification of a metal's relative supply risks, importance to society and the environmental implications of its extraction (Graedel et al., 2012). Assessments of criticality have revealed a number of metals which usually rate highly (e.g. antimony, rare earths, gallium, and indium (BGS, 2015, USDoE, 2011)), potentially requiring metal-specific policy interventions in order to avoid or minimise the risk of supply restrictions in future. These interventions must incorporate some understanding of critical metal resources and supply dynamics. However, studies in these fields must often contend with limited data.

Australia is a global leader in the production of the primary commodities upon which many critical metals are typically produced as by-products (Skirrow et al., 2013), however it is somewhat surprising that little to no critical metal production takes place domestically. Without domestic refining capacity, the critical metals present in Australian mineral deposits are either wasted during mining and mineral processing, or are exported in an unprocessed form overseas. Australian consumers then routinely re-import these metals in the form of finished products like smartphones. However once used, these metals end their life cycles in landfill due to an absence of advanced recycling capacity and current consumer behaviour in Australia (e.g. Read, 2015). This life cycle is generally understood to be materially inefficient, however to date this has not actually been quantified. To determine how Australia might manage its supply of critical metals in future, there is an apparent need to:

1. Quantify the potential resources of critical metals globally and in Australia,
2. Identify the location and form of these resources, and
3. Characterise the life cycle of critical metals once extracted.

It is the aim of this thesis to address these concerns specifically for indium, a metal discovered as recently as 1863, yet already essential to modern society due to its use in LCD displays and renewable energy technologies. It has also been the subject of some debate in the past over its scarcity (Werner et al., 2015). In the following subsections, a description of the aims and methodologies employed in this PhD project is presented, followed by an outline of the succeeding chapters.

This PhD is a component of 'Wealth from Waste' collaboration cluster, a program led by Australia's Commonwealth Scientific and Industrial Research Organisation (CSIRO) and including the University of Technology, Sydney, Monash University, The University of Queensland, Swinburne University of Technology and Yale University. This cluster was established to explore the barriers and enablers to industrial ecology in Australia, determine the nature of Australia's above-ground stocks of metals, identify innovation and business models for advancing industrial ecology, and to identify transition pathways towards a more resource-efficient Australia. Many of these broader themes have driven the research into critical metals presented herein.

1.1 Research questions and tasks

The central aim of this study has been to increase understanding of critical metal resources and usage through an in-depth analysis of indium in Australia. More specifically, this thesis has aimed to answer the following research questions by conducting the tasks listed:

RQ1: What quantities of indium are available in global and Australian mineral deposits?

- **(a) What is currently known about global indium resources?**
- **(b) What quantities are present in known mineral deposits?**
- **(c) What do these quantities suggest about the global supply chain and scarcity?**

- Conduct a review of the state of reporting of indium and other by-product metals by the mining industry. Determine what has been previously estimated.
- Develop a database of indium-containing mineral deposits, collecting data on metal grade, resource tonnage, deposit type, mineralogy and geographical location.
- Where indium is known to be present within a deposit due to mineralogy or drilling data, but indium resource quantities are not reported, develop and conduct a process by which indium grades can be estimated.

RQ2: What wastes of indium are produced during mining and mineral processing?

(a) How much indium can accumulate in the mine wastes of an individual deposit over time?
(b) How much indium has accumulated in mine wastes at the national or global scale over time?

- Review estimates of the efficiency by which indium is produced as a by-product.
- Conduct case studies by collecting mining production and ore grade data and modelling the amount of indium accumulated in mining and processing wastes over time.

RQ3: What are the current stocks and flows of indium in the Australian economy?

(a) What quantities of indium have been imported to and exported from Australia over time?
(b) What quantities of indium have accumulated in-use over time in Australia and how do these compare to mineral resources?
(c) Is recycling a viable option for future indium supply?

- Collect import/export data for indium, the base-metals which host it and indium-containing products to identify flows into and out of Australia.
- Conduct a review of literature on the contents of indium in products and integrate these figures with the results of household surveys and Australian industry sales data to determine in-use stocks.
- Generate waste flows through estimation/review of stock residence times.
- Combine these factors in order to develop a historic, product level dynamic material flow model for indium in Australia.

1.2 Thesis outline

This thesis is structured according to the following chapters:

Chapter 2: Review of Literature

- The societal and academic need for the new research presented in this thesis is justified.
- The many studies conducted to date in the fields of economic geology and industrial ecology concerning critical metals and indium are introduced, and specific knowledge gaps are highlighted.

Chapter 3: Indium: Key issues in assessing mineral resources and long term supply from recycling

- An in-depth review of the issues around estimating the mineral resources of indium is conducted.
- A classification of the key geological deposit types hosting indium mineral resources is developed.
- The extent to which indium has accumulated in mine wastes over time is analysed.

- A review of indium potential from urban mining, i.e. end-of-life product recycling is presented.

Chapter 4: A method for estimating rarely reported metal resources

- Given the many uncertainties associated with previous estimates of the mineral resources of by-product metals, a new method is developed.
- The method is explained in considerable detail, and its potential application to a Pb-Zn deposit database to estimate indium is explained.
- This method involves the systematic creation and classification of resource deposit databases, and proxy analysis for estimating by-product metal grades in deposits where they are present, but not reported.
- The processes by which uncertainty can be quantitatively and qualitatively managed are explained in detail.

Chapter 5: A global assessment of indium mineral resources, scarcity and implications for future supply

- This chapter shows the full application of the methodological framework established and justified in Chapter 4, applied to indium.
- A new database of reported indium deposits is detailed.
- Three databases, totalling 1,512 deposits are analysed in depth to create a global assessment of the quantities of indium present in mineral deposits.

Chapter 6: A material flow analysis of the mining, trade and use of indium in Australia over time

- A top-down, retrospective, product level dynamic material flow analysis of indium is presented for Australia.
- The mining and refining of indium from zinc and copper for over a century are modelled.

- The trade, accumulation in use and accumulation in landfill of indium in Australia is modelled from 1988-2015

Chapter 7: Discussions

- The results of each study presented in Chapters 3-6 are linked and discussed as a whole.
- The research questions posed in Chapter 1.1 are adressed.
- Research uncertainties and avenues for future work and presented.

Chapter 8: Conclusions

- The findings of this thesis are summarised.

2. Review of Literature

Our knowledge of the resources, supply and societal use of metals has grown substantially over time. The field of economic geology, concerned with the quantification of extractable materials in mineral deposits, may be as old as civilisation itself. Yet only in recent years, with the aid of globalisation, computer simulation, the internet and increasing transparency in the mining industry, have we begun to piece together a worldwide picture of what remains to meet the needs of a growing global population. When it comes to analysing what happens to metals once extracted, there is a much shorter history of inquiry. Indeed, the field of industrial ecology, in part concerned with the quantification of anthropogenic stocks and flows of materials, is said to have started as recently as 1989 (Graedel and Lifset, 2016). However like economic geology, this field has also experienced rapid progression in recent decades, as the modelling of anthropogenic systems and the associated movement of metals through their life cycles has grown ever more sophisticated. Industrial ecology has come to be known as a sustainability science, leading it to attract more policy attention of late (e.g. Hill, 2016). With the development of criticality theory and methodologies, industrial ecology has extended into the examination of critical metals. It therefore plays an important role in analysing the use of these metals in specialty and strategically important applications, e.g. renewable energy and military technologies.

Despite much progress in these fields, there are still a great number of uncertainties and challenges faced in understanding how we extract and use metals. In the following sub-sections, these uncertainties and the problems they can create in developing policies to sustainably manage the extraction, production usage and waste management of many metals are outlined. More specifically, the literature which has focussed on the resources, stocks, flows and classification of critical metals is examined. From this, a number of research gaps which must be overcome in order to meet the sustainability challenges of this century, as defined by the Sustainable Development Goals set out by the United Nations, are defined.

A diverse range of studies, methods and ideas are presented, acknowledging the work of many others in the fields of sustainability science, industrial ecology and economic geology. It should be noted that this chapter presents a general overview of the primary research which positions this thesis as a whole. This literature review includes material which is also covered in the papers presented in Chapters 3 to 5 and in Appendix D. The literature reviewed in these sections is summarised below:

- The paper presented in Chapter 3 is itself structured as a review paper. This chapter reviews the many broad issues affecting indium demand, production and resources. It covers the historical processing, wastage, reporting and criticality of indium, as well as providing a brief review of issues around recycling for indium.
- The literature review in Chapter 4 provides and evaluation of the other methods used to estimate the mineral resources of metals, in particular by-product metals. It highlights the shortcomings of these methods, thus establishing the need for the new method developed as a component of this thesis.
- The literature review in Chapter 5 explains the potential outcomes of a supply disruption for indium, the specific benefits of detailed indium resource data and builds upon the review presented in Chapter 3 to provide a more comprehensive collection of definitions for indium deposit types and indium mineralogy.
- Additional co-authored works published during the period of candidature are presented in Appendix D, covering a broad range of themes relating to metal resources and supply.

For additional information on each topic, and in justifying the need for the research presented in this thesis, the reader is referred to the additional reviews of literature in these sections.

2.1 Sustainable resources and development

In 2015, the United Nations agreed on a set of goals which UN member states are expected to use to frame their decision making up to around 2030, in order to ensure prosperity and to protect the planet. Under each of the 17 goals, a number of specific targets are established. For Goal 7 (Ensure access to affordable, reliable, sustainable and modern energy for all) and Goal 12 (Ensure responsible consumption and production patterns), these targets include (Hoballah, 2014, Wu and Wu, 2014):

- *By 2030, ensure universal access to affordable, reliable and modern energy services*
- *By 2030, increase substantially the share of renewable energy in the global energy mix*
- *By 2030, double the global rate of improvement in energy efficiency*
- *By 2030, achieve the sustainable management and efficient use of natural resources*
- *By 2030, substantially reduce waste generation through prevention, reduction, recycling and reuse*
- *Encourage companies, especially large and transnational companies, to adopt sustainable practices and to integrate sustainability information into their reporting cycle*
- *By 2030, substantially reduce waste generation through prevention, reduction, recycling and reuse*

If these and many other targets are to be met, a considerable amount of new renewable energy capacity needs to be constructed in the next 15 years (see also IPCC, 2011). In turn, this will require a number of specialty metals to be supplied, including indium, gallium, tellurium, and selenium. These and other metals are necessary for renewable energy systems to function, but are argued to be at risk of supply disruptions (Candelise et al., 2012a, Candelise et al., 2012b). They must therefore be strategically managed such that their mining, transportation and use entails minimal wastage, and/or that they are recycled or reused where possible. As shown in Fig. 2.1, most metals are not functionally recycled at the end of their useable lifetimes. This suggests that there are significant improvements to be made for how many metals are managed.

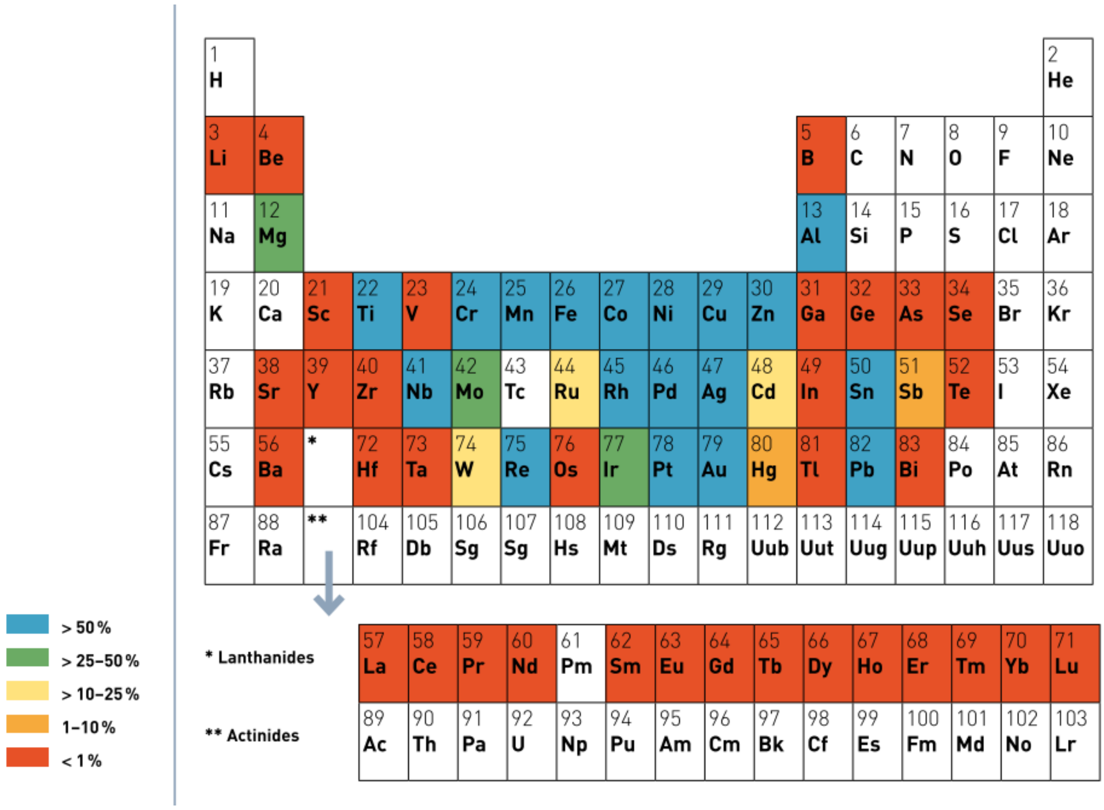
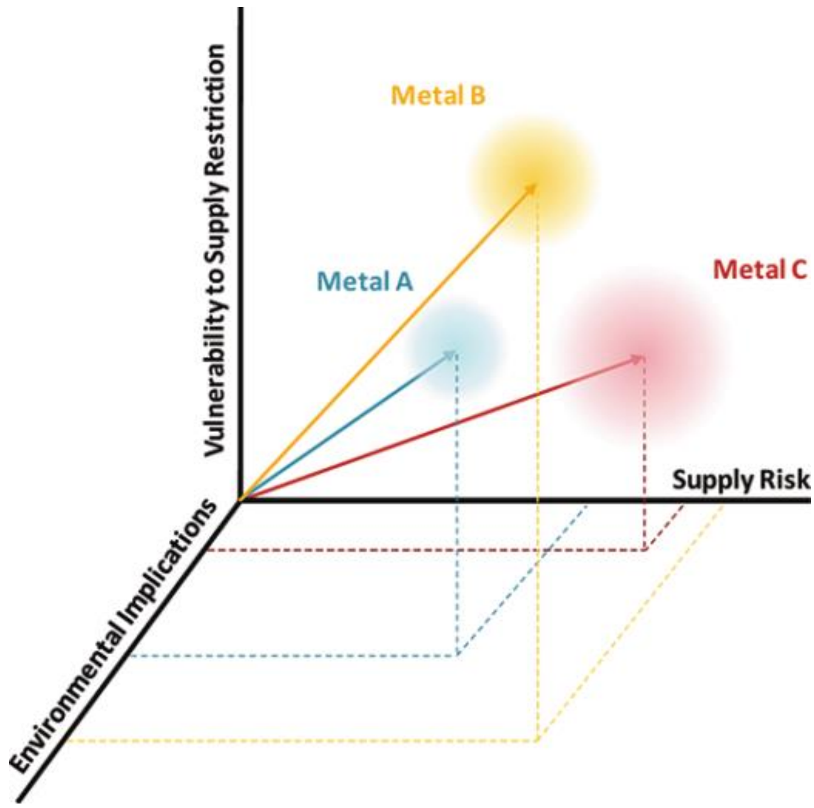


Figure 2.1: End of life recycling rate for 60 metals (sourced from UNEP, 2011)

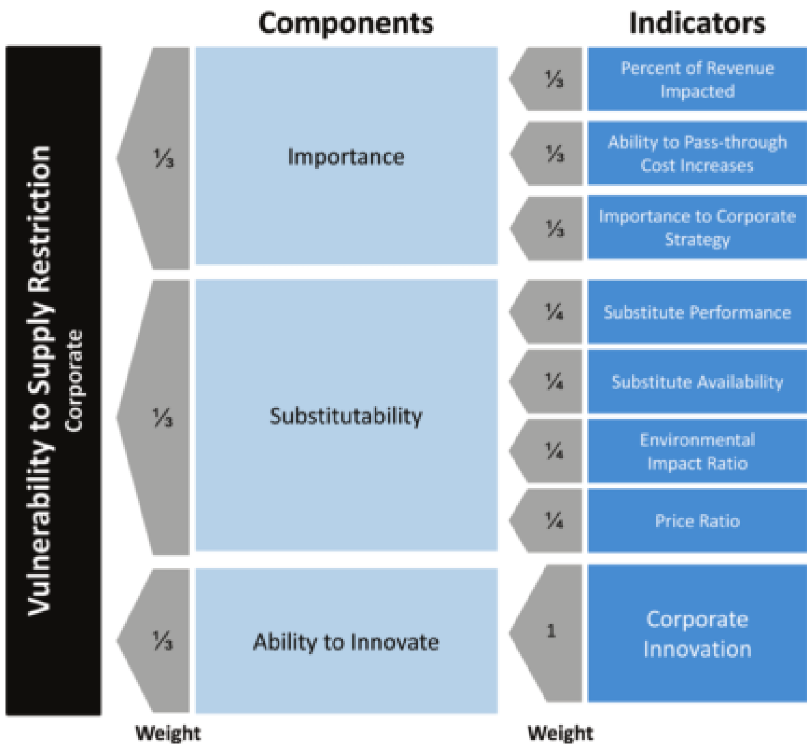
This distinct link between the supply of metals and meeting global sustainability challenges, plus the significant systemic improvements which need to be made in managing these metals, provides a clear mandate for more research into how we use metals, and into what can be done to use them more efficiently. Recently, much of this research has focussed on the potential for various countries to achieve a 'circular economy'. The Ellen MacArthur foundation, a major proponent of this concept defines a circular economy as "*one that is restorative and regenerative by design, and which aims to keep products, components and materials at their highest utility and value at all times, distinguishing between technological and biological cycles*" (Ellen MacArthur Foundation, 2017). Essentially, the circular economy entails the creation of material loops, such that once a metal is mined, it is used and reused or recycled as much as possible to minimise the need for continued extraction from new sources. In theory, doing so reduces the economy's total dependence on natural and energy resources, and also reduces pollution. This concept has been endorsed by the Chinese central government for some time, with some notable programs to increase material recycling and improve product design having been implemented (Yuan et al., 2006), and a circular economy law adopted in 2008 (Allwood et al., 2012). In Australia, the legislative framework needed to implement a circular economy is in its infancy, although momentum is said to be building (Giurco et al., 2014). For domestic policy to be effectively drafted such that that each UN member state achieves a circular economy (or as near to one as is thermodynamically possible), and meets the targets specified under the Sustainable Development Goals, it is a simple but necessary first step to perform baseline research. Intuitively, policies focussing on the management of metals (for example military stockpiling) will require knowledge of how much each metal is annually produced, used, recycled and wasted relative to its total scarcity, and relative to other metals. However to date, much of this information has been missing, particularly for specialty metals, and particularly at the country level at which it would be most policy-relevant (Mudd et al., 2017a). For Australia, it is instructive to first take a brief step back and look at the concept of criticality to understand and establish which metals are indeed critical in this country, before progressing to collecting data on their usage. This process is reviewed in the following section.

2.2 Metal criticality

The concept of criticality begins with the recognition that some metals are more important to society than others. Perhaps they are essential to the function of important technologies, or maybe their supply being disrupted would have greater impacts to the economy. Similarly, the extraction of some metals is far more environmentally harmful than others. For example, the variability in factors such as ore grade and depth between different deposits can mean substantial differences in the amount of energy required to extract a metal (Mudd, 2010). These factors, among others that ultimately determine a metal's impact on society, contribute to what we would term metal criticality. In Fig. 2.2 (a), we see a matrix depicting how metal criticality can be measured by the environmental implications of its extraction, its risk of a supply disruption, and the impact on society if supply were indeed disrupted.



(a)



(b)

Figure 2.2: (a) Metal criticality matrix and (b) Components and indicators of the vulnerability to supply restriction metric. Sourced from Graedel et al. (2012)

Each of these factors is measured by a number of contributing variables. An example for vulnerability to supply restriction is shown in Fig. 2.2 (b). It can be seen, for example, that if a particular metal cannot be easily substituted for its major applications, and if those applications are important to society, we would be more vulnerable to a restriction of the supply of that metal.

When various governments and institutions have measured these factors to determine the strategic importance of various metals, a number of specific elements have consistently rated as being the most critical. These include rare earth elements, antimony, tungsten, indium, tellurium, gallium, germanium and others. Often, this is because their supply is dominated by relatively few countries, usually including China (Fig. 2.3 and BGS, 2015).

United Kingdom[1]	EU low C energy[2]	European Union[4]	United States DoE[5]	South Korea[7]	Japan[8]	Willis and Chapman (2012)[9]	This study[10] *ranking	Score[10]
REE	REE	Antimony	Heavy REE	Gallium	Manganese	Beryllium	*REE	29
Tungsten	Tellurium	Beryllium	Tellurium	Indium	Chromium	Gallium	*Gallium	29
Antimony	Gallium	Cobalt	Indium	Lithium	Nickel	Indium	*Indium	26
Bismuth	Indium	Fluorspar	Lithium	Magnesium	Molybdenum	Magnesium	*Tungsten	23
Molybdenum	Niobium	Gallium	Cobalt	Nickel	Cobalt	PGE	*PGE	22
Strontium	Vanadium	Germanium	Gallium	PGE	Vanadium	REE	*Cobalt	21
Mercury	Tin	Graphite	Manganese	REE	Tungsten	Tin	*Niobium	20
Barium	Selenium	Indium	Nickel	Silicon	Indium	Tungsten	Magnesium	17
Graphite	Silver	Magnesium	Light REE	Titanium	Gallium	Antimony	*Molybdenum	15
Beryllium	Molybdenum	Niobium	Magnesium	Tungsten	PGE	Cobalt	*Antimony	14
Germanium	Hafnium	PGE	Vanadium	Zirconium	REE	Germanium	*Lithium	14
Niobium	Nickel	REE		Antimony	Niobium	Manganese	*Vanadium	13
PGE	Cadmium	Tantalum	**US DoD6**	Chromium	Tantalum	Nickel	*Nickel	13
Cobalt		Tungsten	Zinc	Cobalt	Strontium	Niobium	*Tantalum	13

Figure 2.3: Summary of criticality rankings from between countries (sourced from Skirrow et al., 2013)

A higher rating of metal criticality for these metals technically means that they should be the target of research on resources, supply and usage, yet this has not historically been the case. Indeed, it would appear that there are significant unknowns when it comes to critical metal resources, supply and usage. The final two columns shown in Fig. 2.3 establish which metals are of most strategic interest in the Australian context (albeit estimated with a less robust methodology than is given in Graedel et al., 2012). It indicates rare earth elements, gallium and indium as the top three metals of strategic interest. Rare earth element resources have been studied in some detail in a recent study by Weng et al. (2015). Gallium resources have also been examined to some extent by Frenzel et al. (2016), highlighting indium as a candidate for assessment in this thesis. A review of literature on indium, including a brief history of its demand is provided in Chapter 3. It is worth noting that a more robust study of metal criticality has been conducted for Australia in a co-authored study, although accounting only for 6 metals (see Ciacci et al., 2016, included also in Appendix D).

Having established the metals requiring focus in Australia, the logical following step is to begin collecting data on their usage and scarcity. This naturally begins with an understanding of mining.

2.3 Mining and mineral resources

Earth's crust contains virtually all of the elements in the periodic table, except those elements which are highly unstable and only exist under extreme laboratory conditions. Such elements do not yet have any practical applications, and hence all the metals which are of common use to society must first be mined. These elements have formed as a result of various geological processes over many millions of years, and so relative to human lifespans, they are considered non-renewable. Their extraction takes place largely within the private sector, where mining companies explore Earth's crust for ore. Ore may be defined as "*a naturally occurring solid material containing a useful commodity that can be extracted at a profit*" (Arndt et al., 2015). This aspect of profitability means that in most cases, only those locations where metals are particularly enriched are typically available to society. It is important to also recognise that economic conditions can change, as technological improvements allow for different types of ore to be processed, or new locations to be reached. Hence, what we might have classified as ore in the past can be different to today. These possible changes are technically captured by the multiple classifications for resources used in the mining industry (see Fig 2.4).

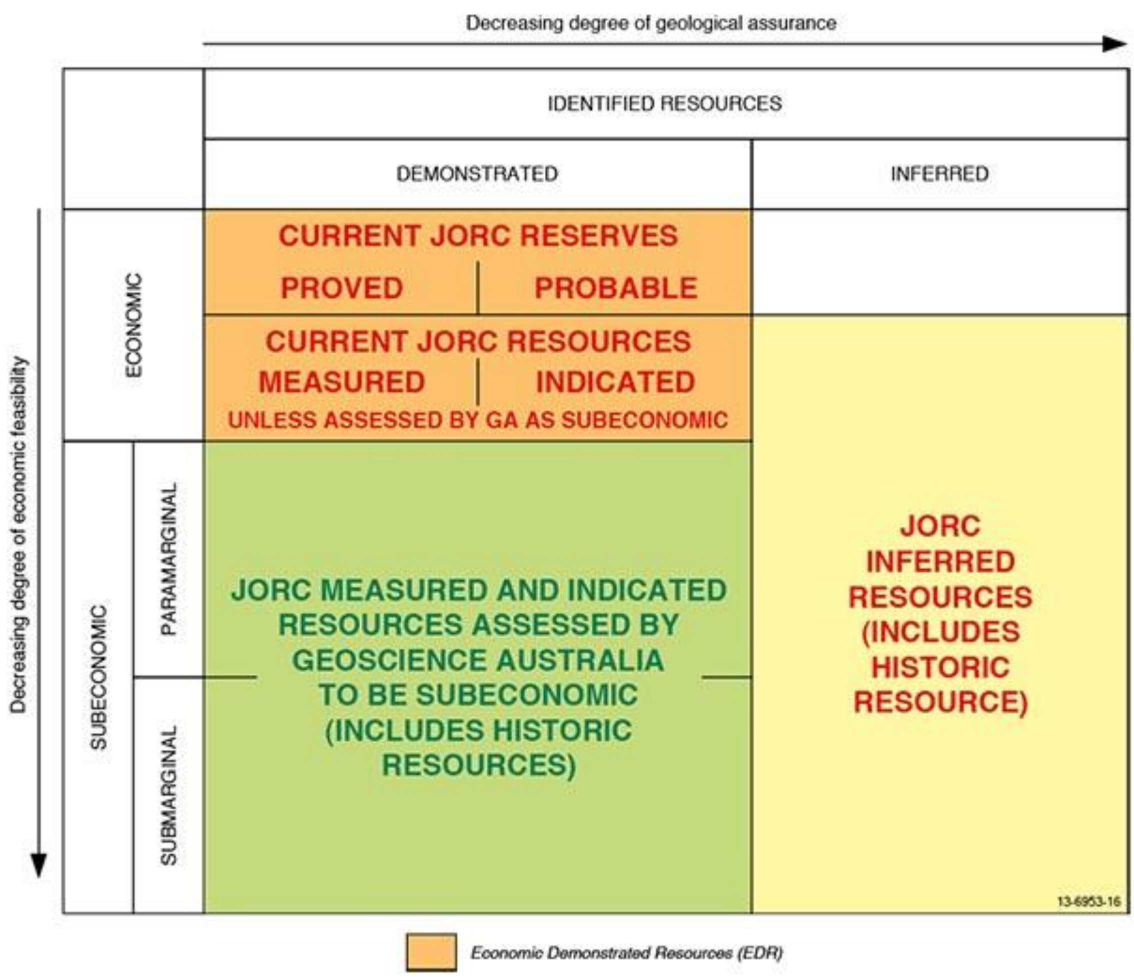


Figure 2.4 Mineral resource classification scheme, based on the McKelvey system used by Geoscience Australia (GA) and the United States Geological Survey (USGS) among others (sourced from Geoscience Australia, 2009)

Generally speaking, the portion of a mineral deposit which is economic to extract for a company under current conditions would be termed "reserves", which are somewhat akin to the bread that a baker has on their shelves for the day. The "resources" are the portion of a mineral deposit which might be extractable in future. There are additional classifications for metals which are not yet discovered. These may be in entirely new locations, or simply beyond what had previously been drilled at a known deposit. However, it is the resources in known location that must be determined in order to properly establish a country's endowment of important commodities. Resource classifications are important in ensuring transparency in the mining industry, and are applied under what are known as mineral resource reporting codes. These codes, such as the Joint Ore Reserves Committee (JORC) code, used in Australia, or the National Instrument NI 43-101 code in Canada, serve as guidelines to ensure that an exploration company has employed the best practice in quantifying and reporting what exists in a deposit (see Geoscience Australia, 2009). The reports or public statements made according to these codes are considered the best sources of information to the public of what the mining industry thinks is still extractable in a project (Mudd et al., 2017a). When looking at resources at a global level, it follows that the best estimates must be derived from a compilation of code-based estimates. This kind of compilation has previously only been created for Ni, Cu, Pb-Zn, Co, platinum-group elements (PGEs) and rare-earth elements (REEs) in the peer-reviewed literature (see Mudd, 2012, Mudd and Jowitt, 2014, Mudd et al., 2017b, Mudd et al., 2013a, Mudd et al., 2013b, and Weng et al., 2015), leaving most of the periodic table unstudied in such a robust manner. This is likely because in their current form, the reporting codes fail to adequately account for metals which are typically produced as by-products. Often, base metal concentrates include smaller quantities of metals which are not accounted for by mining companies, but are nonetheless separated and sold by smelters/refineries downstream. As such, the miner has no economic incentive to report these by-product metals, making it problematic to estimate their global resources. Perhaps unsurprisingly, there have been no code-based compilations of deposits containing indium resources.

In lieu of code-based report compilations, it is common practice to consult governmental agencies such as the USGS or Geoscience Australia (GA), which are generally the most cited sources of mineral commodity-related information. Looking at the most recent Mineral Commodity Summaries published by the USGS, quantitative estimates are not available for hafnium, indium, thallium, scandium, gallium, and cadmium (see Fig. 2.5). Similarly, there appear to be limited estimates for tellurium, strontium, rubidium, niobium, mercury, helium, graphite, cerium, bismuth, beryllium and antimony in this series of publications (e.g. having rough estimates for reserves, but none on resources, USGS, 2016). This lack of clear information within mining company reports or the major research institutions tasked with assessing global resources is clearly problematic. In Chapters 3 and 6, the particular implications of a lack of clear resource information in the context of indium are discussed further.

INDIUM

China returned to being a net exporter of indium in 2015, owing mostly to a lack of domestic investor demand, as well as to higher international prices compared with domestic market prices in the second half of the year and an elimination of the domestic export tariff on indium.

China's ITO production capacity has increased notably in the past few years, along with the country's expanding flat-screen display industry. Within the past year, a China-based company commissioned an ITO production plant in Henan Province, and a Belgium-based ITO producer, in a joint venture with a China-based materials company, constructed a 200-metric-ton-per-year ITO plant in Guangdong, China, which could be commissioned by the end of 2015, at the earliest.

World Refinery Production and Reserves:

	Refinery production[e] 2014	Refinery production[e] 2015
United States	—	—
Belgium	25	25
Canada	65	65
China	460	370
France	43	38
Germany	10	10
Japan	72	72
Korea, Republic of	150	150
Peru	14	15
Russia	5	10
World total (rounded)	844	755

Reserves[6]

Quantitative estimates of reserves are not available.

World Resources: Indium is most commonly recovered from the zinc-sulfide ore mineral sphalerite. The indium content of zinc deposits from which it is recovered ranges from less than 1 part per million to 100 parts per million. Although the geochemical properties of indium are such that it occurs in trace amounts in other base-metal sulfides—particularly chalcopyrite and stannite—most deposits of these metals are subeconomic for indium.

Figure 2.5: Example of USGS reporting of global indium resources (modified from (USGS, 2016))

If indeed resource data are available for a metal, there are still uncertainties which must be contended with. For example, it is known that the published values of reserves and resources are dynamic, i.e. changing over time. There are many cases where we would observe resources becoming reserves over time, as a mine rarely closes upon depletion of its reserves only. Take the Cannington Pb-Ag mine in Queensland, Australia (Fig. 2.6a), which was estimated to contain about 5 Mt Pb in 1997. It has since produced over 4 Mt Pb, but is still estimated to have another 4 Mt Pb remaining. Similar trends can be seen at the national level, as shown in Fig. 2.6b, suggesting that any reports of what remains today is merely an indication of what is available for future generations, and by no means a definitive figure. Nonetheless, a method by which the resources of critical and/or by-product metal resources can be estimated is needed. This method must overcome the uncertainties created by a lack of reporting from conventional sources. It is the intention of the paper presented in Chapter 4 to present such a method, and in doing so, allow research questions 1a, 1b and 1c to be addressed.

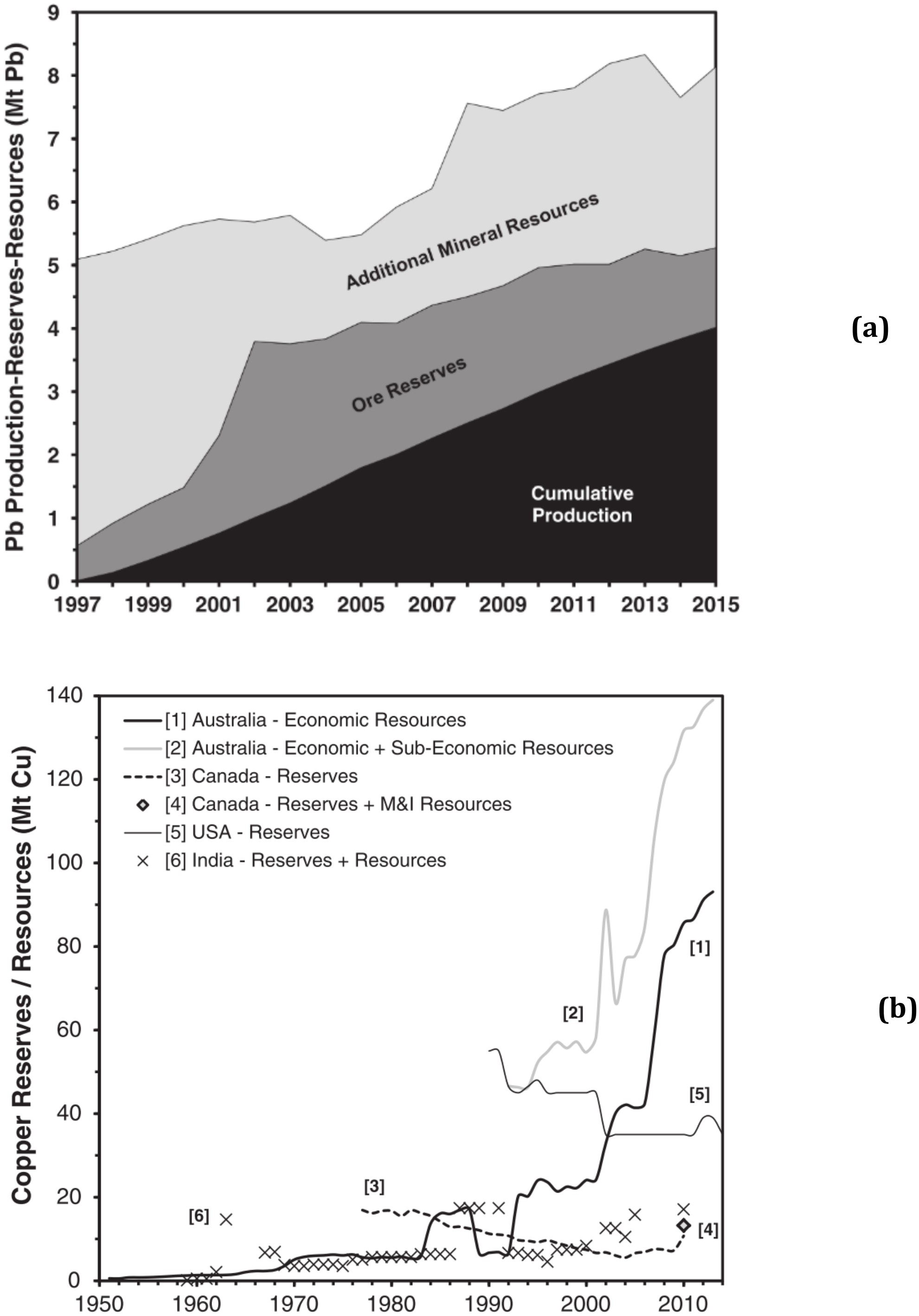


Figure 2.6: (a) Resources, reserves and cumulative production at the Cannington Pb-Ag mine, Queensland Australia, 1997-2015 (sourced from Mudd et al., 2017b), (b) Long-term trends in national estimates of copper reserves and/or resources (sourced from Mudd et al. 2017a).

2.4 Material cycles

Once mined, a metal has entered the early stages of its life cycle. The carbon or nitrogen cycles are well known, and are dominated by climatic, geological and biological processes which transform and transport these elements between various natural and human reservoirs. Given that the cycles of many metals are strongly influenced by human activities, they can change rapidly over short periods of time. For this reason, they must be studied more frequently, and are often not so well understood. The degree to which various elemental cycles are more influenced by human or natural forces was documented by Klee and Graedel (2004, Fig. 2.7). With a greater variety of elements in use for manufacturing new technologies, we can expect that more element cycles will become human-driven over time, and hence require more frequent study. In Fig. 2.7, indium can be seen to be the 19th most human-driven element, although global demand for this metal has increased considerably since 2004 (see Fig. 1a of Chapter 3).

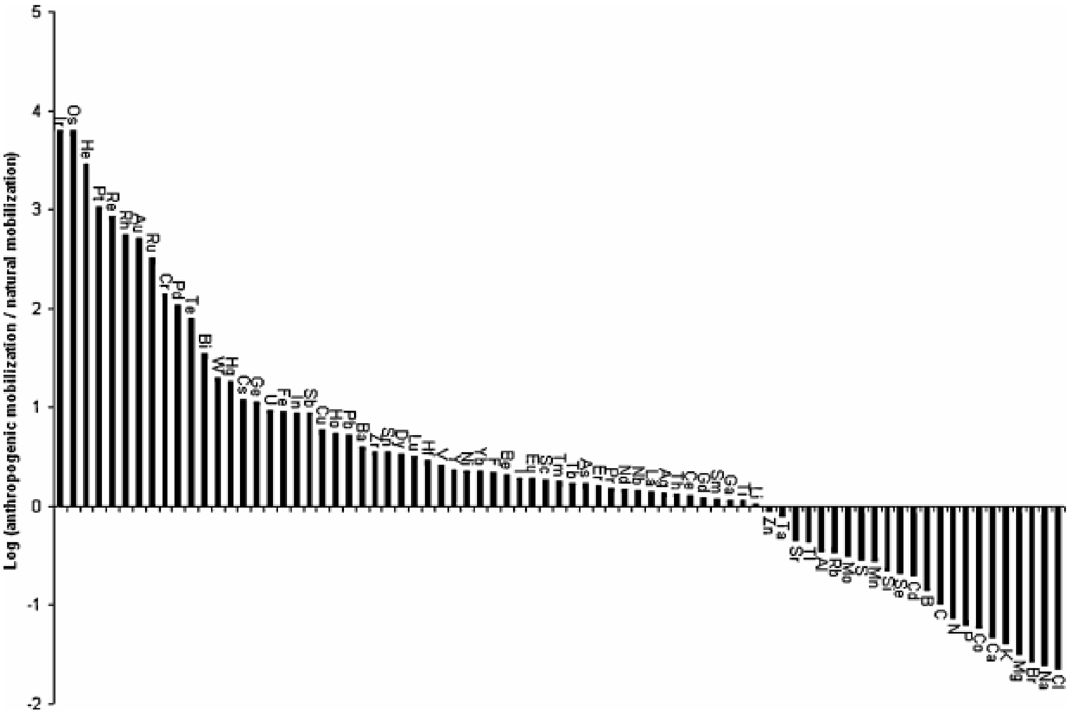


Figure 2.7 Graphic comparison of the ratio of anthropogenic and natural mobilization flows. Human activities mobilize more of each element on the left side of the diagram; natural processes mobilize more of each element on the right side of the diagram (sourced from Klee and Graedel, 2004)

Element cycles are studied using different methodologies depending on whether they are dominated by natural processes or human activities. For example, in a human-driven element cycle, particular focus is placed on how the element is currently used in order to determine where the bulk of it is likely to reside outside of mineral deposits, or whether recycling is a viable option for future supply. Currently the best approaches to documenting and quantifying the anthropogenic life cycle of metals are seen in so-called material flow analysis (MFA) studies. Figure 2.8 shows how the cycle of zinc can be separated into various processes, reservoirs and flows. The methodologies for populating this diagram with real data and managing uncertainty are reasonably well-established (e.g. Buchner et al., 2015, Chen and Graedel, 2015, Fischer-Kowalski et al., 2011, Graedel et al., 2015 and Meylan and Reck, 2016), however obtaining the necessary data for each of these stages remains a persistent challenge, depending on the quality of reporting from various sectors of the economy, and the expertise available on the particular metal of interest.

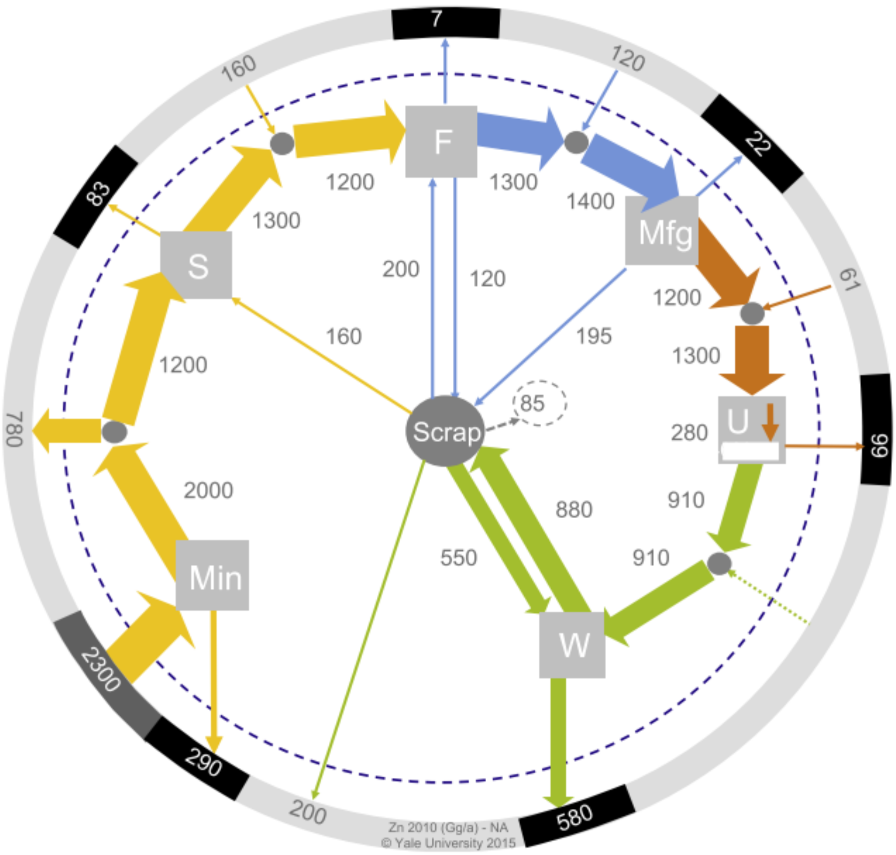


Figure 2.8: Circular material flow diagram developed to visualise the North American zinc cycle (sourced from Meylan and Reck, 2016)

Given the relative recency of national scale studies of material flows, it is perhaps unsurprising that many of the lesser used specialty metals have not been adequately studied, likely due to their lower overall economic value in comparison to say, iron, nickel or zinc. Nonetheless, the need for such studies remains. In the case of indium, the following material flow studies have been identified:

Table 2.1 Summary of studies examining indium material flows

Study	Scope	Comments
Endo et al. (2007)	ITO in Japan / World - 2003	Estimated the depletion of In reserves by 2010. Based on 2003 data, though some projections made to 2025.
Nakajima et al. (2007)	ITO/FPD's in Japan	Confirms major role of ITO, quantifies the environmental consequences of In dissipation. Discusses possibility of In resources depleting before 2020.
Yoshimura et al. (2011)	Japan / World – 2004	Results for Japan extrapolated to the world scale. Determines the greatest recovery potential to be during mining and smelting stages.
Goonan (2012)	United States – 2008 to 2009	Includes historical data on production and stockpiling.
Marwede et al. (2012)	LED products in Europe	Focus on selected target material in LED products, not specifically focussed on In
White and Hemond (2012)	World -2007	Anthrobiogeochemical cycle, looking at non-human flows as well.
Peiró et al. (2013)	Multiple metals, World – 2010	Covers multiple by-product metals and their flows for multiple products for the year 2010. Does not account for trade between countries.
Yoshimura et al. (2013)	Japan - 2008 and World - 2004	Substance flow analysis of indium for multiple product categories. Full cycles depicted only for the years shown only, but for indium in ITO products, multiple year trends are shown.
Zimmermann (2013a)	World, in PV technologies, prospective	A prospective, dynamic analysis of In in photovoltaic systems up to 2050.
Zimmermann (2013b)	World, in PV technologies, historical/prospective	Historical and prospective analysis of in in photovoltaic systems.
Zimmermann and Gößling-Reisemann (2013)	Multiple metals – no particular timeframe	Screening study of metal dissipation. Not a cycle characterisation, but provides data relevant to material flows.
Chang et al. (2015)	Taiwan - 2011	Substance flow analysis of indium in multiple product categories.
Licht et al. (2015)	World - 2011	Substance flow analysis of indium, gallium and germanium at the global level.
Choi et al. (2016)	World – 2008-2058	System dynamics model investigating causal loops under different clean energy deployment scenarios.

It can be seen that although very useful contributions, there are important limitations:

- For the studies conducted at the world scale, a common weakness is that they have not captured the trade of indium between countries
- Often these studies have been limited to a single year of data.

- The studies conducted at the national scale are useful for policy within those countries, however only the United States, Japan and Taiwan are specifically identified here. Thus there are many countries, particularly those which might have the most indium resources, which have not yet been studied in such a way.
- It is common for only selected applications of indium (e.g. thin film applications) to be encompassed in a study.
- A dynamic MFA (i.e. across two or more years) has not been conducted for applications other than photvoltaics in the studies listed.
- None of the identified studies have encompassed robust mineral resource data in addressing themes of scarcity.

In response to these gaps, and in addressing research questions 2b, 3a, 3b, and 3c, it is the focus of Chapter 6 to develop a product-level, dynamic MFA for indium in Australia, covering commodity trade from 1988-2015 and a detailed assessment of indium stocks and flows through mining from the mid to late 1800's. In doing so, a comprehensive study of stocks and flows outside of the current refined indium supply chain, but strongly related to resources, is provided.

2.5 Summary

It has been established that managing the supply of metals necessary for meeting the sustainable development goals will require:

1. An understanding of the strategic importance of various metals, in order to determine which might require some form of policy intervention.
2. A robustly compiled and assessed database of deposits and mineral resources, based on code-based estimates where possible, which enables the location of current and future resources and supplies to be determined. This in turn requires a methodology which is able to account for the lack of reporting of many by-product metals.

3. A detailed, national-level material flow account of the supply, usage and waste management of the metals, preferably with historical information which can be used to identify trends and therefore predict future behaviour.

There are few cases where all of these aspects can be argued to have been well studied for a metal. For example, Pb-Zn deposits have been quite comprehensively documented in Mudd et al. (2017b), and this information may be coupled with a study like Meylan and Reck (2016) to provide a detailed view of Zn resources, supply and usage at the global scale. Given the timing of these studies, it was clearly not possible to combine these perspectives, but nonetheless Zn would be considered as among the more well-studied metals. Cu too can be argued to have been well-studied, having been the subject of significant research on resources (Mudd et al., 2013a), stocks and flows (Glöser et al., 2013), yet previously this could not be said for the critical metals. The research questions in Chapter 1.1 have been framed such that these three aspects are investigated for indium in the following sections.

3. A REVIEW OF INDIUM REPORTING, GEOLOGY, AND LONG-TERM SUPPLY FROM MINE WASTES AND RECYCLING

This chapter presents a review paper, establishing the need to study indium resources, stocks and flows. The nature of indium demand, production and resources is reviewed, showing what data are available on indium, and what trends are evident. The efficiency of indium mining is reviewed, and the amount of indium wasted from the production of zinc over time is also estimated. Additionally, indium deposit type classifications are developed. Importantly, this paper shows that very little agreement could be found among prior studies for indium, suggesting that somewhere between 16,000 and 570,000 tonnes of indium are available in global mineral resources, providing a basis for assessing research question 1. Such a large discrepancy poses significant challenges for the formation of indium-related resource policy. Furthermore, in addressing research question 2, this paper shows through case studies of mine waste in Canada, that several years' worth of global supply can accumulate in a single mine site, highlighting the potential value in future efforts to reprocess tailings. Supplementary information for this chapter is provided in Appendix A.

Indium: key issues in assessing mineral resources and long-term supply from recycling

T. T. Werner*[1], G. M. Mudd[1] and S. M. Jowitt[2]

Indium and other geologically scarce metals are routinely integrated into green technologies and modern consumer electronics. The manufacture of solar cells and liquid crystal displays (LCDs) relies strongly on continued indium supply, yet very little research has been conducted to determine what total resources exist to meet future or even present needs. This paper provides an improved understanding of the nature of indium resources and the current and future production and supply of this critical metal through a summary of global trends in indium production and demand, and through a preliminary account of global code-based reporting of indium mineral resources. Authors also present an overview of the potential for indium extraction from mine wastes and recycled electronics using Canadian and Australian case studies. Our preliminary data suggest that considerable resources are likely to exist in a diversity of deposits globally, which have the potential to meet long-term demand for indium. However, it is clear that a secured future supply of this metal will require some shift of focus from conventional extraction practices. It will be necessary to revisit controls on the conversion of resources to reserves and to supply and the discovery of additional resources to replace those depleted by continuing production. The prospects for indium supply from mine wastes and recycled electronics are found to be substantial, and these sources warrant greater consideration given their probable environmental and social advantages over the discovery and development of new primary indium deposits.

Keywords: Indium, Indium resources, Mineral resources, Urban mining, Critical metals, Critical minerals, Companion metals, Tailings, Slag, Metal recycling, Indium mineralogy, Indium processing, WEEE recycling

Introduction

Throughout history, the supply and utility of metals has been key to the development and progress of human society. Just as the use of bronze or iron once defined an age, the present information age can also be defined by the use of particular metals. Indium is arguably one of today's defining metals, having been discovered relatively recently, yet already embedded into the renewable energy, automotive, aerospace and consumer electronics industries (Moss *et al.* 2011). Demand for indium has grown exponentially since the 1970s, in particular due to the use of indium in liquid crystal displays (LCDs) and solar panels (Schwarz-Schampera, 2014; Tolcin, 2014), although concern over the security of future supply has led some governments and institutions to classify indium as a so-called 'critical metal' (Moss *et al.* 2011; Skirrow *et al.* 2013, European Commission 2014a, 2014b). While indium is certainly one of the least abundant elements in the earth's crust at 0.052 ppm (copper, by comparison, is approximately 520 times more abundant, Skirrow *et al.* 2013), the concern over supply is based on very limited knowledge of future or even current indium resources. Indeed, some research has suggested that stock estimates of critical metals, including indium, are 'almost non-existent' [United Nations Environment Programme (UNEP) *et al.* 2010, p. 25]. There are a few sources of information on indium resources (e.g. Ishihara, Murakami and Marquez-Zavalia 2011; Skirrow *et al.* 2013, Schwarz-Schampera 2014), although none of these contain a global summary of known indium resources using mineral resource reporting standards (e.g. JORC, SAMREC, PERC, NI 43-101, etc.). These standards (or codes) assist in the application of best practice methodology for mineral resource estimation within the minerals industry; this makes them crucial to analyses of resource quantities, as the magnitude of extractable metals ultimately depends on what the minerals industry deems profitable (Mudd, Weng and Jowitt 2013a).

Addressing this gap, the authors generate a preliminary collection of publicly reported indium resources using code-based reporting standards. This is accompanied with a detailed discussion of the factors that influence indium production, demand and availability. Authors further provide descriptions of indium-associated deposit types and discuss the best possible

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approach for future accounting studies, entailing possible statistical inferences from mineralogy, deposit type and historical production. Recognising that the long-term availability of indium should not be restricted to its geological abundance, the authors also analyse other measures that can contribute to increased supply of this critical metal, including:

- Enhanced recovery from the processing of raw indium ores.
- Exploration of new resource types such as mine tailings and slags, including a case study of selected Canadian zinc projects.
- Enhanced end-of-life (EOL) recovery from recycled electronic products, presented via a brief case study of end of life mobile phone recycling in Australia.

Authors conclude by reviewing the nature and sustainability of future indium supply, and by highlighting the future work necessary to solidify our current understanding of indium criticality.

The nature of indium demand, production and resources

A brief history of indium demand

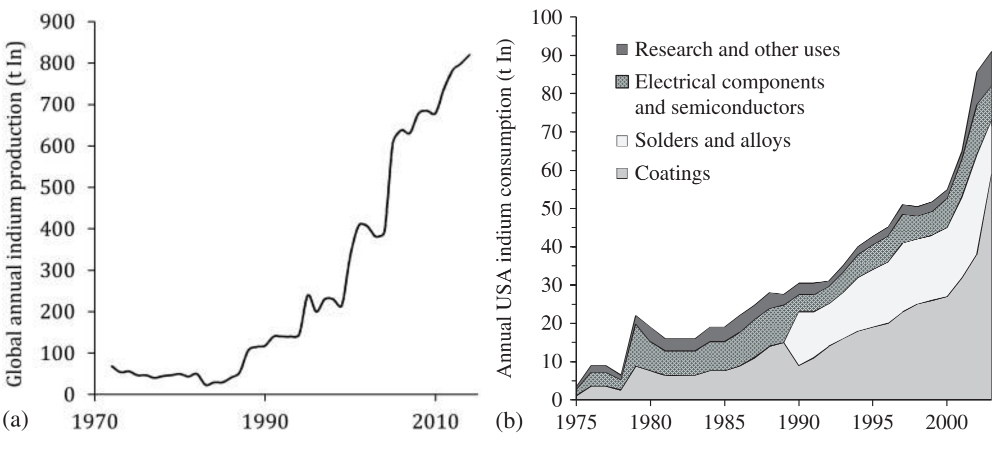
Indium was first discovered in samples of zinc sulphide ore at the Freiberg School of Mines, Germany, in 1863. Its applications were few and were mostly reliant on small-scale research until about 1934 when it was first applied at the commercial scale as a coating for bearings in high-performance aircraft engines (Alfantazi and Moskalyk 2003). Production grew steadily until the latter decades of the twentieth century, when considerable expansion and diversification in indium consumption were observed. This is evident in Fig. 1, which shows the recent growth in total indium production globally (Fig. 1*a*) and the nature of indium consumption in the United States (Fig. 1*b*). These data indicate that indium consumption from the 1990s onwards was dominated by coating applications (Schwarz-Schampera 2014), primarily as a result of the development of copper–indium–gallium selenide (CIGS) solar cells and indium-tin oxide (ITO) coatings applied as thin films to LCDs and flat-panel displays (Fthenakis, Wang and Kim 2009; Li *et al.* 2011). The boom in these technologies has seen an unprecedented growth in indium production, a trend that remains strong to the present day (White and Hemond 2012). Current consumption is dominated, in decreasing order, by Japan, the United States, the United Kingdom, the Republic of Korea and Germany, reflecting these countries' capacity for high-tech manufacturing (Skirrow *et al.* 2013). Demand within these countries and globally shows no signs of decline, with continued growth expected for decades to come (DoE 2011; Skirrow *et al.* 2013).

By-product production and extraction efficiency

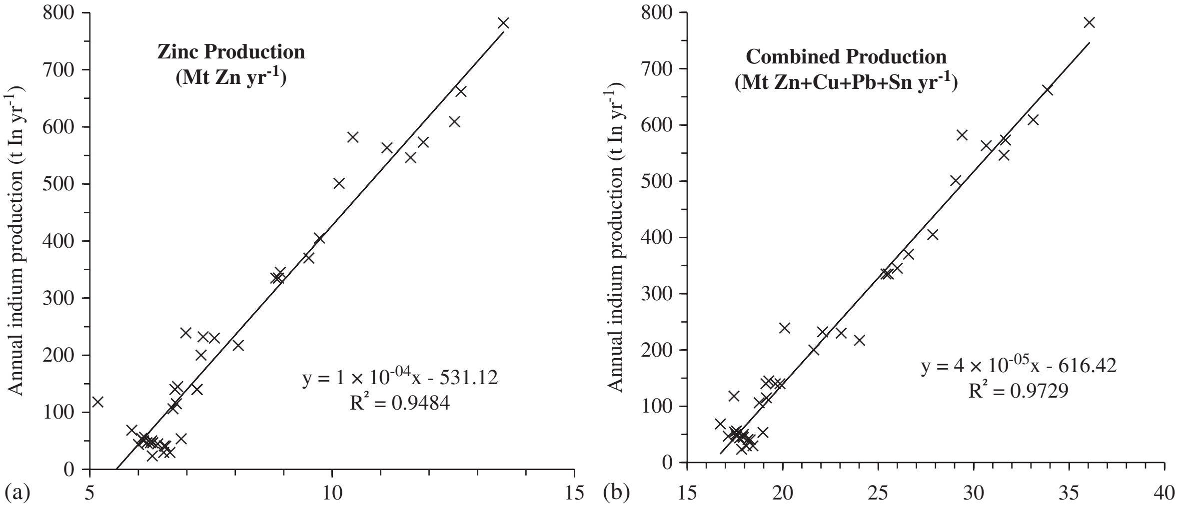
Indium most commonly combines with copper, iron and sulphur to produce roquesite ($CuInS_2$) and indite ($FeIn_2S_4$), although, as is the case for other critical metals like cobalt (e.g. Mudd *et al.* 2013b), these indium-rich minerals are of little commercial interest as they do not concentrate to form mineable deposits. Indium more typically substitutes into the structure of more common base-metal sulphide and oxide minerals such as chalcopyrite, cassiterite and stannite, and sphalerite, all of which are of significant economic interest for primary copper, tin, and zinc production, respectively (Cook, Ciobanu and Williams 2011a; Cook *et al.* 2011b). As such, in addition to being a critical metal, indium is like cobalt in that it is a 'companion' metal, as smaller quantities of indium can be isolated as a byproduct during the processing of primary copper, tin and zinc ores (Felix 2000; Graedel, Gunn and Tercero Espinoza 2014).

At present, about 95% of global indium production is derived from zinc refineries, with <5% derived from copper and tin operations (Schwarz-Schampera 2014). Figure 2 shows the strong correlations between annual indium and zinc (Fig. 2*a*) and combined base-metals production (Fig. 2*b*).

Although lead and zinc are commonly found together in mineral deposits, indium is preferentially associated with zinc (e.g. substitution in sphalerite) rather than lead (i.e. it does not substitute into galena); this is most likely because of the fact that the ionic radius of indium is similar (within ~7%) to Zn but is dissimilar (~30%) to Pb (see Cook *et al.* 2011a, 2011b). This mineralogical difference means that indium therefore mostly deports to zinc concentrates (see Health Steele and Brunswick 6–12 case studies, presented in this paper) rather than any lead concentrates produced during mineral processing. This explains the strong individual correlation with zinc production; however, this does not imply that



1 Global indium production, 1972–2014 (*a*, USGS Var.); indium end-use statistics in the United States, 1975–2003 (*b*, USGS, 2005)

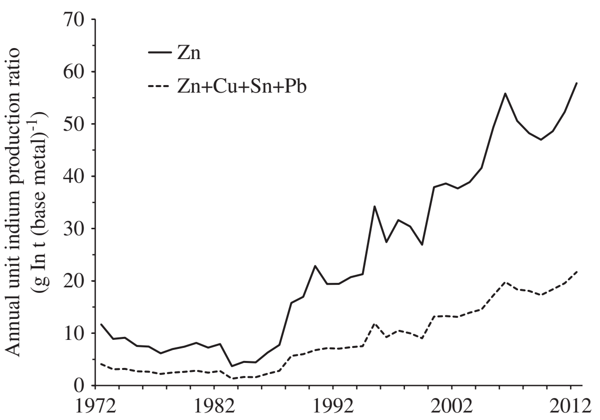


2 Comparison of global base metals and indium production from 1972 to 2012. Correlation with zinc production. *a* correlation with combined base-metal production; *b* Data compiled from USGS (Var.) and BREE (2013)

the production of other base metals can be ignored. The stronger correlation in Fig. 2*b* suggests that Pb, Sn and Cu are still mildly deterministic of indium availability.

It is also important to consider the behaviour of indium production within its context as a byproduct, primarily by considering changes in the amount of indium produced per tonne of base metal. Figure 3 shows that up to the mid-1980s, unit indium extraction was low and in decline, but extraction rose rapidly after this point. Thus, while Fig. 2 shows indium production strongly tied to the production of base metals, Fig. 3 indicates that by the mid-1980s, as indium became more important for new and developing technologies, distinct byproduct behaviour emerged. A breakdown of individual base-metal production in relation to indium is provided in Supplementary Material 1.

The unique behaviour of indium (at least compared to other metals) is also evident when considering rates of growth; global indium production has grown by an order of magnitude since the 1970s, whereas global zinc production, the metal most closely associated with indium (e.g. Figure 2*a*) has only doubled. The growth in indium production was, in fact, unequalled by any metal during this time and far outstripped the growth in the global economy itself (with 60% growth in global population and 160% growth in global GDP, Kleijn 2012, p. 95).



3 Indium produced per tonne of zinc and combined based metals (Zn, Cu, Sn and Pb) from 1972 to 2012, based on global mine production (sources: USGS Var.; BREE 2013)

Yet, the continued growth in production has been met with only marginal improvements to the efficiency of indium separation, as still today only some 20–35% of indium extracted in base-metal ores is separated during refining stages (Table 1).

The remaining indium is lost to wastes from the mining and refining industries such as tailings and slags. It should be noted that Table 1 only includes recovery from refineries and not from milling, meaning that substantial indium would also be lost to tailings (see Heath Steele and Brunswick 6–12 case studies below).

Estimates of indium waste from Zn concentrates

The substantial increases in Fig. 3 suggest not only that more zinc refineries extracted indium to meet growing demands and prices (Speirs, Gross, Candelise and Gross 2011; Schwarz-Schampera 2014) but also that there may have been a substantial quantity of indium in historical refinery slags and wastes that might be a potential future source of this critical metal (Andersson 2000; Mudd, Yellishetty, Reck and Graedel 2014). Here, the authors apply the values presented in Fig. 3 and Table 1 and produce estimates of the potential scale of waste resources from zinc processing, since the production of zinc supports approximately 95% of refined indium production.

The data in Table 1 indicate that recent indium extraction has a recovery rate of ~30–35%; this, combined with a production ratio of 57.8 g In t Zn^{-1} for 2012 (Fig. 3), a total of 13.54 Mt Zn produced during year 2012 [Bureau of Resources and Energy Economics (BREE) 2013], a typical concentration of 50% Zn for a zinc concentrate and 90 and 70–95% efficiency of the extraction of zinc and indium from concentrates, respectively (Andersson 2000; Li *et al.* 2006), yields an average grade of indium in zinc concentrates of 107–138 g In t concentrates^{-1}. This is consistent with the expected values of 70–200 g In t^{-1} suggested by the European Commission (2012) and Schwarz-Schampera (2014), indicating that the extraction efficiency estimates given in Table 1 are within acceptable range of present-day zinc concentrates. Applying the recovery factors of Table 1 to the production of zinc concentrates globally from 1972 to 2012 yields approximate estimates of the total indium that was extracted to zinc concentrates but

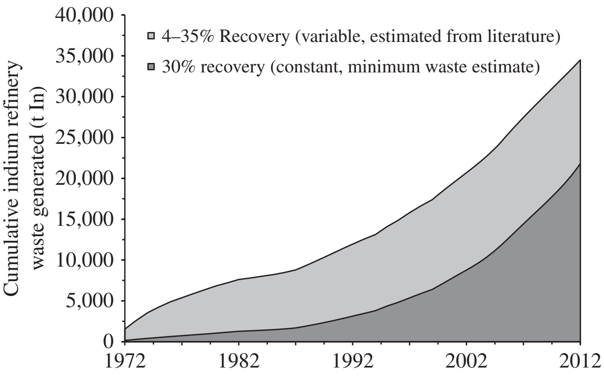
Table 1 Estimated proportions of indium in concentrates that was separated in refineries

Estimate	Source
20% for the period 1993–1997, with potential to increase up to 35–49%	Andersson (2000)
<20%, 'historically'	Phipps *et al.* (2008), p. 58
26% excluding China as at 2010	Moss *et al.* (2011), p. 61
~25% as at 2011	Tolcin (2011), p. 27
~33% as at 2011	Mikolajczak and Jackson (2011), p. 1
35% as at 2012	European Commission (2012), p. 1

not separated during this time, therefore reporting to slag wastes (Fig. 4). It should be noted that assuming already one-third of indium in ore is lost to tailings before this stage (as per case studies), the estimates for total indium waste could in fact increase by one-third above those given in Fig. 4.

The fact that a constant 30% recovery rate for the minimum estimate is generally higher than suggested in the literature (Table 1), particularly for production between 1972 and 1992, suggests that the lower bound illustrated in Fig. 4 is almost certainly a minimum estimate for the likely tonnage of indium that reported to wastes related to zinc production during this period. This conservative estimate could, in some capacity, account for the reduction in tonnage arising from previous efforts to extract indium from wastes, use of mine wastes to back-fill previous pits and/or inefficiencies experienced during their extraction. Figure 4 suggests that there are at least 21 800 t In that accumulated in zinc refinery wastes from 1972 to 2012. If the authors assume that only 25% of this is extractable because of high costs or other complexities, this quantity would still supply world demand from 2014 for 4–5 years (Fig. 6).

Figure 4 also presents a 'most likely' case, which assumes an increasing recovery yield over time, informed by linear regression of selected estimates from Table 1. In this case, some 34 500 t In accumulates from 1972 to 2012. Again, assuming only 25% of this is extractable, world demand for indium from 2014 could be met from this waste source for 6–9 years (Fig. 6). Whether or not it



4 Estimated cumulative tonnage of indium waste from zinc concentrate production, 1972–2012. Most likely estimate (light) based on linear regression of selected efficiency values presented in Table 1, and minimum estimate (dark) based on an assumed continuous rate of indium recovery from concentrates of 30%

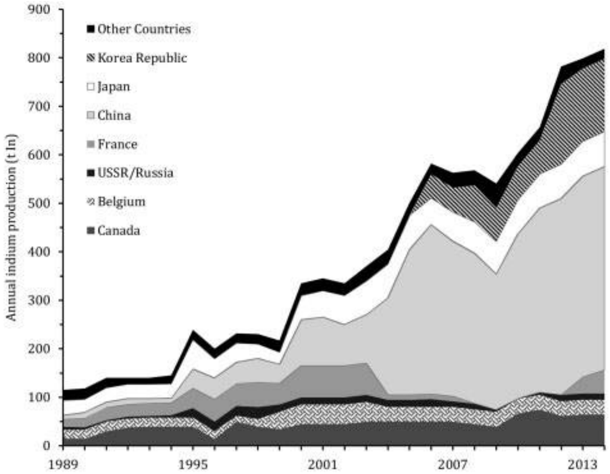
is realistic to extract companion metals such as indium from tailings and slags remains highly speculative, as it depends on a number of environmental, economic and technical factors (UNEP, *et al.* 2011a, 2011b), although any processing approach targeting the extraction of indium from mining and processing waste would also most likely target the extraction of other residual metals that are present in these waste products, providing additional streams of income that makes this approach more attractive. Although there are certainly some uncertainties relating to both the amount of indium within these wastes and the proportion of this indium that could be extracted (among other considerations), this is certainly an area that should be the focus of further research (Guézennec *et al.* 2013, Pan *et al.* 2014), and an area in which more detailed analysis is necessary. As such, the latter part of this paper provides a template by which this detail could be achieved by conducting an indium tailings resource characterisation at the per-deposit level.

Although this analysis has been restricted to zinc concentrates, this is primarily because the vast majority of indium production is a byproduct of zinc production (>95%, Schwarz-Schampera 2014); however, it should also be noted that indium is recoverable from other metal-production circuits, as has been demonstrated at the La Oroya lead refinery, Peru, Hoboken lead smelter, Belgium and from tin slag at Capper Pass, United Kingdom, among others (Felix 2000). Similar indium extraction efficiency and waste analyses are therefore possible for lead, tin and copper production and would add to the available information relating to indium resources in production wastes, but available data are insufficient for thorough analyses. An indication of the relative proportion of indium sent to various concentrate streams is provided in the Heath Steele and Brunswick 6–12 case studies.

Criticality and distribution of supply

Figure 5 depicts the per-country production of refined indium since 1989; these data indicate that global indium production is dominated by Japan, Korea and China, who produced ~9, 19 and 53% of 2013 global production, respectively (USGS 1989–2014). China produces approximately one-quarter of global lead and one-third of global zinc production (BREE 2013), indicating that Chinese indium production is consistent with the makeup of Chinese base-metal production capacity, as well as their considerable manufacturing capacity and demand for electronic goods. Authors have also provided details of the correlations between Chinese indium and base-metals production and trends in byproduct efficiency for China in Supplementary Materials 2 and 3. Additionally, potential shifts in the future distribution of indium supply is indicated in Supplementary Material 4 based on the distribution of known zinc resources.

Countries with major zinc mineral resources and/or existing zinc refineries, such as Australia, India and the USA are notable exclusions from Fig. 5. All have significant potential to produce indium as a byproduct, although this is obviously dependent on the indium content of the zinc concentrate that is being refined, something that in turn is dependent on the concentration of indium within the deposits being exploited in these zinc-production pipelines. In cases where indium is



5 World indium refinery production per country 1989–2014. Data sources: compiled from USGS (1989–2015). 'Other countries' include Germany, Italy, Netherlands, UK, Peru, Brazil and others. It should be noted that not all known indium refinery production is accounted for; for example, the Auby zinc refinery in France began extracting indium in 2012 (Nyrstar 2014), although this is not shown

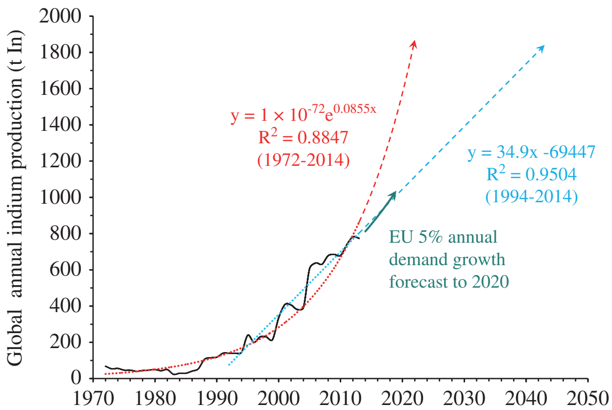
indeed present within zinc concentrates, which is likely in Australia (DoE 2011, p. 58), the lack of any recorded indium production suggests that the cost of establishing or adapting existing zinc refineries to indium-enabled processing facilities has historically outstripped the potential profits to be made (Andersson 2000). Thus, despite the indium resource potential in countries like Australia, India and the USA, impacts on or decisions by individual organisations have so far affected the availability of indium to the global market. The influence of individual organisations was evident in the early-mid 2000s when the closure of indium refinery facilities in France and Japan took place during a time of continued growth in demand. This led to a significant reduction in global supply and subsequent spikes in the price for indium. This in turn affected indium's accessibility to buyers, and hence its ratings of criticality. Another outcome was that, by 2007, production of indium from secondary ITO scrap had increased significantly to overtake primary production (DoE 2011).

The criticality of a metal is often determined by factors of diversity of market and geopolitical stability of suppliers (Graedel *et al.* 2012), meaning that the site of indium production and the global perception of the scarcity of this metal cannot be separated. China's known history of restricting indium exports (Schwarz-Schampera 2014), has created concern in the EU, USA, UK, Japan and Korea over the reliability of supply of this critical element, as is evidenced by their classification of indium among the most critical of commodities (Skirrow *et al.* 2013). The impact of this classification is worth considering, as criticality ratings themselves can influence decisions on whether certain metals should be relied upon for particular applications. For example, general electric considered that rhenium was a highly critical element, meaning that the company developed approaches to minimise the amount of rhenium used in their future jet engine designs (Duclos, Otto and Konitzer 2010). In addition, criticality from a national perspective also influences trade relationships, the degree to which individual countries stockpile elements they consider critical and the exploitation (or not) of local resources of critical elements (Graedel *et al.* 2014). These factors demonstrate possible impacts to the consumption of indium arising from its criticality, although the data presented here indicate that many of these impacts and behaviours are actually based on perceptions of scarcity that are not directly related to the depletion of indium resources. While actual stock quantities have seemingly not inhibited availability thus far, much remains unclear on remaining quantities of extractable indium in base-metal deposits and the extent to which the current low extraction efficiencies can remain. The following section, therefore, presents projected trends in demand for indium and discusses the current state of knowledge on the actual global stocks of indium in extractable locations.

Projections and geological resources

Figure 6 shows extrapolations from historical indium production to indicate future demand. Exponential growth in indium production is clearly evident from 1972 onwards, although the growth in indium production can also be modelled linearly over the last ~20 years. Projecting to 2050, annual production is very likely to exceed 2000 t In per year, with cumulative production of at least 54 000 and 241 000 t In, assuming linear or exponential regressions, respectively. This projection assumes that demand remains stable; i.e., this modelling ignores a series of unknowns, including production bottlenecks, dematerialisation or substitution strategies and peak behaviour related to potential resource depletion and scarcity.

Fthenakis (2009) predicts that a peak in primary indium production is likely at some point between 2025 and 2060, with recycled sources making up a significantly greater proportion of global indium production from the peak point onwards. The USA's Department of Energy highlights that potential price increases may stimulate both improved primary extraction efficiency and secondary recovery and reclamation, which already exceed primary sources (DoE 2011). However, these assessments still suggest that unprecedented primary production of indium will take place, which will have to



6 Historical and future indium refinery production, presented via exponential extrapolation from 1972 to 2013 data, linear extrapolation from 1992 to 2014 (USGS Var.) and projected production to 2020 responding to shorter term demand growth forecasts (European Commission 2014a, 2014b)

be met with further exploitation of geological resources, necessitating continued discovery success and possible adaptation of current zinc production pipelines to enhance indium recovery.

Selected published estimates of stocks are provided in Table 2; these represent a small sample of the many published papers and reports dealing with the concept of indium scarcity, particularly in relation to low-carbon technologies (e.g. Andersson 2000; Phipps, Mikolajczak and Guckes 2008, Murakami and Ishihara 2010; Candelise, Speirs and Gross 2012, Skirrow *et al.* 2013; European Commission 2014a, 2014b), although it is also important to note that these publications are among the few that actually present estimates of expected ore resource/reserve quantities and among the even fewer that are not simply republishing USGS estimates (Candelise *et al.* 2012), which provide limited information in terms of data provenance and reliability. Other examples that discuss the geological availability of indium in terms of crustal abundance or as inferred from trends in production have been published (Fthenakis 2009; Schwarz-Schampera 2014), but these analyses stray even further from the terminology and standards of mineral resource accounting as accepted within the mining and wider resource industry. This clearly indicates the considerable uncertainty surrounding indium resources, a factor that is evidenced by the diversity of estimates given in Table 2. The amount of extractable indium is likely to increase with time as extraction technology improves, prices increase and new deposits are found and/or become economic (as per general trends within the mineral sector; see Mudd 2010), although some of the estimates given in Table 2 are significantly lower than the amount required to meet the demand projected in Fig. 6, whereas other estimates far outstrip this increase in demand with time. This clearly demonstrates that reliance on these sources alone makes it impossible to draw a robust conclusion in terms of defining the extent to which known indium stocks will be able to service future demands for this critical metal. The considerable discrepancy in the estimates has in fact meant that, despite the intentions of the publications listed in Table 2 and others, both the USGS (Tolcin 2014) and government agencies (e.g. Skirrow *et al.* 2013) officially recognise that accurate global estimates for indium stocks are non-existent or not available.

Interestingly, Candelise *et al.* (2012, p. 4975) also note that the USGS itself recognise that an ideal estimation process would entail ‘comprehensive evaluations that apply the same criteria to deposits in different geographic areas and report the results by country’, but that this would require large resources and significant international cooperation. This is supported by UNEP *et al.* (2011a, 2011b, 2012), who highlight the necessity for deposit-by-deposit analyses to provide surety of global stock estimates, yet these analyses have yet to be performed. The next section outlines the preliminary steps necessary to complete this process for indium.

Mineral resource accounting and deposit classification

This section presents a summary of indium resources within mineral deposits classified according to industry standard (e.g. JORC) procedures for mineral resource estimation and outlines a preliminary but robust approach to the development of a global database on indium deposits and resources.

Accounting by mineral resource code reports

Mineral resource reporting codes ensure that the quantities reported distinguish between what is economically extractable at a site (reserves) and what may be economic to extract in future (resources) (Mudd *et al.* 2013a; Graedel *et al.* 2014). In doing so, they encapsulate multiple factors affecting the extractability of the quantities present. These include: the local mineralogy, structural features, logistical considerations

Table 2 Selected published estimates of remaining geological indium stocks

Paper/report	Data source quoted	Indium resource estimate (t In)
United States Geological Survey: Mineral Commodity Summaries – Indium (Tolcin 2008)*	Own research	Reserves#: 11 000 Reserves Base#: 16 000
UNEP *et al.* (2011a, 2011b, 2012)	Own research	Reserves#: 310 000 Reserves Base#: 570 000
Mikolajczak and Jackson (2011)	Not specified	50 000
European Commission (2012)	Own research	12 400 reserves and 95 000 resources in zinc deposits, as at 2010
Murakami and Ishihara (2013)	(Schwarz-Schampera and Herzig 2002; Ishihara, Hoshino, Murakami and Endo 2006; Ishihara *et al.* 2011)	65 183 from 27 selected deposits (not a global resource estimate)
Schwarz-Schampera (2014)	Not specified	31 770 in known deposits 12 500 reserves and 95 000 resources in zinc deposits (total, based on 50 ppm In in deposits) 6 300 reserves and 30 000 resources in copper deposits (total, based on 10 ppm In) Total 125 000 resources and 18 800 reserves

* The USGS last reported indium reserve and reserves base for 2007, with no data since this time. The reporting of USGS estimates for reserves and reserves base are provided in Supplementary Material 5.

Reserves and Reserves Base are USGS categories for classifying mineral resources, which are broadly similar to formal reporting guidelines and industry codes such as JORC, CIM Code, PERC, NI43-101, etc. For further discussion of resource reporting codes and comparisons, see Lambert, Meizitis and McKay (2009); Nickless *et al.* (2014) and CRIRSCO. (CRIRSCO is the ‘Committee for Mineral Reserves International Reporting Standards’ and was established in 1994 as a global effort to harmonise mineral resource reporting codes.)

and access to technology and expertise within the organisation implementing the resource assessment. Errors pertaining to the exclusion of these factors can be further magnified at national or global scales, as has been shown in previous resource accounting studies (e.g. Mudd *et al.* 2013a; Mudd and Jowitt 2014).

Table 3 provides a summary of global indium deposits reported using the JORC (Australian) and NI 43-101 (Canadian) codes. These data indicate that 6051 t In is present within code-based resources associated with these 10 deposits alone, one of which is, rather unusually, a slag waste deposit (Zeehan, Tasmania, Australia). The small number of listed deposits reflects the small scale of the indium market in value terms, in comparison to other metals, as well as a certain bias towards countries likely to employ the use of these types of code-based assessments. However, these data suggest that it is possible to use a select handful of well-classified deposits to obtain *in situ* values for In to give an indication of the potential scale of global resources when considering the size of the host base-metal resources (e.g. zinc, copper, tin, etc.).

No Chinese deposits are present within Table 3, primarily as Chinese data are not readily available and public reporting of mineral resources in China is primarily performed at the provincial or national level by geological survey groups (e.g. Ziran, Qihai, Haiqing and Xiaobo 2012, although this report lacks indium data). However, this is beginning to change as more Chinese mining companies begin to operate internationally and global efforts to harmonise mineral resource reporting codes through CRIRSCO continue. Zheng (2011) estimated that Chinese reserves were in the order of 12 000 t In for 2011.

Although our compiled data are up to date in terms of publicly available information, they certainly highlight that more work is required to determine resource estimates from deposits where code-based tonnage and grades of base metals are known, but indium grades (i.e. g In t^{-1}) are not, usually as a result of the company involved not having or not releasing the data. One possible method to address this is to statistically infer grades using the known range and average indium grades for differing deposit types to determine an average grade for that type. Indium grades are available through high-quality information such as JORC and NI43-101/CIM Code reporting but are also available in lower quality form through published geological field studies such as Tong, Song, He and Lopez-Valdivieso (2008) and Cook *et al.* (2011a, 2011b); this is not to say that the data presented in the latter type of study are of lower quality, but rather it is unclear how representative these data are within sections of or within entire mineralising systems, such as those defined by formal code-based resource reporting. Other possible methods include statistical determination of indium grades using relationships with the base metals present within the same orebody, or analysis of processing behaviour in order to backtrack from production statistics to ore grades, as was briefly discussed above where concentrate grades were inferred from production ratios. These approaches would enable better estimates of the size of the potential orebodies containing indium and therefore allow more robust estimates of global indium mineral resources. This is the focus of current research by the authors of this paper.

Indium-associated deposit types

Indium is known to be present in a variety of different mineral deposit types, especially (as discussed above) those hosting Zn, Cu, Pb and Sn mineralisation with minor amounts of cadmium, silver, gold and bismuth also potentially present. Here, the authors summarise the key deposit types associated with indium, briefly describing the processes by which they form and their significance to global indium resources. Although the presence of indium has been identified in other deposit types (e.g. the Axial Seamount active seafloor hydrothermal vent site in the Northeast Pacific, see Schwarz-Schampera 2014), this section focuses on the main base-metal deposit types that currently appear to contribute the bulk of identified indium resources. A clear definition is necessary for these deposit types, if statistical inferences of indium grade are to be applied in future research that goes one level beyond that of attempts to determine an average content in, for example, 'zinc ores' (e.g. Schwarz-Schampera 2014).

Sediment-hosted Pb–Zn

These deposits are sediment-hosted, have Pb and/or Zn as their primary commodity and, in many cases, represent orebodies that have no direct genetic relationship to igneous activity (Leach *et al.* 2005). These deposits can in most cases be further categorised into the subsidiary classification of sedimentary exhalative (SEDEX) deposits or Mississippi Valley-type (MVT) deposits; however, a number of other classifications exist, including sub-types that include carbonate replacement, Irish-type and Broken Hill-type mineralisation (Leach *et al.* 2005, 2010). Broadly speaking, SEDEX deposits are dominated by Zn–Pb mineralisation with lesser amounts of Cu and commonly with Ba and Ag and form by the venting of hydrothermal fluids onto the seafloor or the replacement of existing sediments by these same hydrothermal fluids. These deposits form at lower temperatures than volcanogenic massive sulphide (VMS) deposits, primarily as the latter are directly linked to magmatism and/or volcanism, whereas the former do not have any clear genetic link to igneous activity (e.g. Leach *et al.* 2005; Robb 2005). In comparison, MVT deposits form as a result of the circulation of high-salinity and relatively low temperature (compared to, e.g. VMS systems) basinal or connate fluids during the diagenesis of sediments in sedimentary basins (e.g. Robb 2005). The deposition of metals in MVT systems is generally restricted to the sedimentary sequences that host the fluid flow.

For the purposes of mineral resource accounting, Mudd *et al.* (2013a) sub-categorised sediment-hosted Pb–Zn deposits exclusively to SEDEX and MVT because of the often limited amount of available information in industry technical reports. In terms of commodities, SEDEX and MVT deposit types are dominated by Pb and Zn, with Leach *et al.* (2005) indicating that these deposits are also important sources of Ag and Cu, and often produce byproduct Mn, Tl, As, Ba, Bi, Ge, Hg, Ni, P and Sb. The Zn-rich nature of these deposits also indicates that these deposit types are an important source of In, as evidenced by the relationship between Zn and In discussed above. This, combined with the substantial quantity of indium within the sediment-hosted Pb–Zn deposit at Malku Khota,

Table 3 Summary of deposits with code-based reporting of indium quantities

Mine name and location	Resources Mt	Reserves Mt ore	Total Mt	Grade g In t^{-1}	Total In t In	Zn%	Sn%	Cu%	Pb%	Ag g t^{-1}	Au g t^{-1}	Res. code	Deposit type	Indium mineralogy	Primary commodities
Malku Khota, Bolivia	485	–	485	5.01	2431	0.05	–	0.02	0.07	23.71	–	C	Sed-Pb–Zn	Jam, owy	Zn, Ag, Pb, Cu
West Desert, UT, USA	59.0	–	59.0	25.98	1533	1.85	–	0.22	–	–	–	C	Skarn	Sph	Zn, Cu
Mount Pleasant (North Zone), NB, Canada	15.2	–	15.2	64.7	983	0.91	0.37	–	–	–	–	C	Gran-rel	Stn, sph and others	Zn, Sn
Geyer Southwest, Germany	12.6	–	12.6	35	439	0.58	0.46	–	–	–	–	A, C	Skarn	Sph	Sn, Zn, Ga
Keg (Main Zone), YT, Canada	39.8	–	39.8	5.77	229	0.77	0.03	0.15	0.26	30.25	–	C	Sed-Pb–Zn	Sph, cp	Zn, Sn, Ag, Pb, Cu
Baal Gammon, QLD, Australia	2.80	3.11	5.91	33.7	199	–	0.2	0.98	–	36.95	–	A	Skarn	Sph, cp	Sn, Ag, Cu
Pingüino, Angentina	10.6	–	10.6	11.5	122	–	–	–	0.62	62.87	0.38	C	Epi	Sph, cp, stn	Ag, Au, Pb
Brunswick 12, Bathurst, NB, Canada	0.79	0.73	1.52	48.75	74.1	7.6	–	0.34	3	92	–	C	VMS	Sph, cp	Sn, Ag, Pb, Cu
Zeehan (Slag), TAS, Australia	–	–	0.46	48	22.0	13.6	–	–	1.5	55	–	A	Slag	Slag	Zn, Ag
Conrad & King Conrad, NSW, Australia	2.65	–	2.65	6.8	18.0	0.53	0.22	0.2	1.33	105.3	–	A	Skarn	Sph, cp, gal, cas, stn	Zn, Sn, Ag, Pb, Cu
Total	**628**	**4**	**633**		**6051**										

Resource code: A = Australian Joint Ore Reserves Committee (JORC) resource code; C = Canadian National Instrument 43–101 (NI 43–101) resource code. Deposit Type: Sed-Pb–Zn = Sediment-hosted Lead–Zinc, Gran-rel = Granite-related, Epi = Epithermal, VMS = Volcanogenic massive sulphide. Indium mineralogy: jam = jamesonite, owy = owyheeite, sph = sphalerite, stn = stannite, cp = chalcopyrite, gal = galena, cas = cassiterite.

Bolivia (Table 3) suggests that indium should also be added to this list of important byproducts.

Granite-related

Granites are coarse-grained igneous rocks formed from highly evolved magmas that are generally the result of extreme fractionation of mafic magmas, the anhydrous melting of the lower crust (e.g. during mantle plume-related underplating) or the melting of igneous or sedimentary rocks during metamorphism. The fine-grained equivalents of granites are termed rhyolites, which are either erupted or intruded near the Earth's surface and are compositionally identical to granites. Both granites and rhyolites are important sources of companion metals (e.g. Weng, Jowitt, Mudd and Haque 2014, 2015), and are known to host indium mineralisation (e.g. Andersen, Stickland and Rollinson 2014; Simons, Andersen and Shail 2013) although the overall current and potential future contribution to global indium supplies is currently unclear. The most significant granite-related indium deposit thus far identified is the Mount Pleasant deposit, New Brunswick, Canada, containing over 983 t In (Table 3), although undoubtedly other granite-related indium mineralisation is yet to be identified or fully quantified.

Volcanic-hosted or volcanogenic massive sulphide (VHMS/VMS)

Volcanogenic massive sulphide deposits are major contributors to global base-metal supplies, for example, hosting over 31.56 Mt Cu (Mudd *et al.* 2013a), as well as significant quantities of Zn, Pb, Ag and Au resources. They form via the discharge of hot, metal-rich hydrothermal fluids that source metals by interaction between modified sea water and source rocks such as epidosites (e.g. Jowitt, Jenkin, Coogan and Naden 2012); these metal-rich fluids then precipitate massive sulphides at or near the seafloor in submarine volcanic environments (Allen and Weihed 2002). Although the focus of this paper is not to provide a detailed explanation of these mineral systems, it is worth noting that Galley, Hannington and Jonasson (2007) highlight that the polymetallic nature of VMS deposits means that although they are often smaller (but higher grade) than, e.g. porphyry deposits, they are resistant to fluctuations in the prices of different metals, making them economically attractive and hence more likely to provide base-metal concentrates from which indium may be extracted. The high metal tonnages and grades of VMS deposits make them particularly relevant to indium mineral resource accounting. Indium concentrations vary considerably in VMS deposits (1–320 g In t^{-1}, data in preparation).

Epithermal

Epithermal mineral deposits form within generally subaerial hydrothermal systems driven by magmatic heat sources at temperatures $<300°C$ and at depths up to 1.5 km below the water table (Simmons, White and John 2005; Tosdal, Dilles and Cooke 2009). These types of deposit form within shallow parts of hydrothermal systems, usually within volcanic arc settings, and may form parts of larger co-genetic epithermal–porphyry–skarn systems (e.g., Jowitt, Mudd and Weng 2013). Epithermal deposits can be enriched in a wide range of metals, most commonly Au and Ag, but (importantly for indium) may also contain Zn, Cu, Pb, As, Sb and Sn (e.g., Simmons *et al.* 2005). Epithermal deposits can be broadly split into low, intermediate and high sulfidation types according to the oxidation state of sulphur in the ore fluid, which also reflects the pH of the system (Hedenquist, Arribas and Gonzalez-Urien 2000); the most important of these, in terms of indium resources, is likely to be intermediate sulfidation type epithermal systems that are associated with Pb and Zn mineralisation, although this does not rule out the presence of indium within other types of epithermal systems. This uncertainty highlights the need for more research in this area.

Skarn

Skarns form during interaction between wall rocks and magmato-hydrothermal fluids derived from plutons and associated deeper magma chamber systems (Einaudi, Meinert and Newberry 1981; Meinert, Dipple and Nicolescu 2005), generally form during contact metamorphism and by a variety of different metasomatic processes (Meinert 1992; Meinert *et al.* 2005). These types of mineral deposits are generally associated with and can form beside or within magmatic plutons and, as mentioned above, may be genetically related to porphyry and epithermal deposits within larger magmato-hydrothermal systems, although this is certainly not always the case. Skarn mineral deposits host a wide variety of metals including W, Sn, Mo, Cu, Fe, Pb, Zn, Au, Ag, Bi, Te and As, with the metals present within a given skarn dependent on differences in composition, oxidation state and the metallogenic affinity of the pluton (e.g., Einaudi *et al.* 1981). Of these different types, it is thought that Zn, Sn and Cu skarns are probably the most important for In mineralisation, as exemplified by the Dachang and Geiju skarns in China, although again more research is required in this area.

Slags

Although they are not naturally occurring, slags represent an important class of anthropogenic mineral deposit type that may be an important future resource of critical metals such as indium. The physical composition of slags is highly dependent on metallurgical processes from which they are derived. In some cases, they are viewed as secondary resources rather than end wastes, although their heavy metal content is often of environmental concern and conventional disposal often entails dumping. This, however, makes them an attractive source of metals, if their removal results in net reductions to local pollution (Shen and Forssberg 2003). At present, the Zeehan resource in Tasmania, Australia is the only known instance of code-based reporting of slag-hosted indium resources. It is somewhat surprising that the In grade of the resource has been reported, given that the quantities shown in Table 3 suggest that the value of indium within the slag represents only about 1% of the total value of the contained metals within the resource, with the majority of the value coming from zinc ($\sim 13\%$ of the total resource value) and silver ($\sim 84\%$), although this is obviously somewhat controlled by the prevailing metal prices.

Indium from mine waste: case studies in Canada

In Fig. 3, it was shown that the production of indium per unit of base-metals has increased substantially in recent decades; however, it was discussed that the economics of indium extraction and processing still leads to extensive wastage. In the previous section, the authors also discussed the need for an exhaustive assessment of indium stocks on a per-deposit basis. Here, two case studies are presented that combine these concepts to identify the potential accumulation of indium resources in the wastes of individual zinc operations in Canada. This was achieved through an analysis of past production, deportment and tailings data, which were available at the Heath Steele and Brunswick 6–12 deposits.

Heath Steele and Brunswick 6–12 are both VMS deposits with indium mineralisation associated with sphalerite and chalcopyrite and produced Zn and Cu concentrates throughout periods of considerable growth in indium demand, but without reported indium production. Theoretically, the quantities of indium not transported to product concentrates from base-metal mining and refinery operations should still reside within the host deposit, or appear in tailings (see Chen and Petruk 1980; Petruk and Schnarr 1981). While it is recognised that the extraction of metals from tailings leads to economic and technical complexities that the market may yet to be properly prepared for (Speirs *et al.* 2011), our analysis is not aiming to present an economic case, but rather to identify the potential quantities of indium that could contribute to future supply as conditions become more favourable for extraction.

Table 4 shows how the indium that is present in the milling feed apportions to various processing streams at the two sites.

The deportment values shown are representative of the production characteristics at the time of sampling (August to September, 1977), meaning that a number of local factors arising from mining different parts of the orebody and changes in technology are likely to affect where and how indium was apportioned to processing streams over time. These potential changes were monitored by tracking ore grades throughout the life of both mines; these data indicate that the base-metal grades at the time of sampling were approximately 9 and 40% below the all-time average at Heath Steele and Brunswick 6–12 mines, respectively, suggesting that our estimates of total indium content are conservative. It should also be noted that Table 4 lists grade and percentage apportionments, meaning that the estimation of process stream indium quantities can be calculated in two ways: as a fraction of what was fed into the mill, or as a fraction of the tonnage of each stream itself.

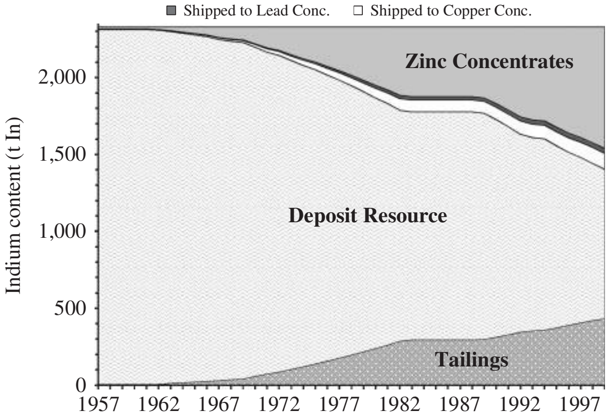
Data for both sites were collected on the quantities of concentrates and product metals produced each year from each mine and were primarily sourced from the Canadian Minerals Yearbook [Energy, Mines, Resources (EMR) 1975–1982], in addition to datasets compiled from Mudd (2009). Where concentrate production data were not reported, the average content of base metals in concentrates from reported years was used to interpolate or extrapolate for concentrate quantities for unreported years. Resource estimates from various years were obtained from a review of processing and geological literature, allowing an original indium quantity known to exist in the deposit at a given year to be used as a reference (Chen and Petruk 1980; Schwarz-Schampera and Herzig 2002, Xstrata 2013). This in turn allowed values from Table 4 to be used to develop the material balance analyses shown in Figs. 7 and 8.

The effect of mining almost the entirety of economically extractable orebodies at Brunswick 6–12 from 1964 onwards has resulted in a significant accumulation of indium (over 3500 t of contained indium metal) in the tailings at the site (Fig. 8). In comparison, the Heath Steele orebody contained $\sim$1,000 t In at the time of closure of the mine with a further $\sim$400 t In in tailings (Fig. 7). Comparing these data with Fig. 4 shows that the indium contained within the tailings of these two sites alone contributes over 10% of the estimates for indium in zinc processing wastes globally. Besides highlighting the conservative nature of that analysis, these case studies suggest that, if all base-metal deposits with indium mineralisation were to be the subject of individual assessments like the ones presented, global estimates of indium stocks could increase enormously, and hence substantially alter perceptions of global indium scarcity.

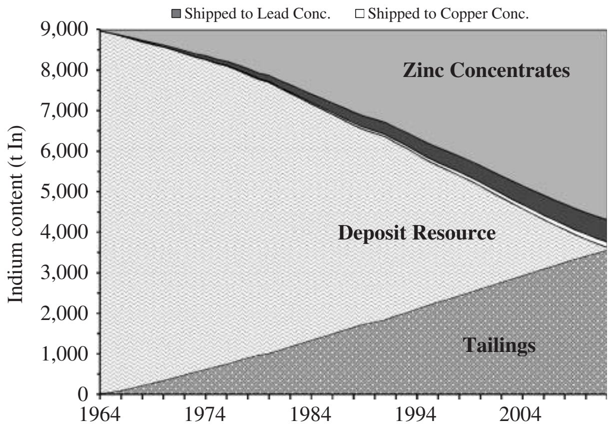
Naturally, it remains to be seen whether the extraction of metal from tailings at Heath Steele, Brunswick 6–12, or indeed tailings generally, represents a more attractive option for future indium supply. Tailings may, in some instances, be used to back-fill mines, making them less accessible and hence further complicating the extraction process. In addition, although the economics of tailings extraction is extremely important in terms of determining the availability of indium, this is beyond the remit of this paper, which aims to simply highlight the enormous size of the potential indium resources available within mine wastes.

Table 4 Measured concentrations and deportment of indium in process streams at the Heath Steele and Brunswick 6/12 mines, adapted from Chen and Petruk (1980) and Petruk and Schnarr (1981)

	Heath Steele		Brunswick 6–12	
Process stream	In grade (ppm)	% Apportionment	In grade (ppm)	% Apportionment
Milling feed	50	100 (feed)	49	100 (feed)
Cu concentrate	200	10.51	76	0.7
Pb concentrate	40	2.3	57	6.7
Zn concentrate	400	51.43	220	58.9
Secondary Zn concentrate	–	–	220	2.7
Bulk sulphide concentrate	–	–	166	2.7
Tailings	20	35.75	30	28.3



7 Indium stocks at the Heath Steele deposit, New Brunswick, Canada from 1957 to 1999



8 Indium stocks at the Brunswick 6–12 deposits, New Brunswick, Canada from 1964 to 2012

Urban mining and end-of-life recycling

In recent years, increasing emphasis has been placed on the concepts of the 'urban mine' and a 'closed-loop' or 'circular' economy, whereby the supply of materials comes from the extraction and reuse of urban material stocks, to the point that traditional mining practice becomes largely obsolete (Brunner 2011). In the case of indium, a closed-loop economy would require that discarded products such as smartphones and decommissioned solar arrays become a central source of supply. Measures to 'close the loop' on indium may contribute to significant improvements in its long-term environmental sustainability (Graedel and Allenby 2010), and so, given the unprecedented quantities in which it is produced at present (Fig. 1), it is worth considering the potential scale of urban indium stocks to see if this is indeed feasible. It has been estimated that currently $<1\%$ of global indium supply comes from EOL recycled materials, and 25–50% from recycled scrap produced during manufacturing (UNEP *et al.* 2011a, 2011b, 2012), suggesting that there is at least some potential for improvement in this space.

Using Australia as a case study, it is estimated that 23.5 million old and unused phones sit within homes and workplaces (Mobile Muster 2014). With 1 t of mobile phones estimated at 13 000 units (Hagelüken 2014), this equates to approximately 1808 t of waste. 'Display waste' is said to contain 174 g In t^{-1} (Buchert, Manhart, Bleher and Pingel 2012), meaning that only approximately 0.3 t of indium could be found, assuming all this stock did in fact contain LCDs. While a number of other sources have not been included in this calculation, for example, indium contents in computers, notebooks, tablets and TVs, as well as any other consumer goods that have been sent to landfill, these quantities are clearly not sufficient to economically outperform traditional mining practices, or even extraction from mine wastes, as the potential resource from a single mine is orders of magnitude higher (e.g. Figure 8).

Geyer and Doctori Blass (2010) assessed the economics of mobile phone recycling and found that the recovery of materials other than the current suite of precious metals and copper would in fact not increase profit margins for recyclers, suggesting that the value of indium is not likely to significantly affect the likelihood of large-scale recovery from phones. More generally, urban mining has been known to face institutional and governmental challenges, as significant economic changes are likely to be necessary to facilitate a transition, and notable discrepancies have been found in terms of subsidies for the mining and recycling industries, despite the apparent benefits of recycling (Johansson, Krook and Eklund 2014). This is particularly true for Australia, which lacks facilities for e-waste collection, faces unique challenges because of distance and transportation and lacks integrated smelter capacity (Khaliq, Rhamdhani, Brooks and Masood 2014). If Australia is to develop this infrastructure, it must further overcome the scarcity of available data on future quantities and physical compositions of products like EOL LCD screens (Salhofer, Spitzbart and Maurer 2011). Should these data become available, there are still numerous technical challenges because of the low concentrations and physical complexity of indium in some electronic products (Gotze and Rotter 2012).

Yet, the future of EOL recycling of indium is not entirely bleak. There are significant drivers leading towards increased metallurgical recovery from waste electrical and electronic equipment (WEEE) globally. These include:

- Concern over the toxicity of electronics in landfills (Cui and Zhang 2008);
- Conventional mining systems themselves facing a number of challenges, including trends of declining ore grades and hence increasing mine wastes and energy consumption (Prior, Daly, Mason and Giurco 2013);
- Exponential growth in demand for consumer electronics and hence their contained metals, combined with decreased efficiency of materials use, contributing to greater waste volumes (UNEP 2013); and
- Increasing focus on critical metals contained in WEEE including palladium, tantalum, cobalt, rare earths and others (Manhart, Buchert, Bleher and Pingel 2012), as well as gold and silver, which contain most of the monetary value of WEEE and can, therefore, drive improved collection and metallurgical recovery from WEEE independent of indium prices. The increased focus on critical metals is also likely to foster increased incidences of stockpiling to ensure stability of supply (Hagelüken 2014).

Additionally, the EU has established an enhanced landfill mining project linked to their landfill directive, which entails specific performance targets (European

Commission 2014a, 2014b), suggesting that there is precedence for overcoming the aforementioned institutional and governmental challenges. Also within the EU, there are indeed examples of economically viable e-waste collection facilities, which extract and sell indium (e.g. Umicore, Hoboken, Belgium, Rombach and Friedrich 2014). Comprehensive economic evaluations and feasibility studies for large-scale urban mining in Australia are yet to be completed, although this process is known to be under way (CSIRO 2014). It has been recognised that a key part of this will be to understand the stocks and flows of metals, which is where the authors believe that the methods outlined in this paper have the potential to make a significant contribution. Case studies have been identified, which track EOL flows of indium and other critical metals in individual product waste streams (e.g. Zimmermann and Gößling-Reisemann 2014), although a comprehensive analysis of Australia's total endowment of indium (and other companion metals) in mineral resources and urban stocks remains a significant gap in knowledge if the potential wealth from companion metals is to be further capitalised upon. This is anticipated to be the subject of future work by the authors of this paper.

Conclusion

This paper has provided:

- A preliminary analysis of global indium resources based on a summary of its reported presence in individual deposits.
- Estimations of indium resources in tailings and slag wastes through analysis of past company, national production and indium deportment datasets.
- A summary of indium-associated deposit classifications.
- A discussion of the potential for supply from recycled products.

It was found that:

Based on the limited code-based reporting of indium resources, a geological resource of this critical metal is likely to exist that is capable of meeting demand well into the twenty-first century, though further research is necessary to estimate precise resource quantities.

The economic factors controlling indium extraction and processing have led to significant wastage in the past, and, given the substantial quantities calculated, it will be important to consider mine wastes as a major contributor to future supply.

There is some potential for indium supply from EOL electronics given expected increases in WEEE quantities and potential price, though significant governmental, institutional and technical challenges must first be overcome.

The concepts and procedures presented in this paper will form the basis of future work in the fields of mineral resource accounting and material flow analysis for companion metals in Australia and globally.

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4. A METHOD FOR ESTIMATING RARELY REPORTED METAL RESOURCES

In this chapter, a new approach for estimating the resources of by-product or critical metals is developed. These metals are not well reported in the mining industry, as shown in the previous chapter and in Mudd et al. (2017a), meaning that many uncertainties can arise in determining the resources of these metals. The method presented here involves the creation of a database, consisting of deposits which a) are reported to contain a metal of focus, and b) are likely to contain this metal, but without it being reported. For the deposits sitting under category b), a “Proxy Method” is proposed where alternate data sources are used to infer metal grades, based on what information about a deposit is available. To demonstrate the application of this methodology, 19 different proxies for estimating indium grade are used against a database of Pb-Zn deposits derived from the co-authored study Mudd et al. (2017b). This article shows that while individual proxies might have been employed in other studies, there can be bias and large variation in results when looking at all possible approaches. It also shows a way of managing this variation, and associated uncertainties. Research questions 1a, 1b, and 1c are addressed.

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The world's by-product and critical metal resources part II: A method for quantifying the resources of rarely reported metals

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ABSTRACT

Estimates of the world's mineral resources of numerous by-product metals remain highly uncertain at best, despite the high criticality of many of these elements to society. This stems from the limited reporting of the concentrations of these elements within mineral deposits by the mining industry, meaning that we require methods to estimate the availability of these resources that overcome this limitation. Here, we present a method for quantifying poorly reported mineral resources of by-product metals that builds upon deposit-by-deposit approaches to global resource estimation, arguably the best-practice approach for well-reported commodities, but also adds the use of proxies for by-product grade estimation. This proxy method allows for deposits with known or inferred by-product metals to also be incorporated within global resource estimates and provides a greater basis for assessing future supply potential.

We demonstrate the application and verification of this methodology with indium, a critical metal for which <1% of identified zinc, tin, and copper deposits potentially hosting indium mineralisation report grades using CRIRSCO (or equivalent) mineral resource reporting codes. The use of the method outlined in this manuscript will allow the global resources of any metal commodity, especially the often under-reported by- and co-product metals that are becoming increasingly essential to modern life, to be quantified to a significantly greater level of accuracy and precision than is allowed by other approaches.

1. Introduction

Entire periods of history have been named after the use of metals like bronze or iron, however we are now seeing changes in which metals are most important to society within lifetimes. This is primarily due to the technologies we use daily becoming more complex and increasingly requiring unique metals to fulfil highly specialised functions (e.g. germanium (Ge) in fibre optic systems or indium (In[2]) in LCD displays). Many of the metals prevalent in modern technology are subject to complicated supply chains, and hence some governments and institutions have begun to consider the criticality of these metals (BGS, 2015; USDoE, 2011; Jowitt, 2015; Skirrow et al., 2013) by quantifying their relative supply risks, importance to society and the environmental implications of their extraction (Graedel et al., 2012). Assessments of criticality have revealed a number of metals as being "most critical" (e.g. USDoE, 2011; BGS, 2015; Skirrow et al., 2013), potentially requiring metal-specific policy interventions in order to avoid or minimise the risk of supply restrictions. These interventions require a better understanding of critical metal resources and supply dynamics; however the reserves and resources of many critical metals remain highly uncertain (Mudd et al., in press-a).

A major factor contributing to this uncertainty is that virtually all critical metals are extracted from mineral deposits as by-products of zinc (Zn), lead (Pb), copper (Cu), nickel (Ni), iron (Fe), titanium (Ti), aluminium (Al), gold (Au), platinum (Pt) and tin (Sn) (Nassar et al., 2015). Although the by-product elements (e.g. indium, Ge, gallium (Ga), selenium (Se) and tellurium (Te)) have significant technological value, they are typically of lesser economic value than the main products of a given mining operation. They are therefore less likely to form part of a company's core business, and hence to be extracted (Willis et al., 2012). With few incentives for mining companies to report the presence of metals that have minimal value to them, there is a global lack of reporting of potential co-products or by-products using mineral resource reporting codes (e.g. JORC, SAMREC, NI 43-101). This is the focus of part I of this study (Mudd et al., in press-a), where we discuss the implications of this lack of reporting and the challenges and uncertainties this absence of reporting has created for global mineral resource accounting, especially of the critical metals. Here, we build on this by presenting a new approach to resource estimation which explicitly

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[2] Although the chemical symbol for the majority of elements are used throughout this paper, we have elected mostly to spell out indium to avoid confusion with the word "in" or the abbreviation of the natural logarithm, "ln".

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addresses the identified uncertainties in order to provide better global resource estimates of by-product and critical metals.

Notwithstanding reporting limitations at the deposit level, numerous attempts have been made to estimate global mineral resources of by-product and critical metals using a number of differing approaches:

- Using reports derived from geological surveys or government mining/resource departments. Such reports have had a tendency to underestimate critical metal resources and often provide little indication of data provenance or uncertainty, although some reports, e.g. by the United States Geological Survey (USGS, 2015a) report/speculate on undiscovered resources, which may limit the risk of underestimation. The quantities published in these reports have on occasion been misinterpreted as representing total extractable global resources (EGR), inflating fears of resource scarcity through the implication that what isn't reported doesn't exist (e.g. Cohen, 2007; Moyer, 2010).
- Applying factors to crustal abundance figures that scale them to a potential extractable resource volume (see UNEP et al., 2011). Measures of a metal's crustal abundance (e.g. grams metal per tonne of the Earth's crust) provide a very coarse indication of the total amount of a metal in the lithosphere. A fraction of this volume is concentrated to economic or sub-economic locations, referred to as reserves and resources, which have clearly defined definitions that are strongly tied to the economics of extraction. Mineralising systems classified as reserves and/or resources are, by definition, anomalous, and so it is methodologically flawed to assume that they can be indicated by crustal abundances. This approach also neglects to consider the mineralogical barrier that was first introduced by Skinner (1976).
- Assuming a fixed ratio between a base metal and a by-product; for example, considering that 50 g indium is present per t Zn within Zn sulphide ores (see Schwarz-Schampera, 2014). This approach may neglect geological variability, for example between deposit types, especially when considering that a single project could contain multiple different deposit types with differing geological and grade characteristics (e.g. Jowitt et al., 2013), and hence most likely differing relationships between base and by product metals. However, this approach can be adapted to be deposit type specific and allows for the quantification of uncertainty, as is well demonstrated by Frenzel et al. (2015), who present an analogous approach using Ge assumed to be contained within sphalerite in Mississippi Valley Type (MVT) deposits. This approach does not necessarily require the use of mineral resource reporting codes to determine the tonnage of critical metal-bearing orebodies.
- Constructing a database of deposits with the reported quantities of the metal in question using mineral resource reporting codes (e.g. cobalt (Co) as a co-/by-product of Ni is well reported and an ideal critical metal for this approach; see Mudd et al., 2013b). This approach is arguably the best practice in mineral resource accounting, and directly indicates resources known to the mining industry to be extractable. It does not, however, explicitly overcome the limitation of reporting, and is likely to only reflect a small portion of the EGR for most critical metals.

Each of these approaches has strengths and weaknesses. However, to the best of our knowledge, no approach has produced global resource estimates that combine:

- Resource assessments based on per-deposit estimates made using mineral resource reporting codes for orebody tonnage (ensuring that resource assessments are restricted to those deposits with known or inferred by-product mineralisation),
- The quantification and/or classification of uncertainty, and
- A direct method of addressing the lack of by-product reporting at the per-deposit level.

Here, we present the Proxy Method, an approach that combines all of these factors and is reproducible for any by-product metal. We demonstrate the efficacy of this approach by applying it to indium, a critical metal for which <1% of identified Zn, Sn, and Cu deposits potentially hosting indium mineralisation report indium grades using CRIRSCO mineral resource reporting codes (see Werner et al., 2016). Our proposed methodology builds upon the methods applied to resource assessments of other primary commodities, including Ni (Mudd and Jowitt, 2014), Cu (Mudd et al., 2013a), Co (Mudd et al., 2013b), REEs (Weng et al., 2015), Au (in prep.) and Pb and Zn (Mudd et al., in press-b), making maximum use of reserve and resource data from CRIRSCO code-based reports. This ensures that considering the world's resources of a given by-product metal uses estimates that are constrained to the small portion of the earth's crust already known to the mineral industry to contain economic or near-economic quantities of host metals from which by-product extraction can be economically leveraged. The processes introduced in the following sections highlight how proxies may be used to infer by-product grade estimation, and how statistical analysis for the quantification of uncertainty can also be embedded into this approach.

2. General design

Our methodological approach aims to produce a comprehensive list of deposits with known or inferred mineralisation of a metal, and to quantify the amount of that metal in each deposit using the best available information. The summed total of per-deposit quantities then represents a new global resource estimate. Depending on the quality of information available, by-product or critical metal resources will have to be determined differently for each deposit. Some deposits may be quantified directly from publicly available information; however, given the limited state of critical metal reporting in mineral resources by deposit, the majority must be quantified through the use of proxy data. Three types of proxy are introduced in this paper that permit by-product metal grades to be estimated, although our approach does not limit the number or type of proxies which can be applied as long as they permit the estimation of by-product metal grades by some justifiable means, allowing new information to be incorporated into the estimation process as it becomes available. Our approach is summarised in Fig. 1 and explained further in the following sections.

3. Reported resource data collection

Our approach first requires that reported data be collected and developed into a resource database. This process entails an extensive review of mining company websites, technical reports, published literature, mineral resource atlases and other national/state mineral occurrence databases with information on mineral deposits containing the by-product element of interest. Any deposit where geological testing notes the by-product as a commodity or potential commodity is added to a list. This review process must be guided by a reasonable understanding of the geology and production characteristics of the metal of interest. For example with indium, which we consider as an example in the following sections, production comes exclusively as a by-product of Zn, Sn and Cu mining, with Zn being the primary source (>95% of global indium production; Schwarz-Schampera, 2014). This is consistent with the strong mineralogical relationship indium shares with Zn (Cook et al., 2011a; Werner et al., 2015), suggesting that deposits with currently known economic quantities of Zn most likely also contain indium, even if not explicitly reported. This understanding enables the efficiency of the review process to be improved.

If a host metal such as Zn is reported as a commodity using CRIRSCO mineral resource reporting codes, there is greater likelihood that the extraction of a by-product can also be economically leveraged. The full list of potential indium-bearing deposits (see the Supplementary data and the Part III paper, Werner et al., 2016, for further analysis) contains entries where the presence of indium is either explicitly reported or is inferred through its mineralisation and reported presence of host metals.

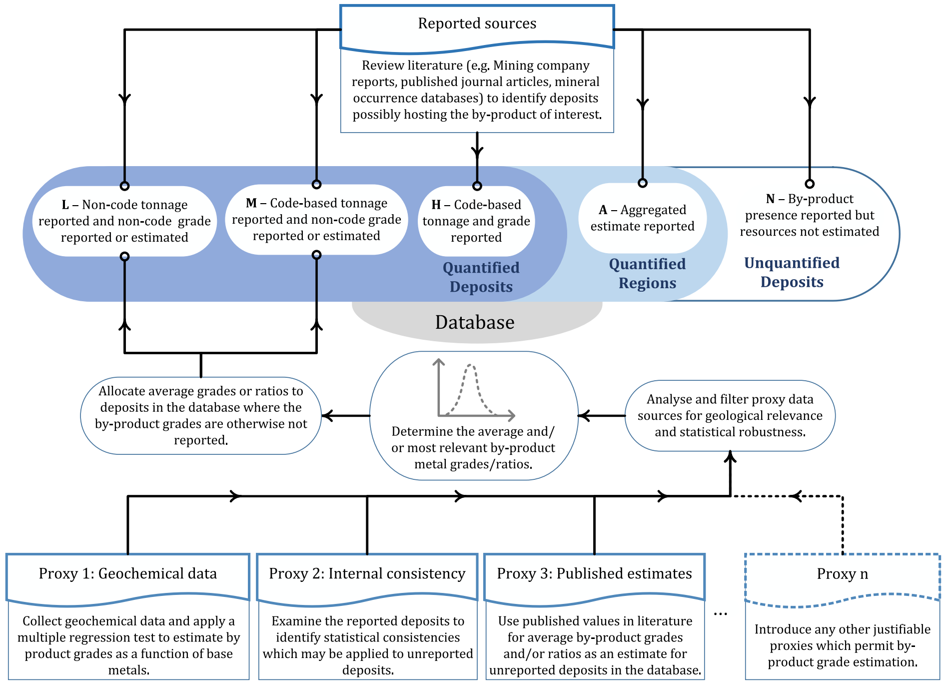

Fig. 1. General method outlined in this study, involving the use of a combination of reported data and proxy data to construct a per-deposit resource database, which itself is organised into differing classifications (quantified deposits, quantified regions and unquantified deposits) and levels of certainty (high, medium, low and aggregated estimates, plus unquantified deposits). Proxy data sources are employed to estimate by-product metal grades where they are otherwise not reported by mining companies or in related literature.

The results produced using this approach can generally be organised according to this distinction of reported vs. inferred deposits. Given the strong association between indium and Zn, the list of inferred deposits used for our indium study were primarily obtained from a collected database of global Pb–Zn resources (Mudd et al., in press-b). The indium in many Pb–Zn deposits with inferred indium mineralisation is unlikely to be economically extractable at present, though in the event that indium prices increase and/or supply becomes restricted through conventional production pipelines, this may change. An economic assessment of by-product/critical metal potential is not the focus of this methodology, rather the estimation of sheer resource quantities in known deposits, which may then support such assessments. We include these currently sub-economic quantities in our database to ensure that the results remain relevant regardless of future price changes. A table documenting 98 deposits reporting a total of ~74,600 t indium is provided in the Supplementary data, and should be considered as additional resources to those presented in the following proxy analysis.

4. Evaluation of resource data quality

When tasked with the quantification of by-products/critical metals within each identified deposit, it is recognised that data quality will vary. A grading system, described as follows, establishes a hierarchy of certainty upon which the resource data are to be interpreted for each deposit or each estimate. It has been developed from an earlier scheme described in Mudd et al. (2013b):

High quality (H): The by-product resource for the deposit has been reported based on the most recent statutory code-based resource estimate available, which contains both a resource tonnage and associated grade. This indicates that the resource estimate for this site has been determined based on a rigorous exploration and resource assessment process, and is representative of the size of the extractable orebody. The values presented under this classification should therefore be considered the most accurate.

Medium quality (M): A recent statutory code-based assessment for the deposit is available to determine the size of the orebody. Yet despite reports of the presence of the by-product at this site, or high likelihood of it being present due to mineralogical associations, no code-based grade estimate is available. The grade of the by-product must be determined through the use of a non-code source or using the proxies described in Section 5. In cases where a low quality estimate is available but a code-based orebody tonnage is also available, the code-based tonnage is applied to update the status of the orebody and the deposit is classified as medium quality. A separate classification, “Medium-calculated” (Mc), may be applied to deposits where the by-product grade has been calculated using the proxies of this study (as per Fig. 1), and applied to a deposit with code-based tonnage.

Low quality (L): Neither by-product grades nor orebody tonnage are available for the deposit under a code-based classification scheme. Grade and tonnage are determined from geological publications, industry and government reports, academic theses or similar works. The data from these publications are not necessarily low quality in themselves, however the database entry receives this classification as it cannot be determined whether the grade and orebody tonnage are truly representative of the mineralising systems. A separate classification, “Low-calculated” (Lc), may be applied to deposits where the grade has been determined from the results of proxy estimation, and applied to a deposit with non-code-based tonnage.

No resource estimate available (N): The deposit has been noted in the literature reviewed for being particularly enriched in the by-product metal in question, although no resource estimates for the orebody have been identified by the authors, and/or insufficient information is available to infer by-product resources using proxies.

Aggregated resource estimate (A): This resource estimate may represent multiple deposits, and/or is representative of a broader region in which by-product resource data were available. The quality of these estimates are to be considered at or below "L", but are included for completeness and representativeness of the literature.

Through the summation of reported and semi-reported quantities of the by-product in identified deposits classified as high, medium and low, a view of the global resource in extractable locations is provided. An additional sense of the scale of potential future resources is provided by the number of deposits classified as 'N' and 'A'.

5. Proxies for grade estimation: case study of global indium resources

Deposits with reported tonnages but unreported by-product grades require a proxy to infer the abundance of the by-product within the deposit. As shown in Fig. 1, three distinct proxy approaches are described in this study. These approaches introduce an additional suite of literature and databases and may be used to classify additional deposits as "M" and "L", depending on the existing quality of tonnage data for the deposit in the database. Using indium as an example, the proxies are described as follows:

5.1. Proxy 1: geochemical databases and multiple regression

This first approach considers that in terms of mineralogy and production, it is possible to find strong statistical and physical links between indium and some common metals (Werner et al., 2015). Similar links have also been found in terms of grades, suggesting that at some level, the grades of metals likely to be reported may indicate unreported indium grades. Table 1 summarises those studies which have considered this through grade relationships between indium and other metals in mineral deposits.

While these studies all affirm some link between indium and one or more metals, they have varying findings that are difficult to consolidate. Hence, they collectively highlight that it can be problematic to infer indium grades for a diverse deposit database based on individual deposit studies, and based on links with a single base metal.

Table 1
Review of literature analysing inter-element relationships between indium and common metals.

Correlation found with indium	Reference
The In/Sn ratio varies from <0.1 to 47 in sulphides.	Yi et al. (1995)
Indium is found when Pb–Zn deposits are Sn rich. Eighty percent of indium in Chinese deposits can be found within the mineral sphalerite (ZnS).	Qian et al. (1998)
Multiple regression coefficients used to accurately determine indium grade, based on base metal grades and the proportion of different minerals present in ore at Neves Corvo, Portugal. Strong associations observed between Sn/stannite (Cu_2FeSnS_4).	Benzaazoua et al. (2002)
In samples of Japanese ore of varying deposit classifications, indium was found to correlate best with Sn, then Ga, and most poorly with Zn.	Ishihara et al. (2006)
Indium becomes relatively more abundant in cases where Cu contents are high and Zn contents are low. This study otherwise provides a good summary of indium mineralogy.	Cook et al. (2011b)
From studies at the Pingüino deposit in Argentina, indium can be strongly correlated with Sn in early stage mineralisation, and with Zn in later stage mineralisation.	Jovic et al. (2011)
Indium correlations were generally low among base metals in ores of the Dulong and Dachang mines, China, however the best correlation was found with Zn.	Ishihara et al. (2011)
The 1000 In/Zn ratio in sphalerite can be a useful indicator of the grade of In in the deposit overall. However, the proportion of sphalerite must be known, which appears not to be widely reported.	Murakami and Ishihara (2013)
In metamorphosed sulphide deposits, strong correlations can be found between indium and Cu, supporting the coupled substitution model.	Lockington et al. (2014)

These studies are therefore not directly applied as proxy data sources. To find more general orebody relationships that could be applied to a diverse deposit database, we collected and analysed large geochemical databases that a) documented the occurrence of indium, b) documented the occurrence of other metals associated with indium in terms of mineralogy or production (i.e. Zn, Sn, Cu, Pb, Au, Pt, iron (Fe), bismuth (Bi), sulphur (S), cadmium (Cd) and arsenic (As)), and c) encompassed multiple deposit types or geological settings. These databases included:

- FOREGS Geochemical Atlas of Europe (Salminen et al., 2005)
- Geochemical Atlas of Australia (de Caritat and Cooper, 2011)
- SedDB — Earthchem.org (Johansson et al., 2012)
- GEOROC (47 individual sources listed from online query for Cu, Sn, Pb, Zn, Bi, silver (Ag), Pt, Fe, S, Cd and indium in whole rocks, sourced via http://georoc.mpch-mainz.gwdg.de/georoc/Start.asp)
- USGS National Geochemistry Database: Rock (USGS, 2015b)
- OSNACA (Ore Samples Normalised to Average Crustal Abundance, Brauhart and Hagemann, 2011)
- Sulphide Ore Geochemistry Database for Volcanogenic Massive Sulphide Deposits of the Paleoproterozoic Flin Flon Belt and Sherridon Area, Manitoba and Saskatchewan. Geological Survey of Canada (Jonasson et al., 2009)
- Geoscientific Data Warehouse mineral resource drilling database, New South Wales, Australia (DIRE, 2016)

Not all geochemistry databases collected were found to contain indium data. For example, a whole rock geochemistry database of Australia (OZCHEM, Champion et al., 2007) was found not to contain any data for indium from over 32,000 data points. Similarly, sources of drillhole assay datasets such as the Geoscientific Data Warehouse for New South Wales, Australia, contain hundreds of thousands of geochemical measurements, but as indium and many other critical metals are not commonly assayed (see also DNRM, 2014), they often cannot provide many data points. As shown later in Table 2, we have managed to find only 51 useful data points from this source. Other databases, sometimes covering the same geographical regions, were however found to be more useful. For example, an assessment of surface geochemistry in Australia (from the Australian Geochemical Atlas) showed that a total of 7902 data points contained 5320 samples with indium concentrations. These data are not sufficient to directly infer average ore grades as they represent surface geochemistry and have similar issues to the crustal abundance approaches outlined above; however, these data may be able to provide an indication of the statistical relationship between elements, which itself can be a reflection of the geochemistry at greater depths (Bradshaw et al., 1974). For this particular database, indium weakly correlates with Zn despite its strong association in terms of production (see Werner et al., 2015), but strongly correlates with Sn (Fig. 2). This may be a function of surficial/regolith processes, where Sn is less mobile than Zn (see Smith and Huyck, 1999).

The fact that higher grades of Sn are associated with higher grades of indium within anomalously enriched samples (i.e. samples most likely related to mineralisation) at the continental scale suggests that this correlation may be used to more accurately estimate the grade of indium where Sn grades are reported. This finding is akin to some studies listed in Table 1, however, Sn itself may not always be reported, suggesting that a correlation which takes into account all potentially reportable base metals which host indium mineralisation, as well as those elements which form minerals with indium has greater utility. Multiple regression techniques provide equations that describe indium grade as a function of multiple element grades. The indium equation generated from each examined database took the general formula:

$$Ingrade\ (g\ In\ t^{-1}) = \sum_{i=1}^{i=n} (x_i \cdot h_n) + c \quad (1)$$

Table 2
Summary of geochemical databases analysed for indium content and statistical reliability.

Geochemical database	Description	No. of useable indium-containing measurements	Elements incorporated into regression (in order of statistical significance)	R^2	Average error	Pass Y/N
GEOROC	"A comprehensive collection of published analyses of volcanic rocks and mantle xenoliths." — Whole rock samples analysed.	55	Sn, Zn, Cu, Ag, Pt, As, Pb	0.953	5%	Y
Forum of European Geological Surveys (FOREGS) Geochemical Atlas of Europe	Geochemical survey of European floodplain sediment, humus, soil, stream sediment and stream water. Subsoil dataset analysed.	785	Ag, Fe, As, Cd, Sn, S, Cu, Zn, Pb	0.315	41%	N
Geochemical Atlas of Australia	Soil/sediment samples gathered from 1315 sites in 1186 catchments across Australia.	1281	Sn, Fe, Cd, Pt, S, Pb, Cu, Zn	0.934	25%	Y
EarthChem.org: SedDB	"SedDB is a data management and information system for marine sediment geochemistry." — indium-containing measurements derived only from whole rock basalt samples	192	Sn, Cd, S, Fe, Ag, As, Zn, Pb, Cu	0.959	3%	Y
USGS: National Geochemistry Database: Rock — ICP-MS[a]	"Geochemical analysis of rock samples collected and analysed by the USGS. This dataset includes and supersedes rock data formerly released as "Geochemistry of igneous rocks in the US extracted from the PLUTO database".	938	Zn, Cu, Cd, Pb, Fe, Ag, Sn, As	0.937	158%	Y
USGS: National Geochemistry Database: Rock — ES[a]		15,439	Pt, Fe, Zn, Cu, Ag, Sn, As, Pb	0.119	53%	N
USGS: National Geochemistry Database: Rock — Unk[a]		8111	Pt, Cd, Pb, Ag, Zn, Sn, As, Cu	0.271	93%	N
NSW Department of Industry, Resources and Energy — Geochemical Database	Drill core/assay database for mineral deposits in NSW, Australia	51	Cd, Ag, Sn, Pb, As, Zn, Cu, Fe	0.882	98%	Y
OSNACA — sediment-hosted Pb–Zn/Mississippi Valley Type (MVT)	"The OSNACA Project (Ore Samples Normalised to Average Crustal Abundance) is an open source project where assay data is freely available online." — Utilises "representative ore samples" from across the globe. 348 deposits included in our regressions.	33	Cu, Sn, Ag, Pb, Cd, Pt, Zn, As, S, Fe	0.920	426%	N
OSNACA sediment-hosted Pb–Zn/Irish type		17	Cd, Ag, Pb, Cu, Zn, Pt, Sn, S, Fe, As	0.937	53%	Y
OSNACA sediment-hosted Pb–Zn/sediment-hosted massive sulphide		24	Sn, Zn, Fe, S, Cu, As, Pb, Ag, Cd, Pt	0.825	83%	Y
OSNACA — orogenic Au		173	Cu, As, Zn, S, Sn, Cd, Pb, Ag, Fe	0.095	83%	N
OSNACA — skarn		25	Ag, Pb, Fe, Cu, Sn, Zn, Cd, As, S	0.728	77%	Y
OSNACA — epithermal		87	Zn, Pb, Fe, Cu, Ag, S, Pt, As, Sn, Cd	0.779	159%	Y
OSNACA — sediment-hosted Stratiform Cu		30	As, Fe, Sn, S, Cu, Zn, Ag, Cd, Pb	0.868	103%	Y
OSNACA — volcanic massive sulphide (VMS)		90	Ag, Cu, Pb, Fe, Sn, Zn, Cd, As, S	0.352	127%	N
OSNACA — IOCG		31	S, Fe, Cd, Pb, As, Zn, Sn, Cu, Ag	0.706	83%	Y
OSNACA — granite-related		20	Cu, S, As, Sn, Ag, Pb, Cd, Fe, Zn	0.961	62%	Y
OSNACA — porphyry Cu		42	Sn, Fe, Cd, Pb, Zn, S, Ag, As, Cu	0.999	77%	Y
Sulphide ore geochemistry database for volcanogenic massive sulphide deposits of the Paleoproterozoic Flin Flon Belt and Sherridon area, Manitoba and Saskatchewan — Geological Survey of Canada Open File 5432	Data from 40 VMS deposits in Manitoba and Saskatchewan, Canada	350	Cu, Sn, Zn, Cd, Ag, S, Pb, As, Fe	0.328	98%	N

[a] ICP-MS = dataset compiled through the application of inductively coupled plasma mass spectrometry. ES = dataset compiled through the application of emission spectroscopy. Unk = unknown methods used to determine quantities presented in this database. Refer to USGS (2015b) for more details.

where x_1 to x_n are the regression coefficients, h_1 to h_n are the measured grades of metals such as Sn, Zn, Pb, Cd, Pt, As, Bi, Fe, Ag, S and Cu (which have production or mineralogical relationships with indium, see Schwarz-Schampera, 2014) and c is the equation's intercept.

The accuracy of this equation was tested to see how well it predicts reported values in the databases listed in Table 2. This entailed applying the equation to all data points and determining the residual values. We chose not to take these residuals directly as indications of error, but rather their percentages, as at higher grades the percentage error will become considerably more relevant than absolute values. For example, if a data point is measured at 0.02 ppm and the equation estimates 0.03 ppm, we take the error to be 33% (calculated from the residual between the predicted and observed values for indium grade as a percentage of the predicted value) rather than 0.01 ppm, as we cannot assume the error to remain this small in magnitude when the equation predicts grades upwards of, for example, 10 ppm per deposit. If an equation was found to predict indium grades with sufficient accuracy (which we arbitrarily set at $R^2 > 0.70$ and average error < 200%), it was deemed acceptable to progress to the next stage of analysis as a potential proxy data source in our deposit database. The results of this testing are presented in Table 2, which shows that the strongest relationship between base/mineral-forming metals and indium was found in the OSNACA database. Fig. 3a–f illustrate in greater detail the ability of the multiple regression equations to match observed data for the cases deemed acceptable for use as a proxy.

5.2. Proxy 2: looking for consistency among reported deposits

For proxy 2, we consider that the deposits reported to contain indium in the literature which make up the database might show consistent characteristics between one another. If there is consistent behaviour among those deposits which are reported, we may be able to make

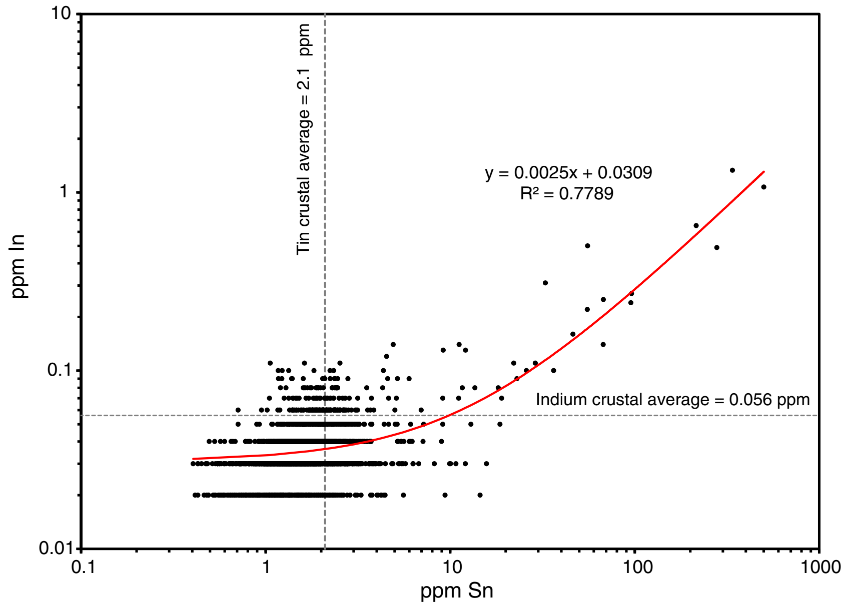

Fig. 2. Correlation between the Sn and indium concentrations within surface soil samples from Australia. Crustal average values taken from upper crust estimates in Rudnick and Gao (2014).

direct inferences for deposits where indium is not reported. The advantage of this approach is that it employs the most geologically relevant sources of data, as all data points are known to represent indium-enriched ore. In the following sections we outline three ways in which reported deposits may be analysed to infer indium grades for other deposits. A table of the deposits used for this proxy along with their resource data, geological characteristics and sources is provided in the Supplementary data.

5.2.1. Finding an optimum ratio between indium and Zn in reported deposits

The first approach we take as part of proxy 2 considers whether unreported indium grades can be determined through only Zn from reported deposits. As over 95% of indium is produced as a by-product of Zn production, consistent ratios between total indium and Zn content may be present in differing deposit types. This approach allows for the use of known production ratios to guide, and question the validity of, the ratios found in our collected deposit database. For example, it is known that currently around 60 g indium are refined for every t Zn mined globally (Werner et al., 2015), thus, taking into consideration extraction efficiencies, one would expect approximately 144 g indium/t Zn to be present in deposits contributing to global production in some way (Mudd et al., in press-a). This could be used to guide the interpretation of results obtained using this proxy.

A total of 48 indium-bearing deposits (10 classified as high quality, 21 medium quality, 17 low quality, see full data in Part III, Werner et al., 2016) were identified that contain reported values for both indium and Zn grades. For each deposit type in the reported database, we identified a value for g In/t Zn that would best produce a 1:1 estimate between calculated and reported indium content. Two primary approaches of unconstrained nonlinear optimisation (see Marthaler, 2013) were undertaken in order to optimise this value, using Goal Seek and Solver functions in Excel, with a third approach adopted as a mean. These approaches are explained with reference to the sample data for a given deposit type in Table 3.

1. *Minimised error*: In Table 3 it can be seen that when an estimate of 3 g In/t Zn is applied across these deposits, a total estimate of 18.5 t In is achieved and this entails a total of 4.5 t In of accumulated error for these three deposits. The minimised error approach determines the value for X g In/t Zn that would reduce the accumulated error from 4.5 t In to as close to zero as possible using Excel's Solver function. In other words, a ratio is obtained such that the "absolute difference" column is set to a mathematical minimum. This however does not constrain the overall estimated content (i.e. the total of the "t In calculated" column). This estimate therefore produces greater precision per deposit, at the possible expense of decreased overall accuracy.
2. *Bulk accuracy*: For this approach, Microsoft Excel's Goalseek function is used to set a ratio between indium and Zn such that the total of the "t In calculated" column matches what is reported. For example in Table 3, it would aim to change 18.5 t In total estimated to be exactly 15 t In. The downside of this is that the degree of over- and underestimation necessary to achieve this bulk accuracy is not constrained (i.e. the "Absolute difference" column is not constrained). Thus, this estimate produces an accurate overall finding, but at the expense of decreased precision per deposit.
3. *Optimised average*: Taken as a mean value between that of the *Minimised error* and *Bulk accuracy* approaches.

The optimised estimate for g In/t Zn may then be applied to deposits of known Zn content to determine their likely indium content. The results of testing for this proxy are presented in Table 4.

It can be seen that despite conscious efforts to minimise error or achieve accurate total estimates using the reported data, it is difficult to find a ratio for indium and Zn that is consistently able to predict indium content, even when disaggregating into deposit types. This is a clear reflection of the limited reporting of indium grade in deposits reporting Zn grades. Despite the ratios being higher than the expected values around 144 g In/t Zn, they still underestimated the reported values. This highlights the considerable diversity of ratios in the reported database, and that they are, in general, particularly high. This result might however be expected, given that the reporting of these deposits in the first place is probably due to their particularly high indium enrichment. The diversity of ratios, even within a single deposit type, are highlighted in Fig. 4 which shows a poor overall correlation, suggesting that there are simply too few reported deposits containing indium for this approach to be considered viable at this stage. The results obtained when this proxy is applied globally are shown in Section 6, and the challenges associated with the use of this proxy are discussed further in Section 7.

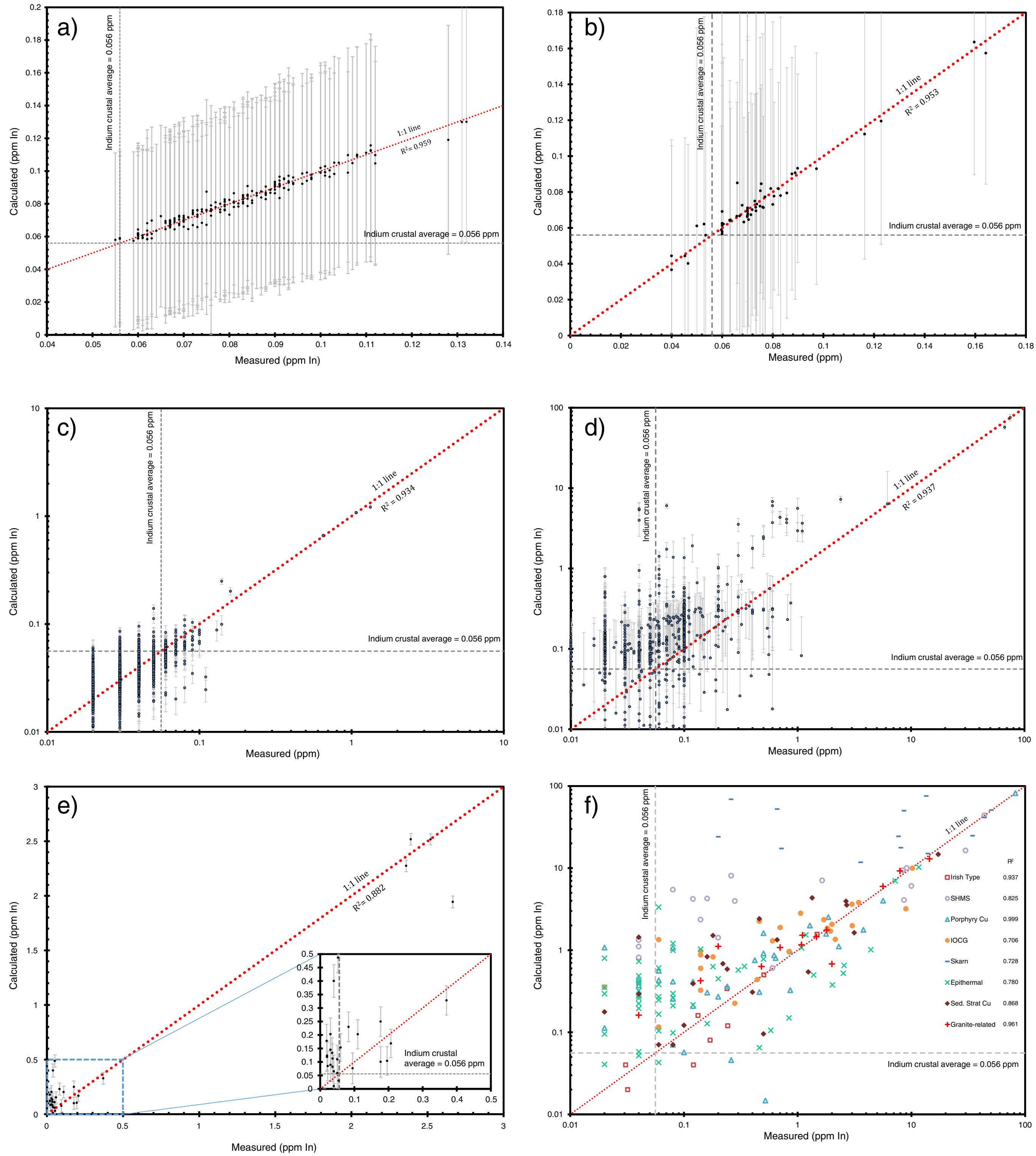

Fig. 3. Correlation between reported and calculated indium grades for the databases deemed acceptable for use as an indium proxy. Grey error bars depict a 95% confidence interval, with black markers indicating the mean value from multiple regression. Indium crustal average of 0.056 ppm obtained from estimates within the upper crust from Rudnick and Gao (2014).

a) Earthchem.org 'SedDB'.
b) GEOROC.
c) Australian Geochemical Atlas.
d) USGS Whole Rock Geochemical Database- ICPMS.
e) NSW drillhole assay database.
f) OSNACA Database (deposit types exhibiting regressions deemed acceptable for use as per Table 2 are shown).

Table 3
Sample table to illustrate the optimisation process.

Deposit name	Mt Zn reported	t In reported	g In/t Zn (actual ratio)	g In/t Zn (estimated ratio)	t In calculated	Absolute difference (t In)
Deposit A	1	4	4	3	3.5	0.5
Deposit B	2	5	2.5	3	6	1
Deposit C	3	6	2	3	9	3
Total		15			18.5	4.5

5.2.2. Multiple regression of Zn, Cu, Pb and Ag from reported deposits

It would be preferable that a single metal grade be sufficient to infer indium grades as this only requires the reporting of that metal, however given the poor statistical performance seen in Fig. 4, we also explore multiple regression from reported deposits, as we have done for the geochemical databases listed in Table 2. The advantage of this approach is that it can produce better statistical accuracy than using only Zn, however it requires a greater number of metals to also be reported for a deposit in order to harness this improved accuracy. In Fig. 5 we see predicted vs. reported indium grades for 28 deposits where Zn, Ag, Pb, Cu and indium grades were all reported. This achieves a modest improvement to an R^2 value of 0.556, at 95% error, however this would not meet the requirements applied to the other databases present in Table 2. In the event that over time, more deposits are reported with indium grades alongside the grades of other common commodities, this proxy may be of greater use. It may indeed apply to by-products/critical metals that are already reported more than indium (e.g. cobalt). Mainly for comparative purposes, we have still included the results of this proxy in the final analysis presented in Section 6.

5.2.3. Analysis of the mineralogy of reported deposits

The final way in which we analyse reported deposits is to consider whether the mineralogy of these deposits reveals any consistent behaviour. For this analysis, each deposit in our reported database is classified according to the minerals reported to be hosting indium, or otherwise the form of the indium mineral (see Supplementary data). Fig. 6 shows the spread of the minerals hosting indium, where reported or otherwise obtainable.

It can be seen from Fig. 6 that grades can be highly variable within particular mineral hosts, and that a greater number of reporting deposits do not reduce statistical uncertainty. Regardless, this information alone could not be used as a proxy as it is highly unlikely that a deposit without reported indium will have information on which minerals are host to it. For mineralogical information to be of use as a proxy, we would then need to link it to other geological factors that would be more likely known about a deposit. In Fig. 7 we disaggregate the mineralogy data according to deposit types, however this only confirms that mineralogy is not a strong indicator of indium grades, at least for the reported deposits in our database. For this reason we have not considered this particular approach further in the following sections.

Given the performance of all approaches under proxy 2, we can generally conclude that it is difficult to make inferences from known deposits when the number of these deposits is severely limited by reporting. This finding assists in the interpretation of other published values. If, for example, it is known that a smaller number of deposits than are presented in our own database are used for an estimate elsewhere in the literature, we can conclude that they would not be able to encompass the geological variability necessary to be particularly accurate. The learnings and applicability of proxy 2 is discussed further in Section 7.

Table 4
Calculated ratios of indium and Zn content by deposit type, and results using the minimised error, optimised average and bulk accuracy approaches.

	No. of sample deposits	Minimised error	Optimised average	Bulk accuracy
Ratio estimates	N	g In/t Zn	g In/t Zn	g In/t Zn
Epithermal	8	47	220	392
Granite-related	0	No data	No data	No data
Porphyry Cu	2	2040	2722	3405
Sediment-hosted stratiform Cu	0	No data	No data	No data
Sediment-hosted Pb–Zn	4	615	801	986
Skarn	7	1223	1958	2693
Tailings	2	1883	1735	1588
VMS	23	474	699	925
Results		t In	t In	t In
Target bulk estimate		39,126	39,126	39,126
Bulk estimate achieved		19,842	29,484	39,126
Final difference		−19,285	−9642	0
Accumulated error		22,881	27,393	33,227

5.3. Proxy 3: applying published values

The final proxy outlined in this study involves the direct application of ratios or average grades found in the scientific or mining literature. Although the reviewed studies are subject to their own methodological uncertainties, the use of this proxy assists in the interpretation of results from the other two proxies by positioning them against current understandings of the metal of interest. Furthermore, it is reasonable to expect that the introduction of a greater number of possible figures from literature may otherwise assist in reducing uncertainty. An example of a value that could be used in this proxy is provided by Schwarz-Schampera (2014), who suggests that 50 g In/t Zn exist in global Zn deposits, and 10 g In/t Cu in Cu deposits. The values for Zn may be applied to the collected databases, as well as previously published deposit databases for Cu (see Mudd et al., 2013a). Although, as discussed earlier, these ratios may neglect geological variability, they still represent some understanding of global resources not otherwise captured in the first two proxies. The published average indium grades and ratios shown in Table 5 highlight how even for a metal like indium, subject to limited reporting in mineral deposits, numerous studies produce findings with potential for application to a Pb–Zn or Cu deposit database.

These data are all compatible for use with deposits in a deposit database reporting Zn or Cu grades, but should be scrutinised on their theoretical basis. It was decided that an assumed grade of 39 ppm, published by Yoshimura et al. (2013) did not have a sufficient theoretical or numerical basis to apply globally, as it was simply an average of 27 selected deposits that were already representative of high indium grades, and hence the direct application of this value as a global average would entail a clear selection bias. Similarly, the UNEP et al. (2011) value of 106 ppm indium was much higher than many deposits already reported for their indium content. Given that the act of reporting is inextricably linked to economic value and hence higher grades (see Mudd et al., in press-a), it therefore seemed unreasonable to assume a global average higher than most of the reported values and so this value was also excluded. This particular study is also based on the use of crustal abundances, and hence is methodologically flawed as we discuss in part I of this series of papers (Mudd et al., in press-a). For the ratio of 144 g In/t Zn presented in Mudd et al. (in press-a), it could be assumed that only higher grade deposits that already contribute to the indium supply chain are accounted for. The ratio of indium in non-producing deposits

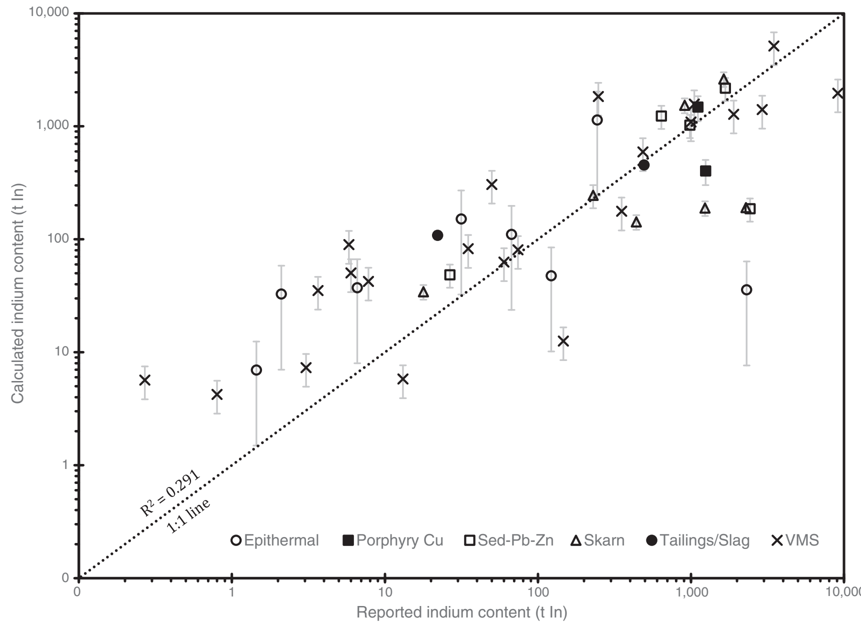

Fig. 4. Results of the optimised average approach, with error bars indicating the spread achieved through the application of the minimised error and bulk accuracy approaches, using 48 deposits of known deposit type, tonnage, and indium and Zn grades.

might therefore be assumed to be much less, potentially sitting at the same as the production ratio at 60 g In/t Zn. The ratio of 144 g In/t Zn could be weighted 0.7, reflecting the percentage of zinc smelters that are indium capable (Mikolajczak, 2009), with a lesser 60 g In/t Zn weighted 0.3 for the remaining, yielding an estimated ratio of ~118 g In/t Zn. This is shown in Table 2 as the Mudd et al. (in press-a): Modified estimate, which was also considered as a potential proxy. The ranges published by Tolcin (2014) and the European Commission (2012) were not directly included as they did not specify an average, but served as guidelines for interpretation of the other proxy estimates, as we show in the next section. The USGS and OSNACA geochemical databases are considered large and broadly representative of a diverse range of deposits. Indeed values taken from the OSNACA database are entirely based on ore characteristics, and hence the direct average grades of indium for these database were considered valuable as additional "literature" based sources.

6. Data aggregation and analysis

6.1. Data aggregation

Up to this point, multiple data sources have been identified which may be used to estimate indium grades in mineral deposits where these grades are otherwise not reported. This is achieved either by:

- Using a combination of the reported grades of Zn, Cu, Pb, Sn, Bi, Cd, S, As, Au, Pt and Fe and deposit type information to infer deposit grades (under proxies 1 and 2)

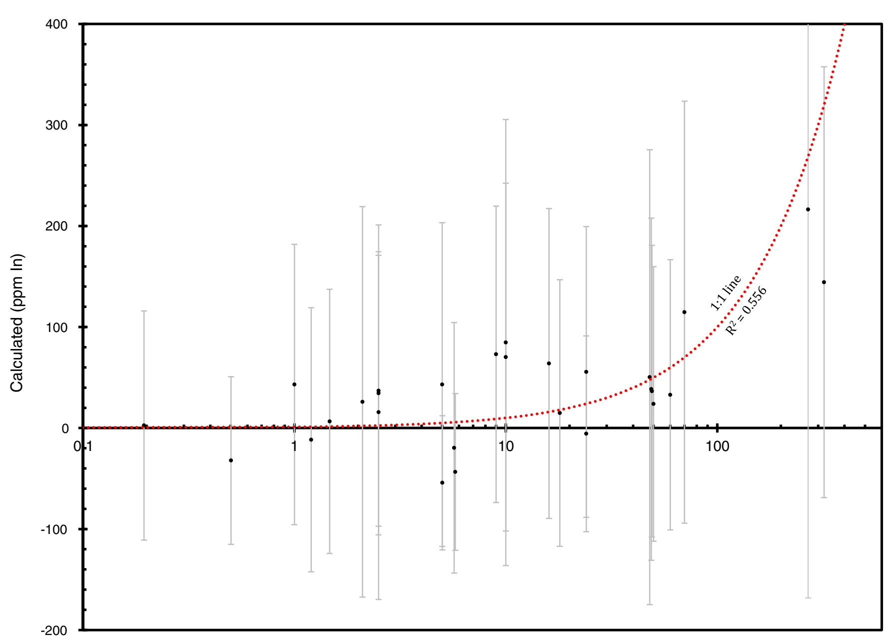

Fig. 5. Results of multiple regression using Pb, Zn, Cu and Ag grades to estimate indium, based on the reporting of 28 deposits in the literature reviewed (see Supplementary data).

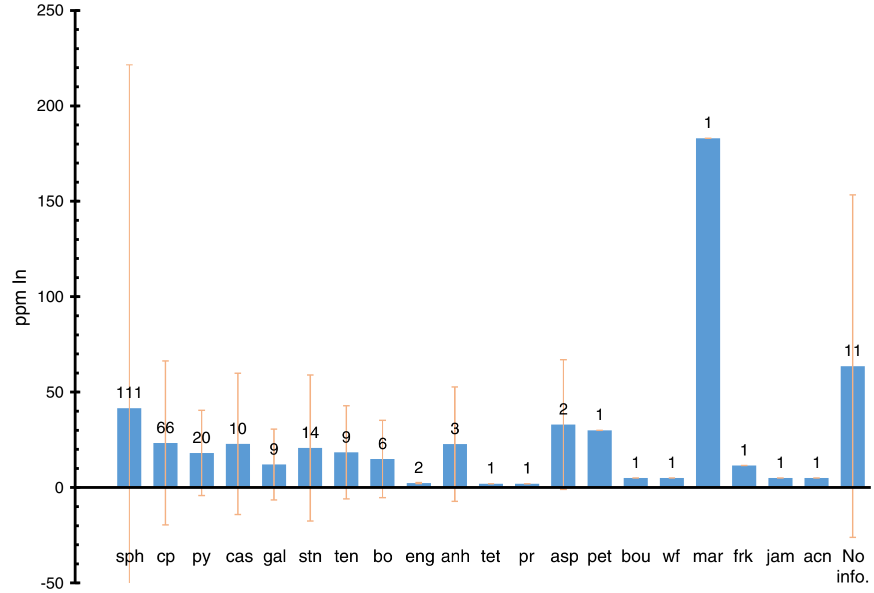

Fig. 6. Distribution of average indium grades according to which minerals are reported to host indium in the reported deposit database. One standard deviation and the number of contributing deposits is indicated. Note that indium may be reported to exist within multiple host minerals for a given deposit. sph = sphalerite, cp = chalcopyrite, py = pyrite, cas = cassiterite, gal = galena, stn = stannite, ten = tennantite, bo = bornite, eng = enargite, anh = anhydrite, tet = tetrahedrite, pr = pyrrhotite, pet = petrukite, asp = arsenopyrite, bou = boulangerite, wf = wolframite, frk = ferrokësterite, jam = jamesonite, can = acanthite, "No info." = grades reported but host minerals not known.

- Applying a fixed grade or ratio between by-product and host metals to all deposits, or differentiating by deposit type (under proxies 2 and 3).
- Assuming a fixed, constant by-product grade for all unreported deposits (under proxy 3).

Using our indium example, 26 individual approaches or estimates (13 under proxy 1, 4 under proxy 2 and 9 under proxy 3) were considered for application to a Pb–Zn deposit database with unreported indium grades (Mudd et al., in press-b). Some were eliminated at an earlier stage due to more obvious weaknesses, or as a result of failing preliminary statistical tests (e.g. the FOREGS geochemical database), leaving 19 actually applied to the Pb–Zn database provided in Mudd et al. (in press-b) as shown in Fig. 8. Note that this deposit database contains 851 entries, however 51 of these have indium contents or grades reported in the literature, and are therefore excluded from proxy analyses, leaving 800 deposits contributing to the estimates shown in Fig. 8, which would be considered in addition to what is reported. Theoretical maxima and minima, as suggested in the indium literature (European Commission, 2012; Tolcin, 2014), are also shown for reference. Each grade is listed in Fig. 8 as a weighted average grade, calculated as per:

$$\frac{\sum_{i=1}^{i=695} C_{i,p}}{\sum_{i=1}^{i=695} T_i} \qquad (2)$$

where $C_{i,p}$ is the content of the by-product in a deposit, i, for a proxy, p, and T_i is the ore tonnage for that deposit. The actual global average

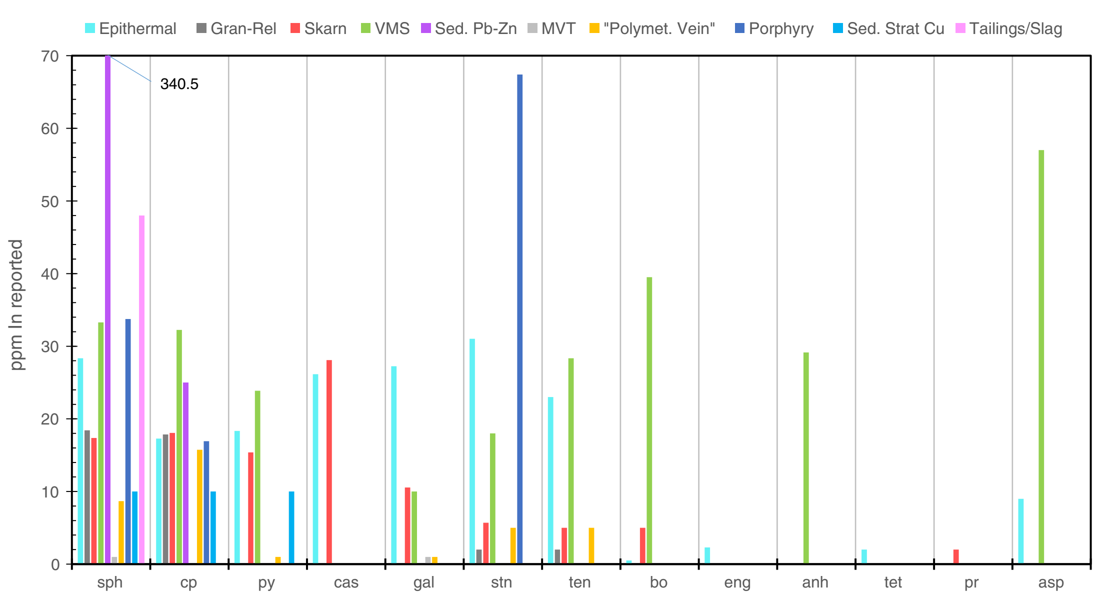

Fig. 7. Average indium grades according to host mineral and deposit type for reported deposits listed in our database. It is indicated here that host mineralogy is not a good proxy for the estimation of indium grades. Minerals are abbreviated as per Fig. 6.

Table 5
Studies estimating global average indium grades or ratios.

Global average indium grade estimate	Reference
106 ppm In	UNEP et al. (2011) — determined using factors from crustal abundance
10 ppm In in Zn ore	Liedtke and Homberg-Heumann (2012) — basis unknown.
1–50 ppm In in base metal sulphide deposits	European Commission (2012) — taken as a reference for theoretical limits
39 ppm In	Yoshimura et al. (2013) — assumed based on a weighted average of 27 selected deposits identified from Murakami and Ishihara (2013)
50 g In per t Zn in Pb–Zn deposits and 10 g In per t Cu in Cu deposits	Schwarz-Schampera (2014) — basis not specified, but assumed to have grounding in some known deposits
0.1 to 100 ppm In in Zn deposits	Tolcin (2014) — taken as a reference for theoretical limits
95 g In per t Zn in Pb–Zn deposits	Licht et al. (2015) — determined using production/efficiency estimates
144 g In per t Zn in Pb–Zn deposits	Mudd et al. (in press-a) — determined using production/efficiency estimates
118.8 g In per t Zn in Pb–Zn deposits	Mudd et al. (in press-a) — modified from the above to account for possible selection bias
6.4 ppm In — calculated average	USGS (2015b) — average of values for the whole database, to represent possible average concentrations in rock at a continental scale
6.55 ppm In — calculated average	OSNACA (Brauhart and Hagemann, 2011) — average of values for the whole database, to represent possible average concentrations across a variety of ore deposits

grade (i.e. $\frac{\sum_{i=0}^{n} G_{i,p}}{n}$, where $G_{i,p}$ is the indium grade for deposit i and proxy p) may also be considered, however the use of actual average grades prohibits the relative comparison of proxies as per Fig. 8, and so all values are normalised to the constant value of total tonnage in the deposit database without affecting the output resource estimates.

It can be seen that most estimates sit within the expected range suggested by Tolcin (2014) and the European Commission (2012), with Schwarz-Schampera (2014) and UNEP et al. (2011) being the exceptions. This is broadly encouraging, and the weighted average grades of most remaining proxies further only range from ~1 ppm to ~12 ppm, which is seemingly conservative given that the highest proxy estimate other than Yoshimura et al. (2013) is still <24% of even the European Commission (2012) maximum (50 ppm). This range still results in large discrepancies for resource estimates, as the total mineral resource tonnage of ~48,519.88 Mt within the 800 deposits considered in the Pb–Zn deposit database combined with a difference in overall grade estimate of 0.1 ppm indium leads to a difference in output resource estimate of ~4852 t In. However, at the very least this approach can assist in restricting the likely range of indium grades from 0.1 to 100 ppm as per Tolcin (2014) or 1 to 50 ppm as per the European Commission (2012) to somewhere in the range of 1 to 14 ppm.

6.2. Data analysis

A direct average of the values shown in Fig. 8 is ~13.52 ppm In, which gives the "Mean" estimate of ~655,882 t In, however this seems to be strongly influenced by the outlying Yoshimura et al. (2013) and UNEP et al. (2011) estimates and it further does not account for the relative merit of individual proxies. The next proposed step is therefore to systematically analyse the strengths and weaknesses of each proxy so that their individual merits can be considered when interpreting these findings and identifying an appropriate mean. Seven criteria are considered in this study to guide the interpretation of proxies. These are listed and described below alongside an indication of how each criterion results in the differential weighting of proxies.

A) *Can reported grades of a deposit be used to estimate the by-product?* — The regressions in proxy 1 (Fig. 3) indicate that higher base metal grades consistently correlate with higher indium grades. A proxy should therefore be able to take one or more known metal grades into consideration when estimating by-product/critical metal grades. If one grade can be incorporated we rate this criteria "Single" and a weighting factor of 0.5 is applied. If more than one grade can be used, then we rate this criteria "Multiple" and a weighting factor of 1 is applied. A weighting factor of zero is otherwise applied for this criterion.
B) *Can deposit type information be considered?* — Analysis of the regressions in the OSNACA database showed that indium grades could be more accurately determined when deposits are disaggregated according to deposit type, suggesting that this is a variable which should ideally be taken into account when

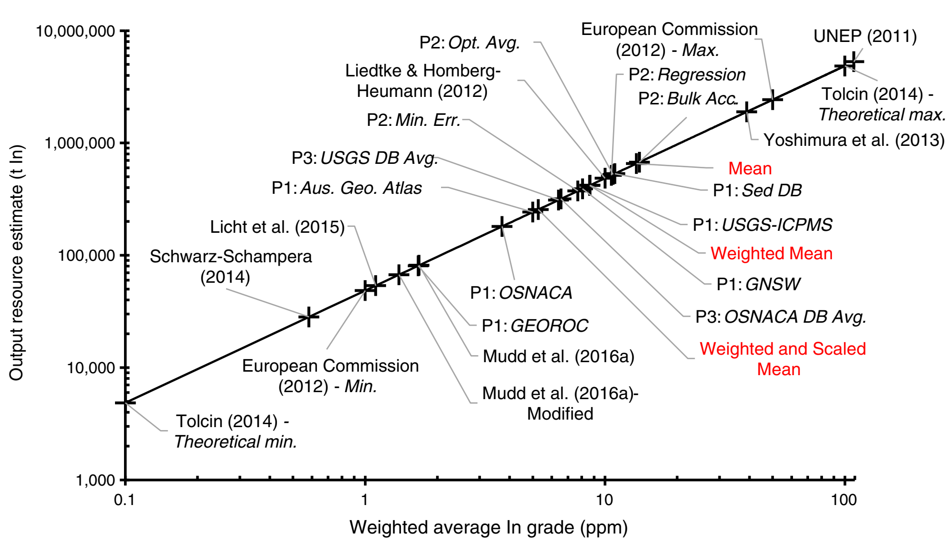

Fig. 8. Summary of grade and indium resource estimates for Pb–Zn deposits identified in Mudd et al. (in press-b). The range of 0.1–100 ppm indium suggested by Tolcin (2014) is set as the absolute bounds of possible average indium grades (termed the theoretical maximum and minimum). The European Commission (2012) publishes a range of 1–50 ppm In. All other estimates shown are used to determine the various mean values, shown in red. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

Table 6
Evaluation of different proxies.

Proxy	A: Accounts for reported metal grades	B: Accounts for different deposit types	C: Incorporates mineralogical information	D: Methodology and/or data sources are trusted and traceable	E: Data based on ore geology/known deposits	F: Rating of internal statistical accuracy	G: General comments or other considerations (not scored)	Percentage influence on "Weighted Mean", resulting from performance against criteria A–G.
Proxy 1: multiple regression								
GEOROC	1	–	–	1	–	0.953	Regressions can accurately predict indium grades, but the uncertainty around the geological relevance of the source data is a weakness of these proxies.	6.4%
Geochemical Atlas of Australia	1	–	–	1	–	0.934		6.4%
EarthChem.org: SedDB	1	–	–	1	–	0.959		6.5%
USGS: National Geochemistry Database: Rock — ICP-MS	1	–	–	1	–	0.937		6.4%
NSW, Australia — Drillhole Assay Database	1	–	–	1	1	0.882	Regressions can accurately predict indium grades, and the source data represent behaviour within ore, however this data is not able to distinguish between individual deposit types.	10.7%
OSNACA (Brauhart and Hagemann, 2011)	1	1	–	1	1	0.742 (average)	Regressions can accurately predict indium grades, the source data represent behaviour within ore, and this data is able to distinguish between multiple deposit types.	10.3% — due to lower average R^2 values than the NSW database, despite being able to account for deposit types.
Proxy 2: analysis of reported deposits								
Optimisation of an In:Zn ratio (minimised error, accurate tonnage, optimised average)	0.5	1	–	1	1	0.291	Highly relevant and appropriate data sources (known indium-hosting deposits), but optimised values are difficult to obtain, resulting in poor statistical accuracy. Further, given that these data represent highly enriched deposits, scaling may be necessary.	8.3% × 3 for each approach
Analysis of mineralogy	–	1	–	1	1	–	While this approach is perhaps the most geologically informed, its statistical grounding is too poor to be applied as a proxy as the standard deviation for average indium grade according to host mineral and deposit type is often greater in magnitude than the mean. As mineralogical information relating to indium is scarcely available for non-reporting deposits and no clear link between host minerals and deposit type was	0% as per criterion G.

(continued on next page)

Table 6 (*continued*)

Proxy	A: Accounts for reported metal grades	B: Accounts for different deposit types	C: Incorporates mineralogical information	D: Methodology and/or data sources are trusted and traceable	E: Data based on ore geology/known deposits	F: Rating of internal statistical accuracy	G: General comments or other considerations (not scored)	Percentage influence on "Weighted Mean", resulting from performance against criteria A–G.
							attainable, it is not possible to apply this proxy to the deposit database.	
Regression based on what's reported	1	–	–	1	1	0.556	While deposit type and mineralogical data are available in the database, the number of deposits with reported grades of multiple base metals required for an effective regression is limited, and hence the integration of deposit type information is restricted.	7.8%
Proxy 3: other published literature								
Yoshimura et al. (2013)	–	–	–	0.5	–	–	This data source is simply an average of 27 selected deposits already mined for indium in Japan. The estimate derived is correspondingly high and would entail a selection bias if applied globally.	1%
UNEP et al. (2011)	–	–	–	0.5	–	–	The methodology is clear and traceable, but based on crustal abundance which makes it problematic to refer to ore. The estimate is also much higher than would be expected even for some of the most indium-enriched deposits.	1%
Liedtke and Homberg-Heumann (2012)	–	–	–	–	–	–	The background to this estimate is unclear.	0% — due to poor ratings against every criterion.
Licht et al. (2015)	0.5	–	–	1	0.5	–	Zn ratio derived from literature, some of which represents known deposits.	3.3%
Mudd et al. (in press-a)/Mudd et al. (in press-a): Modified	0.5	–	–	1	0.5	–	Zn ratio derived from literature, some of which represents known deposits.	3.3% × 2 for each approach.
USGS (2015b)	–	–	–	1	0.5	–	Generalised dataset considered broadly representative of the deposit database and hence the average values are considered as a "literature" estimate under proxy 3. Unlike proxy 1, this approach does not involve a preliminary statistical accuracy test.	2.2%
OSNACA (Brauhart and Hagemann, 2011)	–	–	–	1	1	–	Generalised dataset considered broadly representative of the deposit database and hence the average values are considered as a "literature" estimate under proxy 3. Unlike proxy 1, this approach does not involve a preliminary statistical accuracy test.	4.4%
Schwarz-Schampera (2014)	0.5	–	–	–	0.5	–	This reference appears to incorporate known deposits to infer a single global ratio, however the justification behind the estimate is unclear.	2.2%

considering a proxy's accuracy. This criterion is applied to determine if a proxy permits different estimates based on a deposit type classification. If deposit type information can be incorporated, a proxy is assigned a weighting of 1 against this criterion, and otherwise a weighting of zero is applied.

C) *Can mineralogical information be considered?* — This criterion is applied to determine if a proxy recognises and accounts for differences in grades of by-products for a deposit depending on which minerals are present, and which are thought to host the by-product in question. A weighting value of 1 is applied if mineralogical information can be considered.

D) *Are the data sources traceable and/or is the methodology clear?* — This is necessary to ensure that results are reproducible and are based on source data which are of an acceptable quality. If the methodology and data sources are both known and trusted, a weighting of 1 is applied. If there is information about one but not the other, or the methodology data sources are known but questionable, a weighting of 0.5 is applied. Where the methodology and data sources are both unclear, a weighting of 0 is applied.

E) *Are the data geologically representative of the mineral deposits we wish to apply them to?* — This criteria is applied to establish whether or not the proxy data are based on, or are otherwise representative of, behaviour within ore, as this is the target for deposits requiring a by-product estimate in the collected database. A weighting of 1 is applied only in the case that the data are directly attributable to ore.

F) *Are the data statistically accurate?* — There should ideally be evidence that the by-product grades predicted can match actual/reported values. This is testable for proxies based on regression. The R^2 values, as presented in Table 2, can be used to weight these proxies down by the extent of their uncertainty (i.e. if $R^2 = 0.8$, this criterion weights the proxy to 80% of its original influence). Where no evidence is provided, a proxy is weighted 0 against this criterion.

G) *Are any other factors worth considering which might affect the validity of the data?* — This seventh criterion allows for any additional factors which are not encompassed by the first six criteria, but which still might affect the validity of the data source to be at least qualitatively considered. It is possible that factors considered in this criterion justify the exclusion of a proxy altogether.

Criteria A–C relate to a proxy's ability to allow by-product grades to be estimated for individual deposits based on the specific characteristics of these deposits. This means that the proxy permits differences between host minerals, deposit types or host metal grades to be accounted for in estimating the by-product contents of a given deposit. Criteria D–G pertain to the relevance and theoretical merit of the data sources, which must also be carefully considered when weighting the value of different proxies. Each proxy is listed in Table 6 against the criteria A–G to indicate their relative merits. The final column indicates the influence of each proxy on the weighted mean, according to their performance against the criteria. The percentage contribution is determined by:

$$Weightinginfluence\ (\%) = \frac{\sum_{i=0}^{N} P_i}{N \times \sum_{i=0}^{N} P_i} \times 100 \tag{3}$$

where P_i is the score for a proxy against criterion i, up to N criteria.

When the results for each proxy are weighted according to the final column in Table 6, a more qualitatively and quantitatively appropriate mean is determined. As shown in red in Fig. 5, this is the "Weighted Mean", which sits at an average of 8.63 ppm In across the Pb–Zn deposit database, and results in an estimate of ~418,700 t In across these 800 deposits.

In addition to the weighting of proxies, scaling may also be used if it is believed that a proxy's results are subject to bias. In our case, the values under proxy 2, Yoshimura et al. (2013) and UNEP et al. (2011) could all be scaled, given that they have been calculated from deposits which are theoretically the most enriched, or are otherwise outside of reasonable bounds (e.g. a 100 ppm maximum as per Tolcin, 2014). The average grade estimate from potentially scalable proxies is ~36.08 ppm In, while the remaining proxies collectively average at ~5.06 ppm. Assuming that there is indeed a selection bias found for these proxies, one could scale these results down by their weight average grades versus the average of other proxies which don't require a scale, e.g. for the proxy 2 regression: 5.06/10.63 or 47.6% scale down. When all proxies are considered for possible scaling, a new global estimate of ~255,580.24 t In is obtained, at a weighted average grade of 5.27 ppm In. The weighted average grade should ideally not be applied directly to individual deposits in a database, but used to find a corresponding value for g In/t Zn that achieves this weighted average across the database. This In:Zn ratio can then be applied to individual deposits so that each one is given a separate indium grade that reflects their specific Zn enrichment. This step also permits analysis of grade differences between deposit types, or between countries, which may be of interest for future criticality assessments.

These processes of weighting and scaling allow uncertainty to be encompassed into the final estimates, and for deeper analysis and understandings of the results to arise. Considering the 800 deposits estimated via proxies, plus the 98 deposits with reported indium grade/contents in the literature (see Supplementary data), a total estimate of ~330,000 t In is achieved. Additional deposits of Cu and Sn may also be compiled and analysed to estimate additional indium resources beyond these 898 deposits, however the scope of this paper is primarily to demonstrate the possible application of the proxy method to a deposit database. Further analyses of global indium resources are provided in Part III of this series of studies (Werner et al., 2016).

7. Discussion and conclusions

7.1. Applicability

The processes described in this paper, and illustrated using indium, are not metal specific and may technically be applied to any by-

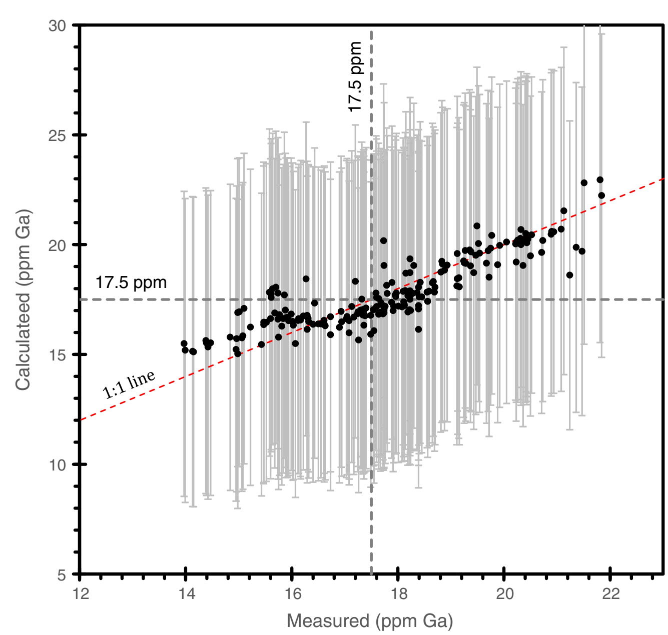

Fig. 9. Use of Sed DB data on Al and Zn to generate a Ga proxy ($R^2 = 0.76$, average error = 4%, Ga crustal average for the upper crust estimated at 17.5 ppm Ga in Rudnick and Gao, 2014).

product/critical metal for which geochemical data, production statistics and/or academic literature are available. This significantly extends the suite of metals for which deposit-by-deposit methods may be applied, an exciting prospect in terms of critical metals research. Furthermore, this method may use existing published databases of base metals, and leverage these databases to identify most, if not all, of the individual deposits from which resource assessments need to be made. For example, existing studies of Cu/Cu–Mo deposits (e.g. Mudd et al., 2013a) may be used to identify Re resources using the same approach outlined for indium here. This reduces the amount of time needed to perform detailed by-product/critical metal resource assessments. The efficiency of assessment is also assisted in that it is possible to use many of the same publicly available data sources as presented in this study to create analogous proxies for other metals. For example, using the Sed DB data, a proxy for Ga grades as a function of Zn and Al grades, its two major base metal hosts, can also be produced (Fig. 9).

Additionally, the method we present allows different proxies, and therefore different types of information sources, to be embedded. This means that the assessment of global by-product and critical metal resources need not be restricted to the types of data presented here. As geochemical and mineral deposit databases become updated, or other new information sources become available, they can be analysed and included in the framework presented here to update the result obtained without significant adjustments needed.

We have demonstrated our method for a metal subject to particularly low levels of reporting (<1% of potentially indium hosting deposits, see Part III, Werner et al., 2016). This has guided the development of the methodology as depicted in Fig. 1. However, for metals subject to an intermediate level of reporting, i.e. by-product metals with a greater market and production volumes, the body of reported data may be sufficiently representative of the geochemical behaviour of that metal, to the point that it may be used as an embedded data check for proxies. Cobalt (Co) would be an ideal candidate for this alternative approach, given the extent to which it is reported in known mineral deposits (see Mudd et al., 2013b). This alternative approach is illustrated in Fig. 10. Additional, better reported by-products like Co might also produce better results for proxy 2 than we see with indium, as proxy 2 relies on the quality of reported data to infer estimates for unreported deposits.

As a final note on applicability, it should be noted that there are various considerations that would lead to the mineral resources listed being converted to actual supply (e.g. by-product processing pathways and recovery rates), and many of these are not considered in the scope of this work. While we have considered the ratio of by-product to host metal using some knowledge of processing behaviour in proxy 3 (discussed further in Part I, Mudd et al., in press-a), this is otherwise not considered in the depth that is presented elsewhere, for example by Frenzel et al. (2015). To be clear, we do not present a methodology that explicitly assesses supply potential. Our approach explicitly aims to quantify mineral resources in known deposits, and overcome a lack of code-based reporting by implementing sound approaches to by-product grade estimation.

7.2. Uncertainties and challenges

7.2.1. General and reporting

There are two types of uncertainty presented in this study. For the reported database, uncertainty is qualitatively indicated via the evaluation and classification of the reporting (i.e. "H" to "L"). This differs significantly from the quantitative uncertainty, yet both must be taken into account when interpreting the results of our methodology. While these indications of uncertainty are not directly comparable to one another, they still present a level of data provenance and reliability rarely seen in mineral resource accounting.

As highlighted by Frenzel et al. (2014), a weakness of deposit-by-deposit assessments of global resources (e.g. in Mudd and Jowitt, 2014 and adapted in this paper) is that they will always produce values below that of the true recoverable endowments, and it is further not possible to know by how much our estimate falls short. This is indeed true, as it can be challenging to identify every single deposit containing a particular commodity from a review of publicly available information (see part I, Mudd et al., in press-a). Similarly, there will be metals in deposits yet to be discovered that clearly cannot be accounted for using our methodology. However, it is believed that in applying proxies to quantify unreported deposits, this study has, at least to some extent, reduced the risk of this underestimation.

The amalgamation of disparate data sources, as promoted by our methodology, also naturally entails uncertainty. When it comes to using reported data to feed directly into the database, the use of tonnage from one study and the grade of another is potentially problematic as there can be little assurance of consistent modelling techniques and assumptions between the studies at the deposit level. The veins and orebodies assessed and actively mined within a broader ore field might, for example, have changed over time, without changing the name of the deposit (see Jowitt et al., 2013; Mudd and Jowitt, 2014).

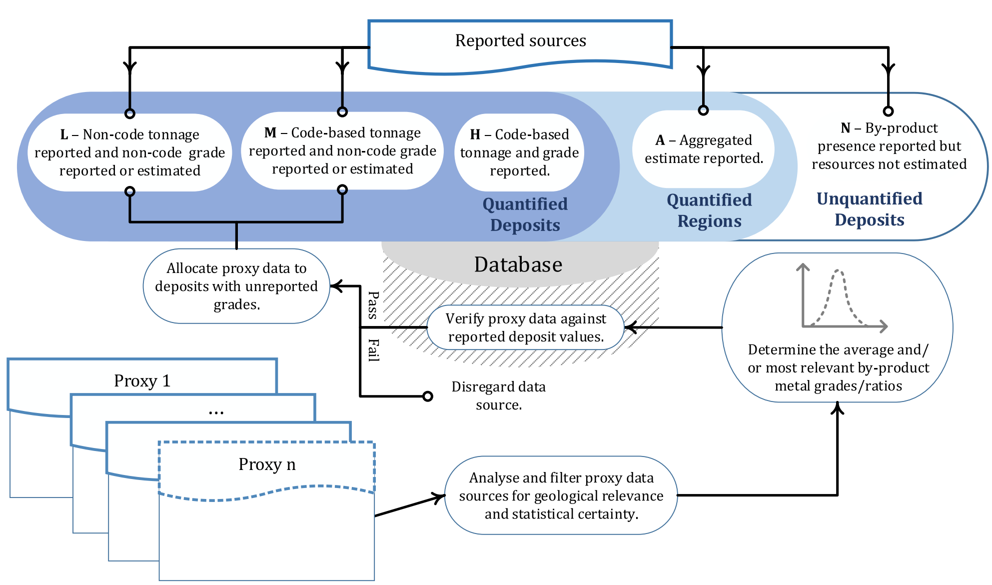

Fig. 10. Alternative approach, incorporating proxy verification against reported data.

Another possibility is that in the time between two reviewed studies, a deposit in question may have been extensively exploited, resulting in higher grade ores within the deposit being depleted. This would have the effect of our study overestimating the ore tonnage and the ore grade. The challenge of interpreting different data sources is highlighted by the Gai(skoye) deposit, Russia, which is reported to contain significant quantities of indium due to its considerable ore tonnage of 380 Mt. Possible questions about the validity of this deposit's estimate arise from the fact that the grade values found for this deposit were published by Prokin and Buslaev (1998), who at the time noted the tonnage of the deposit to simply be ">300 Mt". A later study by Herrington et al. (2005), which was ultimately used for this deposit, clarifies this to be 380 Mt, although it is still unclear whether this has been based on a rigorous exploration and assessment regime for that deposit. Whether or not the original grade estimate is still applicable to this new volume is speculative, however there was no theoretical basis for using any other value. The use of 300 Mt in this deposit would have produced an estimate 1920 t In lower, a significant margin for a single deposit (see Part III, Werner et al., 2016 and Supplementary data). This use of disparate data sources is, however, necessary given the state of reporting of many by-product resources, and is encompassed in our classification scheme for data quality (H, M, L).

Some studies that publish both ore tonnage and grade, or simply total indium content, avoid the uncertainties associated with disparate data sources, although many were still classified as low quality sources given their lack of the use of code-based resource data, and due to their propensity to be easily misinterpreted. The values published for indium by Murakami and Ishihara (2013), for example, could be misinterpreted to represent the remaining resources in a number of deposits, while some more accurately reflect the total endowment of indium in that deposit over time (i.e. the resources, reserves and cumulative production over time). Efforts need to be made where possible to ascertain whether or not the published orebody tonnage is still relevant in the database for deposits classified as L or M.

7.2.2. Proxies

7.2.2.1. Proxy 1. Geochemical databases and multiple regression allow uncertainty to be quantified and visualised, as we have shown in Fig. 3. This statistical uncertainty does not reflect other qualitative aspects such as geological relevance, and is limited in that the correlations shown apply only where all the grades incorporated into the regression are also reported for known deposits, which is not always the case. Thus in terms of statistical uncertainty, proxy 1 has both positive and negative aspects. This proxy is also subject to the nature of measurements made in each database. For example, indium measurements from surface soils in the Australian Geochemical Atlas, were found to clump in a limited set of values (Fig. 3c), which produces greater uncertainty from multiple regression than if the sample showed more variation.

The use of the proxy 1 equations inevitably results in overestimates and underestimates at the deposit level, leading these equations to suggest that for some deposits with very low base metal grades, indium grades are in fact negative. This is clearly not possible, although no deposit-level results were modified (e.g. by setting the minimum per deposit estimate to zero, or to the crustal average) so as to ensure the accuracy of proxy 1 at the global level. This could, however, be done if it could be ensured that uncertainty of global resource estimates is not considerably affected, and remains quantifiable. In the above case, no negative deposits were calculated for the Australian Geochemical Atlas, and ~3% of deposits were counted as negative by the equations of the other databases used.

7.2.2.2. Proxy 2. The development of an In/Zn ratio was considered due to the strong relationships known to exist in the processing of indium and Zn. This relationship can be used to gauge the results of using reported grades of indium within Zn-enriched mineral deposits. Actual grades of indium and Zn were found to vary considerably between reported deposits, making it more difficult to optimise a ratio for each deposit type. The number of deposits where we could identify the type of deposit and the grades of indium and Zn were few, and this appeared to significantly limit the accuracy of the ratios obtained. Furthermore, if a base metal database contains deposit types for which no ratio can be found from reported data, this proxy cannot be applied to those deposits. This suggests that this proxy might be more appropriately applied when the state of reporting is better than it is for indium. This, however, does not exclude it from use with other by-product/critical metals. As shown in Table 4, this proxy was not able to match reported data accurately, and generally underestimated indium resources, despite being one of the higher proxies when used in the Pb–Zn database. Further development of this proxy will be needed to better establish the degree of weighting and scaling needed, and to establish better grounds for the quantification of this proxy's uncertainty. At present, one could use the ratio of H, M and L estimates contributing to the development of the by-product/base metal ratio, however it would take some further work/justification to translate this to a quantifiable weighting. Our results generally suggest that assigning fixed by-product to base metal ratios using known deposits is a highly uncertain practice, despite the common application of this approach in the literature (see Table 5).

Attempts were made to find consistency relating to the minerals reported to be host to indium among the deposits present in the reported database (see Supplementary data). However, as shown in Figs. 6 and 7, there is considerable variability of indium grades between host minerals, and increases in statistical consistency are not seen when disaggregating between deposit types. Even if a statistical link between host minerals and deposit types, or between host minerals and major commodity grades is found, it is still problematic to assume host mineralogy for deposits not reporting indium. This is quite evident from the variety of minerals present in Figs. 6 and 7, and instance of outlying cases such as Malku Khota, Bolivia, where indium is hosted by jamesonite and acanthite (see Supplementary data), but no other locations reported these minerals.

Of the approaches considered under proxy 2, the regression approach produced the best results from a statistical perspective. It is analogous to the approach of proxy 1, but is based on reported deposits, meaning it rates better than most geochemical databases against criterion E (Table 6). While this makes it highly geologically relevant, the state of reporting limits the application of this approach, and indeed a significantly lower R^2 value around 0.5 raises questions about the accuracy of grade estimates at the individual deposit level. At this stage, it would appear that endeavours to perform regressions, as well as optimise an individual by-product to host metal ratio would benefit from improved reporting, significantly more so than attempts to draw mineralogical links.

7.2.2.3. Proxy 3. This proxy involves the use of estimates derived from literature, and is possibly the most problematic in terms of determining the source and quality of estimates. The uncertainty associated with these estimates is therefore difficult to quantify, and indeed, not quantified in our indium example. Given that the results of proxies 1 and 2 both have some level of data provenance from which we can apply scrutiny, data collected under proxy 3 was consistently weighted lower. More analysis is needed to understand the sensitivity of the final results to the values used in proxy 3, and to better understand the quality and source of these data. This would result in a greater suite of sources to be incorporated, as is demonstrated by our analysis of Liedtke and Homberg-Heumann's (2012) estimate of 10 ppm In as an average estimate in Zn ore (see Table 6). While this value is consistent in comparison to the other results obtained, its basis is so unclear that it was weighted to 0% for failing to meet any of our weighting criteria.

The use of literature as a reference tool is still broadly considered a useful addition, as Fig. 8 shows that most resource estimates clumped into a smaller fraction of the theoretical limits for indium in Pb–Zn deposits. Furthermore, from the range of values presented in proxy 3, and an understanding of their sources, one can still be reasonably sure that estimates upwards of ~15 ppm In are a likely overestimate, thus constraining our understanding of the acceptable range of average indium grades more so than the European Commission (2012) range of 1–50 ppm.

7.3. General contribution and conclusions

Detailed inventories of metal-bearing deposits can assist in strategically managing our remaining non-renewable resources. As population growth and technological advancement both accelerate, the need for these inventories will increase. Researchers in the field of economic geology are well positioned to inform society of the state of the world's metal resources, and studies have indeed emerged which enable a good understanding of primary mineral commodities (e.g. Ni and REEs: Mudd and Jowitt, 2014; Weng et al., 2014). Yet for these studies, we see a notable absence of by-products where they could be reported, e.g. Sc in REE deposits (see Weng et al., 2015), and Ga, Ge and indium in Pb–Zn deposits (Mudd et al., in press-b). This has led to urgent, yet broadly unanswered calls on the geoscientific community to respond with further data on these by-products (Herrington, 2013).

In this study, we have introduced a framework by which the global resources of any metal commodity may be estimated in such a way that is reproducible, is leveraged off of existing, publicly available data, does not require overly sophisticated modelling or software, and which explicitly overcomes the lack of reporting by mining companies to produce more detailed and defensible estimates. The uncertainties of these estimates have been discussed, and ways in which they can be qualified (through reporting classifications and multi criteria analysis) and quantified have been embedded into the methodology, which is an improvement upon many previous attempts to characterise global critical metal resources. We have identified avenues for refinement in the methodology which must be explored to further reduce these uncertainties. The application of the methodology to indium has been demonstrated, and the full inventory for indium and further analysis of indium resources is presented in Part III (Werner et al., 2016).

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Appendix A. Supplementary data

Supplementary data to this article can be found online at http://dx.doi.org/10.1016/j.oregeorev.2016.08.008.

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5. A Comprehensive Assessment of Global Indium Resources

In this chapter, the method introduced in Chapter 4 is applied extensively to indium, involving the creation of a reported deposit database, and proxy analysis of two databases containing deposits with Pb, Zn and/or Cu as their primary commodities. This results in a global assessment of indium resources on a per-deposit basis. The way these resources apportion between countries, levels of reporting and deposit types is revealed. This chapter shows that significant quantities of indium exist in countries which do not refine indium, although the Zn concentrates exported from these countries are likely to play some role in the global supply chain. These countries include Bolivia, India and Australia, which would be well positioned to enter the market for refined indium, should the current supply chain experience disruptions. An additional discussion of indium potential from mine wastes is provided, including assessment of what is reported in mine wastes globally, plus an assessment of indium potential in Broken Hill, Australia. This study addresses research questions 1a, 1b, 1c, 2a and 2b. Supplementary information for this chapter is provided in Appendix B.

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The world's by-product and critical metal resources part III: A global assessment of indium

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ABSTRACT

Indium has considerable technological and economic value to society due to its use in solar panels and liquid crystal displays for computers, television and mobile devices. Yet, without reliable estimates of known and potentially exploitable indium resources, our ability to sustainably manage the supply of this critical metal is limited. Here, we present the results of a rigorous, deposit-by-deposit assessment of the global resources of indium using a new methodology developed for the assessment of critical metals outlined in Part II of this study (Werner et al., 2017). We establish that at least 356 kt of indium are present within 1512 known mineral deposits of varying deposit types, including VMS, skarn, epithermal and sediment-hosted Pb-Zn deposits. A total of 101 of these deposits have reported indium contents (some 76 kt of contained In) with the remaining 1411 deposits having mineralogical associations that indicate they are indium-bearing, yielding ~280 kt of contained indium. An additional 219 deposits contain known indium enrichments but have unquantifiable contents, indicating that our global resource figure of 356 kt of contained indium is therefore most certainly a minimum. A limited number of case studies also indicates that a further minimum of ~24 kt indium is present in mine wastes, a total that is undoubtedly smaller than reality given the minimal reporting of mine waste indium concentrations, and the extensive volume of historical mine wastes.

These quantities are sufficient to meet demand for indium this century, assuming current and projected levels of consumption. However, given indium's classification as a critical metal, its supply still remains a concern, and hence we have also discussed the economic viability and spatial distribution of the indium resources identified during this study to further our understanding of the geopolitical scarcity of this critical metal. Our results suggest that the global indium supply chain is fairly adaptable, primarily as the spatial distribution of indium resources deviates significantly from the current supply chains for this metal. Our study provides a stronger basis for future studies of indium criticality, provenance, supply chain dynamics, and stocks and flows in the fields of economic geology and industrial ecology.

1. Introduction

Indium (In) is a soft, silvery-white metal that is extracted from Zn, Sn and Cu mineral deposits. Since its discovery in 1863 and first presentation to the public at the 1867 World Exhibition in Paris (Schwarz-Schampera and Herzig, 2002), it has shifted from being a scientific curiosity to a technological necessity, and is used extensively today in solar panels, liquid crystal displays (LCDs) and touchscreens. It is indeed possible that you are reading this on a screen coated with indium-tin oxide (ITO). The role of In in the consumer electronics and renewable energy industries has fostered unprecedented growth in demand which is likely to continue this century (Werner et al., 2015). However, In is one of the least abundant elements on Earth (at ~56 ppb; Rudnick and Gao, 2014) and has a supply chain that is dominated by relatively few countries. This has led to its classification by multiple government and industry bodies as a critical metal (European Commission, 2014; Jowitt, 2015; Skirrow et al., 2013; Zepf et al., 2014). The significant concerns over the risks of In supply restrictions primarily relate to China's dominance of the In market and its history of imposing export constraints (see also Candelise et al., 2012). If In supply does become restricted, either by geopolitical or geological means, one or more of the following outcomes might be expected, similar to prior cases with cobalt, palladium and the rare earth elements (see Habib, 2015; Mudd et al., 2013b; Sprecher et al., 2015; Weng et al., 2015):

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1. The market price for In increases for a period of typically less than 5 years, until the market responds to adapt the supply situation, leading the price to return to its pre-disruption levels. This was seen in 2010, when Chinese export restrictions caused a tenfold spike in neodymium prices, only for them to return to their pre-disruption prices months later. It is however worth noting that In prices have fluctuated between \$300 and \$800 US dollars/kg in the last decade without such a noted supply restriction.
2. Technological innovation takes place to substitute and/or minimise In in its current end-uses. In a recent study, Graedel et al. (2015) developed a scale by which metals can be rated for their substitutability, where 0 is highly substitutable and 100 is completely irreplaceable. Here, In rated 60, suggesting that there would be some technical challenge in replacing In for its primary end uses, particularly without a loss in functionality/performance for those end uses.
3. In-rich concentrates from existing operations become more frequently processed at In-capable refineries, either by diversion or upgrading of existing facilities.
4. A broader diversification of the In supply chain leads to new mining projects developing in previously non-producing countries, and/or the recovery and reprocessing of In from mine wastes and end-of-life wastes increasing, with a subsequent rise of recycled stocks entering the market. Woodhouse et al. (2012) have examined the thresholds at which the supply of In for photovoltaic (PV) modules could be augmented.

Exponential growth in demand for In since the early 1970's has so far been met with limited shortages, thanks to remarkable improvements in the efficiency of production (see Fig. 1). This is in large part due to greater separation of In from Zn concentrates and adaptations made during the production of ITO, as processing residues from the ITO sputtering process that were originally considered waste are now largely recovered and contribute to nearly half of the In supplied to end users (Duan et al., 2015; Goonan, 2012). The extent to which such improvements in processing efficiency could continue to meet growing demand for In is, however, unclear. There are also uncertainties relating to the In supply chain, which is complicated by its strong dependence on Zn production (Nassar et al., 2015), and the fact that future demand for In might necessitate an oversupply of Zn and/or greater energy required for extraction than could be recovered by the renewable energy technologies dependent on In in the first place (Elshkaki and Graedel, 2015).

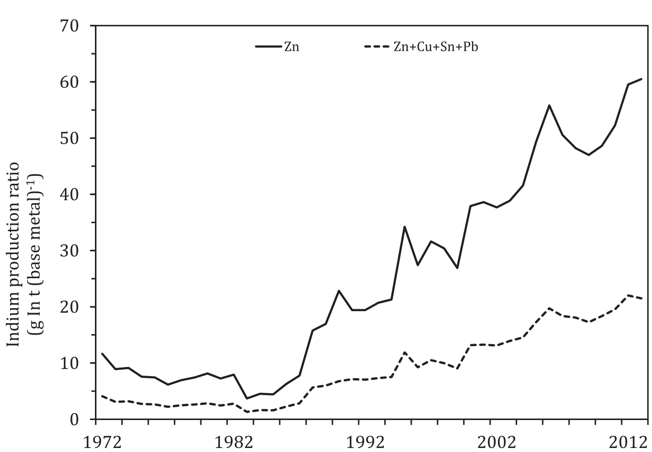

Fig. 1. Grams of refined indium per tonne of mined base metals (global), 1972–2013, showing the significant improvements in production efficiency achieved in response to increasing demand for indium. Compiled from USGS (Var.) and OCE (2014).

Policymakers and industry must therefore act strategically to ensure a sustainable supply of In for both current and future generations. This will require good knowledge of the location, quantity and quality of economically extractable primary and secondary sources of In, wherever available. However, at least in terms of primary In resources, previous studies have not produced clear or well-supported findings. For example, reports in 2007 suggested that In resources would deplete in only 10 years (Cohen, 2007), yet this study, as well as a later study by Moyer (2010) relied on misinterpretations of data originally published by the USGS (see Mudd and Jowitt, 2014). More recent studies assessing total In resources generally contradict this view, but provide little to no quantitative data to support their conclusions. This has led to enormous discrepancies in published global resource estimates (between 16 and 570 kt of contained In from studies published between 2008 and 2014, Werner et al., 2015). These discrepancies might initially suggest that defensible views on In scarcity or resource management are not possible; however, the uncertain results of past studies have remained in use as the only sources to inform policy development and mineral exploration programs. There is a need for a more authoritative assessment of In resources, and this is amplified when considering the many branches of research on In, which include, but are not restricted to: the geochemical behaviour of In (e.g. Lopez et al., 2015; Pavlova et al., 2015), its criticality (e.g. European Commission, 2014; Skirrow et al., 2013), its anthropogenic stocks and flows (e.g. Licht et al., 2015; White and Hemond, 2012; Yoshimura et al., 2013), its lifecycle impacts (e.g. Fthenakis et al., 2009; Marwede and Reller, 2014), its processing and recoverability from wastes (e.g. Alfantazi and Moskalyk, 2003; Li et al., 2006; Zimmermann and Gößling-Reisemann, 2014), and of course its scarcity (e.g. Duan et al., 2015; Weiser et al., 2015; Woodhouse et al., 2012; Zuser and Rechberger, 2011). To varying degrees, these studies require some understanding of In resources, and could produce more refined conclusions in their own respects if they could cite more robustly compiled and assessed resource data. As such, the geoscientific community has been called upon to develop detailed inventories of important commodities to address this uncertainty and assist in meeting the challenges of rising demand (Herrington, 2013).

Here, we provide a more authoritative estimation of the endowment of In in mineral deposits globally by presenting a detailed compilation and analysis of the world's In-bearing deposits, making use of CRIRSCO mineral resource reports where available. The following sections provide a summary of In mineralogy, deposit types, reported and inferred In deposits, and an analysis of the implications for future supply. The methods by which our deposit database has been compiled are described in detail in Part II of this study, which also introduces the data sources employed and explains the quantification/qualification of uncertainty. This approach has been developed in response to the uncertainties identified in Mudd et al. (2016). It is therefore recommended that Part I (Mudd et al., 2016) and Part II (Werner et al., 2017) of this study be referred to in conjunction with the following sections. A study by Mudd et al. (2017) also provides key data on global Pb-Zn resources which have been employed heavily in this study, and should also be referred to for further specific details on Pb-Zn deposits.

2. Indium mineralogy and major deposit types

2.1. Indium mineralogy

Twelve dominant In mineral phases have been identified so far (Schwarz-Schampera, 2014; Schwarz-Schampera and Herzig, 2002), although these minerals are almost always not concentrated in sufficient quantities to make them economic to extract. As such, there are currently no mines that produce or resources that host In as a primary commodity. Some minerals that contain In as a major element, most notably roquesite ($CuInS_2$), are present as microscopic or sub-microscopic inclusions within more common minerals such as chalcopyrite, cassiterite, stannite and sphalerite, enabling In to be extracted as a by-product of primary Cu, Sn and Zn production. Some chalcopyrite within Sn deposits appears to contain high concentrations of In and strong statistical correlations between In and Sn are also present in other deposits such as the epithermal Pingüino deposit in Argentina (Jovic et al., 2011; Schwarz-Schampera and Herzig, 2002). However, the most economically important source for In remains sphalerite, with over 95% of global In production derived from Zn processing pipelines and at least 80% of Chinese In sourced from sphalerite-bearing ores (Qian et al., 1998). This dominance is explained by the fact that In can be easily incorporated into the sphalerite crystal lattice by coupled substitution with Cu ($2Zn^{2+} \leftrightarrow Cu^{+} + In^{3+}$), also suggesting that higher In grades can be associated with elevated Cu concentrations in sphalerite (Cook et al., 2009, 2012). Further discussion of In enrichment characteristics and a summary of studies on In geochemistry are presented in Part II of this study (Werner et al., 2017).

2.2. Major indium deposit types

The majority of important In-associated mineral deposit types have been described in Schwarz-Schampera (2014) and Werner et al. (2015), although some of these descriptions (e.g., polymetallic vein-type deposits) provide no indication of the geological processes that formed these deposits. Clear delineations of deposit types are necessary for the purposes of classification and statistical inferences, as some resource estimation methods require knowledge of deposit type (Werner et al., 2017). We have therefore expanded on the descriptions provided in Werner et al. (2015) to produce a definitive collection of In-related deposit descriptions, all of which were used to classify the reported In deposit database presented in this paper. Each of the deposits within our databases were classified using publicly available sources. Where multiple deposit types are present (e.g. Jowitt et al., 2013) the deposit was classified according to the dominant deposit type. Additional deposit types potentially hosting In mineralisation due to the presence of sphalerite (but not necessarily reported to contain In) are described in Mudd et al. (2017).

2.2.1. Volcanic-hosted or volcanogenic massive sulphide (VHMS/VMS)

VMS deposits form from heated, hydrothermal fluids discharged from vents in submarine volcanic environments at or near to the sea floor, which then precipitate massive sulphides. They source metals from interactions between source rocks such as epidosites and modified seawater (Jowitt et al., 2012). These deposits are major sources of Zn and Cu, as over 275 VMS deposits hosting Pb-Zn mineralisation are identified in Mudd et al. (2017), and at least 31.56 Mt of contained Cu are identified for VMS deposits in Mudd et al. (2013a,b). VMS deposits are known to host a wide variety of metals that are often present at high grades, which makes them economically attractive to extract and somewhat protected from the price fluctuations of individual metals (Galley et al., 2007). They are the most common deposit type among deposits reporting In in our database, hosting grades between 1 and 320 ppm In.

2.2.2. Epithermal

Like VMS deposits, epithermal deposits can host economic quantities of a wide range of metals. These most commonly include Ag and Au, although epithermal deposits can also contain Zn, Cu, Pb, As, Sb and Sn, all of which are particularly important for In (see Simmons et al., 2005). Epithermal mineral deposits form at temperatures up to 300 °C and in shallower depths (less than 1.5 km) within subaerial hydrothermal systems, driven by magmatic heat sources commonly within volcanic arc settings (Simmons et al., 2005). They may form part of greater co-genetic epithermal–porphyry–skarn systems (e.g. Jowitt et al., 2013) and notable In-bearing examples include the epithermal polymetallic deposits of Patagonia, Argentina (Jovic et al., 2015), as exemplified by the Pingüino deposit, which hosts ~122 t In.

The pH of epithermal deposits is reflective of the oxidation state of sulphur in the ore fluids, a factor that is used to sub-classify epithermal deposits into low, intermediate and high sulphidation types (Hedenquist et al., 2000). Although all sulphidation types may host In resources, it is likely that the most important of these for our study are intermediate sulphidation type epithermal systems that host Pb and Zn mineralisation. More research is needed to better understand the links between In mineralisation and these sub-types of epithermal systems.

2.2.3. Skarn

Skarn mineral deposits form during the interaction between magmato-hydrothermal fluids derived from plutons and associated deeper magma chamber systems and (usually sedimentary) wall rocks. This formation arises by a variety of different metasomatic processes that generally occur during contact metamorphism and can take place within or adjacent to magmatic plutons (Meinert et al., 2005). While not always the case, skarns can be genetically related to porphyry and epithermal deposits within larger magmato-hydrothermal systems. They can host a wide variety of metals including Fe, Pb, Zn, Cu, Au, Ag, Bi, Te W, Sn, Mo and As, and the presence of these metals is dependent on differences in composition, oxidation state and the metallogenic affinity of the pluton (e.g. Einaudi et al., 1981). It is thought that Zn, Sn and Cu skarns are probably the most important for In mineralisation, as exemplified by the Dachang and Geiju skarns in China. However, as far as the authors are aware, more research is still required to clarify this.

2.2.4. Sediment-hosted Pb-Zn

The sediment-hosted Pb-Zn class of deposits include orebodies that are not genetically related to igneous activity, are sediment-hosted, and have Pb and/or Zn (rather than e.g., Cu) as their primary commodity (Leach et al., 2005). These deposits represent the world's most important source of Pb and Zn (Mudd et al., 2017) and are therefore of considerable interest for their In content. The two primary subsidiary classifications of sediment-hosted Pb-Zn deposits are sedimentary exhalative (SEDEX) and Mississippi Valley-type (MVT) deposits. Given that both SEDEX and MVT deposits are not directly linked to igneous activity, they typically form at lower temperatures than, for example, volcanogenic massive sulphide deposits, which are formed via magmatism/volcanism (see Leach et al., 2005). SEDEX deposits are formed via the venting of hydrothermal fluids onto the seafloor, which can also result in the replacement of existing sediments. In comparison, MVT deposits form as a result of the circulation of low temperature and high-salinity basinal or connate fluids during the diagenesis of sediments in sedimentary basins (Robb, 2004). This deposition of metals generally occurs during the deposition of the sediments that host fluid flow within MVT systems. These distinctions between SEDEX and MVT deposits can at times be subjective, leading to some disagreement for deposits which show characteristics of both types (Leach et al., 2005). In addition, sediment-hosted Pb-Zn deposits are further classified into sub-types such as Broken Hill- or Irish-type deposits, which can be difficult to classify for the purposes of mineral resource accounting as technical literature often does not contain sufficient information. As such, we categorise only between SEDEX and MVT sub-types in this study, as was also done for the mineral resource accounts presented by Mudd et al. (2013a). In terms of commodities, SEDEX and MVT deposit types are both dominated by Pb and Zn, though they can also be important sources of Ag and Cu. Additional common by-products include: As, Ba, Bi, Ge, Hg, Mn, Ni, P, Sb and Tl. As discussed in Werner et al. (2015), In demand is dominantly met by the production of In as a by-product of Zn, and our database shows that substantial known quantities of In are present in sediment-hosted Pb-Zn deposits (e.g. Malku Khota, Broken Hill and Huari Huari).

2.2.5. Granite-related

The anhydrous melting of the lower crust (e.g. during mantle plume-related underplating), the melting of igneous or sedimentary rocks during metamorphism, or extreme fractionation of mafic magmas can generate highly evolved magmas. When these magmas cool and solidify, they can form either coarse-grained igneous rocks, known as granites, or their compositionally identical fine-grained equivalents, rhyolites, after cooling and solidification. Rhyolites are either intruded near the Earth's surface or erupted. Both granites and rhyolites host In mineralisation (e.g. Andersen et al., 2014) and are known to be significant sources of by-product/companion metals, although In is reported in a limited number of granite-related deposits in our deposit database. Baal Gammon, Australia, is the only instance of a granite-related deposit with full code-based reporting of In resources, although it is likely other granite-related In mineralisation is yet to be identified or fully quantified.

2.2.6. Porphyry deposits

Porphyry deposits are large-tonnage, low- to medium-grade mineral deposits that are genetically linked with porphyry intrusive magmatism (e.g., Kirkham, 1971; Sinclair, 2007; Sillitoe, 2010). They are the world's most important source of Cu as well as being associated with significant Mo, Au, and Ag endowments (e.g. Mudd et al., 2013a,b). As mentioned in the other deposit type classifications above, porphyry deposits are often associated with other mineral deposit types that may be more important in terms of In endowments, such as epithermal, skarn, and also manto mineralization (e.g., Hedenquist et al., 1998; Sinclair, 2007; Sillitoe, 2010). Porphyry deposits are commonly associated with felsic to intermediate arc-type calc-alkaline magmatism (Sillitoe, 1972; Richards, 2003; Sinclair, 2007) and can contain a diverse range of commodities, including Cu, Mo, Au, Ag, Re, PGE, W, Sn, Bi, Zn, In, and Pb (Kirkham and Sinclair, 1995). Typically, porphyry Cu, Cu-Au, and Cu-Mo deposits are not well known for significant In endowments, primarily as these deposits generally contain low concentrations of metals like Zn and Sn that are associated with In enrichments. However, some notable examples of the porphyry-type tungsten–molybdenum subclass of deposits (exemplified by the Mount Pleasant deposit in Canada; Sinclair et al., 2006) are known to contain significant concentrations of In. These In enrichments are most likely associated with a solid solution series between roquesite and sphalerite, a process that generated In-rich sphalerite at Mount Pleasant (Sinclair et al., 2006). This in turn suggests that Zn-rich porphyry deposits may host significant amounts of In that are concentrated in magmato-hydrothermal fluids derived from the silicic intrusions associated with porphyry deposits in a similar fashion to other In-enriched magmatic or magmato-hydrothermal deposits.

2.2.7. Sediment-hosted stratiform Cu

Copper-dominated sediment-hosted stratiform mineral deposits contain thin but extensive zones of disseminated to veinlet stratiform Cu and Cu-Fe sulphide mineralization (Kirkham, 1989; Hitzman et al., 2005). These deposits range in size from giant and supergiant to small deposits (Hitzman et al., 2010), although the former have so far only been discovered in three areas: the Zambian or Katangan Copperbelt, the Kupferschiefer of central Europe, and the Kodaro-Udokan Basin of Siberia (Hitzman et al., 2005, 2010). Generally, these deposits are hosted by siliciclastic or dolomitic sediments and often occur between subaerial and marine sedimentary sequences within sedimentary basins (Hitzman et al., 2005, 2010). The In potential of these deposits remains unclear, as the only currently known instance of a sediment-hosted stratiform Cu, deposit with significant In is the Waterloo deposit in Queensland, Australia (see Supporting information), which contains an estimated 8t of In. This may be because these deposits are dominated by Cu, although Ag and Co are often present and lesser concentrations of Pb, U, Zn, Au, and PGEs have also been reported from these deposits, although commonly in subeconomic quantities (Singer, 1995; Hitzman et al., 2005). The In present within the Waterloo deposit may be a function of the Zn enriched nature of this deposit compared to typical sediment-hosted stratiform Cu deposits, suggesting that this deposit may be a hybrid between sediment-hosted Pb-Zn and sediment-hosted stratiform Cu types of mineralisation, rather than strictly a sediment-hosted stratiform Cu deposit. However, this does not mean that Zn-rich types of sediment-hosted stratiform Cu deposits are not viable targets for In exploration. Further research in this area is certainly warranted.

2.2.8. Tailings and slags

Tailings represent the bulk of the uneconomic or gangue fraction of ore that is discarded during mineral processing and are generated as high volume wastes from milling that often contain sulphide minerals that can be the source of environmental contamination when exposed to oxidation or weathering. In comparison, slags typically form during the cooling of molten solutions of oxides that form as a common by-product/waste of base metal smelting and refining. Although tailings and slags do not occur naturally, we consider these to be relevant classes of anthropogenic mineral deposit types that may be an important future resource of critical metals, including In. The metallurgical processes that produce tailings and slags are optimised for specific conditions at each facility, and hence the physical composition of slags can be highly variable. This composition may additionally change over time due to the exposure of these deposits to the above-ground environment (Lottermoser, 2014). The heavy metal content of tailings and slags is also often a cause for environmental concern, and hence most are considered wastes. If the extraction from these sources leads to net reductions in local pollutant levels and/or the offset of demand from primary production, the extraction of resources held by tailings and slags can be environmentally attractive (Hagelüken, 2014; Shen and Forssberg, 2003). The Zeehan slag resource in Tasmania, Australia (Fig. 2) is one of the very few tailings or slag sites that fully report code-based In resources. Further detail on the history of this site is provided in the Supporting information.

3. Reported indium mineral resources

3.1. Mineral resource accounting

A number of deposits reported to contain In were identified during a comprehensive review of mining company websites, technical reports, published literature, mineral resource atlases and

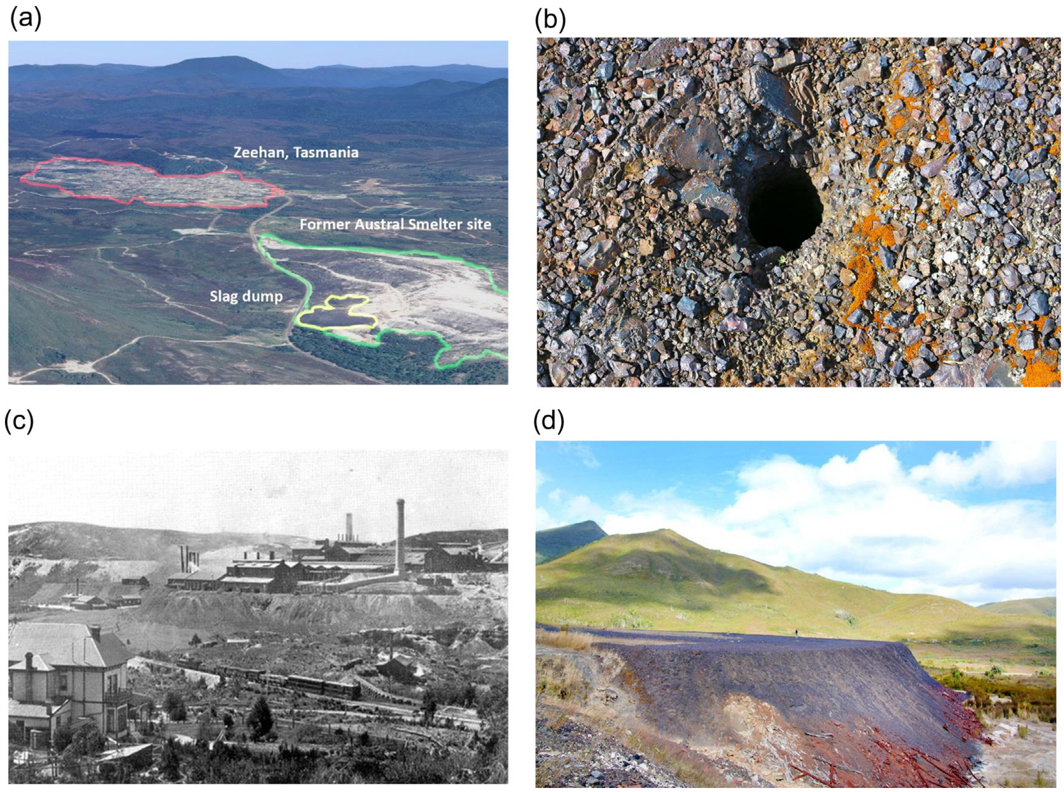

Fig. 2. a) Aerial view of Zeehan township (red), former Austral smelter site (green) and slag dump (yellow). Edited from Google Maps, 2016. b) Drill hole in the slag, reflecting the testing performed to estimate indium contents. c) View of the operating Austral Smelter site ca. 1910. Source: Twelvetrees and Ward, 1910). d) Atop the slag heaps, present da. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

other national/state mineral occurrence databases. The quality of resource data for each deposit varied considerably, meaning that it was necessary to employ a classification scheme to account for variations in data quality. This scheme essentially states that a CRIRSCO code-based classification of the grade and tonnage of a deposit means that the In content of this deposit can be considered with greater confidence than estimates otherwise found in the scientific or engineering literature (e.g. Murakami and Ishihara, 2013). Our scheme is based on (Mudd et al., 2013b; Weng et al., 2015) and is briefly summarised as follows:

- High quality (H): In grade and tonnage is reported and estimated according to CRIRSCO mineral resource reporting codes.
- Medium quality (M): A deposit's tonnage is reported using CRIRSCO mineral resource reporting codes but In grades are not reported but instead are obtained from non-code data sources such as the peer-reviewed scientific or engineering literature. A "medium-calculated", or Mc subtype of this classification is given to deposits that have tonnages reported using codes but with grades calculated using the proxy approach outlined in Werner et al. (2017).
- Low quality (L): Both In grade and ore tonnage are reported in peer-reviewed literature or other non-code study rather than using CRIRSCO mineral resource reporting. Deposits with tonnages reported without using codes and with grades calculated using the proxy approach outlined in Werner et al. (2017) are given a classification of "low-calculated", or Lc.

Of course, not all deposits that contain In have In concentrations or grades measured or reported in the literature, meaning that we have split our assessment into distinct reported and inferred databases. The reported database is presented here and the inferred database is presented in Section 6, and the compilation and analysis methods used to interpret these databases are given in Part II of this study (Werner et al., 2017).

3.2. Reported indium resource database

101 mineral deposits were identified with In reported as a commodity or potential commodity and had sufficient associated data to quantify the amount of contained In within these deposits. The fact that these deposits have reported In grades and tonnages means that they arguably represent the portion of the extractable global resource (EGR) that is most likely to be exploited in the near future (i.e. are likely to meet short term demands). The full list of these deposits along with their deposit type classifications and reported or calculated In contents are given in Table 1, with grade and tonnage data shown in Fig. 3. Data sources and notes on individual deposits are provided in the Supporting information. These data indicate that some ~9470 t of contained In are present in high quality resources, with a further ~12,912 t contained In within medium quality resources and a further ~52,066 t In present within low quality resources. These resources yield a minimum of 76,183 t of contained In, an amount that if In demand continues its linear growth as per the last 20 years (see Werner et al., 2015), would meet global demand to the year 2060.

We examined the way in which the reported In quantities are distributed between individual deposits, countries and deposit types. The majority of the In within our reported database are present in relatively few large deposits (approx. two-thirds of resources, or 48,442 t In, in the top 10 reported deposits). The largest deposit in our database is the Gaiskoye (Gai) deposit in Russia, which contains over 9000 t In. This is more than the bottom 77 deposits within our reported database combined, primarily as a result of the large size of this deposit. The major In deposits within

Table 1
The 101 deposits known to contain In that were quantifiable via the methodology outlined in Part II of this study (Werner et al., 2017) as a result of the availability of reported deposit tonnage and indium grades, or the grade of base metals from which indium grades could be inferred (for deposits rated Mc and Lc).

Country	Mine site name	Tonnage Mt	Grade g/t In	Contained t In	% Zn	% Sn	g/t Ag	g/t Au	% Pb	% Cu	Primary deposit type	Company[1]	Report Quality
Argentina	Pingüino	10.58	11.5	**122**			62.9	0.38	0.62		Epithermal	Argentex Mining Corp	H
Australia	Avebury Mine –Zeehan (Slag)	0.42	48	**20**	13.6		55		1.5		Tailings/ Slag	Intec	H
Australia	Baal Gammon	2.8	38	**106**		0.2	40			1.0	Granite-Related	Monto Minerals/Slow Peak Mining (and formerly, Kagara)	H
Australia	Conrad–King Conrad	3.1	5.7	**18**	0.6	0.2	95.4		1.3	0.2	Skarn	Malachite Resources	H
Bolivia	Malku Khota	485	5	**2431**	0.05		23.7		0.1	0.0	Sed-Pb-Zn	TriMetals Mining Corp (formerly Sth. Amer. Silver Corp)	H
Canada	Mount Pleasant (North Zone)	18.5	67.4	**1246**	0.8	0.3				0.1	Porphyry	Adex	H
Canada	Silver Range – Keg (Main Zone)	39.8	5.8	**229**	0.8	0.03	30.3		0.3	0.2	Skarn	Silver Range	H
Germany	Geyer Southwest, Saxon Ore Mountains	12.6	35	**439**	0.6	0.5					Skarn	Deutche Rohstoff AG	H
Germany	Tellerhäuser	32.2	71	**2286**	0.7	0.4					Skarn	Saxore Bergbau	H
Namibia	Namib Lead-Zinc Project	0.9	29	**27**	5.7		44.8		2.4		Sed-Pb-Zn	North River Resources	H
Peru	Ayawilca	13.3	68	**909**	5.9		14		0.2		Skarn	Tinka Resources	H
USA	West Desert (formerly Crypto)	71.1	23	**1636**	1.9					0.2	Skarn	InZinc Mining (formerly Lithic Resources)	H
Argentina	Pirquitas	33.6	2	**67**	1.5		172.1				Epithermal	Silver Standard	M
Australia	Balcooma – Polymetallic/Zinc	2.3	2.5	**6**	5.5		29.4	0.3	2	1.2	VMS	Kagara	M
Australia	Balcooma – Copper	1.2	2.5	**3**	0.9		15.6	0.1	0.4	2.7	VMS	Kagara	M
Australia	Broken Hill (Main)	33.5	50	**1675**	8.1		75.6		6.3		Sed-Pb-Zn	Perilya	M
Australia	Broken Hill (Rasp)	19.7	50	**984**	6.5		85.1		5		Sed-Pb-Zn	Toho Zinc	M
Australia	Dry River South	0.7	5	**4**	6.9		62.1	0.6	2.5	0.9	VMS	Kagara	M
Australia	Mt Chalmers	3.6	10	**36**			8	0.8		1.2	VMS	Echo Resources	M
Australia	Nightflower	0.2	47.9	**10**	2.2		193.6		4.9	0.2	VMS	Axiom Mining Ltd.	Mc
Australia	Rosebery incl South Hercules	24.8	10	**248**	10.6		121.3	1.7	5.1	0.4	VMS	MMG	M
Australia	Salt Creek	0.5	62.2	**33**	7		52	0.3		2	VMS	Venturex/Venturex Resources	Mc
Australia	Waterloo	0.8	10	**8**	10.2		48.1	1.2	1.6	1.8	Sediment-hosted stratiform Cu	Kagara	M
Bolivia	Bolivar	15.4	150	**2311**	1.1		19.5			0.8	Epithermal	Sierra Metals	M
Bolivia	Porco	2.4	52	**122**	9.2		91		0.8		Epithermal	Glencore-Xstrata	M
Bolivia	Pulacayo (Paya)	20.7	0.3	**6**	1.4		104.3		0.3		Epithermal	Comibol/Prophecy Development Corp.	Mc
Bolivia	San Vicente	6.6	1	**7**	2.6		342		0.3		Epithermal	Pan American Silver Corp.	M
Canada	Akie	68.2	0.4	**24**	5.2		9		1		Sed-Pb-Zn	Canada Zinc Metals	Mc
Canada	Brunswick 12-Bathurst	1.5	48.8	**74**	7.6		92		3	0.3	VMS	Xstrata	M
Canada	East Kemptville (Main and Baby Zones)	80.4	15.4	**1236**	0.1	0.1				0.1	Skarn	Avalon Rare Metals, Inc.	M
Canada	Horne No. 5	67.6	0.05	**3**	0.7			1.8		0.2	VMS	Falco Resources	Mc
Canada	Keno Hill – Onek	0.8	74.9	**61**	13.1		196.6	0.6	1.2		Epithermal	Alexco	Mc
Canada	Kidd Creek	38.0	50	**1900**	4.8		55.3			1.9	VMS	Xstrata	M
China	Qinghai Deerni	20.6	0.09	**2**	0.4					1.2	VMS	Zijin Mining	Mc
Czech Republic	Cinovec	28.1	1	**28**		0.4					Granite-Related	European Metals	M
Kosovo	Drazhnje	4.7	4.9	**23**	4.9		45		2.4			Lydian International	Mc
Papua New Guinea	Eastern Manus Basin – Solwara 1	2.6	57	**146**	0.7		29.6	5.8		7.7	VMS	Nautilus Minerals – PNG Gov	M
Papua New Guinea	Eastern Manus Basin – Solwara 12	0.2	57	**13**	3.6		56	3.6		7.3	VMS	Nautilus Minerals – PNG Gov	M
Peru	Cerro de Pasco	203.1	1.2	**244**	2.6		100.2		1.1	0.2	Epithermal	Volcan Compania Minera	M
Peru	Morococha	15	2.1	**32**	4.6		186.5		1.4	0.5	Epithermal	Pan American Silver Corp.	M
Peru	Santander Project	31.1	0.5	**15**	3.3		22.7		0.5	0.1	Skarn	Trevali	Mc
Portugal	Lagoa Salgada	8.4	1.5	**12**	2.6		52.9	0.8	2.7	0.3	VMS	Portex Minerals Incorporated	Mc
Portugal	Neves Corvo	193.3	18	**3480**	3.8		53		0.9	1.2	VMS	Lundin Mining	M
United Kingdom	South Crofty, East Pool and Agar Mines	3.5	0.4	**1**	0.3	0.3				0.5	Skarn	Celeste Copper	Mc
USA	Bingham Canyon/ Kennecott Copper Mine	859	0.1	**86**			1.6	0.1		0.2	Porphyry	Rio Tinto	M
USA	Santa Rita	356	0.1	**36**				0.02		0.5	Porphyry		M
Australia	Isabel	0.05	140	**7**							Granite-Related		L
Azerbaijan	Filizchay/Filizchai	95	16.9	**1605**	3.63		44.2		1.43	0.6	Sed-Pb-Zn		Lc
Bolivia	Carguaicollo	10	15	**150**							Epithermal	Private Interest	L
Bolivia	Colquiri	17.7	37	**484**							Granite-Related	Comibol	L
Bolivia	Huari Huari	3	1867	**5601**							Sed-Pb-Zn		L

Table 1 (*continued*)

Country	Mine site name	Tonnage Mt	Grade g/t In	Contained t In	% Zn	% Sn	g/t Ag	g/t Au	% Pb	% Cu	Primary deposit type	Company[1]	Report Quality
Bolivia	Potosi	140	29	**4030**							Epithermal		L
Bulgaria	Elatsite/Elacite	100	0.1	**10**							Porphyry		L
Canada	Geco/Manitouwadge	58.4	50	**2920**	3.5		50		0.2	1.9	VMS		L
Canada	Heath Steele	21.4	49.1	**1050**	10.5		40		2.2	0.5	VMS	Noranda – Falconbridge – Xstrata (Current)	L
Canada	Silver Queen (Cole Lake)	0.8	2.5	**2**	17.7		449.1	1.7	6.7		Epithermal	New Nadina Exploration	L
Canada	Silver Queen (Wrinch)	0.6	2.5	**1**	5.5		191.3	3.5	0.8	0.3	Epithermal	New Nadina Exploration	L
Canada	Sullivan-Kimberley	12.8	50	**640**	12		128.7		11.2		Sed-Pb-Zn		L
China	Dachang	75	117	**8775**							Skarn		L
China	Dulong	28	183	**5124**							Skarn		L
Czech Republic	Tisova	3.5	24.4	**87**			13.5	0.2		1	VMS	Canadian International Minerals	L
Georgia	Dambludi	1.87	26	**49**	5.31		30.1	1.9	2.7	0.8	Epithermal		Lc
Germany	Freiberg	4.7	1	**5**							Skarn		L
Germany	Freiberg (Tailings)[2]			**40**							Tailings/ Slag		L
Germany	Rammelsberg	27.2	25	**680**							Sed-Pb-Zn		L
Germany	Pöhla-Globenstein[2]			**1470**							Skarn		L
Greece	St. Philippe/Agios Philippos Mine/Kirki Mine	1.5	7.5	**11**							Epithermal		L
India	Tosham	1	2	**2**							Granite-Related	Hindustan Zinc Limited	L
Ireland	Lisheen	3.8	15.4	**58**	12.6				2.1		MVT	Vedanta Limited	Lc
Japan	Akenobe (6 locations)	17.5	50	**875**							Epithermal		L
Japan	Ashio	24.8	49	**1240**							Epithermal		L
Japan	Ikuno	17.1	64	**1094**							Epithermal		L
Japan	Kosaka	17.8	2.5	**45**							VMS		L
Japan	Omodani	0.4	5	**2**							Skarn		L
Japan	Taishu[2]			**100**							Epithermal		L
Japan	Toyoha (8 locations)	33.7	138	**4651**							Epithermal		L
Japan	Uchinotai	8	4	**32**							VMS		L
Mid-Atlantic Ridge	Broken Spur	"0.1–0.3"	1.9	**0.2**							VMS		L
Mid-Atlantic Ridge	Snake Pit (23°N)	"0.1–0.3"	29	**3**							VMS		L
Mid-Atlantic Ridge	TAG (26°N)	3.8	1.3	**5**							VMS		L
Namibia	Tsumeb (Slag Resource)	2.9	170	**493**	9				2.1		Tailings/ Slag	ZincOx Resources	L
Northeast Pacific	Axial Seamount @ Juan de Fuca Ridge	"<0.003"	7	**0.02**							VMS		L
Russia	Bakr-Tau	1.3	6	**8**	4.7			1.5	0.7	2.6	VMS		L
Russia	Degtyarsk/Degtyarskoye	130.0	0.05	**7**	1.5				0.1	1	VMS	Polymetal International Plc	Lc
Russia	Gaiskoye/Gaiskoe/Gai/Gay	380	24	**9120**	0.7		6.3	0.9	0.1	1.6	VMS	Ural Mining and Metallurgical Company (UGMK)	L
Russia	Komsomolskoye	25	2	**50**	1.8				0.2	1.6	VMS		L
Russia	Letneye/Letnye	6	1	**6**	1.2		13.7		0.6	2.8	VMS	UGMK/State Government (Venture)	L
Russia	Podolskoye/Podolskoe	80.8	6	**485**	1.1				0.1	1.7	VMS	UGMK	L
Russia	Sibaiskoye/Sibai/Sibay	100	10	**1000**	1.6				0.04	1	VMS		L
Russia	Uzelga	69.0	0.2	**13**	2.9		35.0	1.8	0.0	1.4	VMS		Lc
South Africa	Letaba CZ, Murchison Belt	1	60	**60**	9.0		33		0.04	1.5	VMS	Maranda Mining Co.	L
South Africa	Maranda J Mine, Murchison Belt	1.1	320	**352**	23.0		30		0.03	3.1	VMS		L
South Africa	Mashawa, Murchison Belt	0.05	16	**1**	12.1		25		0.03	2.1	VMS		L
South Africa	Mon Desir, Murchison Belt	0.03	9	**0.3**	27.0		5			0.4	VMS		L
South Africa	Romotshidi, Murchison Belt	0.5	70	**35**	23.6		73		0.9	1.5	VMS		L
South Africa	Solomons, Murchison Belt	0.02	24	**0.5**	0.4		5.6		0.01	3.6	VMS		L
South Korea	Ulsan	9	10	**90**							Skarn		L
Southwest Pacific	Southern Lau Basin @ Tonga Subduction Zone	2.5	40	**100**							VMS		L
Sweden	Långban	1.3	1	**1**							Skarn		L
United Kingdom	West Shropshire Orefield (England)	1.0	1	**1**							Sed-Pb-Zn		L
USA	Kingman	4.1	269.2	**1109**	13.2		236.5	9.6	12.0	0.7	Porphyry	ARS Mining	L
			Total:	**76,183**									

[1] Where a company is not listed in this table, it signals that a company or owner of the deposit listed was unknown to the authors at the time the database was compiled.
[2] For the Taishu, Freiberg (Tailings) and Pöhla-Globenstein locations, specific grades and tonnages were unknown, as In resources were reported only as a total quantity in the literature reviewed.

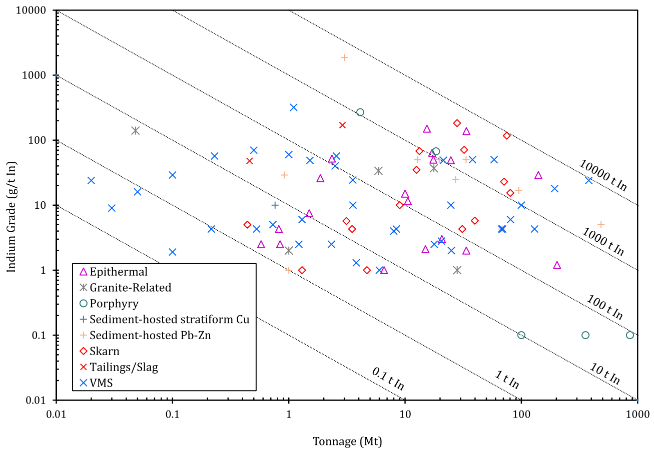

Fig. 3. Grade vs. tonnage for 101 reported indium deposits, classified according to deposit type.

our database are also located within a relatively small number of countries (Fig. 4a), with around 75% of the reported In resources in our database residing in Bolivia, China, Russia, Canada and Japan

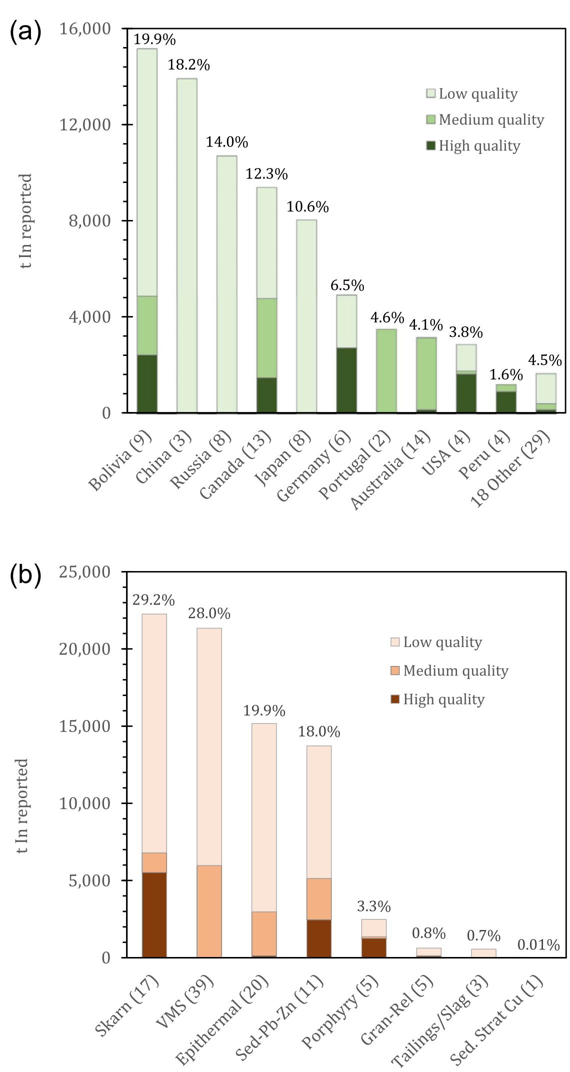

Fig. 4. Apportionment of 101 reported indium deposits according to quality of reporting by (a) country and (b) deposit type. Numbers in brackets indicate the number of reported deposits in each category.

alone. This broadly reflects the state of the current In supply market (which is mostly dominated by production from China, South Korea, Canada and Japan; see later Fig. 8), although the geology of these currently producing countries is not so unique that they form the only potential sources of In. Indeed, some of the countries with major In resources in our reported database (notably Bolivia, Portugal and Australia) do not currently produce refined In, despite exporting In-laden Zn concentrates to countries such as South Korea and Japan (DoE, 2011). Such exporting but non-refining countries need only to develop domestic refining capacity in order to change the global distribution of refined In supply. This appears to be happening in Australia, as the production of refined In from local Zn concentrates is expected to soon commence following upgrades to the Risdon facility near Hobart, Tasmania (Nyrstar, 2014; Fig. 5). It is unclear whether this will affect the volume of In-laden Zn concentrates available to other countries. However, it can at least be concluded that the distribution of resources presented here may be useful to inform the distribution of future In supply, and hence can be used in more detailed future assessments of In criticality.

In terms of deposit types, ~95% of reported resources reside in skarn, VMS, epithermal, and sediment-hosted Pb-Zn deposits, respectively (Fig. 4b), with the remainder mostly present in porphyry and granite-related deposits, or reported mine wastes. With the bulk of reported In present in relatively few deposit types, it would seem that a limited number of process configurations need to be outlined to enhance global In production. VMS deposits contain a large amount of In within our database, although all of these deposits have only medium or low classifications, meaning that there is less certainty that these VMS deposits actually represent the most economically viable source of In. Skarn deposits host the greatest amount of In in deposits with high quality resource reporting, suggesting that they currently represent the most economically viable processing pathway for In, although research into In mineralogy and processing is ongoing (e.g. Lopez et al., 2015; Pavlova et al., 2015).

Changes in the quality of reporting (as per Section 3.1) of In resources between countries is also evident and is shown by the shading in Fig. 4a. The highest quality reporting of In is within deposits in Bolivia, Canada, Australia, the USA and Peru, although less than one third of our reported database is associated with deposits classified using CRIRSCO mineral resource reporting codes and with code-compliant In grades. Some 12% of In resources within our database are from high quality data sources, with a further 16.9% from medium quality sources and the remaining majority (68.3%) from low quality reporting, a clear reflection of In's perception as a lower value by-product (Mudd et al., 2016).

An additional 203 deposits within our database are known to contain In but do not have quantified In grades or tonnages; these deposits are listed in the Supporting information alongside data sources and geological classifications where known. Some 45 of these deposits have reported In grades but no reported total resources, primarily as these deposits have no recently reported tonnage information. Deposits with CRIRSCO compliant tonnages and Pb and Zn grades that are known to contain In but without publicly reported In grades had In grades estimated using the proxy methods outlined in Part II of this study (Werner et al., 2017). These deposits were assigned a ratio of ~465 g In/t Zn, reflecting the weighted and scaled mean ratio between In and Zn determined by Werner et al. (2017). This yielded some, ~191 t In classified as Mc and ~1732 t In as Lc in the reported database, with a further case for the Eagle deposit in Canada having an identified resource tonnage but with insufficient base metal grade data to enable the calculation of an inferred In grade using this proxy approach.

Fig. 5. Nyrstar's Risdon facility near Hobart, Tasmania, Australia. The likely location of Australia's first refined indium production.

Sixteen other Japanese deposits listed by Ishihara et al. (2006) were reported to contain "less than 500 t In", but could not be more accurately quantified; as such, these deposits are not included within our database but we note that they could technically contribute up to 8000 t more In, highlighting that our reported database most certainly represents a minimum estimate of known global In resources.

4. Inferred indium within Pb-Zn and Cu deposits

4.1. Unreported indium resources

Not all In-bearing mineral deposits have In contents that are reported in the technical literature or within mineral occurrence databases. This meant that separate databases of deposits hosting minerals likely to contain In were also necessary to consider. Here, we used a recently compiled database of global Pb-Zn resources containing 851 deposits with quantified Pb-Zn resources (see Mudd et al., 2017 for the full database); some 51 of these deposits have known In concentrations that are already accounted for in our reported database, and hence are not discussed further. The remaining 800 deposits with unknown In concentrations had In resources assigned to them using the estimation proxy approach outlined in Part II (Werner et al., 2017). We justify this estimation approach by the fact that these deposits almost certainly contain some In as a result of the presence of significant (i.e. potentially economic) amounts of sphalerite, which is the source of most (>95%) of In currently extracted (Schwarz-Schampera, 2014). Code-based tonnages are reported for 518 deposits, tailings or stockpiles in this database (resulting in a "medium-calculated" quality estimate with proxies applied), and the remaining 282 have non-code tonnages (resulting in a "low-calculated" quality estimate with proxies applied). The amount of In present in these deposits was estimated using weighted and scaled means involving nineteen different estimates (more details of this approach are given in Werner et al., 2017), corresponding to an average of 5.27 ppm In within Pb-Zn deposits and a ratio of 465.3 g In/t Zn within each deposit. Only 775 deposits in this database contained reported quantities of Zn, meaning that 25 deposits were assumed to not contain any In. The remaining deposits have grades and tonnages shown in Fig. 6.

Another recently compiled database of global Cu resources containing 730 deposits with quantified Cu resources (Mudd et al., 2013a), 611 of which were not present within the other two databases, was also analysed during this study. These deposits likely contain In as they contain significant amounts of chalcopyrite, although the In within these deposits is almost certainly present in lower concentrations than in the Pb-Zn deposits described above (Cook et al., 2011). Here, we use a simpler approach to estimate In quantities within these Cu resources given that less than 5% of In supply is derived as a by-product of Cu and Sn processing combined and research linking Cu and In grades is limited. We apply a single proxy ratio of 10 g In/t Cu as per Schwarz-Schampera (2014), resulting in an average grade of 0.05 ppm In for these deposits. Notably, this is just below the crustal average for In, suggesting that this estimate is highly conservative. In the absence of any other published estimates for Cu deposits, we have not applied any other values. A plot of In grade vs. tonnage for these deposits is also given in Fig. 7, ultimately reflecting the distribution of Cu

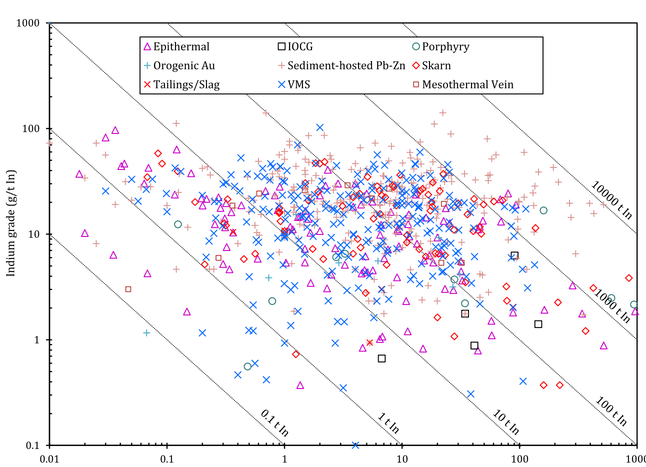

Fig. 6. Inferred indium grade vs. tonnage for 775 Pb-Zn deposits, classified according to deposit type.

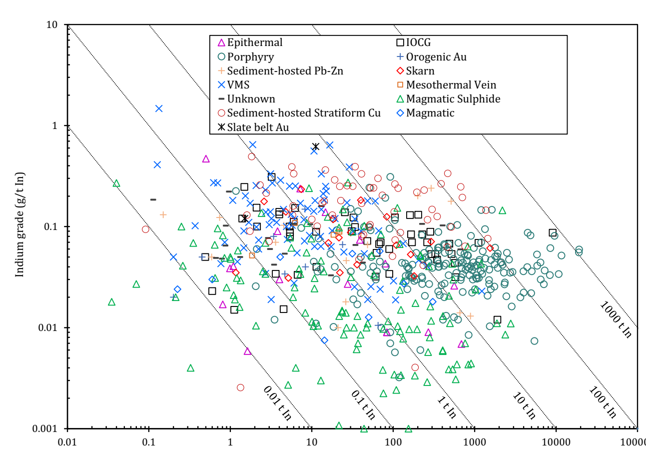

Fig. 7. Inferred indium grade vs. tonnage for 611 Cu deposits, classified according to deposit type.

grades within these deposits. It is therefore similar to Fig. 2 in Mudd et al. (2013a).

5. A summary of global indium resources and their distribution

5.1. Combining each database

The first database of 101 deposits identified during this study have In contents that are explicitly reported in one or more literature sources and yield some 76 kt of contained In. Although less than one third of this is hosted by medium and high quality resources (see Fig. 2), this value represents the most detailed assessment of known In-bearing deposits to date. The secondary databases of 800 Pb-Zn and 611 Cu deposits that have inferred In contents as they contain sphalerite are estimated to contain ∼263 kt In and ∼17 kt In, respectively. This results in a total of 1,1512 known deposits containing some 356 kt of In, representing the current best estimate of the world's In resources in known mineral deposits. Fig. 8 shows how these quantities are apportioned between countries within each database and compares our data to the quantities and distribution of In supply for the year 2015 (see USGS, 2016). Fig. 9 shows a map of the location of the majority of these deposits globally, organised by database. These resources, if entirely turned into reserves and production, are sufficient to meet continuing growth in consumption well into the next century, particularly as significant unreported volumes of In are probably present within known global Pb-Zn deposits. However, enabling this business as usual growth in In consumption will depend on a number of social, technical and geopolitical factors that are distinct from the sheer quantities of In that are available and are already currently known. Additionally, the apportionment of these resources according to deposit type and source database is shown in Fig. 10a, along with a more detailed country breakdown in Fig. 10b. This highlights the potential contributions made by different deposit types beyond what is indicated from the reported database alone, as shown in Fig. 4 (b).

The high criticality of In is strongly influenced by its supply distribution, although Fig. 8 indicates that any restriction of exports from countries like China could be counteracted by other non-producing countries like Bolivia and Australia, who are currently well positioned to enter the market and mitigate the impacts of a supply restriction. This, however, would require investment into In refining capacity in these countries. We have also shown the distribution of known In refining capacity (as per European Commission, 2012) for the year 2010 in Fig. 8. At the time of writing, we are aware of some efforts to establish new In refining capacity at Nyrstar's Risdon facility in Tasmania, Australia (Nyrstar, 2014), which is not shown in Fig. 8.

5.2. Other aggregated estimates

Some resource estimates are not applicable to individual deposits but instead have been compiled for certain regions and here are termed 'aggregate' estimates. It is not possible to directly disaggregate these estimates and apportion these resources to individual deposits in our database, but we include these studies to ensure the completeness of our research. The first notable example is from Zheng (2011), who estimated Chinese In reserves, rather than resources, to be ∼12,000 t In in 2011. This is important as China does not report In resources by public or code-based reporting, meaning that Chinese In resources are almost certainly under-represented within our database, especially as China currently dominates global In supply. Our reported resource database contains 13,901 t In within Chinese resources, suggesting that the available data yield a significant underestimation as, in theory, the volume of resources would be considerably larger than that of the reserves reported by Zheng (2011), particularly if the unreported resources are classified to be inclusive of reserves. It is generally difficult to determine the extent to which these resources have been underestimated, however the resources as reported for our database in Table 1 are 4.35 times larger than reserves on average, so from this we may roughly approximate Chinese resources to be in the order of ∼52 kt In. Alternatively, we could consider that China's total Zn resources are in the order of 115.7 Mt Zn (as reported in Min et al., 2012), and apply a ratio of ∼465 g In/t Zn as per Werner et al. (2017), yielding a similar estimate of ∼53.8 kt In.

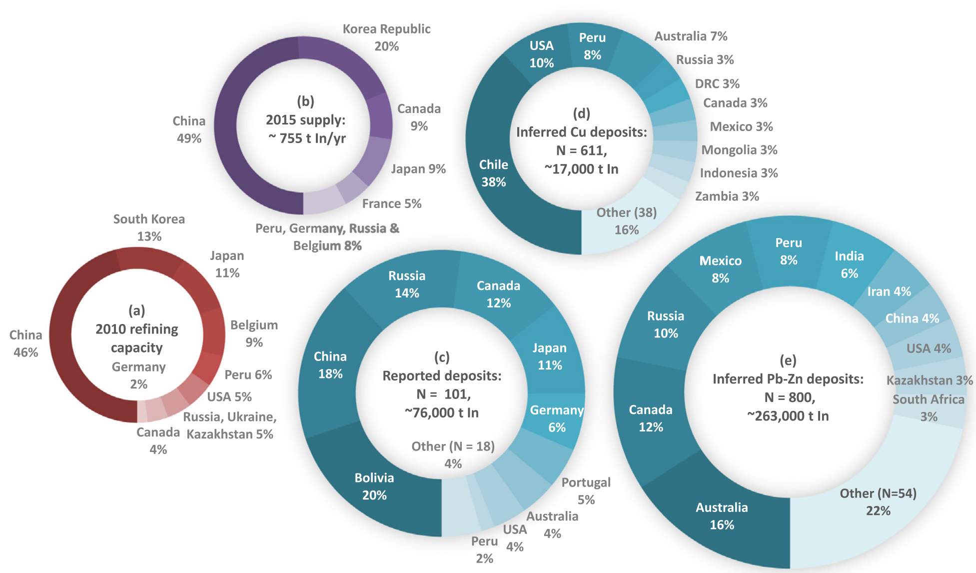

Fig. 8. Country breakdown of (a) refining capacity for indium in 2010 Source: European Commission, 2012), (b) the supply of refined indium in 2015 (Source: USGS, 2016), (c) indium in deposits explicitly reported for their indium content, (d) indium in deposits with inferred content due to their Cu mineralogy, and (e) indium in deposits with inferred content due to their Zn mineralogy, with totals indicate.

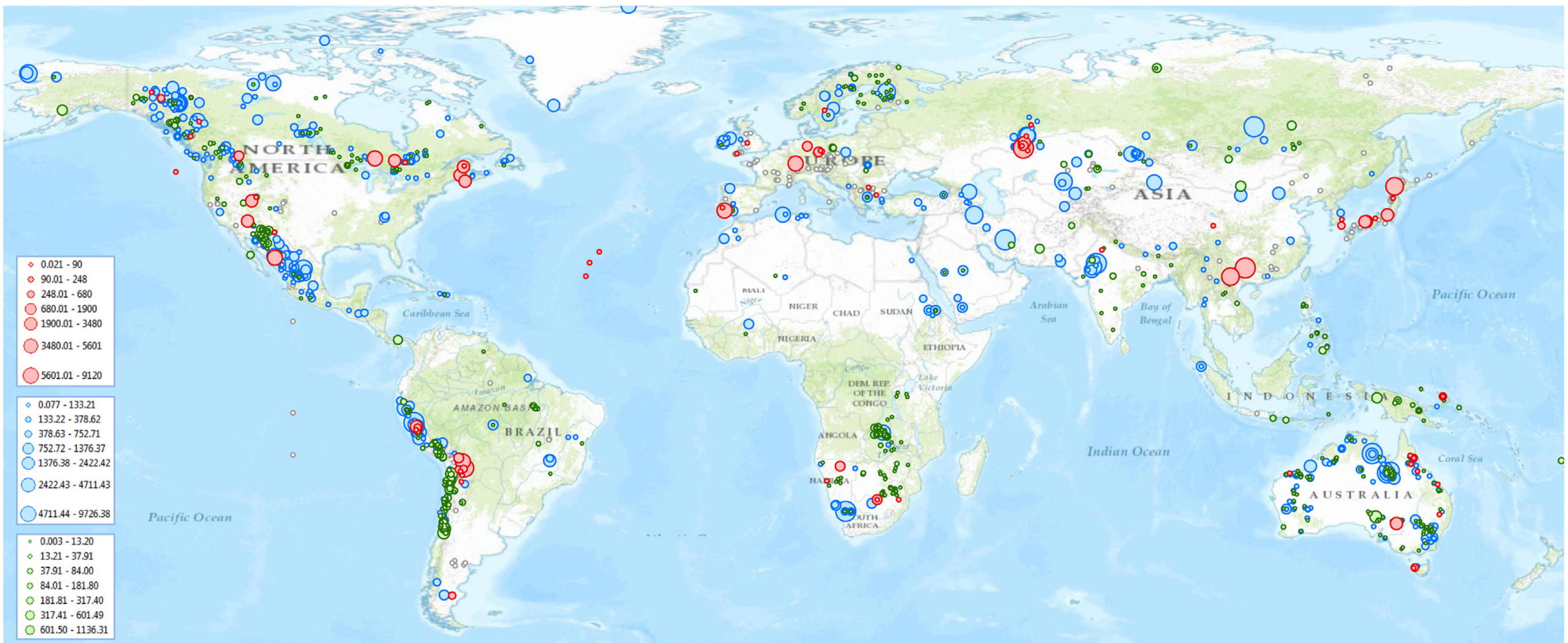

Fig. 9. Global map of indium in reported and quantified (red, N = 101), reported but unquantified (white, N = 219), and inferred Pb-Zn (blue, N = 591) and Cu (green, N = 576) deposits. Quantities depicted in the legend are in t In. Only deposits whose coordinates could be determined at the time of compilation are shown. Location data for deposits are sources from the Australian Mines Atlas, Mindat.org and otherwise estimated from site descriptions or individual site technical reported and company websites. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

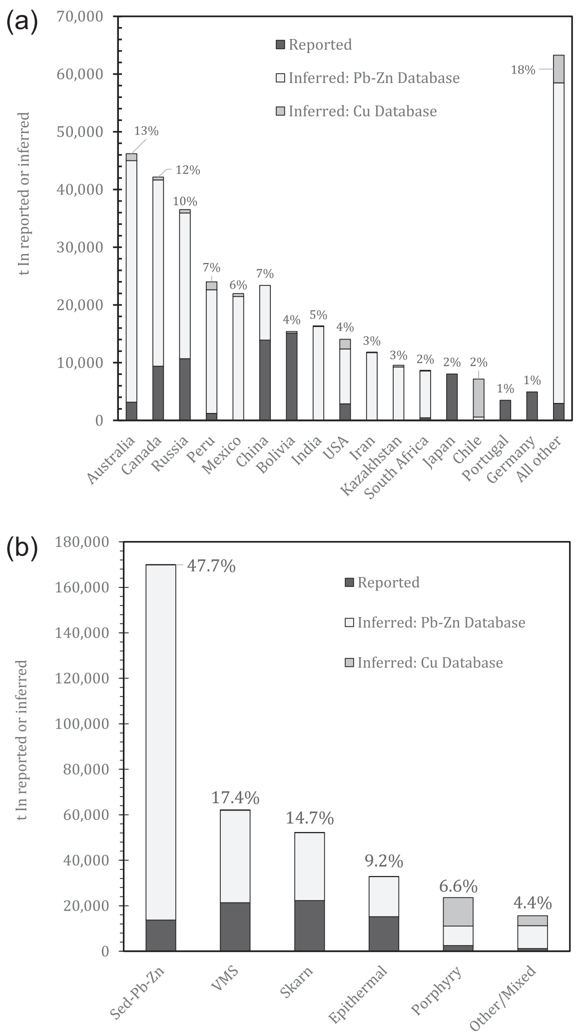

Fig. 10. Distribution of reported and inferred indium resources by (a) major deposit type classification and (b) country, indicating contributions from different deposit databases. Numbers in brackets indicate the number of deposits.

Another aggregate estimate is provided by the EU ProMine database, which was used to identify In as a commodity in multiple European deposits (see INSPIRE, 2011). A summary report for this database indicates that 630 t In resources (i.e., potentially economic for future extraction) are currently present in European deposits, although we cannot identify the individual deposits hosting these quantities. The four deposits within our database that are also listed in the EU ProMine database are reported to contain 8 tonnes of In, suggesting that other European deposits may contain a further 622 t In.

Previous research (e.g. Werner et al., 2015) has also indicated that the processing of In throughout the 20th century has meant that the amount of In in mine wastes could be in the same order of magnitude as that of unexploited mineralisation, although the former may be significantly more difficult to process than the latter (see Mudd et al., 2016). This means that a final aggregate estimate of at least 20,000 t In is present in production wastes derived from Zn processing between 1972 and 2012, in addition to the 3900 t In present in the tailings associated with two Canadian mines (Heath Steele and Brunswick 6–12; Werner et al., 2015). UNEP (2010) otherwise note that there are virtually no estimates of the stocks of any metals in tailings at national or global scales; this is discussed further in Section 6.3. The addition of these aggregate quantities, including 622 t In for the EU ProMine database, leads to a total of ~378,498 t contained In within currently known resources.

5.3. Economic viability and future processing

During this study, we conducted a review of cases where both mineralised material and concentrate In grades were published (see Supporting information) to further our understanding of the economic viability of the In resources identified. These data indicate that In is on average enriched by a factor of 7 between ore and concentrates, although this is dependent on the processing route. This enabled a rough estimation of the concentrate grades of the deposits in our database which, when combined with location data for these deposits, indicates the location of deposits with higher concentrate grades that are more likely to contribute to future In supply (Fig. 11). This is supported by the inclusions of layers indicating the location of known Zn refineries and smelters and countries reported to produce refined In (as per Tolcin, 2014). We further examine the concepts of economic viability through a breakdown of deposit monetary value in Section 6.4.

6. Discussion

6.1. Interpretation of results

The minimum of 76 kt reported contained In and 263 kt contained inferred In in known mineral deposits quantitatively indicates that the In present in known deposits is sufficient to meet the long term demand for this critical metal. More specifically, current consumption of approximately 800 t In/yr may increase to >4000 t In/yr by the beginning of the next century, conservatively assuming a linear growth trend as observed in the last 20 years (see Werner et al., 2015). Assuming no peaks in production and no contribution to global In supply from recycling indicates that deposits with known In contents (76 kt In) could meet this increase in demand to the year 2060. Furthermore, including inferred In resources (281 kt In from the Pb-Zn and Cu databases) means that this increase in demand could be met well into the next century. Of course, the quantities presented in our database must be economically extractable and then processed to a refined product for such an evaluation to be true, and processing would entail inevitable losses of In along the way. Nonetheless, we may conclude that sheer resource quantities will not be a major limitation to the supply of In to the market in the short to medium term. This somewhat contrasts with the results of modelling by Sverdrup and Ragnarsdóttir (2014), who estimated an In production peak in 2020–2045, although they used a very modest resource estimate of approximately 58 kt In.

Hotspots for reported In resources are located in Bolivia, Russia, China, Canada and Japan (Fig. 8). The fact that these deposits have been identified as having elevated In concentrations also indicates that they are worthy of additional study, given their potential role in meeting In demand over the coming decades. Hotspots for In-bearing Pb-Zn deposits where In is not reported are also present in Peru, Canada, Australia, Kazakhstan and Mexico. In addition, the fact that this study only assessed deposits primarily noted for their Zn and Cu mineralogy means that there are likely other deposits (notably Sn deposits) that could also contain significant amounts of In that could be inferred using the methodologies outlined in Werner et al. (2017). These deposits are, however, less likely to form a part of the In supply chain unless drastic pricing changes were observed, and/or advancements in the processing of In from Sn production pipelines take place. Changes in pricing may be triggered by imposed export restrictions from major producing countries such as China, changes in stockpiling behaviour,

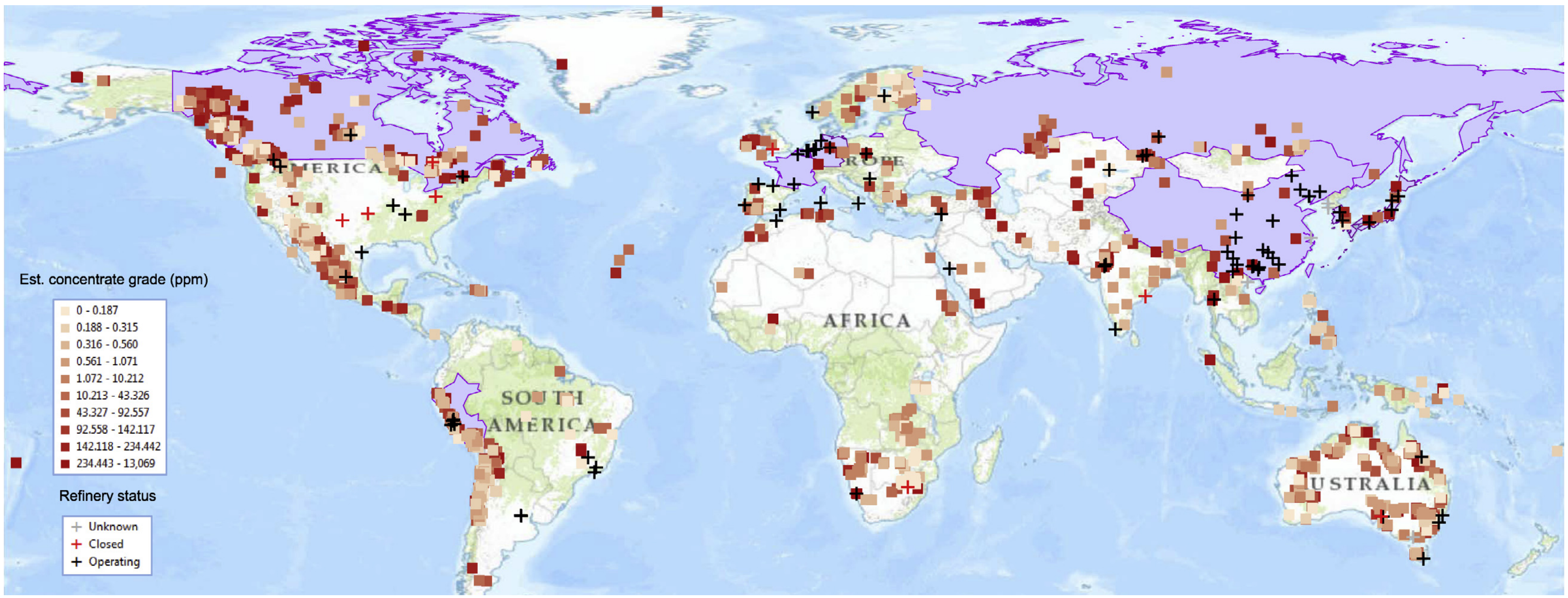

Fig. 11. Estimated concentrate grades of 1268 deposits from all deposit databases used in this study with reported or inferred indium grades and identified location data. Indicates the possibility for currently non-producing deposits to enter the market for indium in future. Countries currently producing refined indium are outlined in purple, with the location of known Zn smelters and refineries also indicated. Location data for deposits are sourced from the Australian Mines Atlas, Mindat.org and otherwise estimated from site descriptions or individual site technical reported and company websites. Zinc refinery status and locations obtained from the SNL Mining and Metal Database. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

broader trends in demand for In-containing products and the technological development of substitutes, which have historically contributed to high volatility in In prices (USGS, 2016).

In terms of deposit types, the distribution of resources in the reported database suggested VMS, skarn, sediment-hosted Pb-Zn and epithermal deposit types to be the major hosts in relatively equal terms. However, it was shown in Fig. 10 that the sediment-hosted Pb-Zn is the by far the dominant deposit type, host to roughly the equivalent of VMS, skarn, epithermal and porphyry deposit types combined when inferred In resources are included. This is primarily due to the amount of Zn in sediment-hosted Pb-Zn deposits in the Pb-Zn deposit database of Mudd et al. (2017), and the introduction of porphyry deposits as a major contributor to In resources comes as a direct result of the amount of Cu reported in porphyry deposits.

The identification of an additional 219 deposits with known In enrichments but with total In contents that could not be quantified further highlights that our findings are not representative of the true quantities present in known deposits but instead represent a minimum value. The effect of these additional deposits on resource estimates is difficult to determine, as although they are numerous, their lack of reporting suggests they have lesser economic value than more well characterised deposits. Paradoxically, some of these deposits have very high reported grades, including the Laochang and Qibaoshan deposits in China that have In grades of 179 and 253 ppm, respectively (see Schwarz-Schampera and Herzig, 2002 and Ye et al., 2011). However, these deposits have unknown resource tonnages that most likely reflect a lack of reporting rather than a lack of In resources within these deposits. The majority of these reported but unquantified deposits were identified in Europe, with lower numbers of deposits in China, the United States and Japan (Fig. 9). This is largely a result of the EU ProMine database, which was useful for cataloguing the presence of In in many locations throughout Europe, however quality resource data were not available for these deposits, and not reported within the database itself. Further, many represent abandoned sites that are unlikely to be extracted.

Our study has made use of a number of data sources of varying degrees of quality in order to build a cohesive view of the state of global In resources. This task was hindered by the mostly limited reporting of In in economic Pb-Zn, Cu and Sn deposits (Fig. 3), which is reflective of the state of reporting of many other critical metals (Mudd et al., 2016). Our results suggest that increased reporting of critical metals would assist in carrying out similar studies in future, providing significant value not only within the field of industrial ecology and in the development of future critical metals policy but also to mining companies who may consider targeting these critical metals.

6.2. Indium deportment and supply potential

While a comprehensive assessment of In supply potential from the deposits in our database is outside the scope of this paper, it is possible to make some inferences. Notably, the comparisons in Fig. 8 indicate strong deviations between the distribution of In resources and the current In supply chain, indicating the capacity to adapt to meet any challenges posed by In supply restrictions via conventional pipelines. This is an important finding in terms of future ratings of In criticality, as the development of new In processing capabilities in countries like Australia and Bolivia could seemingly address any deficits in future In production. However, the effect of increasing production in currently non-producing countries may be dampened as the Zn concentrates exported from these countries already have an important role in the In supply chain. For example, the US Government noted in 2011 that an increase in Japanese and South Korean importation of Zn concentrates from Australia could increase their own refined In production (DoE, 2011).

To learn more about the economic recovery of deposits in countries not currently producing refined In, it is necessary to consider the deportment of In to various mineral concentrates. Unfortunately however, information on the metallurgical processing of In is scarce, as this information is often proprietary. Deportment can also be highly variable, as In may concentrate in other minerals such as chalcopyrite (e.g. in Southwest England, Andersen et al., 2016), meaning it would likely avoid the typical Zn processing route. The few studies on deportment which can be found (see Werner et al., 2015 and Table S1 in the Supporting information), suggest that some 5–35% of milled In typically appears in Zn concentrates. Taking Australia as an example, we can look at the historical production of Zn in Australia's mining sector, apply assumptions on average In grade as per Werner et al. (2017) and see that Australia may have exported some 18.5 kt In in Zn concentrates since 1889. Around half of this is likely to have been sent to processing facilities that have no In capability, meaning it most likely ended up in slags, and the other half possibly making it to production (albeit with downstream processing losses, see Werner et al., 2016).

The likelihood of currently unprocessed concentrates from the operating deposits in our database entering the In supply chain is dependent on numerous economic factors, including the processing capabilities of existing pipelines and the grade of In in these concentrates. We have attempted to characterise these factors in Figs. 9–11, which we believe to be a useful reference for future studies into In provenance. These maps highlight numerous locations outside of currently producing countries containing deposits with elevated calculated In grades as well as existing Zn processing capacity. Hotspots for Zn processing capacity are broadly in keeping with the distribution of global refined Zn production (see OCE, 2014), and locations coloured deeper red in Fig. 11 represent areas with an increased likelihood of In production from concentrates being economic in the event of In price increases. This map was produced using the grades obtained from the "Weighted Mean" estimate presented in Part II of this study (Werner et al., 2017). Green (2012) indicates that concentrates containing at least 300 ppm In are necessary for In to be economically extractable under current market conditions, although it has previously been noted that some concentrates are still processed to recover In at grades of around 100 ppm In (Phipps et al., 2008). This suggests that, assuming an average enrichment factor of 7 between deposit and concentrate grades (see Supplementary Table S1), many of the concentrates derived from deposits in our database are already economically viable, as shown in Fig. 11, which highlights that concentrates already derived from countries like Australia, the United Kingdom and South Africa may be targets for further exploitation.

The current dominant barrier to by-product extraction within Pb-Zn pipelines is that these by-product metals are often outside the source mining company's core business (Willis et al., 2012). This means that the quantification of the adaptability embedded in the In supply chain at a company level (also referred to as 'capacity readiness', Leal-Ayala et al., 2014) requires the identification and cataloguing of In capable facilities that are controlled by companies who also own non In producing facilities. One example of this is the Auby processing facility in France which is operated by *Nyrstar*, who in turn operate processing facilities in Australia which are not currently capable of separating In. Given their plans to establish In refining capacity in Australia (Nyrstar, 2014), it would appear they have confidence in continued growth in the In market. A full analysis of In processing capabilities at the company level globally is outside the scope of this report, however this is expected be the subject of future work by the authors of this paper. Indium refining capacity appears to have been studied at some level by the International Lead-Zinc Study Group, although their reports are not publicly available.

6.3. Mine wastes as a potential augment to indium supply

Previous research has highlighted how the processing of In has led to significant wastage throughout the 20th century. Despite improvements in downstream manufacturing, this largely continues today at the extraction phase, meaning that enormous quantities of In may have accumulated in mine wastes across the globe (Werner et al., 2015). The order of magnitude of mine wastes can be simply inferred by examining historical grade and production data and by combining these data with knowledge of the apportionment of In during the early stages of ore processing. Here, we use Broken Hill, Australia, as a case study for a large mine containing In in a country that does not currently produce In. Annual grade and production data for 1883–2011 indicate that over 10 kt In was milled during this period (data updated from Mudd, 2010). Some 5%–35% may have accumulated in tailings (Fig. 12, see also Supplementary Table S1 for specific references citing In deportment), and our database suggests that the main orebody at Broken Hill currently contains 1675 t In, which is analogous to the quantities calculated for the tailings currently residing at Broken Hill. Extrapolating this calculation to the rest of our deposit database indicates that global tailings resources may rival the contained In within unexploited but known mineral deposits. However, the marginal costs of extraction from these mining wastes are likely to prohibit the economic viability of some of these resources, although the sheer quantities of In (and almost certainly a wide variety of other metals) that are present in these tailings and wastes means that they are certainly worth further investigation. Further data and figures on Broken Hill are provided in the Supporting information.

6.4. Market considerations and estimation of indium-related value within mineral deposits

Indium has to date been exclusively extracted as a by-product, a fact that is reflected in the proportional contribution of In to the resource value of individual deposits (Mudd et al., 2016, 2017). Here, we demonstrate this by plotting normal distributions of the contributions to a deposit's monetary value using 2013 prices (from OCE, 2014; Tolcin, 2014) for deposits with known In contents that are reported with other base metal grades (Fig. 13). It can be seen in Fig. 13 that the median proportional resource value

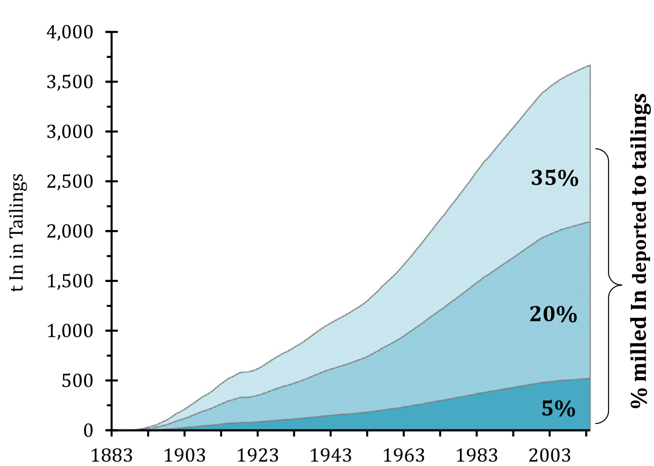

Fig. 12. Potential In accumulated in tailings at Broken Hill, Australia, 1883–2014. Assumes a range of 5–35% of milled In deporting to tailings.

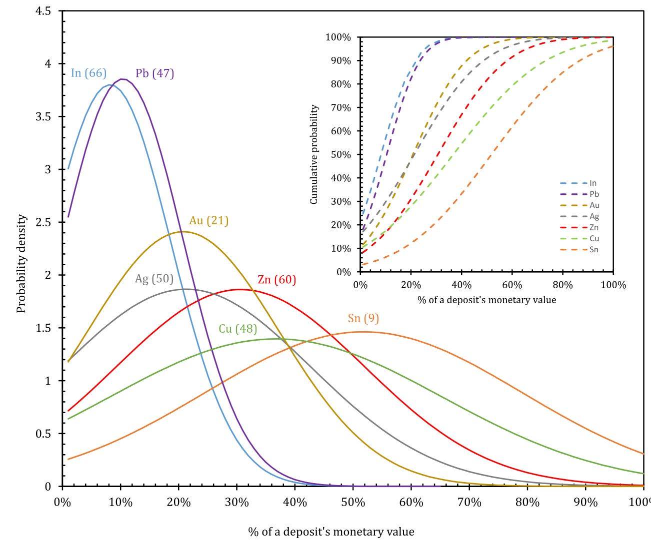

Fig. 13. Distribution of monetary value from 66 deposits reporting indium content and other base metal grades. Numbers in brackets indicate the number of deposits reporting the commodity from which the normal distribution is derived. Insert: Cumulative probability plot, indicating that a metal contributes less than x % of a deposit's resource value, with y % probability).

for In is ~10%. The fact that these curves have been developed from reported deposits that theoretically represent deposits with the greatest potential for profitability from In extraction means that these curves may well represent the upper bounds of proportional In resource value. This is supported by Fig. 13, as the In in these deposits can be seen to contribute a similar proportional resource value to Pb. However, given that many of the deposits in the reported database are not likely to be a part of the In supply chain, this also highlights that the quantity and price of a commodity alone are not enough to reflect the potential profitability of its extraction, when all costs are considered.

A similar distribution of monetary value was produced for the deposits with inferred In content in the Pb-Zn database (see Supporting information). The median resource value of In in these deposits was calculated at ~6%, which verifies the proxy estimates employed in Pt. II (Werner et al., 2016), as it would be expected that the median In value in these deposits is below 10%, given that they have not been reported. Metals commonly have more volatile prices than other commodities, with In being amongst the most volatile (de Groot et al., 2012). However, the demand for In is quite inelastic to fluctuations in the In price, primarily as the costs of In are likely to be very small in comparison to that of other metals present in its primary end uses (O'Neill, 2010). The increased recovery of previously wasted In during fabrication and manufacturing stages led to price drops and stabilised primary production in the last decade, although this has not dampened projections for this decade (e.g. a 14.1% price increase for 2010–20, de Groot et al., 2012). Green (2009) assessed grades and tonnages from a select number of In-bearing deposits and concluded that resources are sufficient to sustain upwards of 1 kt In/year primary production, although there are a limited number of deposits which could provide In as a primary commodity, and then only for a few years. While we have not directly assessed the economics of extracting In from each of the deposits in our database, we have identified deposits with sufficient quantities to supply national or even global markets for many years, suggesting that resource quantities are not likely to be a limiting factor in In becoming a primary commodity. Changes in demand will still be speculative and are dependent on factors such as dematerialisation strategies, changes in technologies such as PV modules (Woodhouse et al., 2012), and by extension the substitutability of In (see Graedel et al., 2015). Further research into the nature and adaptability of the global In market is necessary to understand specific pricing thresholds and the ability of the global resources identified in this study to translate to supply. Ideally, this research would also further explore the potential oversupply of Zn necessary to meet future In demand (as per Elshkaki and Graedel, 2015).

6.5. Uncertainties and other avenues for further research

Although our database is reasonably comprehensive in representing In-bearing deposits, there are undoubtedly some uncertainties relating to the compilation of the reported deposit database and the specific data sources employed for this study. The uncertainties involved in the general methodology employed here are discussed extensively in Part II of this study, including uncertainties associated with the use of proxies for inferring In grades (Werner et al., 2017). Here, we focus on uncertainties and avenues for future work which relate specifically to the data and sources employed for our assessment of In.

Our development of the reported resource database (Table 1 and Supporting information) highlighted that many deposits containing reported In concentrations were no longer operating. This created challenges in terms of understanding the quantities involved in economic terms and determining which values should actually be included within our database. For example, the Toyoha deposit in Japan produced In for many years and the closure of this site strongly suggests that the In at this location is no longer economic; however, the tonnage and grade estimates utilised in this paper were published on the year of its closure (see Ishihara et al., 2006), suggesting that there were still quantities of In remaining at that time. As such, we have included this deposit in our database on the grounds that known quantities in known deposits should be included in a global metal resource assessment, although this undoubtedly introduces some uncertainty. In addition, more detail on the remaining resources associated with individual deposits could be achieved using yearly processing data for the deposits listed in our database to determine how much In had been extracted at a given point in time. This would then generate specific values that would reflect any changes that occurred since the publication of a resource estimate. This process was previously undertaken for the Heath Steele and Brunswick 6–12 deposits in Canada, primarily as these deposits have abundant data relating to In production (Werner et al., 2015), and for Broken Hill during this study, although it is clearly beyond the scope of this paper to undertake the same analysis for the remaining 98 deposits, most of which are, in any case, unlikely to have the necessary data to perform yearly production analyses. It should also be noted that any risk of overestimation from these analyses is undoubtedly counteracted by the fact we have not included Sn deposits within our analysis and the fact that our In estimates provide minimum rather than maximum values, as described above. Including Sn deposits within global In resource estimates would require the compilation of a global database of sn-bearing deposits and the analyses of these deposits to identify and characterise the elevated In grades observed in some Sn mineralisation (Pavlova et al., 2015), a task that is well beyond the scope of this paper.

Another source of uncertainty arises from the seemingly high sensitivity of our results to the work of the relatively few authors who work in this field, such as Shunso Ishihara and Ulrich Schwarz-Schampera, both of whom have contributed extensively to our knowledge of In deposits and resources. Studies such as Murakami and Ishihara (2013), Schwarz-Schampera and Herzig (2002), and Schwarz-Schampera (2014) have provided useful summaries of the state of prior geological research and enabled many 'medium' and 'low' deposits in our reported database to be identified and quantified, which otherwise might not have contributed to our global resource estimates. The input of these sources is another reflection of the state of In reporting, as other studies employing per-deposit analyses of global resources, even for other by-product metals such as cobalt, rely less on academic literature, and more on industry reporting (see Mudd et al., 2013b).

End-of-life wastes, such as consumer electronics which contain In and have become e-waste, have also not been addressed in this study despite much discussion of their potential to augment In supply. For example, Duan et al. (2015) highlight urban mining as a promising solution to meeting China's demand for In, highlighting that China has become a net importer of refined In in recent years despite their status as the dominant exporter of In and their considerable resources, as identified in this study. Improvements in In recovery from end of life wastes are to be expected over time, but approximately 80% of In end uses remain unrecyclable or are dissipated (Ciacci et al., 2015). This suggests that recycling alone will not be enough to meet growing demand, a sentiment echoed by Bloodworth (2014). In addition, Yamasue et al. (2009) found In to be the only case in which the total material inputs required to produce primary In were actually lower than that of In sourced from end-of-life recycling. This suggests that primary In deposits may play a more vital role in the future supply of this metal than other critical metals of interest for their recycling potential. The seemingly conflicting conclusions on the future sourcing of In highlight the conversion of virgin materials or wastes to supply as an avenue for continued research.

7. Conclusions

This study presents a compilation and analysis of 1512 mineral deposits that host In-associated mineralisation, using the best available information to estimate the quantities of In in each of these deposits. This indicates that at least 356 kt (or 380 kt, including regional estimates) of In is present in known mineral deposits and a further minimum of 24 kt In in mining waste projects, which is sufficient to meet demand for In well into the next century, even without consideration of the potential for supply from secondary sources. While China is by far the dominant producer of refined In at present, significant quantities of In exist outside of the current suite of producing countries. This suggests that in the event of a supply restriction, In capacity could be developed elsewhere, although there are numerous challenges in converting dormant resources into supply. Australia and Bolivia notably contain large In resources and have not historically produced refined In, although they may already supply In-laden Zn concentrates to other countries for refined In production. The monetary value of reported and inferred In contents across our databases were also analysed, suggesting that In may contribute ~6% and ~10% of a deposit's resource value in deposits where it is unreported and reported, respectively.

Global In resources are dominantly hosted by sediment-hosted Pb-Zn deposits, which contain the equivalent of VMS, skarn, epithermal and porphyry deposits combined in our deposit databases, indicating a clear relationship between the base metals concentrated in these deposit types (e.g. Zn and to a lesser extent Cu) and In abundances. However, the majority of these quantities are inferred, not reported, highlighting the need for better reporting of by-products/critical metals. Overall, the In grades of many deposits appear sufficient to justify continued improvements in In processing efficiency. Our study also indicates that robustly compiled and assessed resource estimates such as the one presented here should be the preferred basis of policy development and mineral exploration programs. It is hoped that our results may rest questions of 'how much is left', and prompt a shift towards greater focus on the impacts of extraction from different sources. With detailed knowledge of the size and spread of In resources in known deposits, steps can be made towards strategic and sustainable sourcing of In as required for this century's transition to renewable energy, increased use of digital display technologies, and/or other future applications of In.

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Appendix A. Supplementary data

Supplementary data associated with this article can be found, in the online version, at http://dx.doi.org/10.1016/j.oregeorev.2017.01.015.

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6. A MATERIAL FLOW ANALYSIS OF THE MINING, TRADE AND USE OF INDIUM IN AUSTRALIA

In the previous chapter, it was established that Australia hosts more resources of indium than any other country, but does not possess indium refining capacity. This results in a more concentrated global supply chain for indium, which has contributed to its higher criticality ratings. To actually quantify Australia's role in the global supply chain of indium, potentially identify avenues for intervention, and thereby address research questions 2a, 3a and 3b, this chapter presents a retrospective dynamic model of indium stocks and flows in Australia. This model depicts the mining of copper and zinc in Australia back to 1844, and the trade of semi-finished and finished goods containing indium from 1988-2015. This specifically includes estimations of the amount of indium exported and imported in the form of copper and zinc concentrates and all consumer products containing indium, e.g. LCD displays, mobile phones, solar panels, solders and research samples. The quantities of indium which have accumulated in mine wastes, processing wastes, in-use and in landfill are also estimated. Supplementary information for this chapter is provided in Appendix C.

6.1 General approach

A national scale material flow analysis involves the characterisation of a number of stocks and flows along the life cycle of a metal. In Figure 6.1, a generalised life cycle for indium in Australia is shown. The stocks (shown in yellow) represent locations of interest for their future resource value. The flows (grey arrows) are of interest because they determine the rates of stock accumulation. The nodes shown in blue are processing stages, which ultimately affect the direction and magnitude of the flows. It is the aim of this study to characterise, quantify and analyse each of these components. This is done in the following sections on a stage-by-stage basis, beginning first with mineral resources and mining, followed by a discussion on the learnings of the cycle as whole.

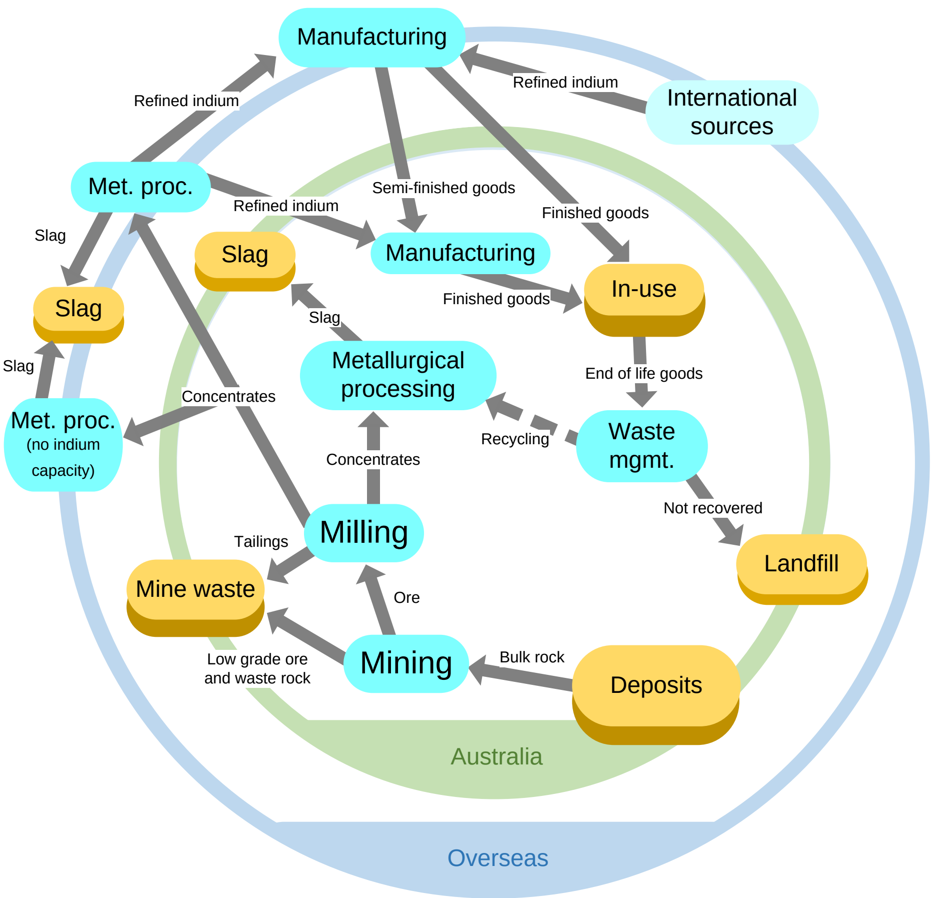

Figure 6.1: Australian and international stocks, flows and processes assessed in this study. Nodes occurring within Australia are placed on the green band and nodes occurring outside of Australia are placed on the blue band.

6.2 Assessing the resources and mining of indium in Australia

6.2.1 Australian mineral resources of indium

The starting point of characterising any metal cycle is to analyse the extraction of that metal from stocks in mineral deposits. As per Chapters 4 and 5, indium mineral resources in Australia were determined as a sum total of estimated contents of indium in individual deposits. For those deposits where contained indium was not explicitly reported by mining companies or in the literature reviewed, indium grades were inferred using the proxy method developed as per (Werner et al., 2017a).

It can be seen in Table 6.1 that approximately 46,000 tonnes of indium are present within 247 Australian mineral deposits. This is an estimate of the current mineral resources of indium, despite well over a century of mining. Of the 247 deposits identified here, indium is only actually reported as an extractable commodity in 14 of them, consistent with what we would expect of a critical metal by-product. These 14 deposits contain ~3,156 t In. Given the strong mineralogical relationships between indium and zinc, it is found that the bulk of resources are present where zinc is reported as the primary commodity, at ~41,846 t In. While there are 109 deposits listed with copper as the main commodity, these are only estimated to contain an additional ~1,207 t In. It should be noted that this list is not exhaustive, as deposits with tin as a primary commodity have not been included, unless specifically reported (as is the case with Baal Gammon, Queensland and King Conrad. NSW). From a list of 37 additional deposits with tin as the primary commodity, it is estimated that ~1,462 t In are present, using the correlation shown in Fig. 2 of Werner et al. (2017a, Chapter 4). These deposits are provided in Appendix C, but were not identified when the remainder of the indium cycle was modelled. This is an area of uncertainty, however it is unlikely to significantly affect the estimates of the remaining cycle, as over 95% of global indium production is derived as a by-product of Zn, as is discussed in Chapter 3.

Overall, the quantities estimated for Australian indium resources are significant, as only about 800 t In are consumed annually per year (Werner et al., 2015). These results alone suggest that Australia could continue to play a role in the global indium supply chain for many years. However, more work is needed to report the quantities of indium in Australian mineral deposits, as only three deposits fully report indium using CRIRSCO mineral resource reporting codes. The bulk of estimates presented in Table 6.1 rate as “Mc” in terms of certainty (see Chapter 4), meaning that codes were used to estimate resource tonnage, but without a reported grade of indium. Additional deposits where indium was reported as a potential commodity, but whose contents could not be quantified are provided in Appendix C.

Table 6.1: Indium deposits in Australia, indicating the present day endowment for purposes of mass balance, adapted from (Werner et al., 2017a), (Werner et al., 2017b) and (Mudd et al., 2017b)

Mine Site Name	Total Mt	ppm In	Total In	%Zn	ppm Ag	ppm Au	%Pb	%Cu	Deposit Type	Company (where known)	Rating	Sources/Notes
Deposits with reported In content:												
Baal Gammon	2.80	38	106		40.0			1.00	Gran-Rel	Monto Minerals/Slow Peak Mining (and formerly, Kagara)	H	Monto Minerals Website 18/03/2014. Mineralisation from McKinnoon & Seidel (88-93), cited in Schwarz-Schampera (2001)
Austral Smelter - Zeehan (Slag)	0.42	48	20	13.6	55		1.50		Tailings/Slag	Intec	H	Australian Mines Atlas 2013-09-30 - Data directly from Intec Annual Report (2007). Zheng et al. (2014) as a reference for possible mineralogy.
Conrad - King Conrad	3.13	5.7	18	0.56	95.4		1.26	0.18	Skarn	Malachite Resources	H	Annual Report 2009 (Last report with resource estimates and no extraction taken place since). Deposit type & Mineralisation from Gore et al. (2007) - "Post-rehabilitation environmental hazard of Cu, Zn, As andPb at the derelict Conrad Mine, eastern Australia".
Isabel	0.048	140	7						Gran-Rel		L	Australian Mines Atlas - Commodity Search. Listed in Qld Industry Publication. Quantities not listed as JORC compliant, but dated to 2014. Described as a pre-JORC resource that, on its own, will "never be mineable" by QLD Gov't (pers. comm.)
Broken Hill (Main)	33.5	50	1675	8.13	75.6		6.25		Sed-Pb-Zn	Perilya	M	Tonnage from Perilya 2012 Annual Report. In grade from Schwarz-Schampera & Herzig (2002)
Broken Hill (Rasp)	19.67	50	984	6.51	85.1		5.03		Sed-Pb-Zn	Toho Zinc	M	In grade from Schwarz-Schampera & Herzig (2002). Tonnage from Australian Mines Atlas - Based on CBH Resource Report '09. Toho Zinc not publishing in Annual Reports.
Rosebery incl South Hercules	24.8	10	248	10.6	121.3	1.69	5.10	0.37	VMS	MMG	M	Tonnage from MMG Annual Report 2013. In grade from Schwarz-Schampera & Herzig (2002) citing Huston et al. (1995)
Mt Chalmers	3.56	10	36		8.0	0.80		1.20	VMS	Echo Resources	M	In grade and mineralogy from Schwarz-Schampera & Herzig (2002). Tonnage from Echo Resources 2010 Annual Report
Waterloo	0.76	10	8	10.2	48.1	1.16	1.63	1.78	Sed. Strat Cu	Kagara	M	Australian Mines Atlas + Kagara 2011 Annual Report. Also listed in Schwarz-Schampera (2002, 2014) - citing Huston et al. (1995)
Balcooma - Polymetallic/Zinc	2.32	2.5	6	5.51	29.4	0.31	2.02	1.24	VMS	Kagara	M	Australian Mines Atlas + Kagara 2011 Annual Report. In grade from Schwarz-Schampera & Herzig (2002)
Dry River South	0.73	5	4	6.89	62.1	0.64	2.51	0.95	VMS	Kagara	M	In grade from Schwarz-Schampera & Herzig (2002) - citing Huston et al. (1995). Tonnage from Australian Mines Atlas + Kagara 2011 Annual Report
Balcooma -Copper	1.22	2.5	3	0.86	15.6	0.10	0.36	2.74	VMS	Kagara	M	Tonnage and grade from Australian Mines Atlas + Kagara 2011 Annual Report
Salt Creek	0.53	61.687	33	7	52	0.30		2.0	VMS	Venturex / Venturex Resources	Mc	*Indium grade calculated via Proxies.*Tonnage from Venturex. Annual Report 2013. High Grades listed, but not with code data. See: http://www.venturexresources.com/investorrelations/Released/Spectacular%20Zinc-Lead-Silver%20Grades%20at%20Salt%20Creek,%20Pilbara%20VMS%20Project%20-%2030-6-10.pdf
Nightflower	0.22	47.494	10	2.20	193.6	0.00	4.91	0.15	VMS	Axiom Mining Ltd.	Mc	*Indium grade calculated via Proxies.*Tonnage from Australian Mines Atlas - citing "2008 ASX (Australian Stock Exchange) Announcement"
Pb-Zn Deposits with inferred In content[1]:												
Bali Hi	0.095	0.0	0.0			0.20	2.08		Epithermal	Artemis Resources (?)	Mc	Minedex (2013-05-05)
Bowdens	88.0	1.8	159.7	0.39	47.4		0.29		Epithermal	Kingsgate Consolidated	Mc	Ann Rep 2013
Carboona	0.035	6.4	0.2	1.37			6.2		Epithermal	unknown	Mc	Aust Mines Atlas (2013-10-01)
Mt Clement-Eastern Hills	0.607	0.0	0.0		26	0.22	2.4		Epithermal	Artemis Resources	Mc	Ann Rep 2009
Nimbus	4.876	6.2	30.1	1.33	79.3	0.29	0.15	0.02	Epithermal	MacPhersons Resources	Mc	Tech Rep (2013-11)
Northampton-Mary Springs	0.145	0.0	0.0				11.46		Epithermal	Prospect Resources (formerly Ethan Minerals)	Mc	Aust Mines Atlas (2013-09-30)
Peterlumbo-Paris	5.9	0.0	0.0		110		0.6		Epithermal	Investigator Resources	Mc	Media (2013-10-15)
Range & Turtle/Copper Ridge	0.131	0.0	0.0		4.6		1.9	1.4	Epithermal	Onslow Minerals	Mc	Minedex (2013-05-05)
Webbs	1.490	7.3	10.8	1.56	245		0.71	0.27	Epithermal	Silver Mines	Mc	Ann Rep 2013
Merlin-Little Wizard	6.715	0.7	4.5	0.14	8.3	0.08	0.02	0.34	IOCG	Inova Res. (formerly Ivanhoe Aust.)	Mc	Tech Rep (2010-10)
Mt Dore	144.3	1.4	202.7	0.30	5.9	0.10	0.05	0.52	IOCG	Inova Res. (formerly Ivanhoe Aust.)	Mc	Tech Rep (2010-10)

Area 55	12.2	0.0	0.0				0.56	0.49	Magmatic sulfide	Hunnan Australia Resources	Mc	Ann Rep 2007 (Compass Res.)
Browns Reef	20.5	9.3	190.8	2	9		1.1	0.1	Orogenic Au	Comet Resources	Mc	Ann Rep 2007
Grants Creek-Wilsons Reef	0.067	1.2	0.1	0.25	16	4.97	0.73		Orogenic Au	Firestrike Resources	Mc	Minedex (2013-05-05)
Trilogy	6.24	5.6	34.8	1.2	47.0	0.8	2.0	1.0	Orogenic Au	Silver Lake Resources	Mc	Ann Rep 2012/2013
Wagga Tank	1.25	15.4	19.2	3.3	68.8	0.66	1.76	0.81	Orogenic Au	MMG51%, Golden Cross Resources49%	Mc	Ann Rep 2001 (GCR)
Tally Ho	0.733	3.9	2.8	0.83	49	0.06	0.9	0.1	Orogenic Au?	Alcyone Resources	Mc	Ann Rep 2008
Kangiara	2.75	6.0	16.6	1.3	24	0.5	1.0	0.18	Porphyry	Paradigm Metals	Mc	Ann Rep 2012
Mt Angelo North	0.124	12.4	1.5	2.7	25	0.20	0.54	0.31	Porphyry	3D Resources	Mc	Minedex (2013-05-05)
Admiral Bay	96.7	11.2	1079.9	2.4	15		2.9		Sedi-Pb-Zn	Kagara	Mc	Ann Rep 2011
Altia	5.8	2.3	13.5	0.5	40		4.0		Sedi-Pb-Zn	Breakaway Resources (now Minotaur Exploration)	Mc	Ann Rep 2012
Angas	0.91	19.5	17.8	4.2	31.1	0.5	1.7	0.22	Sedi-Pb-Zn	Terramin Australia	Mc	Ann Rep 2013
Bulman	1.0	51.2	51.2	11.0			6.5		Sedi-Pb-Zn	Admiralty Resources	Mc	NT DME Fact Sheet 2013-09
Cannington	96.3	13.9	1333.8	2.98	164		4.87		Sedi-Pb-Zn	BHP Billiton	Mc	Ann Rep 2013
Century-Century East	17.6	6.9	121.6	1.5	37		10.1		Sedi-Pb-Zn	MMG	Mc	Ann Rep 2013
Coxco	7.8	19.5	152.4	4.2			1.0		Sedi-Pb-Zn	Admiralty Resources	Mc	NT DME Fact Sheet 2013-09
Dugald River	63	56.3	3545.5	12.1	32		1.8		Sedi-Pb-Zn	MMG	Mc	Ann Rep 2013
Ediacara	17	0.0	0.0		10		1.2		Sedi-Pb-Zn	unknown	Mc	SA DMITRE Website (2014-12-15)
Explorer 108	11.868	15.1	178.9	3.24	11.14	0.3	2.00	0.1	Sedi-Pb-Zn	Westgold Resources (now Metals-X)	Mc	Ann Rep 2013
Flinders Group	0.694	140.5	97.5	30.2			1.4		Sedi-Pb-Zn	Perilya Mines	Mc	Ann Rep 2011
Hera	2.444	17.7	43.2	3.8	16.7	4.1	2.8	0.2	Sedi-Pb-Zn	YTC Resources	Mc	Ann Rep 2013
Jackson-Stella-Chloe Trend	4.7	22.4	105.2	4.8	48.1		2.0	0.2	Sedi-Pb-Zn	Kagara (?)	Mc	Ann Rep 2011 (Copper Strike)
Kamarga-JB	10.4	12.6	130.7	2.7	1		0.2		Sedi-Pb-Zn	RMG	Mc	Ann Rep 2013
Lady Loretta	14.27	74.8	1067.0	16.1	97		5.7		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Lennard Shelf Group	14.99	21.8	326.8	4.69	16		4.28		Sedi-Pb-Zn	North-West Mining & Geology Group (formerly Meridian Minerals)	Mc	Minedex (2013-05-05)
Magellan	51.1	0.0	0.0				4.3		Sedi-Pb-Zn	Ivernia	Mc	Ann Rep 2013
McArthur River	194	42.8	8310.2	9.2	41.0		4.0		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Menninnie Dam	7.7	14.4	111.1	3.1	27		2.6		Sedi-Pb-Zn	Terramin Australia	Mc	Ann Rep 2013
Mt Isa (Open Cut)	427.8	15.7	6713.7	3.4	51.7		2.5		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Mt Isa-Black Star	24.4	21.4	521.9	4.60	61.2		3.11		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Mt Isa-George Fisher North	145.1	38.8	5633.8	8.3	62		3.9		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Mt Isa-George Fisher South (Hilton)	56.0	36.4	2038.1	7.8	106		5.3		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Mt Isa-Handle Bar Hill	8.27	26.5	219.3	5.7	36		2.4		Sedi-Pb-Zn	Glencore Xstrata	Mc	ResV ResC 2013
Myrtle	43.6	19.0	829.7	4.09			0.95		Sedi-Pb-Zn	Rox Resources	Mc	Ann Rep 2013
Nymagee	8.096	3.3	26.4	0.70	9		0.3	1.20	Sedi-Pb-Zn	YTC Resources	Mc	Ann Rep 2013
Oceana	2.145	20.5	44.0	4.41	32.7		1.22		Sedi-Pb-Zn	Zeehan Zinc / Creat Resources Holdings	Mc	Media (2009-03-29)
Pegmont	8.852	6.9	60.9	1.48			3.46		Sedi-Pb-Zn	Pegmont Mines	Mc	Ann Rep 2013
Sandy Creek	24.381	8.4	205.3	1.81	4.57		0.45		Sedi-Pb-Zn	TNG Ltd	Mc	Ann Rep 2012
Silver King	1.6	23.7	38.0	5.1	157		13.9		Sedi-Pb-Zn	MMG	Mc	Ann Rep 2012
Sorby Hills	16.7	3.3	54.4	0.7	52		4.5		Sedi-Pb-Zn	KBL Mining	Mc	Ann Rep 2013

Sunshine	0.288	13.0	3.7	2.8	31		1.5		Sedi-Pb-Zn	Stonehenge Metals (?)	Mc	Aust Mines Atlas (2013-10-01)
Sunter	0.3756	17.7	6.6	3.8	15		1.6		Sedi-Pb-Zn	Terramin Australia	Mc	Ann Rep 2013
Teena	58	51.6	2995.6	11.1			1.6		Sedi-Pb-Zn	Rox Resources	Mc	Media (2016-06-01)
Walford Creek	48.3	4.1	197.8	0.88	20.4		0.83	0.39	Sedi-Pb-Zn	Aeon Metals	Mc	ASX (2014-04-03)
Wonawinta	41.5	0.0	0.0		45		0.5		Sedi-Pb-Zn	Cobar Consolidated Resources	Mc	Inv Pres 2014-03
Maramungee	1.8	20.5	36.9	4.4					Sedi-Pb-Zn	unknown	Lc	Williams & Heinemann (1993)
Woodlawn Underground	10.10	47.2	477.1	10.15	84	0.55	4.03	1.78	Sed-Pb-Zn/VMS	TriAusMin	Mc	Ann Rep 2013
Woodlawn Tailings	11.65	10.6	123.9	2.29	32	0.29	1.35	0.50	Sed-Pb-Zn/VMS Tailings	TriAusMin	Mc	Ann Rep 2013
Djibigan (Manbarrum)	19.93	2.3	46.4	0.5	16.4		0.2		Sedi-Pb-Zn?	TNG Ltd (sold to Legacy Iron late 2013)	Mc	Ann Rep 2012
Brown's-Brown's East	54.5	2.8	153.2	0.60	12.1		3.09	0.43	Sediment-hosted polymetallic	HNC Australia Resources	Mc	NT DME Fact Sheet 2013-09
Federation	0.562	6.5	3.7	1.4	36.4				Skarn	unknown	Mc	Ann Rep 2009
Gossan Dam-Bonnie Rock	0.45	5.8	2.6	1.25	100		1.5		Skarn	unknown	Mc	Minedex (2013-05-05)
Mayfield	4.9	2.8	13.6	0.6	8.3	0.6		0.3	Skarn	Forge Resources / Capital Mining	Mc	Ann Rep 2013 (Capital)
Morrison	1.93	25.1	48.5	5.4	21	0.1	0.3	0.6	Skarn	Kagara (?)	Mc	Aust Mines Atlas (2013-09-30)
Mt Garnet Group	9.633	36.3	349.6	7.80	35.0	0.17	0.39	1.08	Skarn	Kagara	Mc	Ann Rep 2011
Mt Moss	20	1.6	32.6	0.35				0.35	Skarn	Curtain Bros (Qld)	Mc	Qld Met. Ind. Mines 2012
Narrawa	0.209	5.2	1.1	1.11	161.0	2.10	1.32		Skarn	Torque Mining (formerly Frontier Resources)	Mc	Prospectus (2013-07)
O'Callaghans	78	2.3	182.9	0.50			0.25	0.28	Skarn	Newcrest Mining	Mc	ResV ResC 2013
Queenslander	1.59	20.9	33.3	4.5	11		0.1	0.6	Skarn	Kagara (?)	Mc	Aust Mines Atlas (2013-09-30)
Railway Flat	0.9	15.8	14.2	3.40	16.0		0.9	0.2	Skarn	Kagara (?)	Mc	Ann Rep 2011 (Copper Strike)
Red Cap Group	3.832	22.3	85.6	4.80	19	0.06	0.20	0.76	Skarn	Kagara	Mc	Qtr 2012-06
Evelyn	0.00742	17.2	0.1	3.7	343		6.7		Skarn	Crocodile Gold	Lc	Ann Info Form 2013
Allison's Lode	0.057	20.5	1.2	4.41	32.7		1.22		VMS	Zeehan Zinc / Creat Resources Holdings	Mc	Media (2009-03-29)
Anaconda	0.877	9.2	8.0	1.97	6			0.96	VMS	Glencore (?)	Mc	Minedex (2013-05-05)
Barrow Creek-Home of Bullion	2.500	9.3	23.3	2.0	36	0.14	1.2	1.8	VMS	Kidman Resources	Mc	Ann Rep 2015
Belara	2.545	17.4	44.4	3.75	39.61		1.10	0.40	VMS	Ironbark Zinc	Mc	Website (2013-03-22)
Bentley	2.771	44.7	123.8	9.6	139	0.8		1.9	VMS	Independence Group	Mc	Ann Rep 2013
Burnside-Iron Blow	3.175	15.4	48.8	3.3	101	2.1	0.76	0.19	VMS	Crocodile Gold	Mc	Tech Rep (2013-07)
Chakola-Harnett Central	1.22	3.3	4.0	0.7	8.1	0.8	0.4	0.5	VMS	Capital Mining	Mc	Ann Rep 2011
Comstock (Australia)	3.062	6.9	21.1	1.48	45.7		2.10		VMS	Zeehan Zinc / Creat Resources Holdings	Mc	Media (2009-03-29)
Develin Creek-Rookwood	1.75	9.5	16.7	2.05	8.5	0.24		1.71	VMS	Fitzroy Resources	Mc	Ann Rep 2011
Eastman	3.4	7.9	26.8	1.70	14	0.022	0.65	0.66	VMS	Massive Resources Pty Ltd	Mc	Minedex (2013-05-05)
Emull-Lamboo	4.7	20.9	98.4	4.5	19		0.2	0.33	VMS	Northern Star Resources	Mc	Minedex (2013-05-05)
Endeavour (Elura)	17.4	36.1	628.3	7.8	73.6		4.9	0.18	VMS	Toho Zinc	Mc	Tech Rep (2013-04) (Couer Mining)
Fossey-Fossey East	0.425	52.8	22.5	11.4	112	2.3	6.3	0.5	VMS	Bass Metals	Mc	Ann Rep 2013
Golden Grove	26.1	17.9	468.2	3.9	35	0.7	0.3	2.2	VMS	MMG	Mc	Ann Rep 2013
Grieves Quarry	0.2	23.3	4.7	5					VMS	Icon Resources	Mc	Website (2013-03-24)
Hellyer Remnants	0.75	32.5	24.4	7.0	104	1.3	4.1	0.4	VMS	Bass Metals	Mc	Ann Rep 2013

Higgs	0.215	6.0	1.3	1.3	23	3.5	1.5		VMS	Frontier Resources (?)	Mc	Aust Mines Atlas (2013-10-01)
Jaguar	0.475	13.0	6.2	2.8	39			2.2	VMS	Independence Group	Mc	Ann Rep 2013
Jervois Group	1.0	10.3	10.3	2.21	73.2		2.58	0.89	VMS	KGL-Kentor Gold	Mc	Ann Rep 2011
Kangaroo Caves	6.3	15.4	96.7	3.3	12.1			0.5	VMS	Venturex Resources	Mc	Ann Rep 2013
Kempfield	21.8	4.4	95.6	0.9	47	0.12	0.45		VMS	Argent Minerals	Mc	Ann Rep 2012
Koonenberry	5.75	1.6	9.3	0.35	2.30	0.05		1.03	VMS	Ausmon Resources	Mc	Website (2013-03-24)
Kroombit (Cu+Zn/Cu)	5.158	8.7	45.0	1.88				0.15	VMS	Argonaut Resources	Mc	Media (2009-06-11)
Lennon's Find	1.8	23.7	42.7	5.1	82	0.26	1.4	0.2	VMS	Laconia Resources (sold to Musketeer Minerals Pty Ltd late 2013)	Mc	Ann Rep 2013
Lewis Ponds	6.62	11.3	74.7	2.4	69	1.5	1.4	0.2	VMS	TriAusMin	Mc	Ann Rep 2013
Liberty-Indee (Evelyn)	0.657	17.2	11.3	3.7	35.9	0.8	0.3	1.8	VMS	Venturex Resources	Mc	Ann Rep 2013
Manindi-Freddie Well	1.354	28.1	38.1	6.04	3.4	0.25		0.25	VMS	Metals Australia	Mc	Ann Rep 2013
Mariposa	0.574	8.8	5.1	1.90	60.0		5.10		VMS	Zeehan Zinc / Creat Resources Holdings	Mc	Media (2009-03-29)
Mons Cupri	4.607	6.0	27.9	1.3	24.1	0.1	0.5	0.9	VMS	Venturex Resources	Mc	Ann Rep 2013
Mt Ararat	1.3	1.9	2.4	0.4	6	0.5		2.0	VMS	Stavely Minerals	Mc	Media (2015-09-08)
Mt Charter	6.1	2.3	14.2	0.5	36	1.2			VMS	Bass Metals	Mc	Ann Rep 2011
Mt Mulcahy	0.228	8.6	2.0	1.9	33		0.17	2.96	VMS	Black Raven Mining	Mc	Minedex (2013-05-05)
Mulgul-Jillawarra (Abra)	107	0.4	43.3	0.09	9.3	0.11	3.49	0.19	VMS	Hunan Nonferrous Metals	Mc	Ann Rep 2010 (Jabiru Metals; also Minedex)
Onedin (Koongie Park)	4.458	15.1	67.2	3.24	25.97	0.31	0.92	0.81	VMS	Anglo Australian Resources	Mc	Ann Rep 2011
Parkers Hill (Mineral Hill)	4.867	4.4	21.2	0.94	33	1.11	1.68	1.21	VMS	KBL Mining	Mc	Ann Rep 2013
Peelwood North/South	0.895	18.6	16.6	4.0	11.3		0.7	0.8	VMS	Balamara Resources (Sultan Corp)	Mc	Ann Rep 2012
Prairie Downs	2.98	23.0	68.5	4.94	15		1.59		VMS	Prairie Downs Metals (now Brumby Resources)	Mc	Minedex (2013-05-05)
Que River-Que River S Lens	0.68	23.0	15.6	4.9	79	0.7	2.5	1.2	VMS	Bass Metals	Mc	Ann Rep 2013
Quinns-Austin	1.48	6.5	9.6	1.39	3.31	0.24		1.02	VMS	Caravel Minerals (formerly Silver Swan Group)	Mc	Ann Rep 2012
Sandiego	3.53	18.8	66.5	4.05	19	0.27		1.63	VMS	Anglo Australian Resources	Mc	Ann Rep 2011
Stockman	13.986	18.6	260.3	4.0	40	1.1	0.7	2.0	VMS	Independence Group	Mc	Ann Rep 2013
Sulphur Springs	12.831	19.1	244.8	4.1	17.6	0.1	0.2	1.5	VMS	Venturex Resources	Mc	Ann Rep 2013
Sunny Corner	1.5	17.2	25.8	3.7	24	0.3	2.1	0.4	VMS	Argent Minerals	Mc	Ann Rep 2013
Teutonic Bore	1.554	11.6	18.1	2.5	49			1.6	VMS	Independence Group	Mc	Ann Rep 2013
Thalanga Group	4.171	36.9	153.9	7.93	41.4	0.57	2.25	0.93	VMS	Kagara	Mc	Ann Rep 2011
Turner River-Orchard Well/Discovery	2.61	12.4	32.4	2.67	88	0.70	1.07	0.10	VMS	De Grey Mining	Mc	Minedex (2013-05-05)
Whim Creek	0.972	5.1	5.0	1.1	10.3	0.1	0.2	2.1	VMS	Venturex Resources	Mc	Ann Rep 2013
Whundo Cu-Zn / Zn	2.875	6.3	18.0	1.35				0.76	VMS	Fox Resources	Mc	Ann Rep 2013
Burns Peak	0.117	42.3	5.0	9.1	47	0.62	3.1	0.18	VMS	Mancala Resources Pty Ltd	Lc	DPEMP (2014)
Daly River Anomaly A	0.762	46.5	35.4	9.99	14.5		0.3	1.77	VMS	Troy Resources	Lc	Ann Rep 2007

Mt Bonnie	0.65	41.9	27.2	9.0	280	1.7	2.0	0.5	VMS	Crocodile Gold	Lc	Tech Rep (2013-07)
Hellyer Tailings	9.5	11.6	110.5	2.5	104	2.6	3.0	0.2	VMS	Ivy Resources	Mc	Ann Rep 2011 (Bass Metals)
Cu Deposits with inferred In content[2]:												
Mountain of Light-Lyndhurst	2.9	0.077	0.223					0.77	Unknown	Phoenix Copper	Mc	Ann Rep 2010
Mt Carlton	27.6	0.027	0.745		45	1.6		0.27	Epithermal	Conquest Mining	Mc	Ann Rep 2010
Texas-Silver Spur	0.808	0.017	0.014		69.75	0.09		0.17	Epithermal	Alcyone Resources	Mc	Ann Rep 2010
Prospect D	3.16	0.056	0.179					0.56	Gabbro Feeder	Goldstake Explorations	Mc	Ann Rep 2010
Telfer Group	702.3	0.012	8.086			0.86		0.12	Intrusion-Related Au?	Newcrest	Mc	Ann Rep 2010
Camp Dome-17 Mile	6	0.045	0.270					0.45	Intrusion-Related Au?/Skarn?	Mt Burgess Mining	Mc	Ann Rep 2008
Cairn Hill	11.4	0.040	0.456			0.1		0.4	IOCG	IMX Resources	Mc	Ann Rep 2010
Carrapateena	203	0.131	26.593		6.0	0.56		1.31	IOCG	OZ Minerals	Mc	Ann Rep 2010
Clonclurry Miscellaneous	5.4	0.088	0.475			0.62		0.88	IOCG	Exco Resources	Mc	Ann Rep 2010
E1 Camp	48.1	0.072	3.463			0.21		0.72	IOCG	Exco Resources (now Xstrata)	Mc	Ann Rep 2010
Eloise	3.216	0.310	0.997		10	0.8		3.1	IOCG	FMR Investments	Mc	Ann Rep 2009
Ernest Henry	105	0.123	12.940			0.66		1.23	IOCG	Xstrata	Mc	Ann Rep 2010
Gem	0.492	0.050	0.025			0.2		0.5	IOCG	China Yunnan Copper Australia	Mc	Ann Rep 2010
Great Australia	2.1	0.154	0.323			0.13		1.54	IOCG	Exco Resources	Mc	Ann Rep 2010
Hillside	170	0.070	11.900			0.2		0.7	IOCG	Rex Minerals	Mc	Ann Rep 2010
Kalkaroo	62.5	0.055	3.438			0.44		0.55	IOCG	Havilah Resources	Mc	Ann Rep 2010
Kalman	60.8	0.032	1.946			0.15		0.32	IOCG	Cerro Resources	Mc	Tech Rep 2010
Merlin	6.7	0.033	0.223		8.3	0.08		0.33	IOCG	Ivanhoe Australia	Mc	Ann Rep 2010
Monakoff	4.0	0.132	0.528			0.42		1.32	IOCG	Exco Resources	Mc	Ann Rep 2010
Mt Elliott	570	0.044	24.900			0.24		0.44	IOCG	Ivanhoe Australia	Mc	Ann Rep 2010
Olympic Dam	9,075	0.087	789.525		1.5	0.32		0.87	IOCG	BHP Billiton	Mc	Ann Rep 2010
Prominent Hill	285.4	0.089	25.402		2.39	0.79		0.89	IOCG	OZ Minerals	Mc	Ann Rep 2010
Redbank	6.24	0.153	0.955					1.53	IOCG	Redbank Copper	Mc	Ann Rep 2010
Rover 1	5.33	0.100	0.533		2.2	2.1		1.0	IOCG	Westgold Resources	Mc	Ann Rep 2010
Starra Line	31	0.090	2.790			0.80		0.90	IOCG	Ivanhoe Australia	Mc	Ann Rep 2010
Tennant Creek Group	3.61	0.034	0.123			10.16		0.34	IOCG	Excalibur Mining Corp.	Mc	Ann Rep 2010
Mt Colin	1.49	0.247	0.368					2.47	IOCG?	Exco Resources	Mc	Ann Rep 2010
North Portia	11.3	0.089	1.006			0.64		0.89	IOCG?	Havilah Resources	Mc	Ann Rep 2010
White Range Group	33.9	0.099	3.356					0.99	IOCG?	Queensland Mining Corp.	Mc	Ann Rep 2010
Young Australian	2.13	0.100	0.213		1.4			1	IOCG?	Queensland Mining Corp.	Mc	Ann Rep 2010
Corkwood	0.225	0.024	0.005					0.24	Magmatic	Panoramic Resources	Mc	Ann Rep 2010
Panton	14.32	0.008	0.107					0.075	Magmatic	Platinum Australia	Mc	Ann Rep 2010
Pardoo-Highway	50	0.013	0.633					0.13	Magmatic	Segue Resources	Mc	Ann Rep 2010
Wildara-Horn	0.6	0.030	0.018					0.30	Magmatic	Breakaway Resources	Mc	Ann Rep 2010
Brown's / Brown's East	70.5	0.084	5.910		12.1			0.84	Magmatic Sulphide	Compass Resources	Mc	Ann Rep 2007
Copernicus	0.81	0.082	0.067					0.82	Magmatic Sulphide	Panoramic60%, Thundelarra40%	Mc	Ann Rep 2010
Kambalda Field	23.63	0.014	0.339					0.14	Magmatic Sulphide	Various	Mc	Ann Rep 2010

Munni Munni	23.6	0.015	0.348					0.15	Magmatic Sulphide	Platina Resources	Mc	Ann Rep 2010
Radio Hill	9.87	0.076	0.750					0.76	Magmatic Sulphide	Fox Resources	Mc	Ann Rep 2010
Savannah-Sally Malay	5.41	0.078	0.422					0.78	Magmatic Sulphide	Panoramic Resources	Mc	Ann Rep 2010
Thomson River	0.04	0.270	0.011		9.5	2.5		2.7	Magmatic Sulphide	unknown	Lc	historic
Kundip	8.37	0.040	0.335		2.8	2.83		0.4	Orogenic Au	Tectonic Resources	Mc	Ann Rep 2010
Deflector	3.4	0.080	0.272		4.7	5.4		0.8	Orogenic Au?	Mutiny Gold	Mc	Ann Rep 2010
Gabanintha	0.45	0.050	0.023			1.6		0.5	Orogenic Au?	Kentor Gold	Mc	Company Presentation
Big Cadia	42	0.040	1.680			0.38		0.40	Porphyry	Newcrest	Mc	Ann Rep 2010
Boddington	1,531.2	0.010	15.772			0.59		0.10	Porphyry	Newmont	Mc	Ann Rep 2010
Bushranger	52.5	0.035	1.824		1.39	0.04		0.35	Porphyry	Lachlan Star	Mc	Ann Rep 2010
Cadia East	2,347	0.028	65.716		0.47	0.44		0.28	Porphyry	Newcrest	Mc	Ann Rep 2010
Cadia Hill	408	0.012	4.896			0.42		0.12	Porphyry	Newcrest	Mc	Ann Rep 2010
Copper Hill	173	0.031	5.363			0.26		0.31	Porphyry	Golden Cross Resources	Mc	Ann Rep 2010
Marsden	224	0.032	7.164			0.17		0.32	Porphyry	Newcrest	Mc	Ann Rep 2010
McPhillamys	91.94	0.007	0.644			1.00		0.07	Porphyry	Alkane Resources	Mc	Ann Rep 2010
Mt Cannindah	7.43	0.097	0.721			0.38		0.97	Porphyry	Metallica Minerals	Mc	Ann Rep 2010
Mt Unicorn	105	0.006	0.599		3.1			0.057	Porphyry	Dart Mining	Mc	Ann Rep 2010
Northparkes	365	0.062	22.728			0.27		0.62	Porphyry	Rio Tinto	Mc	Ann Rep 2010
Peak Hill	11.27	0.011	0.124			1.29		0.11	Porphyry	Alkane Resources	Mc	Ann Rep 2010
Ridgeway	155	0.038	5.890		0.81	0.73		0.38	Porphyry	Newcrest	Mc	Ann Rep 2010
Rocklands	245	0.021	5.056			0.04		0.21	Porphyry	CuDeCo	Mc	Ann Rep 2010
Spinifex Ridge	843	0.008	7.126					0.085	Porphyry	Moly Mines	Mc	Ann Rep 2010
Temora	142.2	0.032	4.614			0.29		0.32	Porphyry	Goldminco (Straits Res.)	Mc	Ann Rep 2010
Thursdays Gossan	10.6	0.045	0.477					0.45	Porphyry	Beaconsfield Gold	Mc	Ann Rep 2010
Whitewash	71.5	0.010	0.715		1.2			0.1	Porphyry	Aussie Q Resources	Mc	Ann Rep 2010
Yeoval	12.9	0.038	0.490		2.2	0.14		0.38	Porphyry	Augur Resources	Mc	Ann Rep 2010
Barbara	5.33	0.140	0.746		2.5	0.1		1.4	Sediment-hosted Pb-Zn	Syndicated Metals	Mc	Ann Rep 2010
Barbara North	0.74	0.123	0.091			0.1		1.23	Sediment-hosted Pb-Zn	Mt Isa Metals	Mc	Ann Rep 2010
Endeavour	26.2	0.018	0.472		62			0.18	Sediment-hosted Pb-Zn	Toho Zinc	Mc	Ann Rep 2009
Lady Annie	40.55	0.086	3.490					0.86	Sediment-hosted Pb-Zn	China Sci-Tech	Mc	Company Presentation
Mt Isa	200	0.201	40.280					2.01	Sediment-hosted Pb-Zn	Xstrata	Mc	Ann Rep 2010
Napier Range-Wagon Pass	0.59	0.050	0.030		75			0.5	Sediment-hosted Pb-Zn	Meridian Minerals	Mc	Ann Rep 2010
Maroochydore	41.2	0.080	3.296					0.8	Sediment-hosted stratiform Cu	Aditya Birla Minerals	Mc	Ann Rep 2010
Mt Gunson Group	52.05	0.104	5.439		8.8			1.0	Sediment-hosted stratiform Cu	Gunson Resources	Mc	Ann Rep 2010
Mt Gordon	22.1	0.250	5.525					2.5	Sediment-hosted stratiform Cu?	Aditya Birla Minerals	Mc	Ann Rep 2010
Mt Oxide	17.9	0.130	2.327		10			1.3	Sediment-hosted stratiform Cu?	Perilya Mines	Mc	Ann Rep 2010
Kulthor	2.6	0.178	0.462			1.06		1.78	Skarn	Ivanhoe Australia	Mc	Ann Rep 2010
Mt Garnet Field	18.25	0.118	2.150		40.8	0.29		1.18	Skarn	Kagara Zinc	Mc	Ann Rep 2010
Osborne	4.8	0.140	0.672			0.86		1.40	Skarn	Ivanhoe Australia	Mc	Ann Rep 2010
Cleveland-Luina	5.16	0.031	0.160					0.31	Skarn?	Lynch Mining Pty Ltd	Mc	RAM

Einasleigh Group (Cu, PbZnCu)	21.6	0.078	1.680		16.6	0.09		0.78	Skarn?	Copper Strike	Mc	Ann Rep 2010
Canbelego	1.5	0.120	0.180					1.2	Slate belt Au	Helix Resources	Mc	Ann Rep 2010
Cobar-CSA	11.2	0.620	6.944					6.2	Slate belt Au	Glencore	Mc	Ann Rep 2011
Mt Fitch	1.3	0.060	0.078					0.60	Unconformity-related Uranium	Compass Resources	Mc	Ann Rep 2007
Doolgunna-DeGrussa	10.67	0.560	5.975		15	1.9		5.6	VMS	Sandfire Resources	Mc	Ann Rep 2010
Foresthome-Develin Creek	1.75	0.170	0.298		8.5	0.2		1.7	VMS	Icon Resources	Mc	Ann Rep 2010
Home of Bullion	0.133	1.474	0.196					14.7	VMS	Goldstake Explorations	Mc	Ann Info Form 2008
Horseshoe Lights	4.9	0.100	0.490			0.1		1.0	VMS	Horseshoe Metals	Mc	Ann Rep 2010
Iron Blow	3.17	0.019	0.060		100.9	2.08		0.19	VMS	Crocodile Gold	Mc	Ann Rep 2010
Just Desserts (Yuinmery)	1.07	0.182	0.195		2.06	0.78		1.82	VMS	Empire Resources	Mc	Ann Rep 2010
Kanmantoo	32.2	0.090	2.897		3.2	0.2		0.9	VMS	Hillgrove Resources	Mc	Ann Rep 2010
Koonenberry-Grasmere	5.75	0.103	0.592		2.3	0.05		1.03	VMS	Ausmon Resources	Mc	Company Website
Lennons Find	0.853	0.070	0.060		115			0.7	VMS	Jabiru Metals	Mc	Ann Rep 2010
Mt Ararat	0.7	0.270	0.189			0.8		2.7	VMS	Beaconsfield Gold	Mc	Ann Rep 2010
Mt Lyell	33.8	0.119	4.021			0.30		1.19	VMS	Copper Mines of Tasmania	Mc	Ann Rev 2010
Mulgul-Jillawarra	107	0.019	2.015		9.25	0.11		0.19	VMS	Abra Mining	Mc	Ann Rep 2010
Mungana	48.7	0.019	0.925		13	0.7		0.19	VMS	Mungana Goldmines	Mc	Ann Rep 2010
Panorama-Sulphur Springs	19.3	0.120	2.316		16.1			1.2	VMS	Venturex Resources	Mc	Ann Rep 2010
Quartz Circle-Igloo	0.127	0.410	0.052					4.1	VMS	Xanadu Res (now Corazon Res)80%, Cazaly Resources20%	Mc	Ann Rep 2010
Que River-Fossey-Hellyer	2.23	0.060	0.134		103	1.5		0.6	VMS	Bass Metals	Mc	Ann Rep 2010
Red Dome	69.2	0.024	1.661		5	0.63		0.24	VMS	Mungana Goldmines (Kagara Zinc)	Mc	Ann Rep 2010
Roseby Group	132.5	0.068	9.013			0.06		0.68	VMS	Altona Mining	Mc	Ann Rep 2010
Sandiego-Onedin	8.02	0.117	0.938		23.2	0.29		1.17	VMS	Anglo Australian Resources	Mc	Ann Rep 2010
Tritton	27.1	0.173	4.680			0.07		1.73	VMS	Straits Resources	Mc	Ann Rep 2010
Whundo Cu-Zn	1.69	0.110	0.186					1.10	VMS	Fox Resources	Mc	Ann Rep 2010
Whundo Zn	0.767	0.043	0.033					0.43	VMS	Fox Resources	Mc	Ann Rep 2010
Kroombit	6.06	0.028	0.167					0.28	VMS?	Argonaut Resources	Mc	Ann Rep 2010
Muturoo	13.1	0.148	1.939			0.22		1.48	VMS?	Havilah Resources	Mc	Ann Rep 2010
Nifty	60.3	0.179	10.804					1.79	VMS?	Aditya Birla Minerals	Mc	Ann Rep 2010
Parkers Hill	3.0	0.104	0.313		41.4	0.19		1.04	VMS?	Kimberley Metals	Mc	Ann Rep 2010
Tottenham	3.7	0.110	0.407					1.1	VMS?	Mincor Resources	Mc	Ann Rep 2010
Wellington	2.09	0.099	0.207			0.3		0.99	VMS?	Alkane Resources	Mc	Ann Rep 2010
	Total Indium :		46,213									

[1]Pb-Zn deposit database from Mudd et al. (2017b) represents resources estimated as at 2013.

[2]Cu deposit database from Mudd et al. (2013a) represents resources estimated as at 2010.

6.2.2 Quantifying mining activity

Having established the quantity of current day indium resources in mineral deposits, it is necessary to determine the first flows which will make up the indium cycle in Australia. As shown in Fig. 6.1, mining involves the bulk extraction of rock, the separation of quantities not deemed to be ore, the production of concentrates and the disposal of tailings. It is therefore important to know the deportment of indium between mine wastes and the products of mining. Such a calculation requires knowledge of indium deportment in various mineral phases, although it should be noted that this is an area in need of further research. It is outside the scope of this thesis to perform the primary experiments necessary to understand indium deportment in greater depth, however it is possible to find some cases where indium deportment has been reported for individual mines. These studies are listed in Table 6.2 and show that for the mining of zinc, some 62% of milled indium will deport to zinc concentrates, while around 19% appears in tailings on average. Of the portion that is sent to zinc concentrates, some will be processed domestically, i.e. in Townsville, Queensland or Risdon, Tasmania. However, given that these facilities do not operate indium circuits, this portion must be assumed to become slag wastes. This is shown in Fig. 6.1, as no Australian-sourced indium is domestically refined.

To estimate the quantities of zinc and copper mined, and the amount of concentrate exported and imported on an annual basis, historical statistics were used from OCE (2014) and publications from the former Australian Bureau of Mineral Resources, such as the Australian Mineral Industry Reviews (see Mudd, 2010). Knowing the average grade and quantity of base metals mined, the associated indium extracted along with these metals was estimated using the proxy method applied in Chapter 4. Using the average deportment values suggested by the studies in Table 6.2, a material balance was established such that the present day resource total from Table 6.1 is achieved as a result of the mining, milling, processing and trade reported over time. This balance is shown in Figure 6.2.

Table 6.2: Review of literature outlining the deportment of indium as a percentage of milling feed.

Location	Milling Feed	Zn conc.[1]	Pb conc.[1]	Cu conc.[1]	As conc.[1]	Sn conc.[1]	Tailings	Bulk sulphide conc.	Data source and notes
	%	%	%	%	%	%	%	%	
Heath Steele, Canada	100	51.4	2.3	10.5	-	-	35.8	-	(Chen and Petruk, 1980)
Brunswick 6/12, Canada	100	61.6	6.7	0.7	-	-	28.3	2.7	(Petruk and Schnarr, 1981)
Keg - Main Zone, Canada	100	74.3	N/A	8.6			17.1		Silver Range Resources - Keg NI 43-101 Technical Report, 2013
Mount Pleasant North Zone, Canada	100	70.1		Comb.[2]	3.8	5.2	20.9	-	Adex - Mount Pleasant (North Zone) NI43-101 Technical Report, 2013. Tin recovery by flotation.
	100	71.2		Comb.[2]	3.9	3.7	21.3	-	As above. Tin recovery by flotation and gravity separation.
	100	91.7		Comb.[2]			3.2	0.8	As above. Testing conducted at the pilot scale.
Pingüino, Argentina	100	2.3	90.4				7.3	-	Argentex – Pingüino NI 43-101 Technical Report, 2013
West Desert (Crypto), USA	100	58.5							Lithic Resources – NI 43-101 Technical Report on the Crypto Zinc-Coper-Indium Project, Juab County, Utah. Only the proportion that is sent to zinc concentrates is reported.
	100	67.2							As above. Only the proportion sent to zinc concentrates is reported.
	Average %	62.6	4.5	6.6	3.8	14.2	19.1	1.8	
	St. Dev. %	8.7	3.1	5.2	0.04	12.4	12.3	1.3	

[1]Conc. = Concentrate

[2]Comb. = Combined concentrate of Cu and Zn

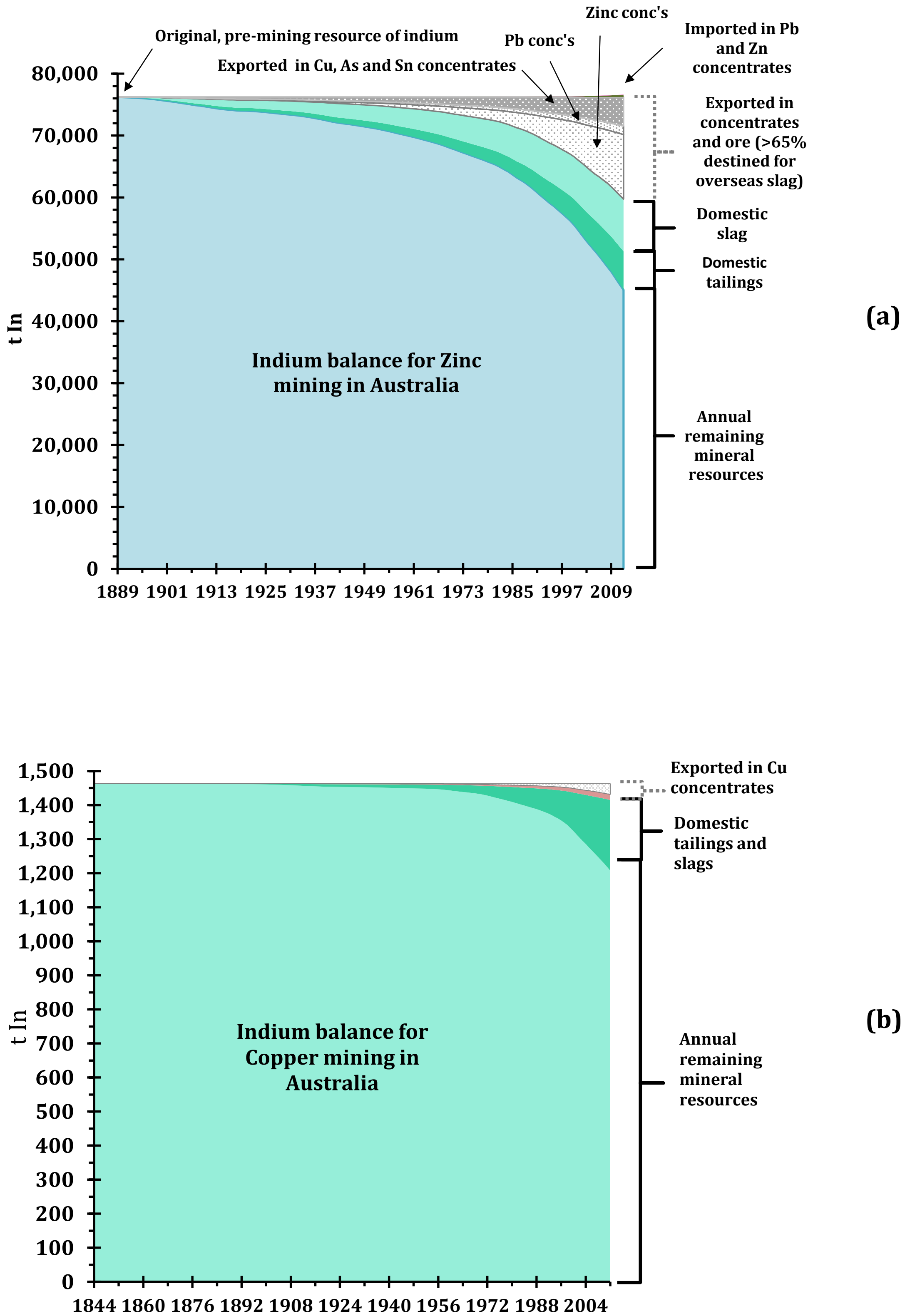

Figure 6.2: (a) Material balance for indium due to the mining of zinc in Australia, 1889-2012, (b) Material balance for indium due to the mining of copper in Australia, 1844-2012.

There are two major findings from Fig. 6.2. The first is that significant quantities of indium (enough to meet several years of global demand) are likely to have accumulated in Australian mining and mineral processing wastes over time. The economic value of indium, and indeed other potential critical metals, present in these wastes may be useful in subsidising their recovery and/or rehabilitation if supply restrictions lead to an increase in prices in future. The second major finding is that the majority of indium embodied in concentrates (plus a smaller portion sold directly as ore) have been exported overseas, totalling over 18,600 t In since trade records began.

As per Mikolajczak (2009), some 30% of the indium-containing concentrates do not reach indium capable smelters. Of the remaining portion that do reach indium-capable smelters, a typical processing efficiency of 50% may be assumed, suggesting that ~35% of indium in concentrates will end up being refined. This value is likely to represent only recent years in which indium has increased in demand. As shown in Table 1 of Werner et al. (2015), other studies estimate a much lower proportion only a few years prior. Using these variable published values of processing efficiency, it is possible to roughly estimate what percentage of indium exported as Australian concentrates was refined in overseas facilities on an annual basis. Following this, global production data available from the USGS (USGS, Var.) can be used to show what percentage of global indium production was ultimately Australian sourced over time (see Fig. 6.3):

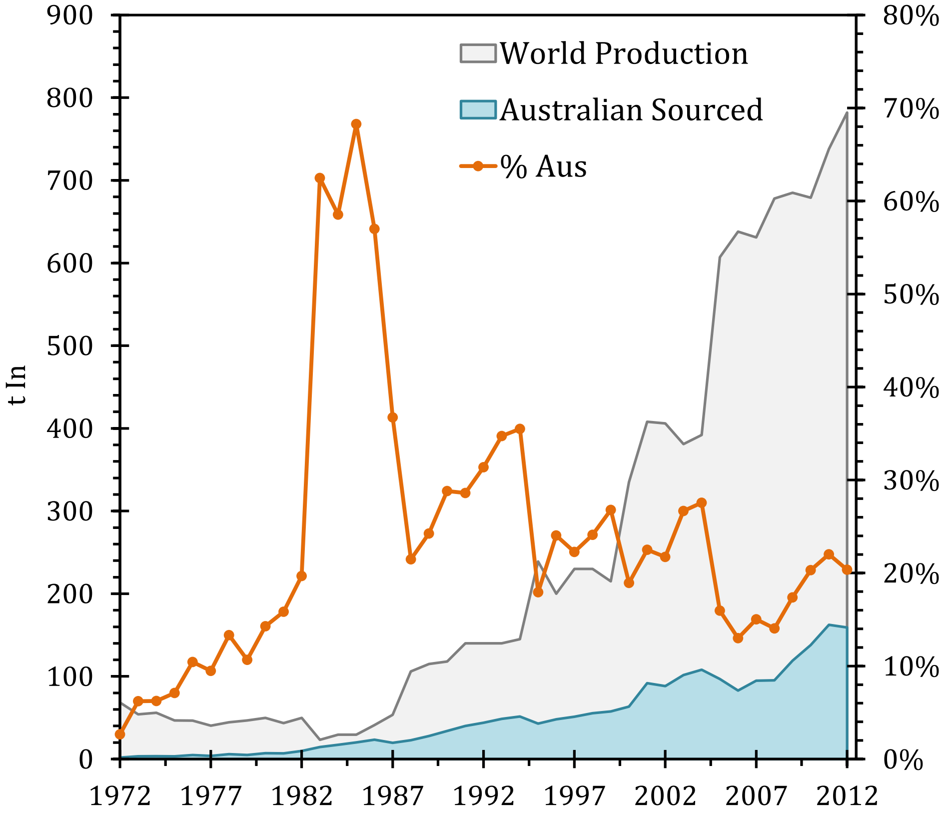

Figure 6.3: Proportion of global indium production estimated to originate from Australian zinc concentrates over time, in absolute and percentage terms.

These data suggest that while Australia has not contributed to the refining of indium over time, it has been a significant contributor to the supply chain nonetheless. If prices are overlain across the values shown in Fig. 6.3, an estimated 1.19 Billion USD worth of indium have been exported from Australia from 1972-2012, adjusted for inflation. However, it is unlikely that Australian miners have seen this value, as the price of specialty metal by-products are often not accounted for in the sale of concentrates (Mudd et al., 2017a), and indeed indium is still significantly under reported as a commodity as shown in Table 6.1. It should be noted that this estimation has been made assuming an average global percentage of what milled indium is refined, it does not track the flows of individual concentrates between specific mining and smelting facilities. Such an analysis is an avenue for further research.

6.3 Commodity trade using United Nations HS tariff codes

As shown in Fig. 6.1, not all of the indium flows within and across Australian borders are associated with mining. Indium is of course also traded in the form of LCD displays, solar panels, soldering materials and the many other applications in which it is used. As well as this, indium will likely be imported in the form of components or intermediates that make up these goods in order to supply domestic manufacturing. To estimate these trade flows, it is necessary to review the commodities traded as per the import/export HS (Harmonised System) tariff codes available from the United Nations Comtrade database. These data have only been extracted back to 1988, however prior to this, global indium consumption was less than 7% of what is it today. Thus, it is evident that Australia would not have traded a significant volume of goods containing refined indium prior to this time, and so the effect of this scoping limitation is somewhat minimised.

6.3.1 Collating codes for trade

So that no trade flows go unnoticed, it is necessary to review literature to determine all of indium's applications. Notable sources include: Chagnon (2000), Felix (2000), Schwarz-Schampera (2014), and USGS (Var.) Following this, the tariff codes which are likely to encompass these applications are identified. This is not always entirely clear, as tariff code descriptions are often not detailed enough to simply be able to search for indium-related terms. Additionally, these codes change periodically, meaning that a particular commodity may have been traded under different codes at different times. It may also be the case that a code has split or merged over time to encompass different products. This has been accounted for in assessing the mass of these trade flows.

The ultimate list of codes and products considered for trade are listed in Tables 6.3 and 6.4. An additional list of potential trade flows containing indium, but which were discounted is provided in Appendix C. These may have been discounted for numerous reasons. For example, one of indium's applications is in nuclear control rods, however Australia only has one nuclear reactor (the ANSTO OPAL research reactor) which only uses hafnium control rods. Therefore, it would be unlikely that Australia has imported any indium under the code 840140 (described as "parts of nuclear reactors"). This example highlights that performing a metal-specific trade analysis requires a significant amount of scrutiny on the use of each code, and an understanding of the state of local manufacturing and consumption to see how likely it is that a given metal or commodity is actually traded in Australia. Other trade flows may be discounted simply due to a lack of information, for example regarding the use of indium as a coating in historical aircraft engines.

Table 6.3: Semi-finished products and corresponding tariff codes containing indium considered for trade

Semi-finished indium product	Code examined	With description
Unwrought In metal and powder (incl. Ge, V, Ga, Hf, In, Nb, Re)	811292	Germanium, vanadium, gallium, hafnium, indium, niobium (columbium), rhenium, & articles of these metals, incl. waste & scrap, powder, unwrought.
Indium tin oxide (ITO)	282590	Hydrazine, hydroxylamine, metal oxides, hydroxides ne) // 282590 (.9000 for ITO). HS -281820, 2819, 2825, 2821, 2824, 2820, 2823, 281700 used.
Indium oxide	282590	Metal bases, oxides, hydroxides, peroxides, nes
Indium hydroxide	282590	Metal bases, oxides, hydroxides, peroxides, nes
Indium sulphamate	283329	Sulphates of metals nes
Indium acetate	291590 + 291529 + 294200	Saturated acyclic moncarboxylic acids, derivs, nes + acetic acid salts except cobalt and sodium + Organic compounds nes
Indium phosphide	284800	Inorganic chemicals, precious metal compound, isotope // Phosphides
Trimethylindium	293190	Organic chemicals/ // Other organo-inorganic compounds. // -Other

Table 6.4: Finished goods and associated tariff codes considered for indium trade, aggregated to general end-use sectors

HS Code	HS Code Description	Likely End use sector
847130	Portable digital data pr	Coatings
851712	Telephones for cellular networks/for other wireless networks, other than Line telephone sets with cordless handsets	Coatings
852851	Other monitors, of a kind solely/principally used in an automatic data processing system of heading 84.71	Coatings
852859	Other monitors, not of a kind solely/principally used in an automatic data processing system of heading 84.71	Coatings
853120	Indicator panels incorporating electronic displays	Coatings
852580	Television cameras, digital cameras & video camera recorders	Coatings
852872	Other colour reception apparatus for television, whether/not incorporating radio-broadcast receivers/sound/video recording/reproducing apparatus,	Coatings
8527-	Radio receivers / radiobroadcast receivers	Coatings
900710	Cinematographic cameras for film <16mm wide	Coatings
	Cinematographic cameras for film >16mm wide	Coatings
8703	Passenger vehicles (In in navigation equipment)	Coatings
760900	Aluminium pipe or tube fittings	Solders & Alloys
800120	Tin alloys unwrought: Alloys of Sn and In, e.g. Field's metal (32.5% Bi, 16.5% Sn, 51% In)	Solders & Alloys
780199	Lead unwrought - nes: Alloys of Pb and In	Solders & Alloys
790120	Zn alloys - unwrought: Alloys of Zn and In	Solders & Alloys
831130	Coated rods and cored wire, of base metal, for soldering, brazing or welding by flame	Solders & Alloys
300640	Dental cements and other dental fillings, bone cement	Solders & Alloys
3810(1010 /1090/90 90)	Indium solder pastes - (Metal pickling preps, solder and brazing flux, etc / Electro-weld cores, coatings, etc)	Solders & Alloys
854140	LED semiconductors	Electrical Compounds and Semiconductors
854140	Photosensitive semi-conductor devices, including photovoltaic cells whether or not assembled in modules or made up into panels; light emitting diodes	Electrical Compounds and Semiconductors
853400	Printed Circuits - In in thermal interface materials	Electrical Compounds and Semiconductors
850610	Manganese dioxide primary cells - Alkaline batteries	Research and Other Uses
701790 (90)	Laboratory, hygienic or pharmaceutical glassware nes	Research and Other Uses
902790 (90)	Microtomes, parts of scientific analysis equipment	Research and Other Uses
903180	Measuring or checking equipment nes	Research and Other Uses
903190	Parts and accessories for measuring, checking equipment nes	Research and Other Uses
382200	Composite diagnostic of laboratory reagents, nes - ceertified reference materials	Research and Other Uses
282739	In Trichloride	Research and Other Uses
283329	In Sulfate	Research and Other Uses

6.3.2 Assigning indium proportions to trade flows

Having identified and collated the trade of indium-containing commodities, it is necessary to determine what proportion of these flows is actually indium. For example, there may be data on the number or mass of mobile phones traded over time, however only a certain portion of this is representative of indium metal itself. A complicating factor is that this portion is likely to have changed over time, as LCD displays were not always present in mobile phones, and the average screen size has also changed considerably. Factors such as these must be considered for all LCD applications, and indeed all other trade flows for which there may be changes noted over time. Of particular note is the emergence of thin-film solar cells such as copper-indium-galium selenide (CIGS), amorphous silicon (a-Si) and cadmium telluride (CdTe) technologies which all contain indium in varying quantities and have all gained varying market shares over time. It can be seen that simply specifying what percentage of a tonne of solar panels traded in a given year is indium can be problematic. Nonetheless, an extensive review of literature has been conducted for as many product categories as possible. In Tables 6.5 and 6.6, some sample calculations are provided for the estimation of indium traded in the form of solar cells under the HS tariff code 854140 – photosensitive/photovoltatic/LED semiconductor devices. These highlight the detail necessary to assign indium proportions to trade flows. The content estimates and data sources for other indium trade flows are provided Appendix C.

Table 6.5: Estimation of the proportion of indium in thin film solar cells on a mass basis

Value	Units	Source
CIGS		
22.5	kg In / MW	McLellan et al. (2016)
18.99	kg In / MW	Moss et al. (2011)
16.5	kg In / MW	Zimmermann and Gößling-Reisemann (2014)
28.49	kg In / MW	Andersson (2000), and data for unit weight and power rating per area from Solar Design Tool (2016)
0.014	wt% In	Average of the above estimates, assuming 155000 kg total unit weight / MW for CIGS panels Solar Design Tool (2016)
0.084	wt% In	von Gries and Wilts (2014)
0.049	wt% In	Average of the above two estimates. Ultimate value used.
a-Si		
0.0022	wt% In	Goe and Gaustad (2014) and Solar Design Tool (2016)
0.0028	wt% In	Zimmermann (2013a) and Solar Design Tool (2016)
0.0025	wt% In	Average of the above two estimates. Ultimate value used.
CdTe		
0.015	wt% In	Zimmermann (2013a) and Solar Design Tool (2016)

The sum of columns D, F and H in Table 6.6 represents the total proportion of indium traded in solar panels for each year. However, the description of the HS code 854140 implies that other commodities besides just solar panels (e.g. LED semiconductor devices) are also traded under this code. It is therefore necessary to apply a factor that accounts for what proportion of this code purely represents solar panels (separate calculations must be performed for indium present in LED devices). The relative market shares of solar panels versus LED semiconductor devices is generally difficult to determine, as market share information of this kind is often proprietary. For this study, a constant proportion of 20.68% was applied for all years, following data suggesting that this was the average proportion for June/July shipment records in India (see Limra Exim, 2016). Additional sources to verify this proportion were not found, making this estimate quite uncertain. Nonetheless, these percentages are subsequently applied to the actual trade volumes reported under the tariff codes (with larger uncertainty bounds).

Table 6.6: Estimation of the market share of the various thin film solar cells technologies in relation to one another, and the broader photovoltaic market

	A	B	C	D	E	F	G	H
Source	(Adiboina, 2006)	(Adiboina, 2006)		Table 6.5	Zimmermann (2013a)	Table 6.5	Zimmermann (2013a)	Table 6.5
Year	Market share of thin films among all PV	Market share of CIGs among thin films	Calculated overall CIGS market share =A x B	Calculated In% attributable to traded CIGS cells = C x 0.049%	Overall a-Si market share	Calculated In% attributable to traded a-Si cells =E x 0.0025%	Total market share of CdTe	Calculated In% attributable to traded CdTe cells =G x 0.015%
1988					25%	0.00060%		
1989					25%	0.00062%		
1990					26%	0.00064%		
1991					27%	0.00066%		
1992					24%	0.00059%		
1993					21%	0.00052%		
1994					18%	0.00044%		
1995					15%	0.00037%		
1996					12%	0.00029%		
1997					12%	0.00030%		
1998					12%	0.00030%		
1999					12%	0.00030%		
2000					12%	0.00030%		
2001	2.5%	3%	0.03%	0.00002%	10%	0.00024%	0.13%	0.000020%
2002	2.9%	7%	0.05%	0.00005%	9%	0.00022%	0.25%	0.000039%
2003	3.7%	10%	0.08%	0.00007%	6%	0.00016%	0.38%	0.000059%
2004	3.7%	13%	0.14%	0.00010%	5%	0.00011%	0.50%	0.000077%
2005	4.9%	15%	0.20%	0.00010%	4%	0.00011%	0.70%	0.000108%
2006	6.2%	18%	0.48%	0.00029%	5%	0.00012%	1.10%	0.000170%
2007	9.9%	21%	0.44%	0.00020%	5%	0.00012%	1.10%	0.000170%
2008	14.4%	23%	0.47%	0.00010%	5%	0.00013%	1.40%	0.000217%
2009	24.7%	26%	0.67%	0.00010%	5%	0.00013%	2.70%	0.000418%
2010	20.6%	25%	1.29%	0.00024%	6%	0.00015%	4.70%	0.000728%
2011	21.8%	30%	2.19%	0.00049%	5%	0.00012%	6.40%	0.000991%
2012	25.9%	32%	4.03%	0.00083%	3%	0.00008%	9.00%	0.001394%
2013	30.9%	34%	3.37%	0.00078%	5%	0.00012%	5.30%	0.000821%
2014	31.7%	35%	4.46%	0.00117%	6%	0.00015%	5.50%	0.000852%
2015	32.9%	37%	6.19%	0.00202%	7%	0.00018%	5.78%	0.000895%

6.3.3 Estimating total indium trade quantities

The imports reported under HS 854140 are shown in Table 6.7, alongside the final percentage content estimates derived from above. It shows that overall, the combination of imports plus increasing market share means that only in recent years have the quantities of imported indium in solar cells become significant. Exports are also accounted for in determining net flows in Australia, however exports reported under this particular code are orders of magnitude lower than imports, which are shown for illustrative purpose here.

Table 6.7: Calculations of total indium imported in the form of solar panels under the code 854140

Year	kg of imports reported under HS 854140	% In in solar panels	Total In imported (kg)
1988	206,115	0.0001290%	0
1989	268,028	0.0001330%	0
1990	223,053	0.0001371%	0
1991	212,537	0.0001219%	0
1992	289,389	0.0001066%	0
1993	456,069	0.0000914%	0
1994	284,435	0.0000762%	0
1995	341,300	0.0000609%	0
1996	361,097	0.0000613%	0
1997	324,060	0.0000617%	0
1998	362,777	0.0000688%	0
1999	453,926	0.0000755%	0
2000	560,182	0.0000685%	0
2001	515,852	0.0000756%	0
2002	625,764	0.0000748%	0
2003	777,945	0.0001069%	0
2004	1,071,692	0.0001019%	0
2005	1,886,567	0.0001163%	1
2006	3,020,452	0.0001780%	4
2007	4,195,176	0.0003071%	11
2008	7,786,931	0.0004522%	31
2009	24,048,571	0.0007266%	171
2010	62,710,570	0.0005362%	330
2011	87,838,236	0.0006448%	545
2012	106,939,903	0.0008347%	883
2013	77,632,782	0.0010447%	795
2014	88,930,953	0.0011870%	1042
2015	84,778,634	0.0013429%	1110

These calculations, performed for each commodity, enable the total trade of indium in semi-finished and finished goods to be calculated on an annual basis.

6.4 Analysis and verification of the net imports of semi—finished and finished goods

In replicating the analysis shown in Chapter 6.3 for each of the HS tariff codes shown in Tables 6.3 and 6.4, a more complete sense of how Australia has consumed indium over time is achieved. These results are shown in Figures 6.4 and 6.5, which show the first product-level dynamic MFA results at a national scale for all indium applications. The uncertainty bounds are defined by the absolute range of estimates used for indium contents in trade flows. They are large relative to the magnitude of the values, reflecting the uncertainty and general unavailability and of market share and product content data for many product categories. Nonetheless, these uncertainty bounds are not so large that they restrict conclusions from being made. For example, it is clear that the magnitude of the consumption of indium is significantly lower than that of the amounts exported in concentrates, and which appears in mine wastes. It is also clear which commodities have become more important carriers of indium over time.

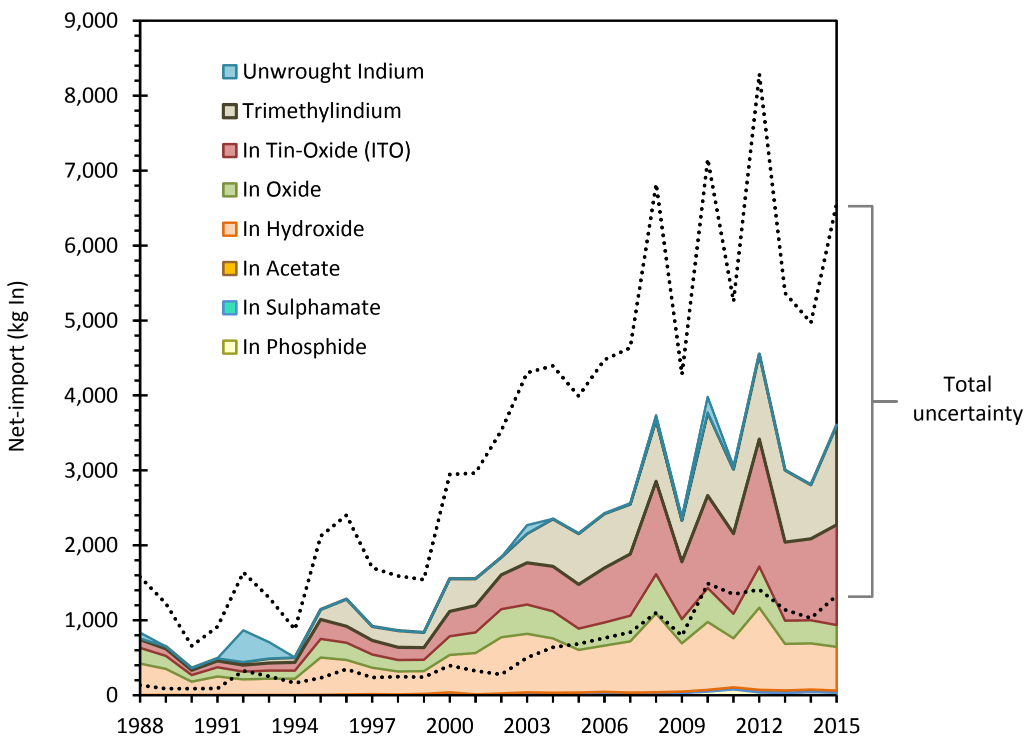

Figure 6.4: The trade of indium in semi-finished goods, 1988-2015.

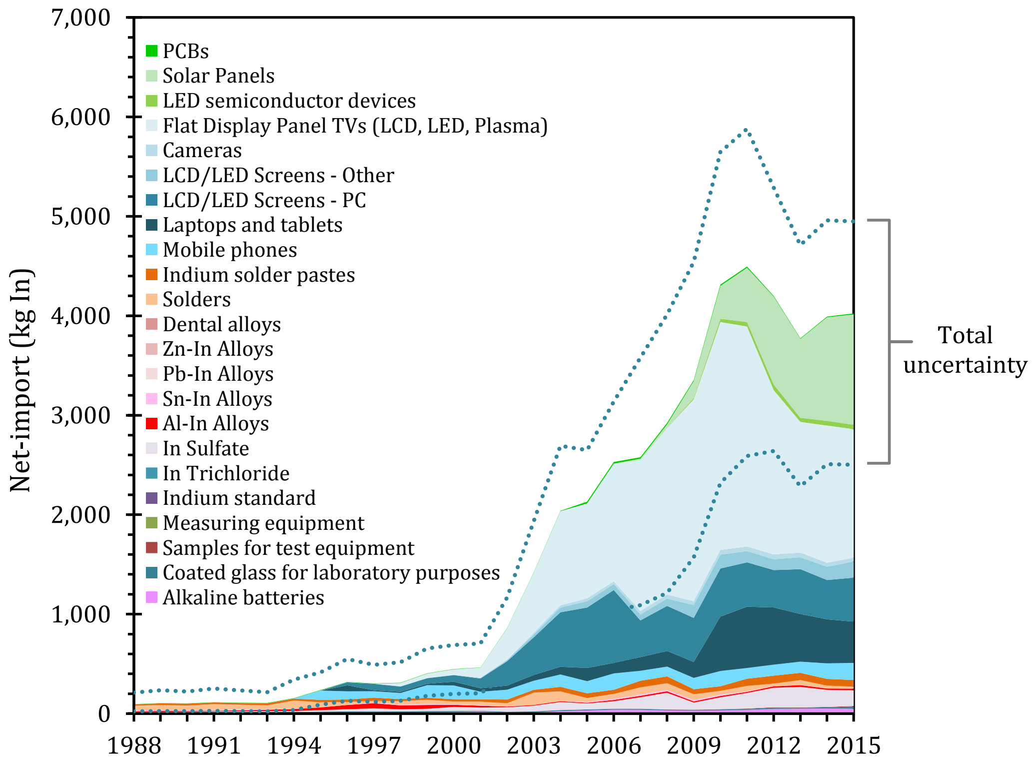

Figure 6.5: The trade of indium in finished goods, 1988-2015.

In some cases, it is possible to cross check that the estimates are reasonable by comparing results to other sources. Figure 6.5 shows that the indium in solar panels has become a major component of net imports in a relatively short period of time. This is at least partly verified by the fact that the number of solar installations recorded during this period has also increased considerably (see Fig. 6.6.).

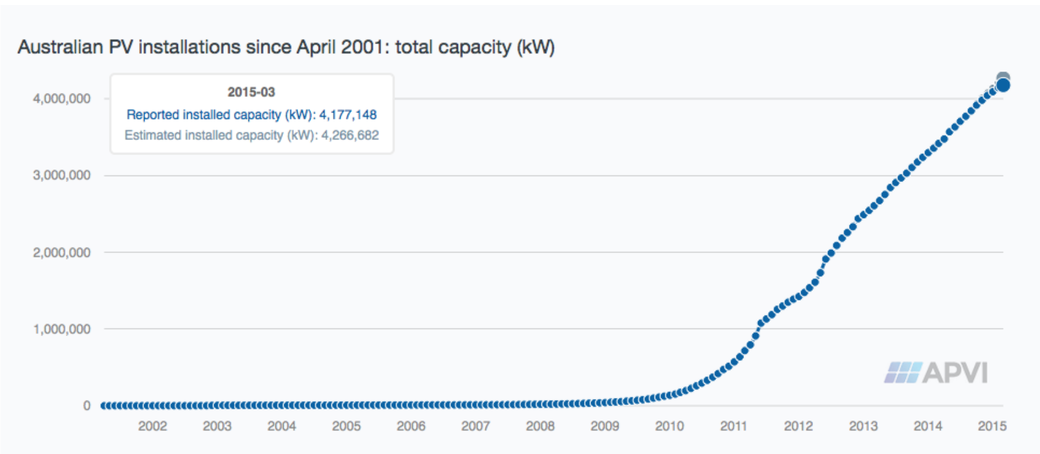

Figure 6.6: Photovoltaic installations in Australia 2001-2015 (Directory, 2015)

A second potential way to very these data are to aggregate the product-level estimate up to general end-use categories, and compare to the end-use statistics available from the USGS. This is done in Fig. 6.7. It can be seen that there is general agreement between the two figures, which show an increasing dominance of indium use in the coatings category. Discrepancies may arise as a result of assigning individual products to a different category, however it is not clear how the USGS have apportioned individual product categories to end-use categories in order to verify this. Nonetheless, the general consistency is somewhat encouraging in that the analysis of Australia is at least in keeping with what would be expected from general trends of consumption, and consumes proportionally what would be expected in comparison to the United States.

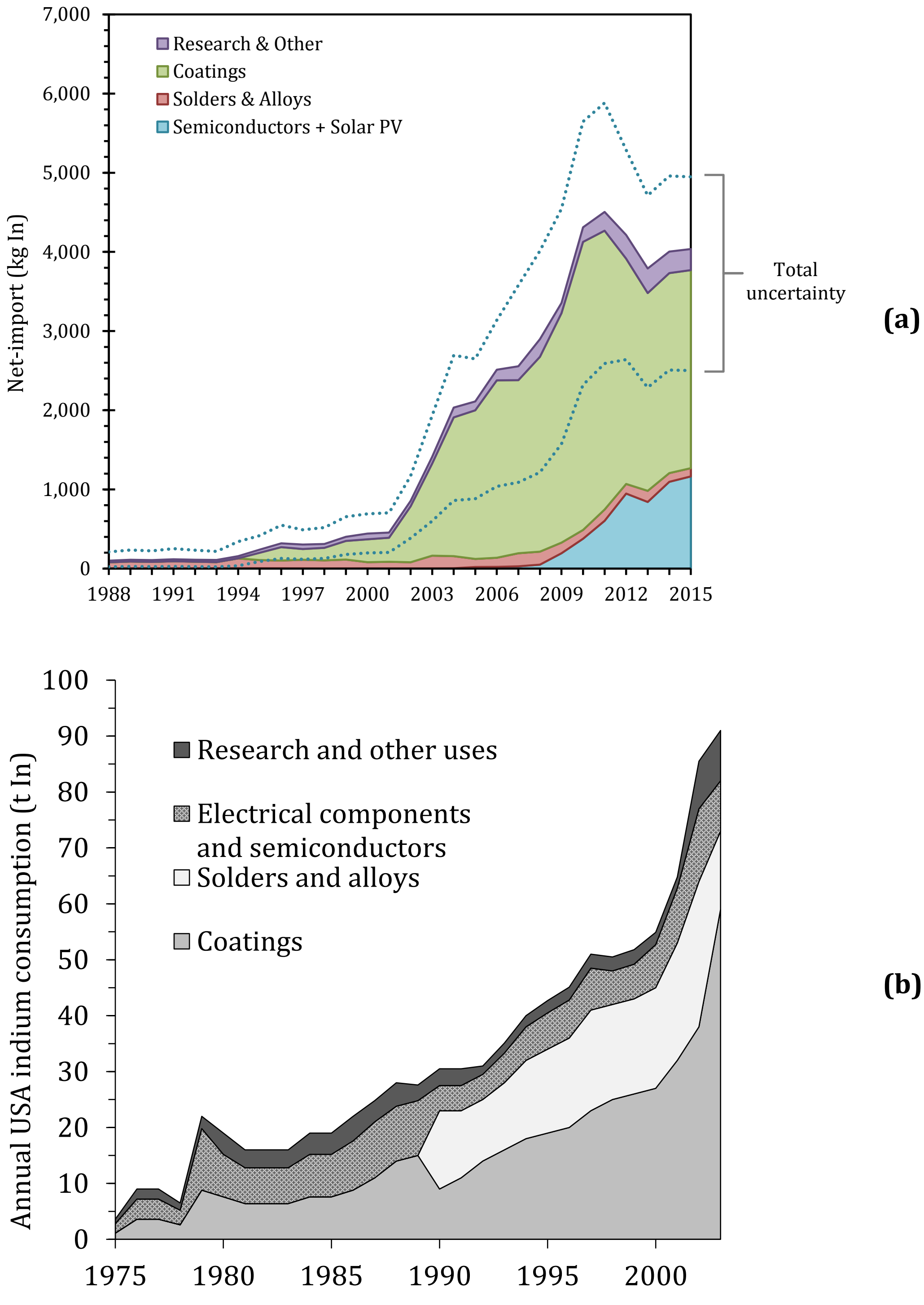

Figure 6.7: (a) Australian consumption of indium per end use sector (this study) (b) End use statistics 1975-2005 in the United States (USGS, 2005), cited in Werner et al. (2015)

6.5 Estimating and verifying in-use stocks and waste flows

Net imports do not directly indicate what stocks of indium are in use, but this must be known in order to estimate the value of recycling as a potential policy direction. The amount of indium which enters use in a given year is represented as the sum of what is imported directly into use, plus that which enters use from domestic manufacturing.

6.5.1 Additions to stocks in use from domestic manufacturing

The quantities of indium directly entering use from net imports of finished goods are already shown in Fig. 6.5, however the indium in semi-finished goods must be processed and manufactured before entering use. The indium imported in semi-finished goods is all assumed to enter domestic manufacturing. Ideally, quantities of semi-finished goods would be matched against actual domestic production statistics of devices which contain indium, however these were not available while conducting this analysis. Instead, a review was conducted to ensure that at the very least, domestic manufacturing capacity does exist for many of the product categories for which the imports of semi-finished goods were recorded (see Table 6.8).

Table 6.8: Australian suppliers and/or manufacturers of indium-containing goods (Ferret, 2016)

Semi-finished product category	Number of suppliers identified
Semiconductors	17
LCD Desktop monitors	17
Alkaline batteries	18
Solders/Solder Pastes	9
Thermal interface materials	3
PCBs	44
Solar panel systems	8

Each of the intermediate indium flows recorded as per Fig. 6.4 were apportioned according to typical usage ratios. It was for example assumed that for each kg of unwrought refined indium ingot that was imported, 70% would go to ITO manufacturing, 16% for electrical compounds/semiconductors, 8% to alloys and 1% to stockpiles (e.g. for military/strategic purposes), as per Licht et al. (2015). This is representative of a global average, and not necessarily the situation in Australia. However, lacking the specific information to make these estimates at the domestic scale, this assumption is necessary. Similarly, the portion of indium ingot attributed to ITO processing is added to direct ITO imports, and then typical processing efficiencies are applied for the use of this sum value of ITO in LCD products, solar panels and architectural glass. With all semi-finished goods allocated to the domestic production of finished goods, domestic manufacturing can be included in what flows into use on an annual basis. Fig. 6.8 shows how semi-finished imports have been apportioned to the production of finished goods in this study.

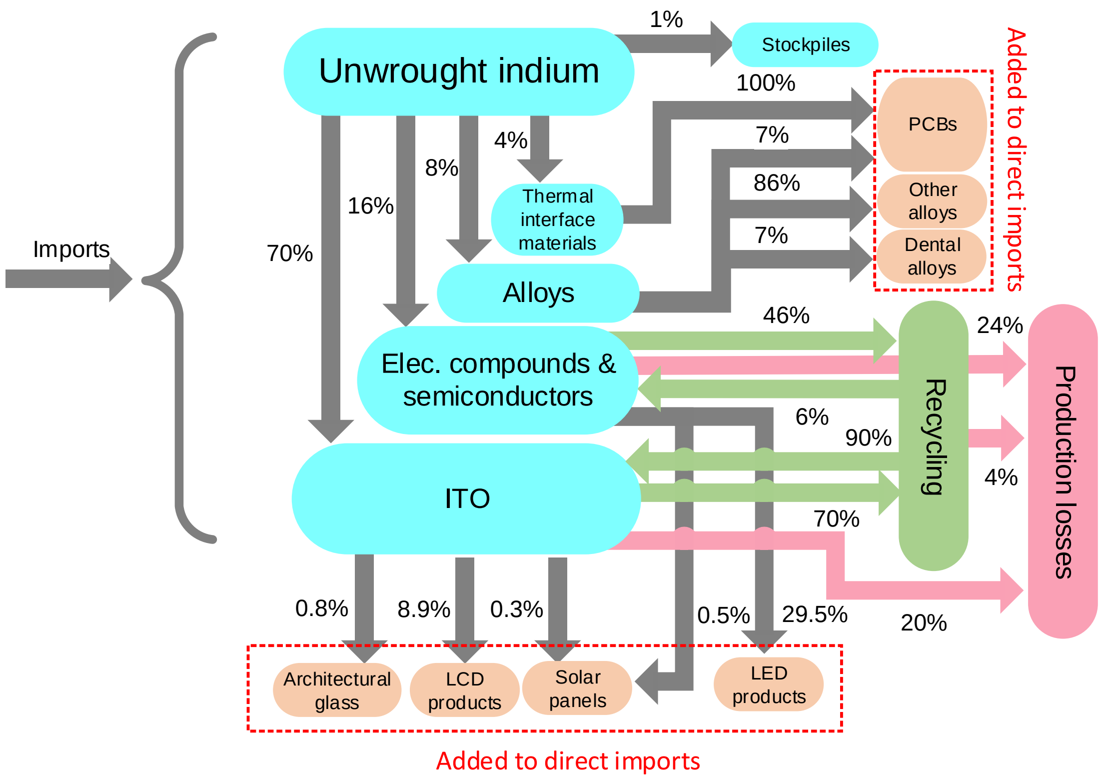

Figure 6.8: Apportionment of semi-finished imports to finished goods, taking into account production efficiencies. Apportionment estimates are derived as an average of Chang et al. (2015), Licht et al. (2015), Nakajima et al. (2007) and Yoshimura et al. (2013)

6.5.2 Calculation of in-use stocks on an annual basis

With the total annual inflows derived from trade and local manufacturing determined, it is possible to calculate in-use stocks on an annual basis. Typically, in-use stocks are estimated by one of two main approaches: top-down, where stocks are essentially estimated as the difference between inflows and outflows; and bottom-up, where stocks are determined as a sum of individual estimates (e.g. the average amount of steel in a building, multiplied by the number of buildings in the study area). These approaches are described in greater detail in Gerst and Graedel (2008) and Chen and Graedel (2015). For the top down method, which is applied here, it is necessary to use lifetime probability functions for each product category. There are a number of ways in which these functions can be derived. The LiVES database, created by the National Institute for Environmental Studies in Japan, is an often cited source of this information (see Oguchi et al., 2010). However, these data are based on average product lifetimes derived from consumer surveys conducted mostly in Japan, and consumer behaviour can differ between countries. Given that mobile phone usage and hoarding behaviour is relatively well studied in Australia, it was possible to create an up to date and spatially relevant function of mobile phone lifetimes for Australia. This was done in a co-authored study not directly included in this thesis, but which is provided in Appendix D (see Golev et al., 2016). Lifetime probability functions specify the probability that a product is still in use a given number of years after it was first imported. Table 6.9 shows the lifetime distribution parameters used in this study. Some product categories which contain indium have not been directly studied for average lifetimes. In these cases, a proxy estimate was necessary, meaning that the lifetime of a similar type of product was applied to these categories.

Table 6.9: Lifetime distribution variables and sources used for the estimation of in-use stocks

Lifespan distribution							Calculations	
End-use	**Reference information**	**Proxy**	**Average lifetime**	**Scale parameter**	**Shape parameter**	**Distribution type**	**Average lifetime**	**Scale parameter**
Laptops + Tablets	Balde et al. (2015)	Laptop - UNU Key 0303		5.2	1.5	Weibull	4.69	5.20
Mobile Phones	Golev et al. (2016)	Mobile Phones		4	2.7	Weibull	3.56	4.00
LCD/LED Screens - PC	Balde et al. (2015)	Flat Display Panel Monitors - UNU Key 0309		7.5	2.5	Weibull	6.65	7.50
LCD/LED Screens - Other	Balde et al. (2015)	Flat Display Panel Monitors - UNU Key 0309		7.5	2.5	Weibull	6.65	7.50
Smaller LCD displays	Balde et al. (2015)	Flat Display Panel Monitors - UNU Key 0309		7.5	2.5	Weibull	6.65	7.50
Cameras	Balde et al. (2015)	UNU Key 0406		8.2	1.4	Weibull	7.47	8.20
Flat Panel TV's	Balde et al. (2015)	Flat Display Panel Monitors - UNU Key 0309		7.5	2.5	Weibull	6.65	7.50
Other Portable Electronics	Balde et al. (2015)	UNU Key 0402		8	0.8	Weibull	9.06	8.00
Cameras for Film <16 mm	Balde et al. (2015)	UNU Key 0406		8.2	1.4	Weibull	7.47	8.20
Cameras for Film >16 mm	Balde et al. (2015)	UNU Key 0406		8.2	1.4	Weibull	7.47	8.20
Motor Vehicles	Harper et al. (2014)	Brass and bronze (vehicles)	12.00		3.5	Weibull	12.00	13.34
Solders and Solder Pastes	Harper et al. (2014)	Solders and alloys	13.40		2.15	Weibull	13.40	15.13
Al/Sn/Pb/Zn and Dental Alloys	Harper et al. (2014)	Solders and alloys	13.40		2.15	Weibull	13.40	15.13
Solar Panels	Zimmerman (2013)	Photovoltaic cells - Note that a min and max of 2.6 and 14.41 for the shape parameters are suggested for a sensitivity analysis. Similarly, a lifespan of 20 to 30 years is a suggested range for sensitivity.	28		5.38	Weibull	28.00	30.37
LED Semiconductors	Harper et al. (2014)	Electrical components and semiconductors	13.40		2.15	Weibull	13.40	15.13
PCBs / Thermal interfaces	Harper et al. (2014)	Electrical components and semiconductors	13.40		2.15	Weibull	13.40	15.13
Domestically produced LCD screens from imported ITO	Balde et al. (2015)	Flat Display Panel Monitors - UNU Key 0309		7.5	2.5	Weibull	6.65	7.50
Batteries	Nomura (2005)	Batteries	10.9		2.2	Weibull	10.90	12.31
Research samples, standards and glass	Harper et al. (2014)	"Other" end-use category	5		0.5	Normal	5.00	2.50

When these product lifetime functions are applied to all the indium trade flows shown in Fig 6.4, in addition to the quantities added from local manufacturing, an annual estimation of in-use stocks of indium is provided. This is shown in Fig. 6.9, which also takes into account the effect of metal dissipation, i.e. the potential for losses to accumulate over time (see, Ciacci et al., 2015).

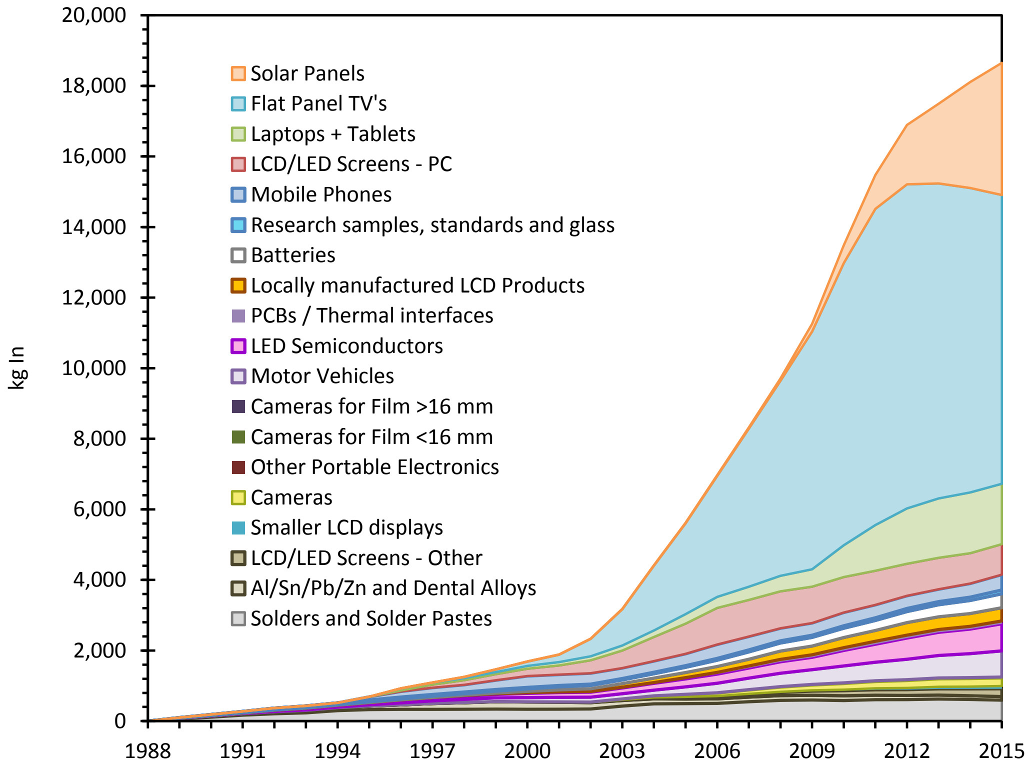

Figure 6.9: Product level accumulation of in-use stocks of indium over time in Australia

To examine the uncertainty of these estimates, it is important to conduct a sensitivity analysis, e.g. by varying the results according to the maximum and minimum possible trade estimates, and by varying the potential product lifetimes as well. At the time of preparing this thesis, such an analysis has not yet been completed, and is expected to be the subject of further study. However, it was possible to somewhat verify these results by comparing them to a bottom-up study of in-use stocks which was co-authored and is presented in Appendix D (Zhu et al., 2017). This study involved the creation of a survey which asked Australian householders of the number of electronic devices they owned. Using the responses to this survey, ownership of electronic devices was then correlated with demographic variables, allowing the number of electronic devices to be mapped at a high spatial resolution. Using the results of studies which have estimated the average contents of indium and other critical metals in electronic devices (e.g. Chancerel et al., 2015), it was possible to map indium stocks among the ten electronic devices surveyed (see Fig. 6.10):

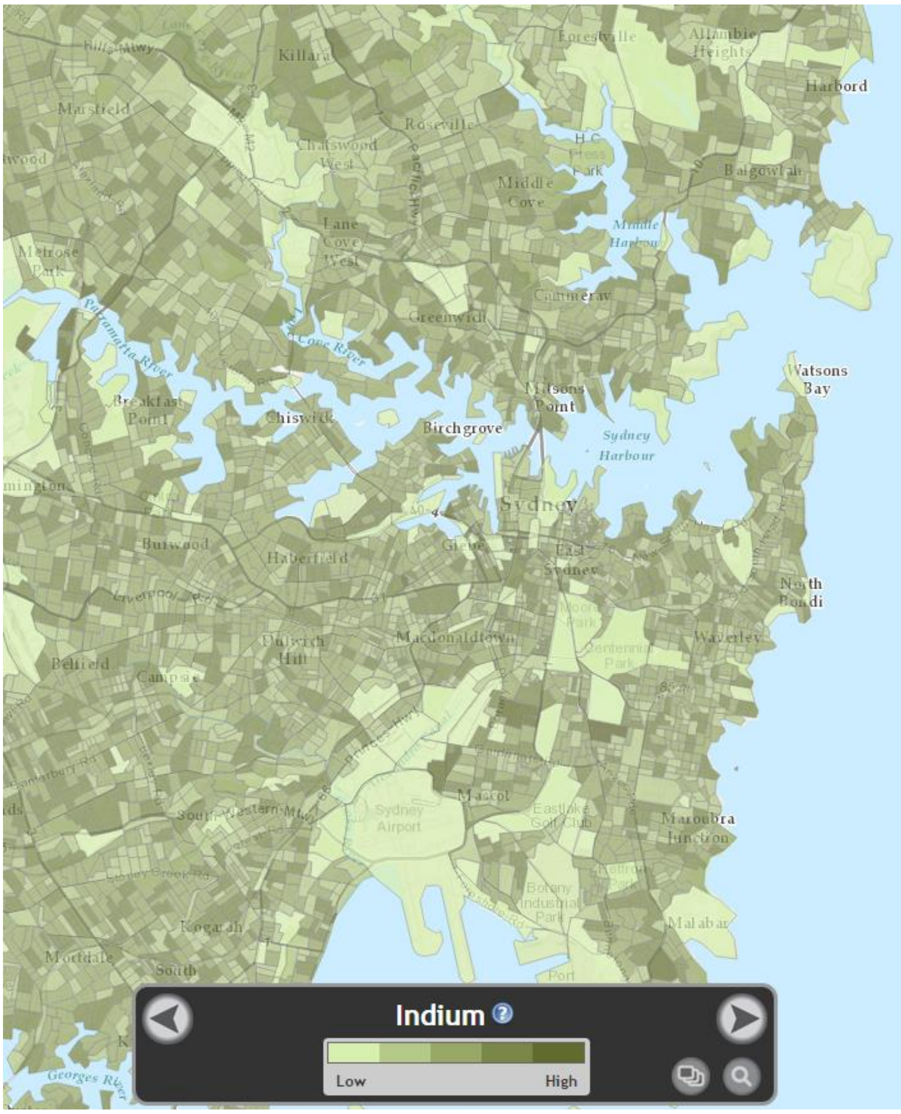

Figure 6.10: Map of indium stocks in the Sydney metro area, derived from a household survey of 10 categories of electronic devices. Image taken from the Recyclable Resources Atlas, developed in conjunction with Zhu et al., (2017).

As four of the same products are assessed, it is possible to compare some results from Zhu et al. (2017) with this study. Using our estimates for in-use stocks in 2011 and comparing those to Zhu et al. (2017), which used 2011 census data, it can be seen in Table 6.10 that there is very good agreement overall. Considering the number of uncertainties in both studies and the large difference between the two approaches, this is a highly encouraging result.

Table 6.10: Comparison of in-use stocks estimates from our study versus that of Zhu et al. (2017)

Product	This study: Top down (kg In, 2011)	Zhu et al. (2017): Bottom-up (kg In, 2011)
Smartphones	365	101
Tablets + Laptops	1295	1539
PC Monitors	967	750
Flat Panel TVs	8959	8921

In analysing the results of our in-use stocks assessment, it can be seen that flat panel televisions have come to dominate. However, indium stocks in these devices have levelled out, and in some cases begun to decrease again. This phenomenon is known as saturation, where an increase in population or affluence no longer leads to significant increases in the stock of a given metal. It may be that given the rapid uptake of LCD, LED and plasma televisions in recent years, Australian consumers have now filled this demand gap in households, and new LCDs are mainly being purchased to replace old ones, rather than CRT type screens. The dominance of indium in-use as flat panel TVs also suggests that any efforts aimed at recovering indium from end of life goods in Australia should target the collection of flat panel televisions. This comes despite the large number of mobile phones in Australia, and noted hoarding behaviour of these devices as is identified in both parallel studies co-authored on this topic (Golev et al., 2016; Zhu et al., 2017).

This study suggests that solar panels are likely to be the major potential source of indium available for recycling in the coming years, however it is very clear that even if Australia were to pursue recycling as an option for critical metal recovery, the amounts that could be obtained are orders of magnitude smaller than even that of a single deposit (see Table 6.1). This is discussed further in Chapter 7. The completed indium cycle is represented in Fig. 6.11.

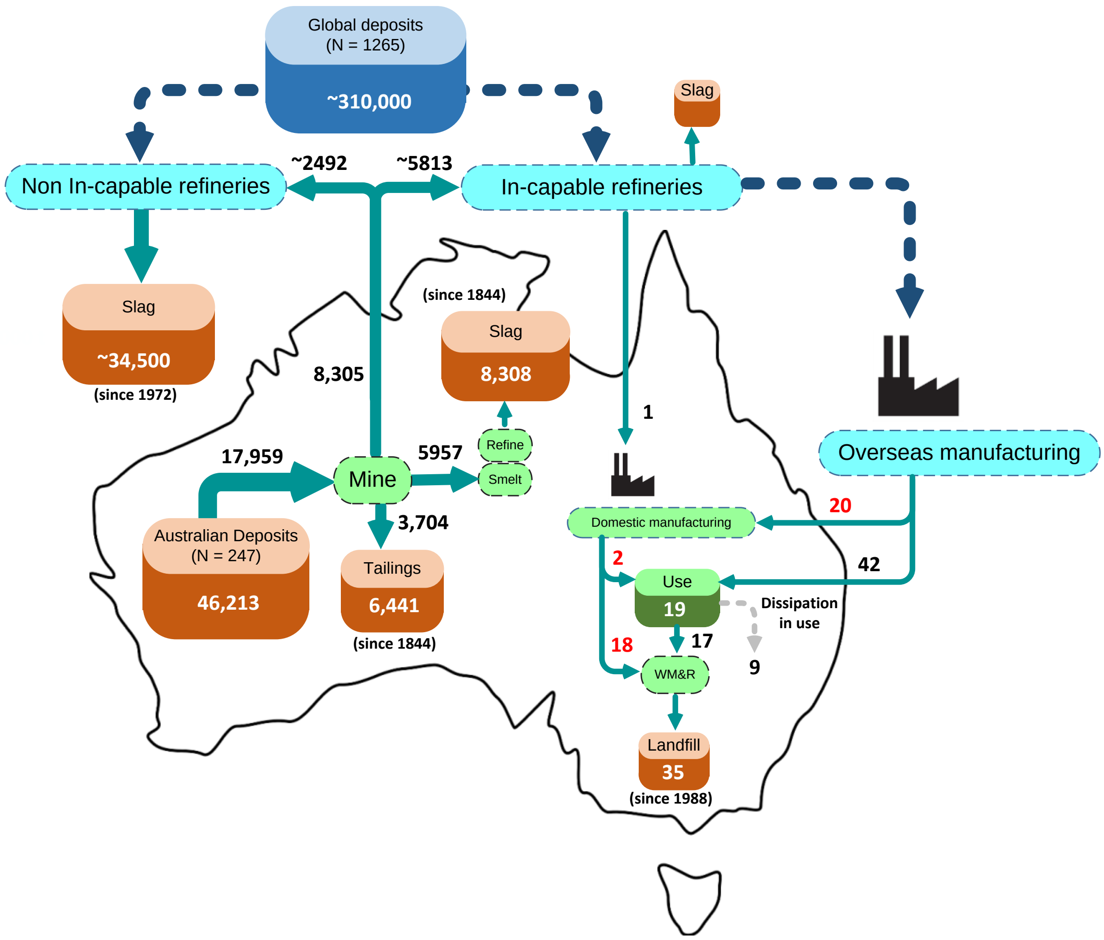

Figure 6.11: Completed indium cycle for Australia, accounting for mining, mineral processing, trade, manufacturing, use and disposal

There are numerous uncertainties embedded into this diagram. Many of the deposit resource estimates are subject to the uncertainties described in Chapter 4. The deportment averages could having significant bearing on what quantities accumulate in tailings and slags, and the market share data and product content are so scarce that often global averages had to be used, which may in multiple cases not reflect the trade situation for Australia. Fortunately however, these uncertainties have little effect on the overall conclusions that are drawn. Although sensitivity analyses are still to be conducted in future work, it is clear that regardless of assumptions on market share and indium content in products, the amount of indium present in use will remain orders of magnitude smaller than that of mineral deposits and mine wastes. At this stage, this study does not venture beyond making broad conclusions such as these. It is also safe to comment that there is virtually no circularity either, as Australia lacks the advanced recycling capacity to recover indium. While studies have rated indium as being highly critical in Australia, the cycle suggests no noticeable efforts to address this. It is recommended that should Australia wish to intervene and reduce its dependence on imports of refined indium, it would be more advantageous to develop capacity in the earlier stages of the cycle, rather than the latter stages. Such a finding runs contrary to what might be suggested from much of the literature looking at indium scarcity and which calls for increased urban mining for indium (Candelise et al., 2012b, Duan et al., 2015). However, as is discussed in Chapter 3, it is also likely to be uneconomic to recover indium from end-of life electronics using current technologies, and given the scale of the end of life resources identified here. The links between this MFA and broader questions on the resources and future supply of critical metals are discussed in the following chapter.

7. Discussions

To briefly recap where this thesis began, it was established in Chapter 2 that many governments and institutions have recognised the importance of critical metal supplies on their futures. To develop effective policies for managing critical metals, the need for detailed data on resources, stocks and flows was highlighted. In response, this thesis has presented a series of related studies examining the resources, usage and supply of indium, demonstrating what can be learnt about other critical metals too. The specific findings of each published study are provided in their respective chapters. As such, the broader themes linking these studies, and the new knowledge that can be drawn from this thesis as a whole are discussed here. In addition, a number of uncertainties and avenues for future research are identified. The discussions that follow are segmented according to the overarching research questions listed in Chapter 1.1.

7.1 On the global resources of indium and other critical metals

The first series of studies published in Chapter 3 had the aim of demonstrating what was previously known about indium, in response to research question 1a. With respect to resources, it was found that large discrepancies existed among recent studies. Indeed, estimates published by highly reputable groups such as the USGS and UNEP ranged between 16,000 and 560,000 t In. This is clearly not accurate enough to glean useful information on the scarcity of this metal, given that global consumption sits at around 800 t In / year. Yet, these highly uncertain estimates remained highly cited, leading to a largely unfounded view of indium scarcity that had made its way into the public domain. Indeed, studies like Cohen (2007) and Moyer (2010) used the very lowest of these estimates to suggest that in only a few years, we'd be running out of many technologically important metals.

If such uncertainty existed for indium, a question was raised as to whether other metals were experiencing this same level of uncertainty, and if so, why? To explore this further, a separate co-authored study (Mudd et al., 2017a) was conducted to review the state of resources and reporting of critical metals more generally. It showed that the resources of many critical metals were equally uncertain, with this uncertainty largely derived from the way resources are reported within the mining industry. Essentially, many mining companies are not paid for the value of potential by-products present in the concentrates they export. Given that the reporting of resources is dependent on economic value to the miner, there are no incentives to report these by-products. This resulted in a perceived view that the quantities of the by-products not reported are not actually present (see Mudd et al., 2017a). Ultimately, identifying that many critical metals are under reported provides great optimism that the resources of many critical metals may be significantly larger than otherwise thought. This finding is consistent with other more recent studies which recognise that the limitations to future critical metal supply are likely to be more related to environmental limitations and supply chain complexities than overall scarcity (e.g. Løvik et al., 2016, Nassar et al., 2015). However, this doesn't mean global mineral resource estimates aren't still needed. Indeed, as discussed in Chapters 4 and 5, detailed resource assessments can provide useful information for resource policymaking, criticality assessments, studies of supply chain dynamics, and can reveal more about geopolitical-related scarcity. Thus, it was recognised that a new approach to estimating critical metal resources was needed which is robust, defensible and repeatable. Furthermore, this method must overcome the specific uncertainties relating to a lack of reporting. In Chapter 4, such a method was developed. It showed that the resources of by-product metals can be estimated using code-based resource estimates, where available, and that proxy data sources (such as geochemical databases) can be used to estimate by-product metal grades. While the data are still generally scarce with respect to indium and other critical metals, this method provides ways for the data that are available to be used for per-deposit resource assessments. In addition, the method provides ways in which the data can be classified according to different levels of quality, and also ways to quantify uncertainty. Fundamentally, this method relies on being able to find statistical correlations between metals which are reported, and those that aren't. If there is no statistical correlation between geological concentrations of primary commodities and their by-products, it is difficult to determine the resources of these by-products. This is an unavoidable weakness of the method that may restrict which metals can be studied in as much detail as has been demonstrated in Chapter 5.

In applying the newly developed method to indium, over 356,000 t In can be attributed to 1512 individual deposits globally, answering research question 1b. This study did not include a comprehensive compilation of deposits with tin as their primary commodity, but nonetheless it suggests that significant quantities exist in economically extractable locations, enough to meet global demand this century. This quantitatively confirms that indium is not scarce, despite being one of the most geologically scarce metals on a relative scale. The primary contribution of this study is not that global resources are significantly more than the USGS estimate, but rather the confirmation that the global resources of a number of by-products can indeed be estimated in detail, permitting the analysis of differences between geological deposit types and countries. It is also possible to understand the grade distribution of indium deposits as well, potentially highlighting which locations are more or less likely to contribute to the indium supply chain in future. Such aspects are likely to be of more practical interest than our estimate of the global endowment of indium.

Indium resources are discussed in greater depth in Chapter 5, however a major finding is that there are significant discrepancies between the distribution of current refined supply, and that of resources. With reference to research question 1c, this suggests that if there was indeed a supply disruption, other countries could move to develop refining capacity elsewhere. Certainly Australia would be well positioned in this respect, having a large endowment of zinc resources and already having domestic refineries which could be adapted to process indium. Notably, countries like Bolivia and India were also identified as potentially becoming major suppliers in future. The analysis of other critical metals, and determination of which countries might be likely to enter the supply chain for these metals is anticipated to be the subject of future work.

7.2 On the quantities of indium from mine wastes

In Chapter 3, deportment studies conducted in the 1980's in Canada, coupled with annual production and ore grade data were used to estimate the amount of indium that could accumulate in two deposits: Heath Steele and Brunswick 6-12. This showed that several thousand tonnes could accumulate in these two otherwise obscure deposits. It is unfortunate that more deportment studies are not available to conduct such estimates on a deposit scale. Knowing how much of the indium that is milled appears in various product or waste streams has enormous bearing on the quantities that accumulate in tailings, and hence might be available for future recovery. This is exemplified from the mere handful of deportment studies identified in Table 6.2, which address research question 2a. Some 5% to 35% of milled indium can appear in tailings depending on the mine. Applying these values to the Broken Hill deposit in Australia (again using historical production and ore grade data) yields results somewhere between 500 and 4000 t In accumulated in tailings over time (see Fig. 12 in Chapter 5). This a broad range, yet still useful in that even the lowest estimate yields a significant amount relative to global consumption. Thus, in response to research question 2a, very large quantities of indium are likely to have accumulated in individual deposits over time, however the data necessary to verify this in greater detail are lacking. There is a very clear need for more research into the deportment of critical metals during mining and mineral processing.

The quantities of indium present in mine and processing wastes were also examined at the national level in Chapter 6, in consideration of research question 2b. Assuming an average proportion of 20% of milled indium deporting to tailings (Table 6.2), and using production and grade data for the mining of zinc and copper in Australia as far back as records are available, it was shown that ~6,234 t In have accumulated in zinc-related tailings and ~207 t In for copper-related tailings (Fig. 6.2). These quantities rival that of what is estimated for slags. Given that Australia's two zinc refineries (Risdon, Tasmania and Townsville, Queensland) do not possess indium refining capacity, and have processed indium-laden zinc concentrates for many years, there have also been significant quantities of indium estimated for domestic slag in the MFA study shown in Chapter 6 (some 8,308 t In). These quantities are particularly large, especially for only two locations, and suggests serious potential for secondary recovery. When it comes to the resources sector, Willis et al. (2012) have found that the dominant barrier to the processing of a by-product is that a company is simply not in the market for that product. As such, determining whether or not a by-product is likely to be processed in future requires some understanding of the ownership of various mines and processing facilities. Interestingly, the zinc refinery at Risdon is currently operated by Nyrstar, who are already involved in the indium supply chain through their facility in Auby, France. In recent company publications, they have also expressed an intention to begin processing for indium and germanium at their Australian facility, highlighting that they may take advantage of these stocks, and allow Australia to enter the supply chain for refined indium in future (see Nyrstar, 2014).

Of course, having the sufficient quantities of by-products, alongside a company with the expertise and market position to process a by-product are not the only factors affecting whether or not a by-product is produced. As highlighted in Mudd et al.

(2017b), the reprocessing of mine and processing wastes can be problematic, as these wastes are often rich in elements which were unrecoverable from the original flotation facility. Cases in which wastes have been reprocessed can be found (e.g. Broken Hill, Australia), although generally these are rare. It has been argued that the mining industry should consider the reprocessing of tailings more seriously, as this is one of the primary ways in which it can contribute to the emergence of a circular economy (Golev and Lebre, 2016). However, as discussed further in Chapter 7.4.1, more research is needed to validate the extractability of critical metals from mine-related wastes.

7.3 On the use and recycling of critical metals in Australia and globally

Despite the very clear findings that global resources of indium will not be a limiting factor to meeting supply for the foreseeable future, indium remains a critical metal. This is largely because its supply remains concentrated in select countries. For other countries concerned about the future supply of indium, options include (among others):

- Fostering the development of new refining capacity domestically, or in other trusted locations,
- Researching substitutes for indium in its primary applications in order to reduce demand,
- Stockpiling to safeguard from future restrictions, and
- Recycling from secondary sources such as electronic goods.

Multiple studies have examined the recovery of indium from electronic waste as a way for countries to become less reliant on imports (Buchert et al., 2012, Cucchiella et al., 2015, Felix et al., 2012, Hagelüken, 2014), with some highlighting that a focus on product design and waste management strategies are virtually essential to managing the future supply of indium (e.g. Pongrácz, 2014 and Duan et al., 2015).

Certainly there are trends in favour of this approach. Notably, the European Union has recently passed legislation known as the waste electrical and electronic equipment (WEEE) directive, which specifies increasing targets for the management and recovery of e-waste for EU member states (European Union, 2016). This means that increasing volumes of indium-containing products are likely to be recovered in future, and Europe has indeed developed capacity for refining indium from these goods (notably at the Umicore facility in Belgium, Rombach and Friedrich, 2014). There is also thought to be significant space for improvement in the recycling of indium, as globally, the end-of-life recycling rate for indium is less than 1%. This also goes for another 19 elements in the periodic table, many of which have been rated as critical (Reck and Graedel, 2012). However, it should be recognised that different metals have different goalposts in terms of what recycling rates should be achieved. In Fig. 7.1, it can be seen that ~80% indium is considered "currently unrecyclable" in the applications in which it is used. This means that with current technologies, efforts to pursue the secondary recovery of indium could at best achieve a 20% recovery of indium stocks in use.

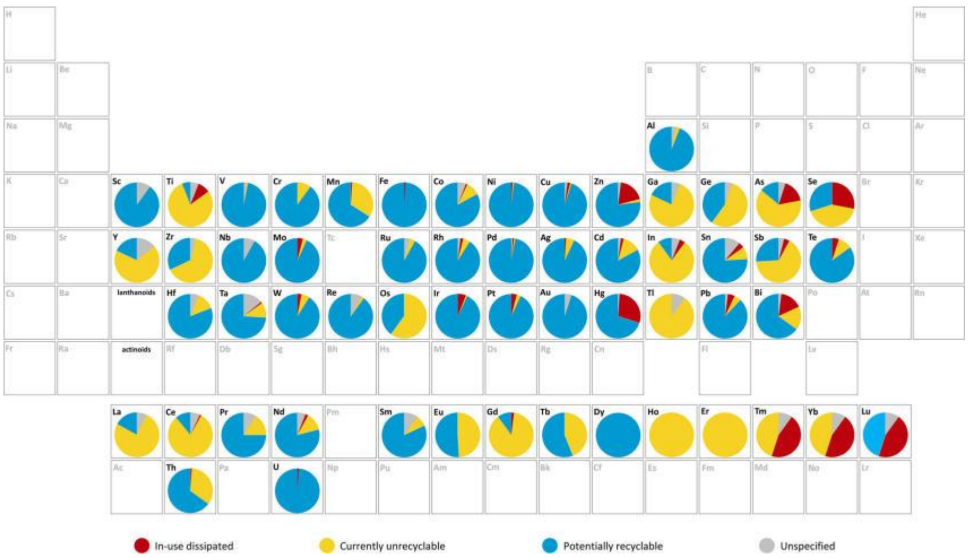

Figure 7.1: Distribution of the elements as per their dissipation and recyclability, from Ciacci et al. (2015)

This raises the clear question of what exactly the stocks of indium in-use are. For Australia, the answer to this question is clearly provided in Fig. 6.9, showing some 19,000 kg In have accumulated in use since 1988 (already taking in-use dissipation into account). It is possible to make some inferences about what stocks might possibly accumulate in other countries from this estimate. Fig. 7.1 shows the per capita stocks of indium over time. It can be seen that the rapid growth in the use of LCD screens over time has led to a quadrupling of in-use indium stocks per capita in the last decade. However, there is a noticeable plateauing of this value, suggesting that Australia may be approaching a saturation of indium stocks.

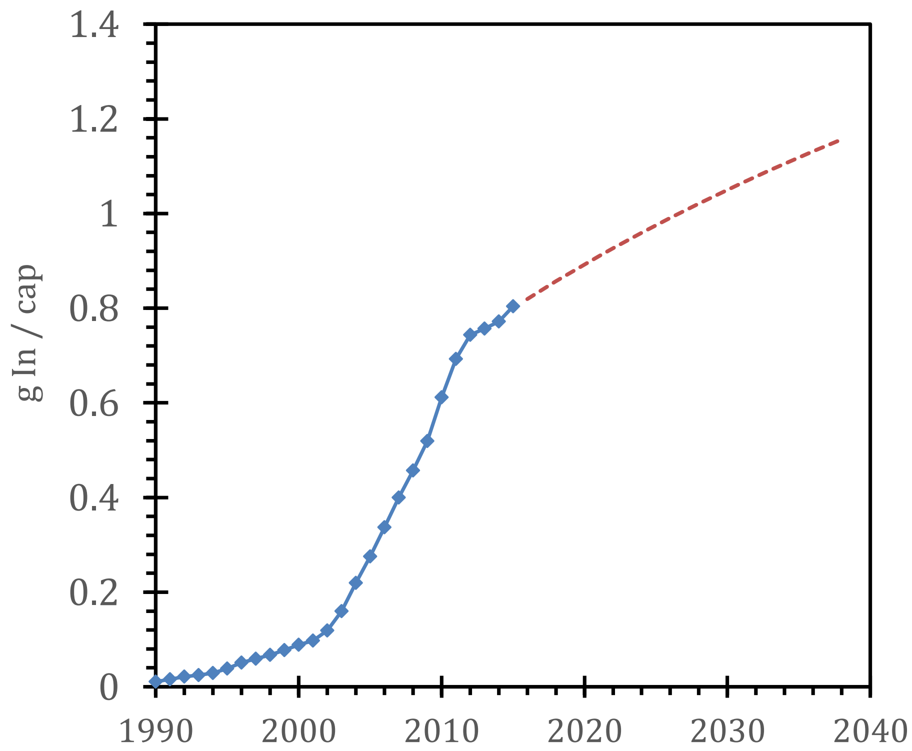

Figure 7.2: Per capita stocks of indium in use in Australia over time, with potential increases if Australia transitioned to 100% solar PV in dwellings following recent trends

This seems dependent on the future of solar photovoltaic technologies in this country, as LCD stocks have certainly exhibited saturation behaviour, but solar PV has exhibited a notable growth trend in recent years. Certainly with solar PV installations becoming cheaper (Candelise et al., 2013), there is reason to believe that in-use stocks of indium will continue to increase in this category. Statistics from the Australian PV Institute (2016) show that nationally, 19.7% of dwellings in Australia have solar PV installed. This suggests that it is possible the proportion of indium attributed to solar panels might peak at five times the current value. If it is assumed that all other product categories are in fact saturated, and indium in solar panels increased to this saturation level, we might see a peak at around 1.2 g In per capita. This projection is shown in red in Fig. 7.2. Given that Australia ranks second in the human development index (Jahan et al., 2015), this might be considered somewhat of a maximum value which could be used as a benchmark for other countries. For example, if China were to reach this ultimate saturation level of 1.2 g In / cap, its current population (~1.382 Billion) would result in a maximum of 1,659 t In in use. Similarly, if the global population at approximately 7.5 Billion were to reach this saturation figure, there would be ~9,000 t In in use across the globe. These dispersed resources would be sufficient to meet perhaps a decade of global demand at most, but as per Fig 7.1, only ~20% of this is currently recyclable, and would still need to be collected, sorted and processed. Disregarding collection losses, 1,800 t In might therefore be extractable from in-use stocks. Ultimately, this global estimate is very small, and in fact comparable to a single indium-bearing deposit (see Werner et al, 2017a; 2017b). This quite emphatically counters the argument that urban mining is a realistic way to secure future indium supplies (addressing research question 3c), at least until new technologies arrive to make the reprocessing of electronic wastes considerably more attractive. It may be that some countries without indium deposits might still consider recycling to meet future needs, but this is highly speculative. Despite the clear finding that recycling is not the ideal solution for future indium supply, there are still many ways in which supply of critical metals can be made more sustainable. Indeed, the concepts of industrial ecology, and the pursuit of closed materials loops can be made across many stages of a metal's life cycle, not just in end-of-life. In the short to medium term, policies relating to indium supply should focus on the sustainability and efficiency of mining as a primary target.

7.4 Additional uncertainties and opportunities for future research

Each study conducted in this thesis is the subject of large uncertainties which have limited the scope of conclusions which could be drawn. For the most part, these have been discussed in the preceding chapters. In addition, the focus of the co-authored Mudd et al. (2017a) study was to describe the uncertainties associated with estimating the resources of critical metal resources. The uncertainties of estimating tailings resources are also discussed in the co-authored Mudd et al. (2017b). In this section, the more general uncertainties faced in developing this thesis as a whole are discussed, with particular reference to how future work may be able to address these. The most frequent and common uncertainties across all studies have related to the limited availability of data on indium. Indeed, many of the studies conducted in the fields of economic geology and industrial ecology have understandably focussed on primary commodities such as steel first, with the research on the resources and material flows of many by-products only progressing more recently. As such, it would be expected that many of the uncertainties and avenues for further research discussed here would also apply to many other critical and/or by-product metals.

7.4.1 In assessing the resources, mining and processing of critical metals

Very little data for the milling, smelting and refining (i.e. deportment) of critical/by-product metals were found to be publicly available. While methods have been developed in this thesis to draw conclusions from the limited data that are available, it is possible to avoid much of the uncertainty of these methods by improving the reporting mechanisms in the mining industry. All the same, given that a new method for estimating by-product metals was established in this thesis, it makes sense for it to be applied to other metals. Specifically, it should be applied to all by-product metals for which there exists a strong statistical relationship between primary commodity and by-product. Such a study for gallium, germanium and rhenium is anticipated to be the subject of future work. A version of the method presented in Chapter 4, incorporating fewer proxies, was co-authored on antimony, gallium and germanium in Australia (Yellishetty et al, 2016, Appendix D).

Less than 1% of the deposits which could potentially report quantities of indium have actually done so (see Chapter 5). It cannot be the case that these 1% are the only deposits which currently play a role in the supply chain of indium. Indeed, many deposits which may be essential sources of existing primary indium supply do not report it, and it is not even clear from the literature reviewed if the mining companies that operate these deposits are aware. As discussed in Chapter 6, it would be of considerable benefit to conduct a study which identifies the specific deposits which are already contributing to the supply chain for indium, so that criticality assessments, and assessments of adaptability in the supply chain could perhaps be more targeted (e.g. by determining the likelihood of a particular company or deposit having its supplies interrupted). In addition, more publicly available databases are needed in order to leverage assessment of by-product metal resources. For example, it is possible to examine the by-products of metals like Cu, Pb-Zn, and Ni (see Mudd and Jowitt, 2014, Mudd et al., 2017b, and Mudd et al., 2013a), but not explicitly those of metals like Sn, Al, Fe and Ti, as comprehensive global databases for these metals are not publicly available and/or regularly maintained. The study of indium presented here is also limited by this, as discussed in Chapter 5.

The uncertainty in estimating by-product metals in tailings mainly comes from limited deportment data. Yet the solution to this may be similar to that of resource data, as the code-based reports listed in Table 6.2 included results of testing to develop the plant setup and examine recovery rates, which ultimately provided the information necessary to determine the deportment values shown. Clearly, there is scope for future research into how mineral resource reporting codes such as JORC or NI 43-101 might be revised in future to reveal more about by-product mineral resources and deportment.

At present, the methods to estimate the tonnage of mine and processing wastes using production data are reasonably good, as studies comparing production-based estimates (e.g. Figures 7 and 8 in Chapter 3) to code-based estimates of tailings tonnage show good agreement (Mudd et al., 2017b). However, there are methodological improvements which could be made in assessing the quantities of by-products in these wastes. Evidently, the amount of critical metals that can accumulate in the form of slag at a refinery depends on the various mines that contribute to it. Each contributing mine will have varying grades of metals shipped in concentrates, and each may be subject to different rates of deportment. As such, it is instructive to examine the relationship between specific mines and processing facilities, such that material flows can become location-specific. From the literature reviewed, this has not yet been studied within the MFA community. Some preliminary efforts have been made to find historical documents which track the relationship between mines and refineries in the zinc industry in Australia (see Fig. 7.3). However, more work is needed to attribute quantities of by-products in these material flows, and to find sources of data that permit more recent years to be studied.

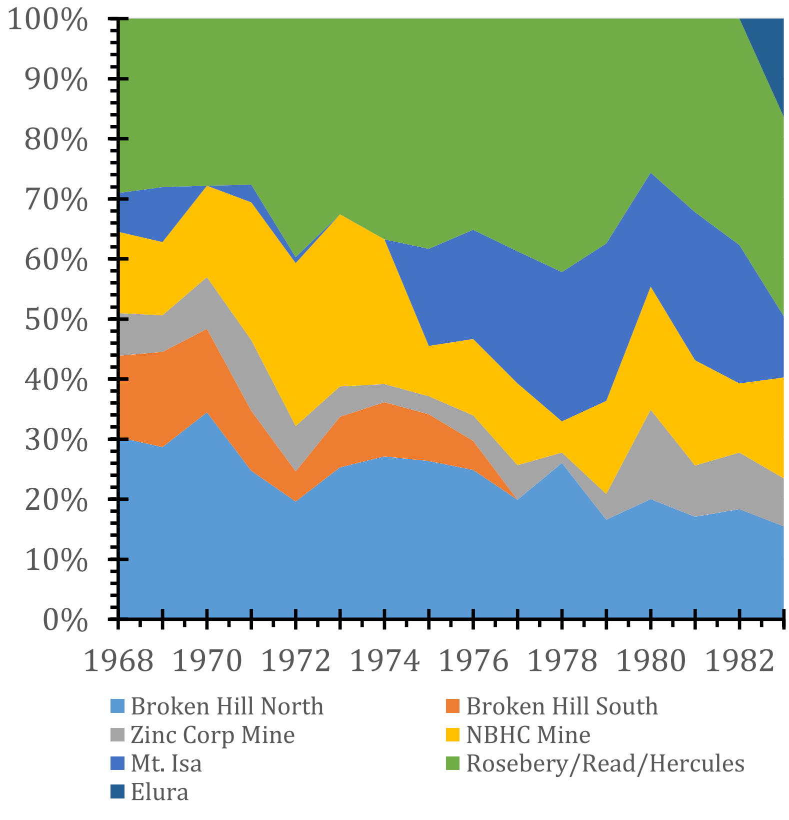

Figure 7.3: Source of Zn concentrates for the zinc refinery at Risdon, Tasmania, 1968-1983. Compiled from the Australian Mineral Industry Reviews (BMR, Var.)

An obvious follow up to the finding that tailings and slags may be plentiful sources of indium (as per Chapters 3, 5 and 6) is to assess how extractable these quantities actually are. A review of processing options and cost/benefit analyses of mine waste reprocessing has been outside the scope of this thesis, however such information would be of crucial importance in developing recommendations to the private sectors and/or policymakers on where best to source indium in future. At best, the findings in this thesis can serve as a strong theoretical basis upon which to explore mine wastes as a potential source in future, however much remains to be studied before this can be fully promoted. A life cycle assessment comparing the extraction of a unit of indium from tailings versus slag versus conventional zinc concentrates appears to be the most practical study to conduct in light of this thesis.

7.4.2 In examining the downstream material flows of indium

Chapter 6 showed quite strongly that the viability of securing future indium supplies through recycling is limited by sheer resource quantities in Australia, and likely at the global level as well. Interestingly, this view was not so strongly held during the earlier stages of this study. Indeed, from the review paper shown in Chapter 3, it was suggested that "*the prospects for indium supply from mine wastes <u>and</u> recycled electronics are substantial"* (Werner et al., 2015). Certainly the many issues surrounding urban mining as a means of securing future indium supply are discussed in Chapter 3, however in retrospect, this statement perhaps indicated too much of a reliance on the idea of a circular economy being broadly applicable to indium. This thesis has shown several cases where achieving end of life material loops is less preferable than improving recovery rates in early stages of processing. This was supported by economic assessments suggesting that recycling indium from e-waste is currently not economically viable (Geyer and Doctori Blass, 2010), and other studies which have shown that recycled indium might in fact have a greater total material requirement (TMR) than primary indium (Yamasue et al., 2009). All the same, quantities and profitability alone do not answer the question of whether or not a potential resource should be extracted. Similar to the assessment of mine waste potential, a life cycle assessment comparison on recycling indium versus primary extraction would be necessary to ensure that all aspects of sustainability are considered in determining the future of indium supply.

While some inferences have been made about the material flows of indium in other countries in Chapter 6, there is a need for other major players in the indium supply chain to be studied in the same detail as is provided for Australia (e.g. the EU and China). In particular, it is worth conducting similar dynamic MFA studies in order to determine if the maximum saturation value of 1.2 g In /capita is also suggested. The use of econometrics can only go so far in linking the results of one study to another. Although strong correlations between indium in use and GDP or population can be found in Australia, these do not necessarily hold true for all countries. At this stage, it appears unlikely that consumer behaviour would differ so strongly between countries, that others would maintain orders of magnitude more indium in use on a per capita basis. However, this is speculative without further study.

8. CONCLUSIONS

This thesis has presented a series of studies examining the resources, stocks and flows of indium in Australia and globally. The research presented herein begun with a review of indium resources, deposit types, history of use, accumulation in mine wastes and urban mining potential. This and a separate co-authored study established a need for further study into how by-product metals can be more reliably estimated. In response, a new method was developed and described in Chapter 4. In Chapter 5, this new method was fully applied to indium to generate a new estimate of global resources. To estimate the stocks and flows of indium in Australia, a retrospective product level dynamic material flow analysis for Australia was conducted in Chapter 6. The primary findings of these studies are summarised here in relation to the research questions listed in the beginning of this thesis:

RQ1: What quantities of indium are available in global and Australian mineral deposits?

(a) What is currently known about global indium resources?

Prior to the research presented here, relatively little had been known about global indium resources. This stemmed largely from limited reporting of indium among deposits which hosted it. Studies ranging from 16,000 to 560,000 t In globally could be found, providing little to no indication of overall scarcity, nor of the proportion of resources held by different countries. Information is available about many individual deposits, however no comprehensive global compilations of these deposits had been

(b) What quantities are present in known mineral deposits?

Over 356,000 t In can be found in 1,512 known mineral deposits. These resources are apportioned between a great number of countries, although Australia, Canada, Russia, Peru, Mexico, China, Bolivia and India account for two thirds of global resources. The majority of indium resources are held in sediment-hosted Pb-Zn deposits, which comprise the equivalent of all other deposit types combined.

(c) What do these quantities suggest about the global supply chain and scarcity?

Given that current global consumption of refined indium sits at around 800 t per year, these quantities are sufficient to meet demand for indium well into the next century. However, there is scope for the distribution of the supply chain to change considerably in this time, as large quantities of indium reside in the mineral deposits of countries not currently refining indium. These countries may however contribute significantly to existing demand through the supply of indium-laden zinc concentrates.

RQ2: What wastes of indium are produced during mining and mineral processing?

(a) How much indium can accumulate in the mine wastes of an individual deposit over time?

Case studies indicate that large mines can accumulate anywhere from a few hundred to several thousand tonnes of indium over time, as anywhere from 5 to 35% of milled indium typically appears in tailings. If these deposits have shipped concentrates to processing facilities without indium processing capacity, as has been done throughout Australia’s extensive mining history, even larger quantities can accumulate in slags. Our dynamic material flow analysis has shown that over 8,000 t In has likely accumulated in the slags of only two zinc refineries over time.

(b) How much indium has accumulated in mine wastes at the national or global scale over time?

It is estimated that 6,441 t In and 8,308 t In has accumulated in tailings and slags in Australia, respectively. These quantities are akin to the mineral resources held by Bolivia, India or the USA. This suggests that for countries with long mining histories, the amount present in mine wastes could rival that of their mineral resource endowment. More research is needed to assess the economic and environmental implications of recovering indium and other critical metals from mine and mineral processing wastes.

RQ3: What are the current stocks and flows of indium in the Australian economy?

(a) What quantities of indium have been imported and exported to/from Australia over time?

An estimated 10,409 t In have been exported in zinc ore and concentrates over time, with only about 35 t In imported. Another 32 t In have been exported in copper concentrates. Since 1988, only 21 t In are estimated to have been imported as semi-finished goods, with another 42 t In imported for end uses. Australia is clearly a net producer of indium, and has been estimated to be the source of nearly 20% of global demand in recent years.

(b) What quantities of indium have accumulated in-use over time in Australia and how do these compare to mineral resources?

Approximately 19 t In have accumulated in use in Australia since 1988. The majority of these stocks are held in various LCD products, however these stocks appear to be reaching saturation. The indium contained in solar panels is becoming increasingly significant, and is likely to be the major component of future increases of in-use stocks. Australia's entire in-use stocks of indium constitute <0.05% of the amount present in Australia's mineral deposits. For comparison, the slag heap at Zeehan, Tasmania, has been estimated to contain 22 t In using a code based methodology, which is 16% more than Australia's in-use stocks.

(c) Is recycling a viable option for future indium supply?

Recycling is not a viable option for primary future indium supply, at least from the perspective of sheer resource quantities. There is simply not enough indium in Australia, or possibly worldwide, to warrant recycling as a serious alternative to the current supply chain. Of course, the recycling of electronic goods may still be warranted for many social and/or environmental reasons. An analysis of the environmental or economic benefits of recycling indium is needed to fully address the viability of this option. In the meantime, it appears that aiming to improve the recovery of indium through mining and refining operations would be a more reasonable avenue.

This thesis began by stating that we live in a period where the demand for materials is greater than ever before. It is in this environment that researchers in the fields of economic geology and industrial ecology have an obligation to learn as much as possible about how to sustainably manage future metal supplies. The studies conducted in this thesis show that despite many data limitations, there is much which can be learned about critical metals, and it is indeed anticipated that the methods and procedures employed in this thesis will be applied to many other by-product/critical metals in future.

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APPENDICES

Appendix A: Supplementary information for Chapter 3

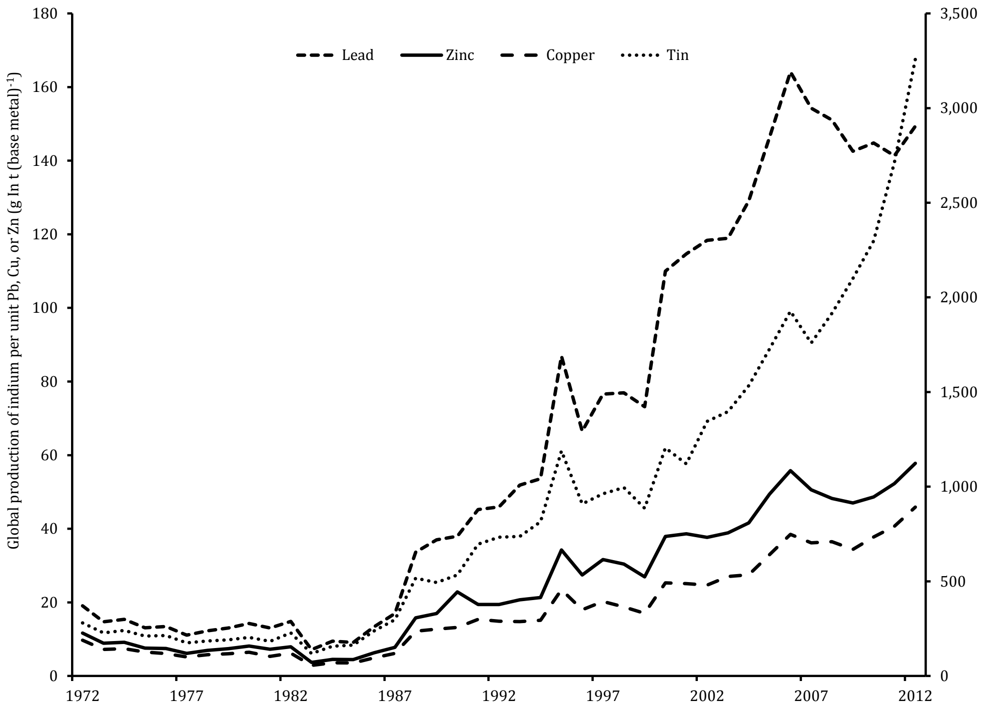

Figure A1: Ratio of indium produced per unit lead, zinc, and copper at a global scale, 1972-2013.

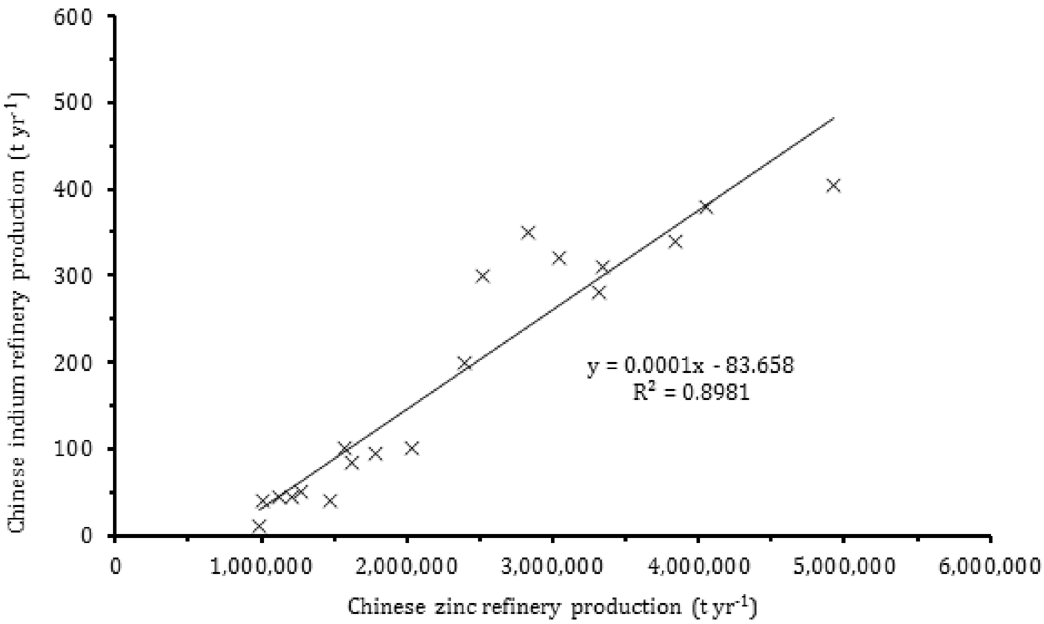

Figure A2: Correlation of zinc and indium refinery production in China, 1994-2012.

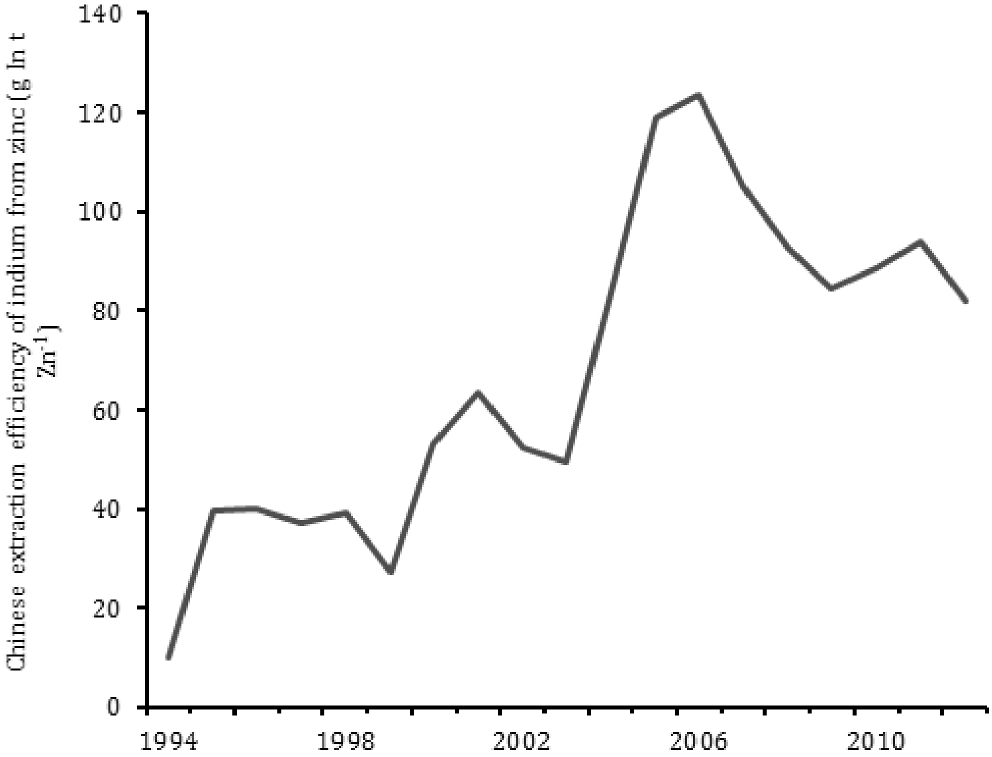

Figure A3: Extraction yield efficiency of indium from zinc in China, 1994-2012

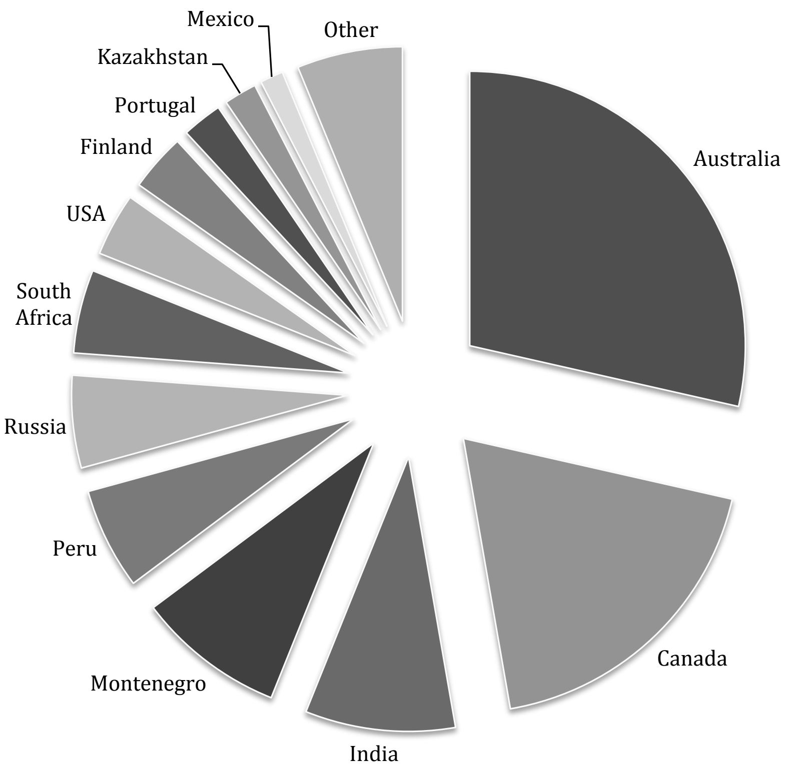

Figure S4: Spatial distribution of future indium potential as determined by magnitude of code-based reporting of zinc resource tonnage as at 2010. "Other" includes Ireland, Algeria, Sweden, Bolivia, Namibia, Brazil, Burkina Faso, Spain, Argentina, Chile, Armenia, Botswana, Turkey, and Honduras. It should be noted that many other base metal producing countries, including China, are not shown due to insufficient information.

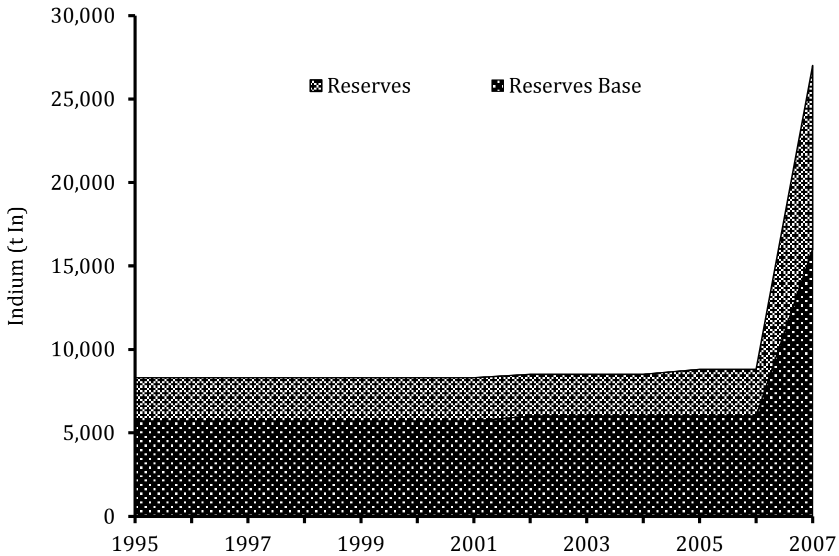

Figure S5: Published USGS estimates for global indium reserves and reserves base (analogous to resources, as per JORC-code classification), 1996-2008.

Appendix B: Supplementary information for Chapter 5

B1: History of the Zeehan slag resource

Mining of the Zeehan field, Tasmania first became active around the late 1880's to early 1890's, leading to the establishment of the Zeehan township itself. The Tasmanian Smelting Company (also referred to as Tasmania Smelters) began operating its Austral Smelter beside Henty Road, just south of the Zeehan township in 1889, and closed in 1913. The site was abandoned by Tasmanian Smelters in 1948 and the slag dumps remain to this date, leading to significant environmental degradation in the Austral Creek and the Little Henty River. The older technologies employed for metallurgical extraction failed to remove some valuable metals such as silver, and were not focussed on the extraction of other by-products such as indium. In 1988, Dragon Resources drilled the slag heaps and estimated approximately $70 million of base metal value but registered no production. The slag is now owned by the metallurgical company Intec Environmental, who was granted a lease to extract 100,000 tonnes from the ~400,000 tonne stockpile in 2010. Some pilot scale extraction and processing has already taken place, though this has not come without controversy, as some locals have felt that the slag dumps represent a part of the history of the region (see Twelvetrees and Ward, 1910; Kidd, 2011; McGregor, 2016).

B2: Selected examples of indium enrichment from ore to concentrates

Table B1: Review of studies used to estimate how indium enriches in concentrates:

[1] Feed grade (ppm In) (Source as per Table S4)	[2] Zn concentrate grade (ppm In)	Enrichment factor [2]/[1]	Location	Reference for concentrate grade.
50	390	7.8	Heath Steele, Canada	Chen and Petruk (1980)
50	243	4.9	Brunswick 6-12, Canada	Petruk and Schnarr (1981)
11.4	320	28.1	Keg, Canada	Silver Range Resources - Keg NI 43-101 Technical Report, 2013
50	270	5.4	Kidd Creek, Canada	Rodier (1990)
18	190	10.6	Neves Corvo, Portugal	Santos et al. (2010)
270	4112	15.2	Mount Pleasant, North Zone, Canada	Amended Adex Mount Pleasant (North Zone) NI43-101 Technical Report, see McCutcheon et al. (2013)
110	330	3	Dulong, China	Tong et al. (2008)
110	307.5	2.8	Dulong, China	Tong et al. (2008)
140	840	6	Toyoha, Japan	Shunso Ishihara et al. (2011)
150	306	2	Maranda J / Romotshidi / Letaba CZ, South Africa	Schwarz-Schampera et al. (2003)
2.5	14.4	5.8	Uchinotai, Japan	Ishihara and Endo (2007)
26	11	0.4	Uchinotai, Japan	Ishihara and Endo (2007)
152	499	3.3	Porco, Bolivia	Shunso Ishihara et al. (2011)
150	584	3.9	Bolivar, Bolivia	Shunso Ishihara et al. (2011)
37	213	5.8	San Lorenzo, Bolivia	Shunso Ishihara et al. (2011)
	Average:	**7.0**		

B3: Additional data and sources for the reported indium database

Table S2 lists the reported but unquantified deposits in full, and indicates the data source utilised to identify that they are indium-bearing. Multiple deposits have been listed in sources such as Schwarz-Schampera (2014, 2014), which in turn cite other studies. These others studies have been named here, however the full references are available in the source study.

Table B2: Other deposits noted for their indium content, but which could not be quantified. Ore tonnage, indium grade, status and deposit type are listed as reported/identified.

Country	Mine site name	Tonnage Mt	Grade gt^{-1} In	Status	Deposit Type	Sources / Notes
Argentina	Cerro Negro District					Identified in the USGS Mineral Resource Data System - citing: "(Deposit:: ALVARADO, BENJAMIN, 1970 , IRON ORE DEPOSITS OF SOUTH AMERICA, IN SURVEY OF WORLD IRON ORE RESOURCES -- OCCURRENCE AND APPRAISAL: NEW YORK, UNITED NATIONS, P. 302 - 380.)(Deposit:: DIRECCION NACIONAL DE MINERIA Y GEOLOGIA, 1991, REPORT ON THE ARGENTINE MINING SECTOR, ARGENTINE SELECTED MINING PROJECT: BUENOS AIRES.)(Deposit:: ANGELELLI, VICTORIO, 1984, YACIMIENTOS METALIFEROUS DE LA REPUBLICA ARGENTINA, VOL. II: BUENOS AIRES, COMISION DE INVESTIGACIONES CIENTIFICAS, P. 393-704.)(Deposit:: HOSKOLD, H. D., 1904, OFFICIAL REPORT UPON THE MINES, MINING, METALLURGY, AND MINING LAWS ETC., OF THE ARGENTINE REPUBLIC: MINISTRY OF AGRICULTURE, COMMERCE AND INDUSTRIES, NATIONAL SECTION OF MINING AND GEOLOGY, BUENOS AIRES, SOUTH AMERICAN BANK NOTE COMPANY, 474P.)(Deposit:: SCHALAMUK, I.B., AND LOGAN, M.A.V., 1994, POLYMETALLIC AG-TE-BEARING PARAGENESIS OF THE CERRO NEGRO DISTRICT, FAMATINA RANGE, V. 32, P. 667-679.}"
Argentina	Gonzalito				Epithermal	Jovic et al. (2015) - Characterized by veins, veinlets, breccias and stockworks. Mineralization is composed of sulphides containing Zn, Pb, Cu, In, Ag, Cu (Sn, Bi, Sb, As) with minor quartz and carbonates as gangue.
Argentina	La Luz				Epithermal	Jovic et al. (2015) - Characterized by veins, veinlets, breccias and stockworks. Mineralization is composed of sulphides containing Zn, Pb, Cu, In, Ag, Cu (Sn, Bi, Sb, As) with minor quartz and carbonates as gangue.
Argentina	Angela				Epithermal	Jovic et al. (2015) - Characterized by veins, veinlets, breccias and stockworks. Mineralization is composed of sulphides containing Zn, Pb, Cu, In, Ag, Cu (Sn, Bi, Sb, As) with minor quartz and carbonates as gangue.
Argentina	Toruel				Epithermal	Jovic et al. (2015) - Characterized by veins, veinlets, breccias and stockworks. Mineralization is composed of sulphides containing Zn, Pb, Cu, In, Ag, Cu (Sn, Bi, Sb, As) with minor quartz and carbonates as gangue.
Argentina	San Roque			Target Outline	Epithermal	Dill et al. (2013)
Australia	Ann			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015) - Geological Survey of Queensland - MINOCC Database
Australia	Arbouin			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015)
Australia	Black Sparkle			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015)
Australia	Khartoum Tin Project			Deposit	Granite-Related / Vein-Stockwork Sn-W	In mineralisation from Aus Mines Atlas - Commodity Search - Geoscience Australia (2015). Auzex yet to produce tonnage or grade estimates.
Australia	Orient Camp East Group			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015)
Australia	Unnamed 167715			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015)
Australia	Weinert			Adandoned		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015). Listed in Qld Industry Publication.
Australia	West Orient			Deposit		Aus Mines Atlas - Commodity Search - Geoscience Australia (2015)
Austria	Ausservillgraten				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available. Refer to INSPIRE (2011) hereafter.
Austria	Bleiberg-Kreuth		1		MVT	In grade from Schwarz-Schampera & Herzig (2002). Also listed on China-Resources.net (Maness, 2010)
Austria	Förolach				MVT	Pimminger et al. (1985), cited in Schwarz-Schampera and Herzig (2002)
Austria	Haufenreith, Kogl (St.Kathrein / Offenegg), Arzberg-Rauchenberg, Arzberg, Kaltenberg-Burgstall				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Hochobir				MVT	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Knappenkar				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Knappenstube (Zwickenberg), Zwickenberg, Strieden				VMS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Lamprechtsberg (Ettendorf),Ettendorf			Old industrial mine, abandoned deposit	Skarn	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Panzendorf				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Pirkachgraben, Pirkach-Hochstadel				MVT	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Stottergraben, Lengholz,Lengholz				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Tessenberg-Thurnbach, Gerichtsbachgraben, Panzendorf-Rain			Old industrial mine, abandoned deposit	VMS/Sed-MS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Austria	Tösens, Platzertal Mine			Old workings	Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Bolivia	Chorolque Mine				Polymetallic Vein-Type	Maness (2010) and Murakami and Ishihara (2013). With W-Sn-Au ore being mined all the way to the peak at 5,600 meters above sea level, Chorolque is one of the highest metal mines in the world. Part of the deposit is in Nor Chichas province. Well-known among Bolivia collectors for its unique specimens of cassiterite casts after crystals of apatite and bismuthinite.
Bolivia	Colquechaca				Polymetallic Vein-Type	Listed in Schwarz-Schampera (2002) and Maness (2010). Indium associated with cassiterite.
Bolivia	Pulacayo Tailings		7	Feasibility	Tailings/Slag	In detected in assays of these tqailings. Reported on the Prophecy website: <http://www.prophecydev.com/prophecys-pulacayo-tailings-piles-samples-assayed-up-to-1200gt-ag-7gt-au-154-5gt-in-indium/>, Accessed Aug, 2015.
Bolivia	Reserva					Listed in Murakami & Ishihara (2013)
Bolivia	San Cristobal		11	Satellite		In grade from Ishihara et al (2011)

Bolivia	San Luis		5		Polymetallic Vein-Type	Listed in the USGS Bulletin No. 1975 (1992) and Legendre (1994), cited in Schwarz-Schampera and Herzig (2002) and Schwarz-Schampera (2014)
Bolivia	Siete Sayos			Active		Listed in Murakami & Ishihara (2013). Active Zn-Pb-Sn-Ag mine between the Animas and Chocaya mines, on the same vein system. Sometimes considered to be part of the Animas mine. NB: The Animas, Siete Suyos, Oploca, and Chocaya mines all worked the same system of veins and have very similar mineralogies. For this reason it is generally impossible to tell which mine a specimen actually came from, and "errors" are common both on collectors' labels as well as in the scientific literature, so many collectors just refer to it generically as the "Chocaya-Animas" mine.
Bolivia	Tatasi			Closed		Listed in Murakami & Ishihara (2013)
Brazil	Mangabeira Massif				Vein-stockwork Sn-W	Listed in Botelho and Rogers (1990), cited in Schwarz-Schampera & Herzig (2002) and Schwarz-Schampera (2014).
Brazil	Ouro Belo			Exploration		Listed online at the Crusader Resources website: <http://www.crusaderresources.com/2011/10/2070/>, Accessed Aug, 2015.
Brazil	Pitinga			Active		Listed in China-Resources.net (Maness, 2010, Unofficial website). This region's major commodities are Bauxite, Lanthanides, U3O8, Rare Earths and Tin. The Geog. Coordinates assigned to this entry are for the Tin entry in the SNL Database.
Bulgaria	Chelopech			Producing industrial mine	Epithermal	From EU ProMine Database see Cassard et al. (2015 - tonnage available but no grades offered for In
Bulgaria	Eastern-Central Sredna Gora / Srednogorie (5 locations)		0.5		Epithermal/Porphyry Cu	Hristova et al.(1986) cited in Schwarz-Schampera & Herzig (2002) and Schwarz-Schampera (2014). Listed Also in Maness (2010)
Bulgaria	Elshitsa			Old industrial mine, abandoned deposit	VMS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Bulgaria	Krassen / Krasen				Epithermal	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available for Indium, but other grades available.
Bulgaria	Madan ore field / Madan district				Low-sulphidation Epithermal- to mesothermal polymetallic-Ag veins	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Bulgaria	Malko Tarnovo ore field (6 locations)				Skarn	Listed in CMH Text and older '02 text.
Bulgaria	Panagjuriste / Panagyurishte (Sredna Gora)		3	Operating	Epithermal	In grade from Hristova et al. (1986), cited in Schwarz-Schampera (2002).
Bulgaria	Radka				VMS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Bulgaria	Vozdol			Subeconomic deposit	Epithermal	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Canada	Eagle	0.9		Exploration Drilling Only	Epithermal	Victoria Gold Website / Technical Report (2015-05) for tonnage. Avino Website. 6 cores drilled, of which 4 showed indium concentrations ranging from 0.1ppm to 89.4ppm. We cannot assign a grade based on the fact that it's epithermal because we have no zinc data for this site.
Canada	Equity Silver		2	Closed	Epithermal	In from Schwarz-Schampera (2002) citing (Yi et al 1995); Taylor (1996)
Canada	Miramichi			Deposit	Porphyry-Sn	Avalon website - last checked 16-Apr-2015. "Early exploration stage" http://avalonraremetals.com/projects/miramichi/ Drilling has taken place to suggest Indium presence, however no code based ppm values available. See Govt of New Brunswick site: http://www.gnb.ca/0078/minerals/pdf/geotin4-e.pdf
Canada	Mount Pleasant (West Zone)			Deposit/Exploration	Granite-Related	Geodex Website - last checked 16-Apr-2015. Few assays picking up In at up to 312ppm in a sphalerite -rich float found in the eastern part of the claim block. For both mount pleasant sites, see extensive list of Mineralogy from Schwarz-Schampera (20'02) - p. 201
China	Bainiuchang		63			In grade from Ye et al (2011), cited in Cook et al (2011)
China	Chahe				Skarn	Listed in Murakami & Ishihara (2013). Sn-Cu Deposit
China	Dabaoshan		253	Operating		In grade from Ye et al (2011), cited in Cook et al (2011).
China	Fankou			Operating		China-Resources.net (Maness, 2010)
China	Gejiu			Reserves Development	Skarn	Listed in Schwarz-Schampera (2002, 2014), Maness (2010). And the USGS Mineral Commodity Data System
China	Hongtuoshan				VMS	Mentioned as In-containing in Schwarz-Schampera (2014). Medium-scale volcanogenic Zn-Pb-Cu massive sulfide (VHMS) deposit, consisting of chimneys, veins, and stratiform ore bodies in the lower and middle parts of the Hongtoushan Formation of the Archean Anshan Group. The Hongtoushan Formation consists of biotite-plagioclase-gneiss and amphibole-plagioclase gneiss with intercalations of felsic gneiss and magnetite quartzite. Ore minerals are mainly pyrite (50%), pyrrhotite (20%-30%), chalcopyrite (1%-10%), sphalerite (1%-15%), and minor galena, cubanite, and chalcocite. The ore minerals occur in masses, breccia, bands, and disseminations.
China	Huogeqi Copper Deposit					USGS Mineral Resource Data System - citing "Porter, Kenneth E."
China	Jinziwo					Listed in Murakami & Ishihara (2013). Cassiterite-sulphide type tin-polymetallic deposit.
China	Jubankeng					Listed in Murakami & Ishihara (2013)
China	Laochang		179	Advanced exploration	VMS	In grade from Ye et al (2011), cited in Cook et al (2011). Also listed in CMH Text and older '02 text. Mnrlgy: Schwarz-Schampera(2002 - citing Yanf and Mo (1993). Not that the indium grade from Schwarz-Schampera & Herzig (2002) was 5 ppm for this deposit. Went with the most recent estimate. Zinc deposit.
China	Qibaoshan		10	Active	Skarn	In grade from He et al. (1984), cited in Schwarz-Schampera (2002). Also listed in USGS MRDS and Schwarz-Schampera (2014)
China	Tonglushan Copper Mine					Identified using the USGS Mineral Resource Data System - citing "Porter, Kenneth E."
Czech Republic	Dlouha Ves, Abertamy, Bohemian Massif		5		Polymetallic Vein-Type	In grade from Schwarz-Schampera (2002). Also listed in the EU ProMine Database - no tonnage or grade avail.
Czech Republic	Jáchymov, St Joachimsthal, Svornost mine, Joachimstal			Old industrial mine, exhausted deposit	Five-element (Bi-Co-Ni-Ag-U) associated (or Co arsenide-polymetallic) veins	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Czech Republic	Kutná Hora / Kutna Hora (Zone de Turkank, Bohemian Massif)		10	Old industrial mine, abandoned deposit	Polymetallic Vein-Type	In from Schwarz-Schampera (2002), citing Hak et al. (1983), Zak et al. (1991). Also listed in the EU ProMine Database
Czech Republic	Plavno Zone		0.5		Skarn	Identified in the EU Promine Database. In grade from Hak et al. (1979), cited in Schwarz-Schampera & Herzig (2002). Also listed in Schwarz-Schampera (2014)
Czech Republic	Pohled, Havlickuv Brod		5		Polymetallic Vein-Type	Identified in the EU Promine Database. In grade from Hak and Johan (1962), Zak et al. (1991), cited in Schwarz-Schampera & Herzig (2002). See also Schwarz-Schampera (2014)
Czech Republic	Pribram Mine, Bohemian Massif		5	Old industrial mine, exhausted deposit	Polymetallic Vein-Type	Dlouha - see Johan (1962) for BohMas and Pohba and Ilavsky (1968) for Pribram cited in Schwarz-Schampera & Herzig (2002). Also listed in Schwarz-Schampera (2014)
Czech Republic	Western Carpathians (Region)					Listed in China-Resources.net (Maness, 2010, Unofficial website)
Czech Republic	Zlaty Kopec polymetallic Tin-Zinc-Indium Project			Target outline		Listed in Appendix 5b of European Metals June 2014 Quarterly Report as a mining tenement interest.

Czech Republic	Žulová / Zulova, Bohemian Massif		52		Polymetallic Vein-Type	Identified in the EU ProMine Database. See Losos et al. (1994), cited in Schwarz-Schampera & Herzig (2002), and Schwarz-Schampera (2014)
Finland	Getmossmalmen				Vein-stockwork Sn-W	Cook et al (2011). See also Schwarz-Schampera & Herzig (2002) and Schwarz-Schampera (2014)
Finland	Jungfrubergen				Polymetallic Vein-Type	Cook et al (2011). See also Schwarz-Schampera & Herzig (2002) and Schwarz-Schampera (2014)
Finland	The Sarvlaxviken Area					Identified in Cook et al (2011). Also studied in Valkama et al. (2016b)
Former Yugoslav Republic of Macedonia	Sasa, Sase			Producing industrial mine	VMS/Sed-MS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Former Yugoslav Republic of Macedonia	Toranica, Ruen, Toranitsa, Svinja Reka, Kozja Reka				VMS/Sed-MS	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Former Yugoslav Republic of Macedonia	Zletovo, Dobrevo				Low-sulphidation Epithermal- to mesothermal polymetallic-Ag veins	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Charrier (2 locations)			Old industrial mine, exhausted deposit	VMS	Listed in Schwarz-Schampera (2014), Maness (2010) and Picot & Pierrot (1963) - Note that the details provided here may correspond to a second from the EU ProMine database. The first deposit first one mentioned in Picot and Pierrot (1963) might be different - as it is a Vein-stockwork Sn-W deposit w' sphalerite.
France	La Telhaie		1		Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) citing Cantinolle et al (1985) - Also listed in the EU ProMine database, though without tonnage or grade.
France	La Villeder			Old industrial mine, abandoned deposit	Beach sands and offshore placers	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Les Bormettes			Old industrial mine, abandoned deposit	Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Les Chaillats, Chaillat			Old workings	Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Les Clochettes		5		Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) - citing Picot (1973) - Also in EU ProMine Databse (No tonnage/grade)
France	Echassières (Les Montmins, Beauvoir, Les Colettes)				Supergene industrial rock and mineral deposit	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Montbelleux - Luitré (District)			Old industrial mine, abandoned deposit	Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Montgros			Old industrial mine, abandoned deposit	Polymetallic Vein-Type	Indium mineralisation associated with "BPGC" (sphalerite, pyrite, galena, chalcopyrite) vein in mafic volcanosedimentary environment
France	Montmirat - Lozerette			Old industrial mine, abandoned deposit	Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	EU Promine Database - see Cassard et al. (2015). Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba
France	Saint Daumas - Pic Martin, Pic Martin				Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	EU Promine Database - see Cassard et al. (2015). - Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba
France	Saint-Martin-la-Sauvet District		1.0	Old small-scale mine, abandoned deposit	Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) citing Johan (1988) - Indium mineralisation associated with "BPGC" (sphalerite, pyrite, galena, chalcopyrite) vein in mafic volcanosedimentary environment
France	Vaulry - Cieux, La Glaieule, Ban?che		5.0	Old prospect	Vein-stockwork Sn-W	In from Schwarz-Schampera & Herzig (2002) citing Cantinolle et al. (1985). Also identified in the EU ProMine Database
France	Vialas			Old small-scale mine, abandoned deposit	Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
France	Yvrandes			Old workings - Lead mine.	Indium mineralisation associated with "BPGC" (sphalerite, pyrite, galena, chalcopyrite) vein in mafic volcanosedimentary environment	From EU ProMine Database - see Cassard et al. (2015). No other resource data available
Germany	Altenberg			Deposit of unknown status	Vein-stockwork Sn-W / Porph	Listed in Schwarz-Schampera (2014) - Critical Minerals Handbook
Germany	Marienberg				Polymetallic Vein-Type	Seifert et al. (1992, 1994, 1999), Seifert et al. (1997), cited in Schwarz-Schampera (2002). See also Seifert (2006) - No tonnage in EU Promine Databse (Cassard et al., (2015). Also listed in Maness (2010) unofficial report from ChinaResources.net
Germany	Oelsnitz / Ölsnitz		1.0		Skarn	In from Schwarz-Schampera & Herzig (2002) - citing Doering et al (1994) and Kaempf (1982). Also listed in Maness (2010) unofficial report from China-Resources.net
Germany	Pechtelsgrün			Dormant deposit	Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Germany	Sauberg Mine, Greifenstein Rocks, Ehrenfriedersdorf				Porphyry Sn deposit (stock or dome-related)	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Hungary	Nagybörzsöny		2.5		Epithermal	In from Schwarz-Schampera & Herzig (2002) - citing Panto and Panto (1972). Also listed in Schwarz-Schampera (2014) and ChinaResources-Net (Maness, 2010). Mnrlgy: Schwarz-Schampera (2002) - Also in EU Pro-Mine Database.
India	Amjhore Mine					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.

Indonesia	Merapi Volcano				Epithermal	Listed in Schwarz-Schampera & Herzig (2002) - citing Kevalieris (1994). And Schwarz-Schampera (2014) and ChinaResources-Net (Maness, 2010).
Ireland	Lisheen	3.8		Operating	MVT	Sphalerite from Lisheen found to be highly indium-enriched - (Wilkinson et al., 2005), citing Marsh et al. (2015). Note: No MVT deposits with reported indium grade enable a good estimation of the grade of indium in this deposit. Must remain unclassified for now.
Italy	Palinuro Seamount		17		V/Sed-MS	In from Schwarz-Schampera (2002, p. 204) - citing Puchelt (1986). Also listed in Schwarz-Schampera (2014), Maness (202) and the EU ProMine Database
Japan	Agenosawa		9		Epithermal	In from Ishihara et al (2006). Said to contain <500t In. Grade from Yoshida (2010) - available online at http://www.researchgate.net/publication/259559983_The_characteristics_of_the_particular_occurrence_of_rare-metals_associated_with_base-metal_deposits_in_the_northern_Tohoku_district_Japan.
Japan	Chichibu				VMS	In from Ishihara et al (2006). Said to contain <500t In
Japan	Goka		20		Epithermal	In from Ishihara et al (2006). Said to contain <500t In. In grade and mineralogy from Schwarz-Schampera & Herzig (2002) - citing Murao and Furuno (1990). Also listed in China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Japan	Hoei				Skarn	In from Ishihara et al (2006). Said to contain <500t In
Japan	Kawayama				VMS	In from Ishihara et al (2006). Said to contain <500t In
Japan	Kuga				Skarn	In from Ishihara et al (2006). Said to contain <500t In
Japan	Miyatamata				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Nakakoshi				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Nishizawa				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Oppu				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Shinkiura				Skarn	In from Ishihara et al (2006). Said to contain <500t In
Japan	Suttsu			Closed - 1962	Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Tada Mine			Closed - 1973	Epithermal	In from Ishihara et al (2006). And China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Japan	Takatori				Skarn	In from Ishihara et al (2006). Said to contain <500t In
Japan	Tatemata				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Yoichi				Epithermal	In from Ishihara et al (2006). Said to contain <500t In
Japan	Akita, Hanawa			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Akita, Koaska			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Akita, Matsumine			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Akita, Shakanai			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Besshi			Past producer		Ishihara & Endo (2007) - Past Producer. Also the site of a Cu Refinery.
Japan	Sazare Mine			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Fukoku		5		Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) - citing Shimizu & Kato (1991)
Japan	Fukuzawa / Fukazawa					Listed in USGS Mineral Resource Data System and China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Japan	Shimokawa			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Hitachi			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Katsutoyo		5		Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) citing Murao et al. (1991). Also listed in Chinaresources.net - Maness (2010) unofficial list. Contains <500t In (Ishihara 2006)
Japan	Kawazu (akaRendaiji) Mine					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Japan	Sangenya					Listed in Schwarz-Schampera & Herzig (2002) - citing Murao et al. (1991)
Japan	Shirataki Mine, Kochi Prefecture			Past producer		Ishihara & Endo (2007) - Past Producer
Japan	Kuroko		10		VMS	In from Schwarz-Schampera (2002, 2014) - citing Murao et al (1991), Kooiman & Ruitenberg (1992)
Japan	Kyushu (5 locations)				Polymetallic Vein-Type	Listed in Schrawz-Schampera (2002) and (2014)
Japan	Yanahara			Past producer		Ishihara & Endo (2007) - Past Producer - "High In values up to 207 ppmwere obtained in Cu-pyrrhotite-rich ores at Yanahara hosted in the Permian rhyolites".
Japan	Seto		5		Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) - citing Murao et al. (1991). Listed also in Ishihara (2006). Also listed in Chinaresources.net unoffical document - Maness (2010)
Kazakhstan	Akchatau			Producer		From the USGS Mineral Resources Data System, citing "Buckingham, David A."
Kazakhstan	Asubalak					USGS Mineral Resource Data System - citing "Alaska Field Operations Center (AFOC)"
Kazakhstan	Kansay		0.5		Skarn	In from Schwarz-Schampera & Herzig (2002) - citing Kulikova (1966). Listed also in Critical Minerals Handbook (Schwarz-Schampera 2014) and China Resources.net (Maness, 2010).
Kazakhstan	Kyzyl Espe, Batystau, Akchagyl, Karagayly (4 Locations)		2		Skarn	In from Schwarz-Schampera & Herzig (2002). Listed also in Critical Minerals Handbook (Schwarz-Schampera 2014) and China Resources.net (Maness, 2010). Mineralogy: Schwarz-Schampera & Herzig (2002) - citing Miroshnichenko (1962)
Kazakhstan	Ognevka					From the USGS Mineral Resources Data System, citing "Alaska Field Operations Center (AFOC)"
Kazakhstan	Sarykan, Kumyschkan, Kurgashinkan (3 Locations)		1		Skarn	In from Schwarz-Schampera (2002, 2014) - citing also Kulikova (1966)
Kosovo	Trepca			Limited production		Listed in China-Resources.net (Maness, 2010, unofficial website) and Eu ProMine Database. Refer also to Kepuska et al (2001) and Kolodziejczyk (2016).
Kyrgyzstan	Sarybulak					From the USGS Mineral Resources Data System, citing "Paidakovich, Matthew E.", "Jenness, Jane E."
Mexico	El Tecolote (Reyna del Cobre)			Target outline		Azure resource drilling report. Available online at: <http://www.azureminerals.com.au/reports/Tecolote-JOGMEC-AzureJointVenture.pdf>
Mongolia	Dajing Tin-Polymetallic Deposits					Listed in Murakami & Ishihara (2013) and China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Mongolia	Mongon Tolgoi Mine / Meng'ntaolegai Ag-Pb-Zn deposit					Listed in Murakami & Ishihara (2013) and China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Pacific Ocean	East Pacific Rise (12°58'N)		7.8		V/Sed-MS	In from Schwarz-Schampera & Herzig (2002) citing Moss and Scott (1996)
Pacific Ocean	East Pacific Rise (16°43'S)		10			In from Schwarz-Schampera & Herzig (2002) citing Moss and Scott (1996)
Pacific Ocean	East Pacific Rise (21°N)		3.5		V/Sed-MS	In from Schwarz-Schampera & Herzig (2002) citing Moss and Scott (1996)
Pacific Ocean	East Pacific Rise (7°24'S)		3.6			In from Schwarz-Schampera & Herzig (2002) citing Moss and Scott (1996)
Peru	Carahuacra				Polymetallic Vein-Type	Listed in both Schwarz-Schampera texts. Refer to Soler, P 1987 work.
Peru	Hercules					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Peru	La Promesa			Exploration drilling		Solitario Website - 17-03-2014. Exploration drilling only. No code based estimates at this stage. Cores from 0 to 430 ppm In.
Peru	Sayapullo			Reserves development	Polymetallic Vein-Type	Listed in both Schwarz-Schampera texts. Refer to Soler, P 1987 work.

Poland	Upper Silesia (Region)					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Portugal	Panasqueira, Barroca Grande			Producing industrial mine	Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Russia	Deputaskoe / Deputatskoe / Deputatsky		5		Vein-stockwork Sn-W	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Ivanov and Rozbianskaya (1961)
Russia	Ege-Khaya					USGS Mineral Resources Data System - citing "Alaska Field Operations Center (AFOC)"
Russia	Ilintas			Producer		USGS Mineral Resource Data System - citing "Paidakovich, Matthew E.", "Jenness, Jane E."
Russia	Kitel / Kitelskoe / Kitelskoye				Skarn	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Smirnov (1989)
Russia	Pitkäranta District				Skarn	See Valkama et al. (2016a). Some data available, indicating In presence in sphalerite, although not enough information to make a resource estimate.
Russia	Kudryavy/Kudriavy/Kudryavyi Volcano, Kuril Islands		20		Epithermal	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Kovalenker et al. (1993)
Russia	Lifudsin				Polymetallic Vein-Type	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Zabarina et al. (1961)
Russia	Mutnovsky, Kamchatka		10		Epithermal	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Lattanzi et al (1995)
Russia	Prasolov / Prasolovskoye		20	Reserves Development	Epithermal	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) citing Kovalenker et al (1993)
Russia	Pravourmiiskoe		30	Operating	Vein-stockwork Sn-W	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) citing Semenyak et al (1994)
Spain	Julio Cesar, Cartagena		2.5		Epithermal	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Oen et al. (1975, 1980)
Sweden	Gasborn		10		Skarn	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources (Maness) List and the EU ProMine Database. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Kieft & Damman (1990).
Switzerland	Binntal / Binn Valley (Region)					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
Switzerland	Lengenbach Quarry, Im Feld, Binnen				Carbonate-hosted deposit	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
Tajikistan	Altyn-Topkan			Operating		Murakami & Ishihara (2013) and China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
United Kingdom	Alston Moor, Alston Block (Nenthead)				Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Brynyrafr			Deposit of unknown status	Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Cardigan-Montgomery			Old small-scale mine, abandoned deposit	Brittle-fault related vein and silicified breccia deposit (sphalerite, pyrite, galena, chalcopyite : "BPGC") and/or F, Ba	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Cligga, Cligga Head				Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Cornwall (2 locations)				Skarn	Listed in Schwarz-Schampera (2014) and Andersen JCØ, Dendle JE, Stickland RJ (2009): Indium in main stage polymetallic mineralisation in West Cornwall. Applied Earth Science 118: 24-25.
United Kingdom	Dolcoath				Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Geevor / Geevor Mine			Old industrial mine, abandoned deposit	Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Great Work Mine / Leeds Mine				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Northern Pennines District				MVT	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Pengenna				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Penhawger				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Redmoor / Red Moor (Cullington)				Vein-stockwork Sn-W	Sn-specialised granite (greisen) and porphyry deposit enriched in indium - no tonnage or grades available in the EU ProMine Database.
United Kingdom	Redruth District				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Saint George (St. George)		1		Sn-specialised granite (greisen) and porphyry deposit enriched in indium	Schwarz-Schampera & Herzig (2002). Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Legendre (1994)
United Kingdom	Sancreed Peurance				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.

United Kingdom	South Crebor				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	St. Agnes		1		Sn-specialised granite (greisen) and porphyry deposit enriched in indium	Schwarz-Schampera & Herzig (2002). Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Legendre (1994) - Mineralogy notes also listed in the EU Promine Database
United Kingdom	St. Michael's Mount /Saint Michael				Vein-stockwork Sn-W	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Thorntwhite / Thorntwaite				Unspecified deposit type	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Tresavean				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Trewethen				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Truro, Wheal Jeanne				Beach sands and offshore placers	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Wheal Jane, Mount Wellington			Old industrial mine, abandoned deposit	Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
United Kingdom	Wheal Mary Consols				Sn-specialised granite (greisen) and porphyry deposit enriched in indium	From EU - ProMine Database - see Cassard et al. (2015) - No tonnage or grade estimates available.
USA	Bay State			Prospect		Indium listed as a secondary commodity in the USGS Mineral Commodity Search Map
USA	Bisbee (Tailings)			Tailings		Found online: (See blog report: <http://arizonageology.blogspot.com.au/2013/08/rare-earth-element-indium-reported-in.html>, viewed)
USA	Black Copper Queen			Prospect		Indium listed as a secondary commodity in the USGS Mineral Commodity Search Map. See also Airzona Department of Mines Report
USA	Campbell			Past Producer		From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Columbus			Past Producer		Indium listed as a secondary commodity in the USGS Mineral Commodity Search Map
USA	Damascus District			Past Producer		From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Ducktown District					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	Dutch Flat District (32 locations)			Past Producer	Stream placer; hydrothermal vein	From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Forest Hill District (26 locations)			Past Producer	Stream placer; hydrothermal vein	From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Gladstone			Prospect		From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Hall Valley Property					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	Iron Hill Mine			Reserves Development		China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	Junction Mine			Past Producer		From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Lady Bug Mine			Past Producer		Indium as Secondary Commodity on the USGS Mineral Commodity Search Map
USA	Leadville District & City			Reserves Development		China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	McLaughlin Gold Deposit			Closed 1996	Epithermal	Listed in Schwarz-Schampera (2002, 2014). Also China Resources.net (Maness, 2010) - unofficial publication. Mineralogy taken from Schwarz-Schampera & Herzig (2002) - citing Rytuba (1992). Closed 1996 - See http://www.portergeo.com.au/database/mineinfo.asp?mineid=mn590. Resource estimate is from 1994- two years before closure. Not known how much was mined after that. Hence values not included.
USA	Michigan Bluff District (61 locations)			Past Producer		From USGS Mineral Commodity Search Map - Indium listed as a Tertiary Commodity here.
USA	Shooting Star					Only qualitatively mentioned at this website: https://arswebsite.wordpress.com/about/
USA	Star Mine (Coeur d'Alene district, Idaho)					China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	Tri-State district mines (Mississippi Valley [type])				MVT	China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.
USA	Twentieth Century Mine			Producer		Indium listed as Secondary Commodity on the USGS Mineral Commodity Search Map
Uzbek and Tadzhik Republics	Karamazar		35		Skarn	In from Schwarz-Schampera & Herzig (2002). Still listed in Critical Minerals Handbook (G. Gunn ed., 2014). Also listed in Maness (2010): China Resources.net and Schwarz-Schampera & Herzig (2002), citing Badalov and Rabinovich (1966)
Vietnam	Nabod Ores (2 Locations: Dien Mine + Phudo Mine)					Ishihara et al (2009): "Preliminary note on indium contents of lead-zinc ores in the northern Vietnam". 59(1), 23～28, 2009 - also listed on China Resoures.net - Maness (2010) - unofficial publication of indium-bearing deposits.

Table S3 provides extra notes and data sources on the individual deposits identified for the “reported” database presented in our study. Once again, multiple deposits have been listed in sources such as Schwarz-Schampera (2014, 2014), which in turn cite other studies. These others studies have been named here, however the full references are available in the source study. The full references of the primary studies employed during the development of the deposit database are provided below.

Table B3: Additional notes on quantified deposits. Status is as reported or identified during the review of literature.

Country	Mine site name	Status	Notes / Sources
High quality			
Argentina	Pingüino	Deposit	Research conducted in 2009/10 but published in Argentex Technical Report - Jan 31 '13
Australia	Avebury Mine - Zeehan (Slag)	Operating	Australian Mines Atlas 2013-09-30 - Data directly from Intec Annual Report (2007)
Australia	Baal Gammon	Slag	Monto Minerals Website 18/03/2014. Mineralisation from McKinnoon & Seidel (88-93), cited in Schwarz-Schampera (2001)
Australia	Conrad - King Conrad	Deposit	Annual Report 2009 (Last report with resource estimates and no extraction taken place since). Deposit type & Mineralisation from Gore et al. (2007) - "Post-rehabilitation environmental hazard of Cu, Zn, As andPb at the derelict Conrad Mine, eastern Australia".
Bolivia	Malku Khota	Deposit	TriMetals Mining Website, viewed 17-03-2014, and Technical Report, available at <http://www.trimetalsmining.com/upload/Technical_Reports/110511_MK_PEA_Update_SEDAR.pdf>
Canada	Mount Pleasant (North Zone)	Deposit	Adex Technical Report Oct '13. Also Mineralisation from this report. See p 6.3: "Greisen vein SW-Sn-W is the same as Porph-Sn"
Canada	Silver Range - Keg (Main Zone)	Deposit	Silver Range Resources Website at: <http://www.silverrangeresources.com/s/SilverRange.asp?ReportID=585668>, viewed 18-03-2014. and NI 43-101 Technical Report
Germany	Geyer Southwest, Saxon Ore Mountains	Deposit	See Deutche Rohstoff publication: http://www.rohstoff.de/wp-content/uploads/2011/10/2009-10-08-Geyer-factsheet.pdf also http://www.rohstoff.de/en/geschaftsbereiche/hightech-metalle-seltene-erden-zinn-wolfram/zinn/
Germany	Tellerhäuser	Deposit	See Saxonore Bergbau statements, available online at: <http://www.anglosaxony.com/projects/germany/tellerhauser> - viewed 16/12/2015. Claims JORC compliant resource estimate.
Namibia	Namib Lead-Zinc Project	Deposit	North River Resources news article availabe online at: <http://www.advfn.com/news_North-River-Resources-Plc-Maiden-JORC-Resource-at_54459136.html>
Peru	Ayawilca	Deposit	Tinka Resources Technical Report 2015-03.
USA	West Desert (formerly Crypto)	Deposit	InZinc Mining Technical Report 2014-05. Updated this option from the Original "Crypto" techincal report from Lithic Resources (which is now InZinc Mining)
Medium quality			
Argentina	Pirquitas	Deposit	In grade from Schwarz-Schampera & Herzig (2002) citing Paar et al. (1998). Tonnage from Silver Standard Resources Annual Information Form 2013
Australia	Balcooma - Polymetallic/Zinc	Deposit	Australian Mines Atlas + Kagara 2011 Annual Report. In grade from Schwarz-Schampera & Herzig (2002)
Australia	Balcooma -Copper	Deposit	Tonnage and grade from Australian Mines Atlas + Kagara 2011 Annual Report
Australia	Broken Hill (Main)	Operating	Tonnage from Perilya 2012 Annual Report. In grade from Schwarz-Schampera & Herzig (2002)
Australia	Broken Hill (Rasp)	Operating	In grade from Schwarz-Schampera & Herzig (2002). Tonnage from Australian Mines Atlas - Based on CBH Resource Report '09. Toho Zinc not publishing in Annual Reports.
Australia	Dry River South	Operating	In grade from Schwarz-Schampera & Herzig (2002) - citing Huston et al. (1995). Tonnage from Australian Mines Atlas + Kagara 2011 Annual Report
Australia	Mt Chalmers	Historic Mine	In grade and mineralogy from Schwarz-Schampera & Herzig (2002). Tonnage from Echo Resources 2010 Annual Report
Australia	Rosebery incl South Hercules	Operating	Tonnage from MMG Annual Report 2013. In grade from Schwarz-Schampera & Herzig (2002) citing Huston et al. (1995)
Australia	Waterloo	Deposit	Australian Mines Atlas + Kagara 2011 Annual Report. Also listed in Schwarz-Schampera (2002, 2014) - citing Huston et al. (1995)
Bolivia	Bolivar	Operating	In grade from Murakami & Ishihara (2013). Tonnage from Sierra Metals Technical Report. (2013). Also listed in China-Resources.net (Maness, 2010). Mineralogy from USGS. Bulletin No. 1975 (1992) & Roskill Info Services (1996)
Bolivia	Porco	Operating	In grade from Murakami & Ishihara (2013). Tonnage from Glencore-Xstrata Reserves-Resources Report 2013. Estimate reduced from 997 t In which was published in Murakami & Ishihara (2013)
Bolivia	San Vicente	Operating	In grade from Schwarz-Schampera & Herzig (2002). Tonnage from 2013 Pan American Silver Annual Information Form. Further information on this deposit available at USGS Bulletin No. 1975 (1992) and Kitakaze et al. (1983)
Canada	Brunswick 12-Bathurst	Closed 2013	Tonnage as at 31 Dec '12 (4 months before close) - Xstrata Reserves-Resources Publication. Mineralisation from p. 174 of Canadian geology pdf files (citing Chen & Petruk 1980). In hosted in Sph and Cp
Canada	East Kemptville (Main and Baby Zones)	Deposit	Avalon 2013 Financial Report p.55. http://www.avalonraremetals.com/news_media/display/index.php?id=12341. Published Oct 31 2014. Indium grades not published in NI 43-101 estimate, but weighted average drilling results used for medium quality estimate.
Canada	Kidd Creek	Operating	In grade from (Hannington 1999) - cited in Schwarz-Schampera & Herzig (2002). Tonnage from Xstrata Reserves-Reseources 2012 "Supplied up to 25% of world production as at 1999 (Herzig 1999)" Known In producer. See USGS (2004)
Czech Republic	Cinovec	Old mine workings	In grade partially from Schwarz-Schampera & Herzig (2002) - cited also in Murakami & Ishihara (2013). Listed in China-Resources.net (maness, 2010). See also Novak et al (1991). Further grade data from EU Promine database. Tonnage available online from: <http://www.brandonhillcapital.com/media/52002/eminitiationnotemastercopy.pdf>
Papua New Guinea	Eastern Manus Basin - Solwara 1	Pre-production / Active	In from Schwarz-Schampera & Herzig (2002). Tonnage from Nautilus Minerals Annual Report (2013) - Broader Eastern Manus Basin listed in Critical Minerals Handbook: Schwarz-Schampera (2014) and Schwarz-Schampera & Herzig (2002) - citing Binns and Scott (1993), Parr and Binns (1997). Solwara 1 and 12 are in the EMB. See further online <ftp://rock.geosociety.org/pub/reposit/2011/2011342.pdf>
Papua New Guinea	Eastern Manus Basin - Solwara 12	Pre-production / Active	In from Schwarz-Schampera & Herzig (2002). Tonnage from Nautilus Minerals Annual Report (2013) - Broader Eastern Manus Basin listed in Critical Minerals Handbook (Schwarz-Schampera, 2014) and Schwarz-Schampera & Herzig (2002). See also Herzig (1999), Binns and Scott (1993) and Parr and Binns (1997). Solwara 1 and 12 are in the EMB. See furhter online at: <ftp://rock.geosociety.org/pub/reposit/2011/2011342.pdf>
Peru	Cerro de Pasco	Operating	Originally operated 1955-1976. SNL Database suggests operating again. In grade from Schwarz-Schampera & Herzig (2002). Tonnage from Volcan Compania Minera 2013 Annual Report. See further in Petersen (1965), Putzer (1976)
Peru	Morococha	Operating	In grade from Schwarz-Schampera (2002) - citing Petersen (1965), Soler (1987). Tonnage from Pan American Silver Corp NI 43-101 Technical Report (June '14).
Portugal	Neves Corvo	Operating	In grade from Gaspar and Pinto (1992), Leistel et al. (1998), Schwarz-Schampera & Herzig (2002). Tonnage from Lundin Mining Reserves-Resources report 2013.
USA	Bingham Canyon / Kennecott Copper Mine	Operating	Grade and mineralogy from Rose (1967) & Sawkins (1990) cited in Schwarz-Schampera & Herzig (2002) Note that Briskey (2005) find average of 11ppm from 42 samples of Chalcopyrite. Tonnage from Portergeo - JORC compliant mineral resource statement as at 31 Dec 2011. Also contains Molybdenum at 0.20 ppm Reserves and 0.12 ppm Resources

USA	Santa Rita		In grade from Rose (1967) , cited in Schwarz-Schampera &Herzig (2002). Referred to as Central District. Tonnage from Portergeo consultants (online) - JORC compliant mineral resource statement as at 31 Dec 2011.
Australia	Nightflower	Deposit	Indium grade calculated via Proxies.Tonnage from Australian Mines Atlas - citing "2008 ASX (Australian Stock Exchange) Announcement"
Australia	Salt Creek	Deposit	Indium grade calculated via Proxies.Tonnage from Venturex. Annual Report 2013. High Grades listed, but not with code data. See: http://www.venturexresources.com/investorrelations/Released/Spectacular%20Zinc-Lead-Silver%20Grades%20at%20Salt%20Creek,%20Pilbara%20VMS%20Project%20-%2030-6-10.pdf
Bolivia	Pulacayo (Paya)	Feasibility	Indium grade calculated via Proxies.Listed in Fesser (1968). Tonnage from Apogee Silver Technical Report 2013-01
Canada	Akie	Reserves Development	Indium grade calculated via Proxies. Tonnage from Canada Zinc Metals Technical Report (Mar-2012). This report suggests that the area is enriched in indium, however no grades are provided.
Canada	Horne No. 5	Reserves Development	Indium grade calculated via Proxies. Listed as being indium-enriched on China-Resources.net (Maness, 2010). Tonnage from Falco Resources Technical Report (2014-05).
Canada	Keno Hill - Onek	Operating	Indium grade calculated via Proxies. Tonnage reported online from Alexco Resources website: <http://www.alexcoresource.com/s/news.asp?ReportID=510307> , <http://www.alexcoresource.com/s/news.asp?ReportID=510314>
China	Qinghai Deerni	Operating	Indium grade calculated via Proxies. Tonnage from Zijin Mining Annual Report 2013. Mentioned as In-containing in Schwarz-Schampera (2014).
Peru	Santander Project	Operating but w' unexploited In	Indium grade calculated via Proxies. Tonnage from Travel NI 43-101 Technical Report (2012). Doesn't include Indium, but drilling has found evidence of some enrichment.
Portugal	Lagoa Salgada	Reserves Development	Indium grade calculated via Proxies. Tonnage from Portex Minerals NI 43-101 Technical Report (Jan 2012). Indium suggested in Schwarz-Schampera (2014 and Oliviera et al (2011) - noting "Academic papers on the deposit show readings of Indium as high as 90 ppm (or 90g/t)" -
United Kingdom	South Crofty, East Pool and Agar Mines	Old industrial mine, abandoned deposit	Indium grade calculated via Proxies. Listed in the EU - ProMine Database. Tonnage from Celeste Copper Technical Report (October 2012)
		Low quality	
Australia	Isabel	Adandoned	Aus Mines Atlas - Commodity Search - Geoscience Australia (2015). Listed in Qld Industry Publication. Quantities not listed as JORC compliant, but dated to 2014. Described as a pre-JORC resource that, on its own, will "never be mineable" by QLD Gov't (pers. comm.)
Azerbaijan	Filizchay / Filizchai		Azerbaijani Geographical Society (2016) lists In as a potential commodity. See <http://gsaz.az/en/articles/view/92/mineral-resources-of-azerbaijan>, viewed 19 June 2016. Location approximated to the Balakan region of Azerbaijan.
Bolivia	Carguaicollo	Operating	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Matthews (1991)
Bolivia	Colquiri	Limited Production	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Campbell (1947
Bolivia	Huari Huari		In from Murakami & Ishihara (2013). Also listed in China-Resource.net (Maness, 2010, Unofficial publication) and Schwarz-Schampera & Herzig (2002)- citing USGS Bulletin No. 1975 (1992), Roskill Info Servies (1996)
Bolivia	Potosi	Operating	Murakami and Ishihara (2013)
Bulgaria	Elatsite/Elacite		In from Schwarz-Schampera & Herzig (2002) - citing Hristova et al. (1986)
Canada	Geco/Manitouwadge	Closed	In from Schwarz-Sachampera (2002). Tonnage from Galley et al 2007 (based on Hannington et al 1999) . Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Cabri et al. (1985) & Petersen (1986)
Canada	Heath Steele	Closed	In from Schwarz-Schampera & Herzig (2002) citing Chen & Petruk (1980). Republished in Murakami and Ishihara (2013) - Note that grades of other metals is taken from the most recent grades (i.e. 1999) in published production data. Production from Canadian minerals yearbook was used to determine a decline in ore grades at this site, reflecting more recent updated grade estimate for indium.
Canada	Silver Queen (Cole Lake)	Reserves Development	In from Schwarz-Schampera & Herzig (2002) - citing Eckstrand et al. (1995). Non-code tonnage- Tech Report 1996-02
Canada	Silver Queen (Wrinch)	Reserves Development	In from Schwarz-Schampera & Herzig (2002) citing Eckstrand et al. (1995). Non-code tonnage- Tech Report 1996-02
Canada	Sullivan-Kimberley	Closed - 2001	Non code-base data obtained from Mineral Depopsits Canada (Goodfellow & Lydon, 2007). In from Schwarz-Schampera & Herzig (2002) citing Yi et al (1995). Reserves "exhausted" and mine closed in 2001.
China	Dachang	Reserves Development	In from Murakami & Ishihara (2013). Also listed in the Critical Minerals Handbook (Schwarz-Schampera, 2014) and Schwarz-Schampera & Herzig (2002)
China	Dulong	Reserves Development	In from Murakami & Ishihara (2013) - See also Tong et al. (2008) for flotation information at this site and Qian (1998)
Czech Republic	Tisova	Past producer. New exploration	Testing reveals a pre-NI 43-101 resource estimate. See Canadian International Minerals website: http://www.cin-v.com/tisova-cu-co-vms-czech-republic.html, viewed 9 January 2017.
Georgia	Dambludi		Tvalchrelidze & Morizot (2003). Mineral Resource Base of the Southern Caucases.
Germany	Freiberg	Industrial wastes (slags, bottom ash)	In from Schwarz-Schampera & Herzig (2002), republished in Murakami & Ishihara (2013). Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Seifert (1994; 1999), Seifert et al. (1997)
Germany	Freiberg (Tailings)	Tailings	In content from Martin et al. (2014)
Germany	Rammelsberg	Closed	In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014). Mineralogy from Schwarz-Schampera & Herzig (2002) citing Kraume (12955), Wealter (1986), Sperling (1986)
Germany	Pöhla-Globenstein		See Table 36 from Hösel (2003), cited also in Andersen et al. (2016). Also listed in the EU ProMine Database
Greece	St. Philippe / Agios Philippos Mine / Kirki Mine		In from Schwarz-Schampera & Herzig (2002) also listed in Critical Minerals Handbook (Schwarz-Schampera 2014) and China-Resources.net (Maness, 2010) List. Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Skarpelis (1995)
India	Tosham	Reserves Development	In from Schwarz-Schampera (2002, 2014) and cited again in Murakami & Ishihara (2013) - Mineralogy from Seetharam (1986) and Murao et al. (1995)
Japan	Akenobe (6 locations)	Closed	In from Murakami & Ishihara (2013). Listed in China-Resources.net (Maness, 2010) & Critical Minerals Handbook (Schwarz-Schampera, 2014). Mineralisation from Schwarz-Schampera & Herzig (2002), P. 17. Listed also in Ishihara (2006)
Japan	Ashio		In from Murakami & Ishihara (2013), also listed in Maness (2010) unofficial publication. Said to contain >500 t In (Ishihara et al, 2006)
Japan	Ikuno	Smelter operating	In from Murakami & Ishihara (2013) - Mineralogy from Schwarz-Schampera & Herzig (2002) - citing Shimizu & Kato (1991). Listed also in Ishihara (2006)
Japan	Kosaka		Ishihara & Endo (2007)
Japan	Omodani		In from Schwarz-Schampera citing Shimizu & Kato (1991). Also on China-Resources.net (Maness, 2010). Mineralisation from Shimizu & Kato (1991)
Japan	Taishu		In from Ishihara et al (2006). Said to contain 100-150t In. Selected 100 t In to be conservative.
Japan	Toyoha (8 locations)	Closed	In from Schwarz-Schampera (2014) - citing Ishihara (2006) - "Reserves exhausted" - but this assessment is from the year of closure. - Mineralogy from Yoshie et al. (1986), Ohta (1989) Kooiman and Ruitenberg (1992)
Japan	Uchinotai		Ishihara & Endo (2007) See also online <https://www.gsj.jp/data/bulletin/58_01_02.pdf>
Kosovo	Drazhnje	Exploration drilling	Reported to contain Sn-associated In enrichments in stannite group minerals present in sphalerite. Resources reported in the 2009-03 Lydian International Technical Report. Resources are based on the (former) Yugoslav reporting code. Identified in Kolodziejczyk (2016), which notes mineralogical relationships. See http://www.mindat.org/loc-258157.html for location. In content calculated using proxies as per Werner et al. (2017).

Mid-Atlantic Ridge	Broken Spur		In from Schwarz-Schampera & Herzig (2002). Also listed in China-Resources.net (Maness, 2010). Tonnage estimated from Hannigton et al (online) at: <ftp://rock.geosociety.org/pub/reposit/2011/2011342.pdf>
Mid-Atlantic Ridge	Snake Pit (23°N)		In grade from Schwarz-Schampera & Herzig (2002) - citing Fouquet et al. (1993). Tonnage estimated from Hannigton et al: ftp://rock.geosociety.org/pub/reposit/2011/2011342.pdf
Mid-Atlantic Ridge	TAG (26°N)		In from Herzig et al. (1998) and Humphris et al. (1995) and Schwarz-Schampera & Herzig (2002). Also in China-Resources.net (Maness, 2010).
Namibia	Tsumeb (Slag Resource)	Slag / Reserves Development	Listed in Alfantazi & Moskalyk (2003) - Also contains germanium (0.026%) gallium (approximately 0.02%)
Northeast Pacific	Axial Seamount @ Juan de Fuca Ridge		In grade from Schwarz-Schampera & Herzig (2002) citing Schwarz-Schampera & Herzig (1997). Tonnage estimated from Hannigton et al (online) at: <ftp://rock.geosociety.org/pub/reposit/2011/2011342.pdf>
Russia	Bakr-Tau		Grade and mineralogy from Prokin & Buslaev (1999). Tonnage from Herrington et al. (2005). See also Schwarz-Schampera & Herzig (2002) and republished in Murakami & Ishihara (2013), Also listed on China-Resources.net (Maness, 2010.)
Russia	Gaiskoye / Gaiskoe / Gai / Gay	Expansion / Active	In grade from Schwarz-Schampera & Herzig (2002) citing Prokin & Buslaev (1999). Tonnage from Herrington et al. (2005)
Russia	Komsomolskoye	Operating	Grade, tonnage and mineralogy from Prokin & Buslaev (1999). See also Schwarz-Schampera & Herzig (2002) and republished in Murakami & Ishihara (2013), Also listed on China-Resources.net (Maness, 2010.)
Russia	Letneye / Letnye	Satellite	Grade and mineralogy from Prokin & Buslaev (1999). Tonnage from Herrington et al. (2005). See also Schwarz-Schampera & Herzig (2002) and republished in Murakami & Ishihara (2013), Also listed on China-Resources.net (Maness, 2010.)
Russia	Podolskoye / Podolskoe	Reserves Development	Grade and mineralogy from Prokin & Buslaev (1999). Tonnage from Herrington et al. (2005). See also Schwarz-Schampera & Herzig (2002) and republished in Murakami & Ishihara (2013), Also listed on China-Resources.net (Maness, 2010.)
Russia	Sibaiskoye / Sibai / Sibay	Operating	Grade, tonnage and mineralogy from Prokin & Buslaev (1999). See also Schwarz-Schampera & Herzig (2002) and republished in Murakami & Ishihara (2013), Also listed on China-Resources.net (Maness, 2010.)
South Africa	Letaba CZ, Murchison Belt	Closed	Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003)
South Africa	Maranda J Mine, Murchison Belt		Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003) - "Among the highest grades for Indium in the literature"
South Africa	Mashawa, Murchison Belt		Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003)
South Africa	Mon Desir, Murchison Belt		Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003)
South Africa	Romotshidi, Murchison Belt		Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003)
South Africa	Solomons, Murchison Belt		Listed in Schwarz-Schampera (2003, 2010, 2014). Grades and Tonnage from Schwarz-Schampera (2003)
South Korea	Ulsan		Listed in Schwarz-Schampera (2002, 2014) - citing Imai and Choi (1984). Also listed in China-Resources.net (Maness, 2010)
Southwest Pacific	Southern Lau Basin @ Tonga Subduction Zone		Listed in Schwarz-Schampera (2002, 2014) - citing Schwarz-Schampera & Herzig (1997). Tonnage and grade from SS'02 - citing Schwarz-Schampera (2000). Cited also in Ishihara & Endo (2007)
Sweden	Långban		Listed in Schwarz-Schampera (2002, 2014) - citing Burke and Kieft (1980). Also listed in China-Resources.net (Maness, 2010). Also in EU ProMine DB. Tonnage from Allen et al. (1996)
United Kingdom	West Shropshire Orefield (England)	Old industrial mine, abandoned deposit	In from Schwarz-Schampera & Herzig (2002) - citing Pattrick and Dorling (1991), Pattrick et al. (1993) - Indium mineralisation associated with "BPGC" (sphalerite, pyrite, galena, chalcopyrite) vein in mafic volcanosedimentary environment
USA	Kingman	Grassroots / Active	Not compliant w' NI 43-101. Sourced from Technical Report (2008-07). See also ARS Mining online at: <http://arswebsite.wordpress.com/about/>
Russia	Degtyarsk / Degtyarskoye	Reserves Development	Indium grade calculated via Proxies. Listed in Murakami & Ishihara (2013) and USGS Opoen File Report 2005-1294D. Non-code Tonnage from Herrington et al. (2005)
Russia	Uzelga	Adandoned	Indium grade calculated via Proxies. USGS Open File Report 2005–1294D, citing "Herrington (2004)". Non-code tonnage from Herrington et al. 2005

B4: Per-country production of indium, 1989-2015

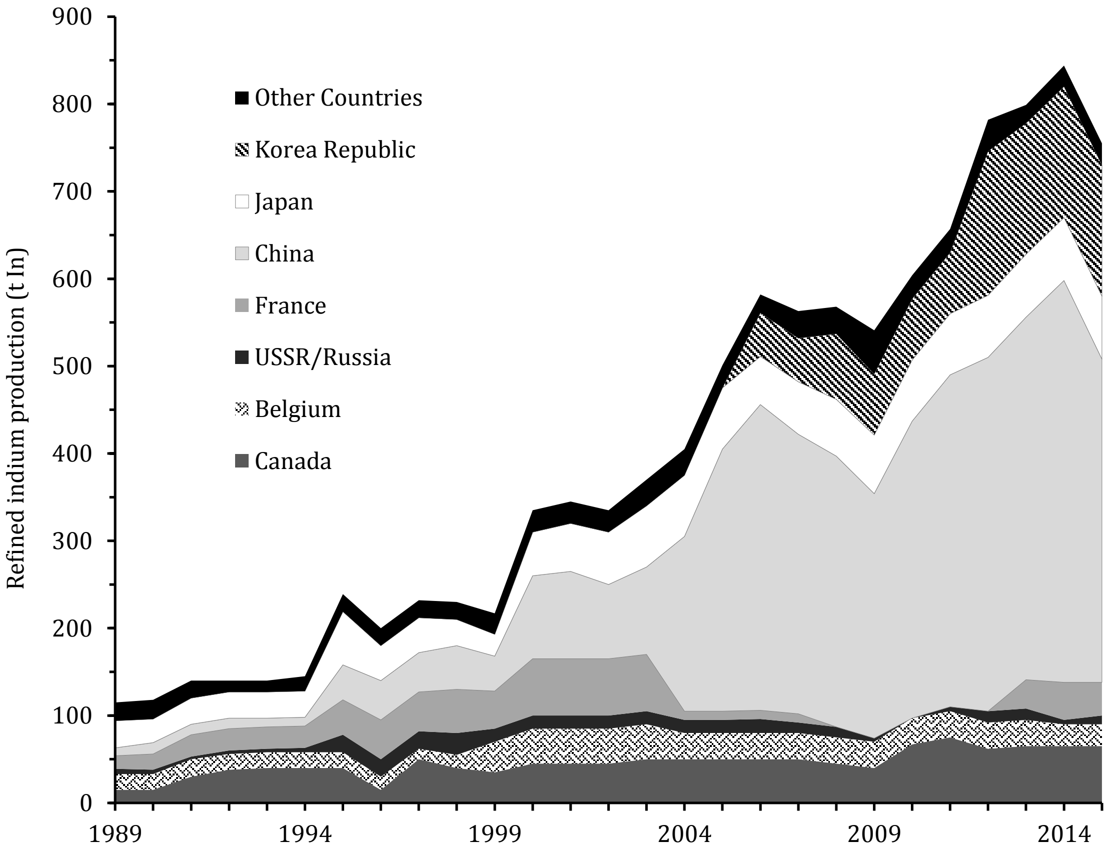

Figure S1: Country primary production of indium, 1989-2015, updated from (Werner et al., 2015), compiled from (USGS, 1989-2015).

B5: Broken Hill Production Data

Table B4: Broken Hill production data, updated from Mudd (2007, 2010). Indium grade of 50 ppm In taken from Schwarz-Schampera & Herzig (2002) assumed to be a maximum at 50 ppm, and declining at the same rate as Zn grades from 1995 (date of original estimate) to 2014.

Year	t Pb produced	t Zn produced	t Ore Milled	t Concentrates Produced	t Tailings Produced	Est. In grade in ore	t In to tailings (5% deportment)	t In to tailings (20%)	t In to tailings (35%)
1883			107.5	0	108	50	0.00	0.00	0.00
1884			9,314	0	9314	50	0.02	0.09	0.16
1885			11,505	0	11505	50	0.03	0.12	0.20
1886	2,023		14,986	4045	10941	50	0.04	0.15	0.26
1887	10,073		53,447	20145	33301	50	0.13	0.53	0.94
1888	18,453		128,000	36906	91093	50	0.32	1.28	2.24
1889	27,704		164,084	55408	108676	50	0.41	1.64	2.87
1890	33,380		195,627	66760	128866	50	0.49	1.96	3.42
1891	69,614		478,639	139229	339410	50	1.20	4.79	8.38
1892	61,839		409,582	123678	285905	50	1.02	4.10	7.17
1893	73,372		500,000	146745	353255	50	1.25	5.00	8.75
1894	55,403		653,108	110806	542303	50	1.63	6.53	11.43
1895	57,061		525,846	114122	411724	50	1.31	5.26	9.20
1896	90,492		833,492	180984	652508	50	2.08	8.33	14.59
1897	106,123	10,329	1,028,152	232903	795249	50	2.57	10.28	17.99
1898	102,686	6,198	905,062	217768	687294	50	2.26	9.05	15.84
1899	191,761	48,043	1,283,317	479607	803710	50	3.21	12.83	22.46
1900	168,432	35,519	1,446,812	407904	1038909	50	3.62	14.47	25.32
1901	149,263	37,519	1,299,080	373563	925517	50	3.25	12.99	22.73
1902	134,664	29,903	1,134,927	329133	805794	50	2.84	11.35	19.86
1903	137,572	36,063	1,131,591	347271	784321	50	2.83	11.32	19.80
1904	167,739	22,394	1,363,752	380264	983488	50	3.41	13.64	23.87
1905	164,211	42,076	1,480,595	412573	1068022	50	3.70	14.81	25.91
1906	138,704	41,723	1,286,331	360855	925477	50	3.22	12.86	22.51

1907	191,465	96,012	1,679,337	574954	1104383	50	4.20	16.79	29.39
1908	172,593	112,139	1,470,965	569464	901501	50	3.68	14.71	25.74
1909	154,922	151,955	1,047,814	613753	434061	50	2.62	10.48	18.34
1910	185,626	209,183	1,263,583	789619	473964	50	3.16	12.64	22.11
1911	206,127	241,397	1,508,147	895047	613100	50	3.77	15.08	26.39
1912	226,710	232,281	1,665,894	917982	747911	50	4.16	16.66	29.15
1913	239,887	219,628	1,821,943	919029	902914	50	4.55	18.22	31.88
1914	192,412	168,387	1,465,037	721598	743440	50	3.66	14.65	25.64
1915	155,566	133,601	1,528,677	578334	950343	50	3.82	15.29	26.75
1916	139,148	120,583	1,036,347	519463	516885	50	2.59	10.36	18.14
1917	155,764	160,103	1,048,952	631735	417217	50	2.62	10.49	18.36
1918	188,641	184,653	1,285,386	746587	538799	50	3.21	12.85	22.49
1919	62,232	66,098	422,046	256660	165387	50	1.06	4.22	7.39
1920	6,825	10,237	45,455	34125	11329	50	0.11	0.45	0.80
1921	77,982	141,691	323,223	439347	0	50	0.81	3.23	5.66
1922	141,978	198,662	650,305	681279	0	50	1.63	6.50	11.38
1923	153,204	145,505	892,594	597418	295175	50	2.23	8.93	15.62
1924	140,539	109,951	1,067,485	500979	566505	50	2.67	10.67	18.68
1925	174,713	141,004	1,315,039	631434	683605	50	3.29	13.15	23.01
1926	171,547	145,772	1,317,498	634636	682862	50	3.29	13.17	23.06
1927	191,318	166,097	1,417,245	714829	702416	50	3.54	14.17	24.80
1928	172,674	141,923	1,196,658	629195	567463	50	2.99	11.97	20.94
1929	187,848	148,384	1,248,856	672464	576392	50	3.12	12.49	21.85
1930	192,554	121,527	1,330,508	628162	702346	50	3.33	13.31	23.28
1931	131,126	75,399	888,073	413051	475023	50	2.22	8.88	15.54
1932	161,299	117,523	1,115,984	557644	558340	50	2.79	11.16	19.53
1933	175,794	125,682	1,219,784	602953	616831	50	3.05	12.20	21.35
1934	187,150	139,050	1,273,674	652400	621274	50	3.18	12.74	22.29
1935	189,536	143,959	1,328,528	666990	661538	50	3.32	13.29	23.25
1936	188,948	143,427	1,377,286	664748	712537	50	3.44	13.77	24.10
1937	199,494	149,300	1,517,604	697588	820017	50	3.79	15.18	26.56
1938	224,900	167,577	1,608,700	784953	823746	50	4.02	16.09	28.15
1939	218,641	147,530	1,447,036	732343	714693	50	3.62	14.47	25.32
1940	217,851	164,032	1,452,688	763766	688922	50	3.63	14.53	25.42

1941	226,844	174,622	1,492,073	802933	689141	50	3.73	14.92	26.11
1942	207,228	153,546	1,346,791	721549	625242	50	3.37	13.47	23.57
1943	176,177	130,141	1,191,467	612638	578829	50	2.98	11.91	20.85
1944	171,039	130,438	1,185,013	602953	582059	50	2.96	11.85	20.74
1945	151,273	120,463	1,082,009	543473	538536	50	2.71	10.82	18.94
1946	154,846	124,740	1,145,227	559172	586055	50	2.86	11.45	20.04
1947	150,018	122,929	1,135,825	545895	589930	50	2.84	11.36	19.88
1948	167,279	138,322	1,253,870	611203	642667	50	3.13	12.54	21.94
1949	163,215	141,743	1,286,567	609917	676650	50	3.22	12.87	22.51
1950	169,027	142,642	1,260,399	623338	637060	50	3.15	12.60	22.06
1951	162,203	140,960	1,289,240	606326	682914	50	3.22	12.89	22.56
1952	167,115	144,615	1,363,076	623460	739615	50	3.41	13.63	23.85
1953	211,816	189,064	1,676,293	801760	874533	50	4.19	16.76	29.34
1954	229,810	212,702	1,858,177	885023	973153	50	4.65	18.58	32.52
1955	229,719	213,905	1,949,390	887246	1062144	50	4.87	19.49	34.11
1956	230,425	227,407	2,096,576	915664	1180912	50	5.24	20.97	36.69
1957	260,158	241,724	2,272,084	1003763	1268321	50	5.68	22.72	39.76
1958	238,752	209,302	1,930,551	896108	1034443	50	4.83	19.31	33.78
1959	240,157	201,881	1,914,312	884076	1030235	50	4.79	19.14	33.50
1960	228,958	232,390	2,033,322	922695	1110627	50	5.08	20.33	35.58
1961	205,898	224,553	1,938,791	860904	1077888	50	4.85	19.39	33.93
1962	295,413	245,788	2,260,196	1082402	1177794	50	5.65	22.60	39.55
1963	333,717	269,887	2,497,625	1207209	1290416	50	6.24	24.98	43.71
1964	301,071	260,097	2,400,874	1122337	1278537	50	6.00	24.01	42.02
1965	300,580	274,716	2,530,471	1150592	1379879	50	6.33	25.30	44.28
1966	284,379	275,796	2,497,484	1120351	1377133	50	6.24	24.97	43.71
1967	283,445	297,883	2,601,473	1162656	1438818	50	6.50	26.01	45.53
1968	249,176	275,522	2,428,051	1049396	1378655	50	6.07	24.28	42.49
1969	281,595	339,981	2,745,139	1243151	1501987	50	6.86	27.45	48.04
1970	275,647	284,506	2,813,426	1120307	1693119	50	7.03	28.13	49.23
1971	253,573	304,504	2,604,046	1116155	1487890	50	6.51	26.04	45.57
1972	245,524	299,709	2,859,829	1090466	1769363	50	7.15	28.60	50.05
1973	242,676	280,176	2,429,307	1045704	1383603	50	6.07	24.29	42.51

1974	225,063	255,984	2,466,983	962094	1504889	50	6.17	24.67	43.17
1975	242,370	280,490	2,835,084	1045720	1789364	50	7.09	28.35	49.61
1976	216,151	265,763	2,791,762	963828	1827934	50	6.98	27.92	48.86
1977	231,106	273,639	2,673,148	1009490	1663658	50	6.68	26.73	46.78
1978	223,694	250,838	2,697,671	949064	1748607	50	6.74	26.98	47.21
1979	224,745	269,601	2,800,480	988692	1811788	50	7.00	28.00	49.01
1980	209,653	250,367	2,730,202	920040	1810162	50	6.83	27.30	47.78
1981	189,684	224,421	2,715,026	828210	1886816	50	6.79	27.15	47.51
1982	217,072	286,879	3,023,170	1007902	2015268	50	7.56	30.23	52.91
1983	196,938	267,911	2,958,723	929698	2029025	50	7.40	29.59	51.78
1984	163,713	211,207	2,550,968	749840	1801128	50	6.38	25.51	44.64
1985	180,397	232,288	2,979,566	825370	2154196	50	7.45	29.80	52.14
1986	136,890	146,898	1,946,697	567576	1379121	50	4.87	19.47	34.07
1987	139,615	212,603	2,539,297	704436	1834861	50	6.35	25.39	44.44
1988	154,696	208,113	2,535,606	725618	1809988	50	6.34	25.36	44.37
1989	154,724	218,473	2,545,619	746394	1799225	50	6.36	25.46	44.55
1990	147,128	222,442	2,488,949	739140	1749809	50	6.22	24.89	43.56
1991	156,150	228,173	2,554,807	768646	1786161	50	6.39	25.55	44.71
1992	163,720	205,498	2,569,445	738436	1831009	50	6.42	25.69	44.97
1993	159,502	187,455	2,397,899	693914	1703985	50	5.99	23.98	41.96
1994	141,747	192,317	2,489,315	668128	1821187	50	6.22	24.89	43.56
1995	131,726	163,967	2,506,628	591386	1915242	*50*	6.27	25.07	43.87
1996	135,447	189,964	2,627,139	650822	1976317	*49*	6.40	25.61	44.83
1997	128,396	191,376	2,729,692	639544	2090148	*48*	6.48	25.93	45.38
1998	78,656	184,723	2,769,186	526758	2242428	*46*	6.40	25.61	44.83
1999	103,323	190,765	2,760,226	588176	2172050	*45*	6.21	24.84	43.47
2000	93,642	180,420	2,861,269	548124	2313145	*44*	6.26	25.04	43.81
2001	75,424	178,974	2,737,539	508796	2228743	*43*	5.82	23.27	40.72
2002	44,620	112,027	1,767,411	313293	1454118	*41*	3.65	14.58	25.52
2003	61,800	170,400	2,171,000	464400	1706600	*40*	4.34	17.37	30.39
2004	53,700	131,100	1,994,000	369600	1624400	*39*	3.86	15.45	27.04
2005	70,300	130,000	1,940,000	400600	1539400	*38*	3.64	14.55	25.46
2006	75,900	131,500	2,075,800	414800	1661000	*36*	3.76	15.05	26.34
2007	47,000	65,700	1,493,400	225400	1268000	*35*	2.61	10.45	18.29

2008	44,700	75,900	1,770,900	241200	1529700	*34*	2.99	11.95	20.92
2009	54,100	58,800	1,528,000	225800	1302200	*33*	2.48	9.93	17.38
2010	51,200	63,600	1,637,600	229600	1408000	*31*	2.56	10.24	17.91
2011	49,500	63,600	1,749,200	226200	1523000	*30*	2.62	10.50	18.37
2012	51,400	64,200	1,692,400	231200	1461200	*29*	2.43	9.73	17.03
2013	43,240	59,550	1,604,000	205580.8	1398419	*28*	2.21	8.82	15.44
2014	53,760	71,402	1,600,000	250323.2	1349677	*26*	2.10	8.40	14.70

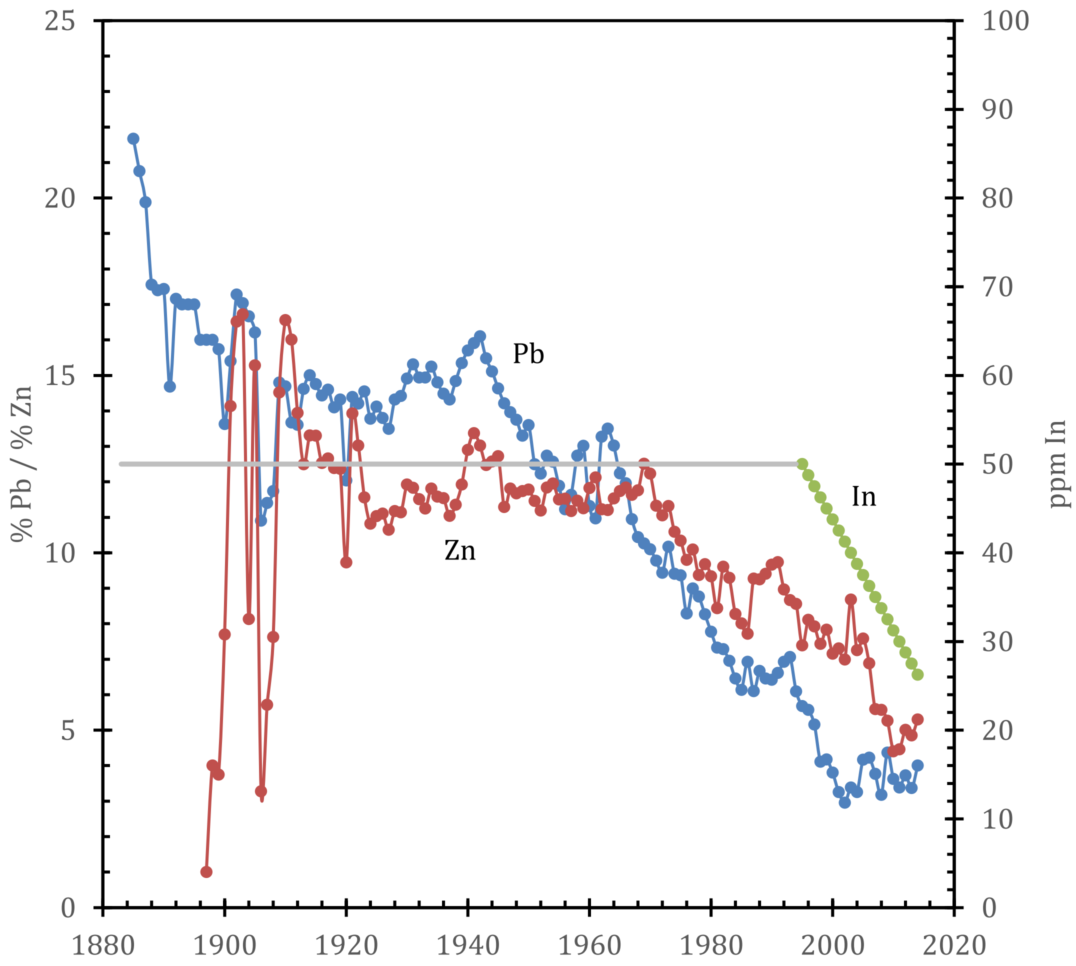

Figure B2: Ore Grade for Pb, Zn and In 1883-2014 use for projecting tailings estimates as shown in Fig. 12 of the main report. In grades were capped at 50 ppm to maintain conservative estimates for accumulated In in tailings, from 1883-1994, despite increased Zn grade during this time.

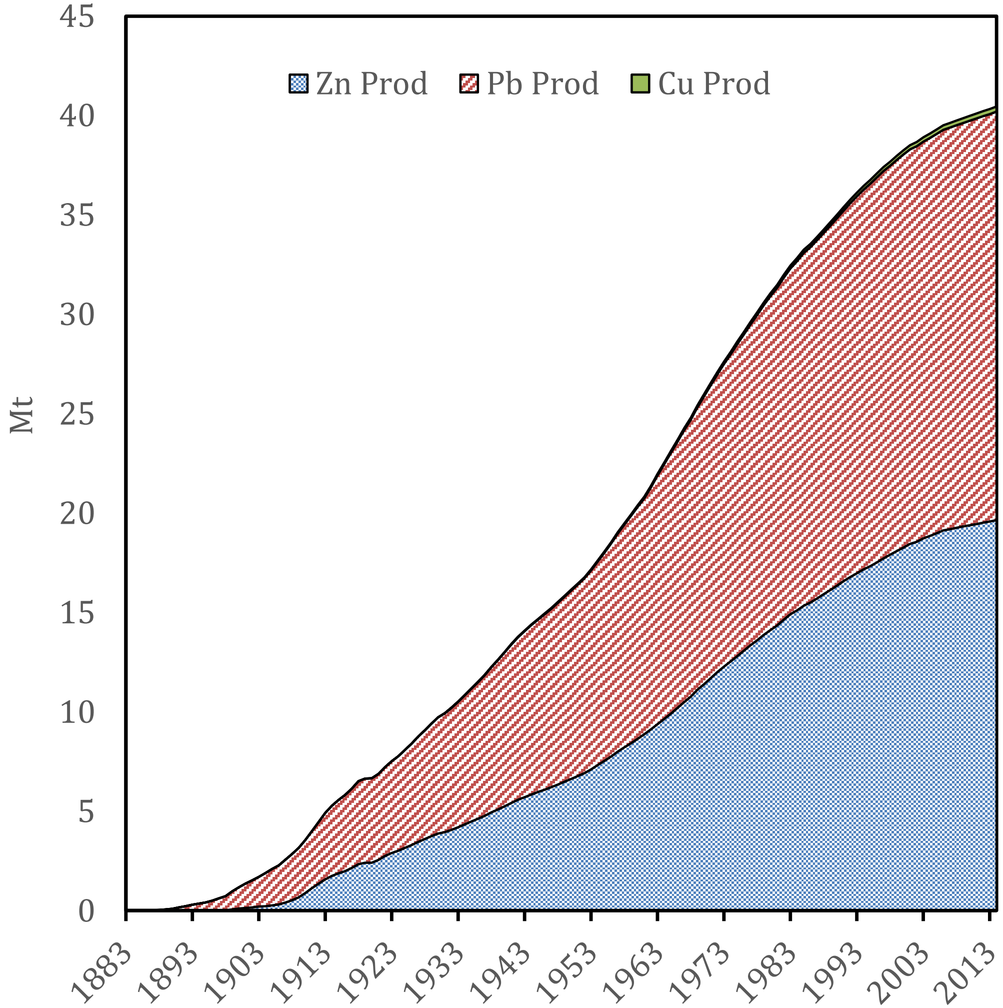

Figure B3: Cumulative Pb, Zn and Cu production from Broken Hill, 1883-2014. Data updated from Mudd (2007, 2010).

B6: Proportional monetary value deposits with estimated In contents in the Pb-Zn deposit database

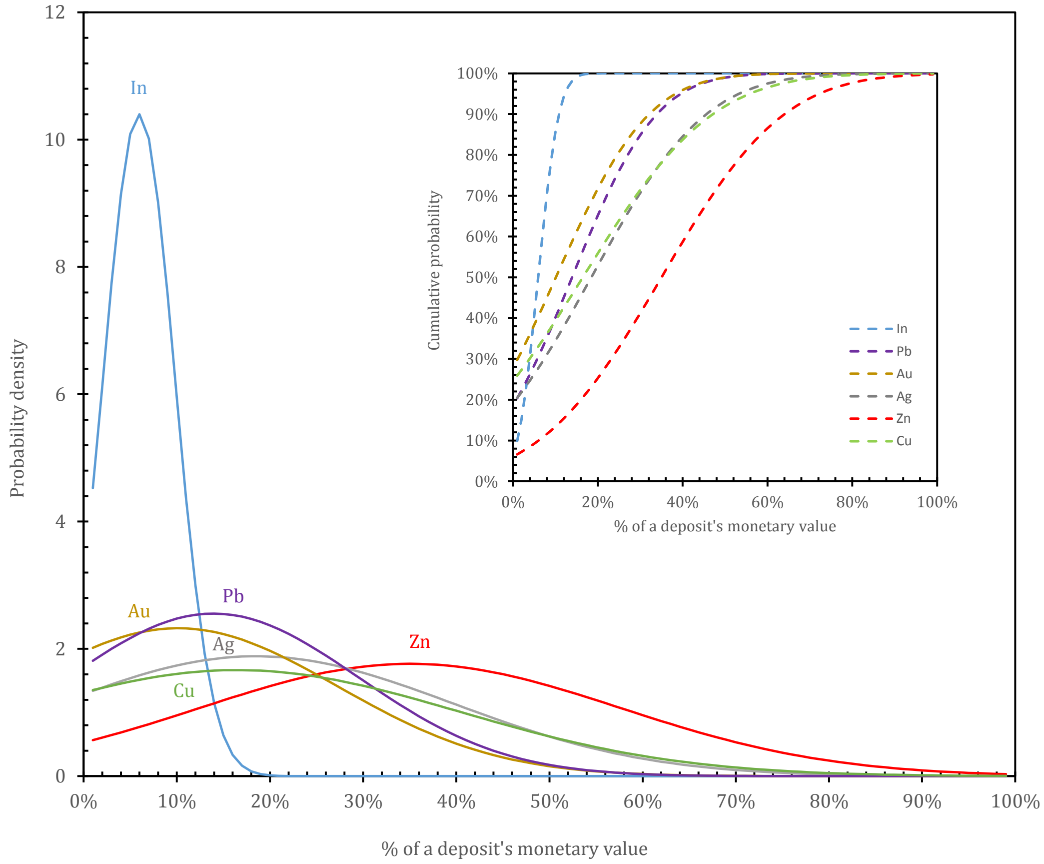

Figure B4: Distribution of monetary value from deposits where indium contents have been inferred using the Pb-Zn deposit database available from Mudd et al. (2017).

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Appendix C: Supplementary information for Chapter 6

C1: Assessing mineral deposits of indium in Australia

Table C1: Tin Deposits in Australia, with indium contents estimated as per Werner et al. (2017a).

Mine / Deposit	Status	Mt ore	%Sn	%Cu	g/t Ag	%WO3	%Pb	%Zn	kt Sn	kt Cu	t Ag	ppm In	t In (est)	Company
Mt Garnet-Windermere	Deposit	2.103	0.55						11.6			13.781	28.981	Consolidated Tin Mines
Collingwood	Care & Maint.	0.703	1.28						9.0			31.973	22.477	Metals X
Mt Garnet-Gillian	Deposit	3.001	0.78						23.4			19.531	58.616	Consolidated Tin Mines
Mt Garnet-Pinnacles Group	Deposit	1.871	0.41						7.6			10.196	19.076	Consolidated Tin Mines
Mt Garnet-Deadman's Gully	Deposit	0.402	0.49						2.0			12.281	4.931	Consolidated Tin Mines
Conrad-Greisen	Deposit	0.479	0.13	0.02	40.6				0.6	0.1	19	3.281	1.570	Malachite Resources
Doradilla (3KEL-Midway)	Deposit	7.81	0.28						21.9			7.031	54.911	YTC Resources
Mt Paynter	Deposit								0.0			0.031	0.000	unknown
Taronga	Deposit	46.757	0.145						67.8			3.656	170.939	AusNiCo (res. not JORC compliant)
White Rock	Deposit								0.0			0.031	0.000	Paradigm Metals
Renison Bell	Mine	9.685	1.58	0.27					153.5	25.9		39.651	384.018	Metals X
Renison Bell (tailings)	Mine	19.999	0.45	0.21					90.0	42.0		11.281	225.607	Metals X
Mt Bischoff	Mine	1.667	0.54						9.0			13.523	22.543	Metals X
Cleveland-Luina (Sn-Cu)	Deposit	5.157	0.7	0.31					36.1	16.0		17.531	90.407	Lynch Mining Pty Ltd (data Aust Mines Atlas 2013-09-30)
Cleveland-Luina (Sn-W)	Deposit	3.8	0.05			0.28			1.9			1.281	4.867	Lynch Mining Pty Ltd (data Aust Mines Atlas 2013-09-30)
Montana-Keemskirk	Deposit	0.36	1.6						5.8			40.031	14.411	Stellar Resources (data Aust Mines Atlas 2013-09-30)
Queen Hill	Deposit	1.6	1.2						19.2			30.031	48.049	Stellar Resources (data Aust Mines Atlas 2013-09-30)
Mt Lindsay	Deposit	43	0.2			0.1			86.0			5.031	216.329	Venture Minerals
Sweeneys-Birthday	Deposit	0.562	0.5					1.4	2.8			12.531	7.042	unknown (data Aust Mines Atlas 2013-09-30)
Aberfoyle-Lutwyche	Deposit								0.0			0.031	0.000	Minemakers
Royal George	Deposit								0.0		0	0.031	0.000	Minemakers
Oonah	Deposit	0.44	1.25	1.48	136				5.5	6.5	60	31.281	13.764	TnT Mines
Great Pyramid	Deposit	5.2	0.18						9.4			4.531	23.561	TnT Mines
Heemskirk	Deposit								0.0			0.031	0.000	Stellar Resources
Kara	Deposit	13.07				0.034			0.0			0.031	0.404	Tasmania Mines
Moina	Deposit	24.6	0.1			0.1			24.6			2.531	62.260	TnT Mines
Anchor	Deposit	8.8	0.18						15.8			4.531	39.872	TnT Mines
Federation	Deposit	0.562	0.5		36.4			1.4	2.8			12.531	7.042	unknown (formerly Stonehenge Metals; data 2009 Ann Rep)
Molyhil	Deposit	4.71				0.28			0.0			0.031	0.146	Thor Mining
Yanco Glen	Deposit	3.445				0.11			0.0			0.031	0.106	Carpentaria Exploration
Bielsdown-Wild Cattle Creek	Deposit	1.59				0.045			0.0			0.031	0.049	Anchor Resources
Watershed	Deposit	49.32				0.14			0.0			0.031	1.524	Vital Metals
Mt Mulgine	Deposit	8.18				0.21			0.0			0.031	0.253	Hazelwood Resources
Big Hill-Cookes Creek	Deposit	16.22				0.16			0.0			0.031	0.501	Hazelwood Resources
O'Callaghans	Deposit	78		0.28		0.33	0.25	0.50	0.0			0.031	2.410	Newcrest Mining
Mt Carbine	Deposit	59.4				0.12			0.0			0.031	1.835	Carbine Tungsten (formerly Icon Resources)
Dalcouth	Deposit	0.1024	0.34						0.3			8.531	0.874	MGT Resources
Extended	Deposit	0.0095	0.35						0.0			8.781	0.083	MGT Resources
Wolfram Camp	Care & Maint.	1.42				0.60			0.0			0.031	0.044	Deutsche Rohstoff (formerly Planet Metals)

Table C2: Additional unquantifiable projects identified to potentially contain indium in Australia:

Deposit Name	Source / Notes
Khartoum Tin Project	In mineralisation from Australian Mines Atlas - Commodity Search. Auzex yet to produce tonnage or grade estimates.
Ann	Australian Mines Atlas - Commodity Search - Geological Survey of Queensland - MINOCC Database
Arbouin	Australian Mines Atlas - Commodity Search
Black Sparkle	Australian Mines Atlas - Commodity Search
Orient Camp East Group	Australian Mines Atlas - Commodity Search
Unnamed 167715	Australian Mines Atlas - Commodity Search
Weinert	Australian Mines Atlas - Commodity Search. Listed in Qld Industry Publication.
West Orient	Australian Mines Atlas - Commodity Search

C2: Assessing the trade of indium in Australia

Table C3: Other codes identified to possibly include indium, but not considered for Australia

HS Code	Description
392690	Plastic articles nes - Two instance indium foils traded under this code in India. Likely that similar quantities traded in Australia would be << 1kg.
852691	Radio navigational aid apparatus – Traded under 8526.91.0030 in Australia.
7326	Articles of iron or steel nes
690919	Ceramic laboratory & technical ware except porcelain
580137	Special woven fabrics; tufted textile fabics; lace; tapestries; trimmings; embroidery // woven pile fabrics and chenille fabrics, other than fabrics of heading 58.02 or 58.06 // - Of man-made fibres : // -- Warp pile fabrics
441872	Articles of wood // Other, Multilayer
401693	Gaskets, washers and other seals of vulcanised rubber
321290	Tanning or drying extracts - other
2505590	Natural sands // other (e.g. STANDARD CALIBRATION MATERIAL FOR R&D PURPOSE - INDIUM DSC CALIBR STD TEMP & ENTH OF FUS - 1 NO@288 USD EACH United States Hyderabad Air Cargo KGS 0 19,314 19,313,920)
980200	Laboratory chemicals
681599	Articles of mineral substances // other
3403/3405	Lubricating preparations / Polishes and creams
7115/7114/7110	Articles of precious metals / articles of platinum
741999	Articles of copper, nes
841990	Part, laboratory/industrial heating/cooling machinery. - Includes apparatus for the deposition of indium onto products such as solar cells, but may not contain indium itself.
382490	Chemical products and preparations of the chemical or allied industries (including those consisting of mixtures of natural products), not elsewhere specified or included
840710	Aircraft engines

Table C4: Content estimates for the electrical compounds and semiconductors end use sector

	1			2			3		
	LED semiconductor devices			Solar Panels			PCBs		
HS CODE	854140			854140			853400		
Period	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top
EXPORT									
1962	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1963	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1964	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1965	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1966	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1967	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1968	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1969	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1970	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1971	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1972	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1973	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1974	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1975	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1976	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1977	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1978	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1979	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1980	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1981	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1982	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1983	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1984	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1985	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1986	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1987	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1988	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0003%	0.0003%	0.0001%	0.0006%
1989	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0003%	0.0003%	0.0001%	0.0006%
1990	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0003%	0.0003%	0.0001%	0.0006%
1991	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1992	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
1993	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0002%	0.0003%	0.0001%	0.0006%
1994	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0002%	0.0003%	0.0001%	0.0006%

1995	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
1996	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
1997	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
1998	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
1999	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0002%	0.0003%	0.0001%	0.0006%
2000	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
2001	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0002%	0.0003%	0.0001%	0.0006%
2002	0.000053%	0.0000400%	0.000067%	0.0001%	0.0000%	0.0001%	0.0003%	0.0001%	0.0006%
2003	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
2004	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0003%	0.0001%	0.0006%
2005	0.000053%	0.0000400%	0.000067%	0.0001%	0.0001%	0.0002%	0.0017%	0.0003%	0.0031%
2006	0.000053%	0.0000400%	0.000067%	0.0002%	0.0001%	0.0004%	0.0017%	0.0003%	0.0031%
2007	0.000053%	0.0000400%	0.000067%	0.0003%	0.0002%	0.0006%	0.0017%	0.0003%	0.0031%
2008	0.000053%	0.0000400%	0.000067%	0.0005%	0.0002%	0.0009%	0.0017%	0.0003%	0.0031%
2009	0.000053%	0.0000400%	0.000067%	0.0007%	0.0004%	0.0015%	0.0017%	0.0003%	0.0031%
2010	0.000053%	0.0000400%	0.000067%	0.0005%	0.0003%	0.0011%	0.0017%	0.0003%	0.0031%
2011	0.000053%	0.0000400%	0.000067%	0.0006%	0.0003%	0.0013%	0.0017%	0.0003%	0.0031%
2012	0.000053%	0.0000400%	0.000067%	0.0008%	0.0004%	0.0017%	0.0017%	0.0003%	0.0031%
2013	0.000053%	0.0000400%	0.000067%	0.0010%	0.0005%	0.0021%	0.0017%	0.0003%	0.0031%
2014	0.000053%	0.0000400%	0.000067%	0.0012%	0.0006%	0.0024%	0.0017%	0.0003%	0.0031%
2015	0.000053%	0.0000400%	0.000067%	0.0013%	0.0007%	0.0027%	0.0017%	0.0003%	0.0031%
IMPORT									
1962	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1963	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1964	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1965	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1966	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1967	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1968	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1969	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1970	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1971	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1972	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1973	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1974	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1975	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1976	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1977	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1978	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1979	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1980	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1981	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1982	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%

1983	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1984	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1985	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1986	0.000053%	0.0000400%	0.000067%	0.0000370%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1987	0.000053%	0.0000400%	0.000067%	0.0001249%	0.000125%	0.000125%	0.0003%	0.0001%	0.0006%
1988	0.000053%	0.0000400%	0.000067%	0.0001290%	0.000129%	0.000129%	0.0003%	0.0001%	0.0006%
1989	0.000053%	0.0000400%	0.000067%	0.0001330%	0.000133%	0.000133%	0.0003%	0.0001%	0.0006%
1990	0.000053%	0.0000400%	0.000067%	0.0001371%	0.000137%	0.000137%	0.0003%	0.0001%	0.0006%
1991	0.000053%	0.0000400%	0.000067%	0.0001219%	0.000122%	0.000122%	0.0003%	0.0001%	0.0006%
1992	0.000053%	0.0000400%	0.000067%	0.0001066%	0.000107%	0.000107%	0.0003%	0.0001%	0.0006%
1993	0.000053%	0.0000400%	0.000067%	0.0000914%	0.000091%	0.000091%	0.0003%	0.0001%	0.0006%
1994	0.000053%	0.0000400%	0.000067%	0.0000762%	0.000076%	0.000076%	0.0003%	0.0001%	0.0006%
1995	0.000053%	0.0000400%	0.000067%	0.0000609%	0.000061%	0.000061%	0.0003%	0.0001%	0.0006%
1996	0.000053%	0.0000400%	0.000067%	0.0000613%	0.000061%	0.000061%	0.0003%	0.0001%	0.0006%
1997	0.000053%	0.0000400%	0.000067%	0.0000617%	0.000062%	0.000062%	0.0003%	0.0001%	0.0006%
1998	0.000053%	0.0000400%	0.000067%	0.0000688%	0.000066%	0.000071%	0.0003%	0.0001%	0.0006%
1999	0.000053%	0.0000400%	0.000067%	0.0000755%	0.000070%	0.000081%	0.0003%	0.0001%	0.0006%
2000	0.000053%	0.0000400%	0.000067%	0.0000685%	0.000061%	0.000076%	0.0003%	0.0001%	0.0006%
2001	0.000053%	0.0000400%	0.000067%	0.0000756%	0.000070%	0.000081%	0.0003%	0.0001%	0.0006%
2002	0.000053%	0.0000400%	0.000067%	0.0000748%	0.000074%	0.000075%	0.0003%	0.0001%	0.0006%
2003	0.000053%	0.0000400%	0.000067%	0.0001069%	0.000095%	0.000119%	0.0003%	0.0001%	0.0006%
2004	0.000053%	0.0000400%	0.000067%	0.0001019%	0.000106%	0.000098%	0.0003%	0.0001%	0.0006%
2005	0.000053%	0.0000400%	0.000067%	0.0001163%	0.000144%	0.000089%	0.0017%	0.0003%	0.0031%
2006	0.000053%	0.0000400%	0.000067%	0.0001780%	0.000225%	0.000131%	0.0017%	0.0003%	0.0031%
2007	0.000053%	0.0000400%	0.000067%	0.0003071%	0.000387%	0.000228%	0.0017%	0.0003%	0.0031%
2008	0.000053%	0.0000400%	0.000067%	0.0004522%	0.000572%	0.000332%	0.0017%	0.0003%	0.0031%
2009	0.000053%	0.0000400%	0.000067%	0.0007266%	0.000962%	0.000491%	0.0017%	0.0003%	0.0031%
2010	0.000053%	0.0000400%	0.000067%	0.0005362%	0.000715%	0.000357%	0.0017%	0.0003%	0.0031%
2011	0.000053%	0.0000400%	0.000067%	0.0006448%	0.000853%	0.000436%	0.0017%	0.0003%	0.0031%
2012	0.000053%	0.0000400%	0.000067%	0.0008347%	0.001043%	0.000627%	0.0017%	0.0003%	0.0031%
2013	0.000053%	0.0000400%	0.000067%	0.0010447%	0.001271%	0.000818%	0.0017%	0.0003%	0.0031%
2014	0.000053%	0.0000400%	0.000067%	0.0011870%	0.001366%	0.001008%	0.0017%	0.0003%	0.0031%
2015	0.000053%	0.0000400%	0.000067%	0.0013429%	0.001487%	0.001199%	0.0017%	0.0003%	0.0031%
Data Sources / Notes	Share of Solar panels vs. LED's determined from ratios for India Jun/Jul 2015 Limra Exim (2015) - see http://www.limraexim.com/hscode-854140-classification-data.aspx. 51.4% of this code is estimated to be LEDs. Of this, we estimate 0.052% are InGaN-based LEDs (Indian share - Zauba.com), with an In content in the InGaN Layer ranging from 15-25% (Lin et al., 2000). The InGaN component is positioned within the die of an LED package (see Marwede et al., 2012). We have assigned a maximum 1% for the weight of InGaN within in a lamp, accounting for portions within the die, and the die within the overall package, but the results can be quite sensisitive to this assumption.			Andersson (2000), Zimmerman & Gossling-Resiemann (2014), Moss et al. (2011), McLellan et al. (2016), Zimmerman (2013), Goe & Gaustad (2014), and specifications for specific models identified from SolarDesignTool.com. Thin film market share data from Adiboina (2006). Share of Solar panels vs. LED's determined from ratios for India Jun/Jul 2015 Limra Exim (2015) - see http://www.limraexim.com/hscode-854140-classification-data.aspx			Indium is used as a thermal interface material in PCBs. According to the India trade records, obtained form Zauba.com. Interfaces represented 128/40943 = 0.312% of this flow. While it is possible that In is present in other flows not labelled as such, indium is still in relatively niche usage for heat management in PCBs (see: http://www.indium.com/blog/thermal-management-using-indium-metal-the-heat-spring.php). For this reasson we make the assumption that indium is present in only this 0.312% portion. Given that it is applied as a thin film/foil, we further only assume a range of 1-5% by weight to represent pure indium. Note also that the Indium Corporation describe this application as "more recent", hence we limit these values to the last 10 years, with reductions assumed beyond this.		

Table C5 (a): Content estimates for the solders and alloys end use sector

	1			2			3			4		
	Al-In Alloys			Sn-In Alloys			Pb-In Alloys			Zn-In Alloys		
HS CODE	760900			800120			780199			790120		
IMPORT												
1962	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1963	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1964	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1965	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1966	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1967	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1968	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1969	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1970	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1971	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1972	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1973	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1974	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1975	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1976	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1977	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1978	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1979	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1980	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1981	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1982	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1983	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1984	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1985	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1986	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1987	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1988	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1989	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1990	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1991	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1992	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1993	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1994	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1995	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1996	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1997	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1998	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
1999	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2000	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2001	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2002	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2003	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2004	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2005	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2006	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2007	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2008	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%

2009	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2010	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2011	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2012	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2013	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2014	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
2015	0.001365%	0.000318%	0.002413%	0.00413%	0.00075%	0.00750%	0.0000041%	0.0000008%	0.0000075%	0.0000041%	0.0000008%	0.0000075%
Data Sources	Al-In alloys known to be traded under this code in India. 1.27% of records traded under this code were indium alloys. Al-In alloys tested for a range of 0.025% to 0.19% In by Despic et al. (1976). 110 records of Al-In alloys out of 8661 trade records leads to 1.27% estimate for the market share for India - assumed to represent Australia too due to the lack of location-specific data. Trade data obtained from Zauba.com			Indium-Tin alloys made up 1 in 95 shipment records for Tin from Zauba.com, representing approximately 0.015% of this flow in monetary terms. Indium content in In-Sn alloys ranges from 5-50%			Uncertain estimates. Based on In-Sn alloys, but a reduction factor of 1000 is applied to imports and exports to account for Australia's considerable Pb-Zn refining capacity that would likely take up a very large portion of the export flows, and given that no indium is reported under these commodity flow for India, despite our understanding that Pb and Zn based alloys containing indium do exist.			Uncertain estimates. Based on In-Sn alloys, but a reduction factor of 1000 is applied to imports and exports to account for Australia's considerable Pb-Zn refining capacity that would likely take up a very large portion of the export flows, and given that no indium is reported under these commodity flow for India, despite our understanding that Pb and Zn based alloys containing indium do exist.		

Table C5 (b): Content estimates for the solders and alloys end use sector (continued)

	5			6			7		
	Solders			Dental alloys			Indium solder pastes		
HS CODE	831130			300640			3810		
IMPORT									
1962	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1963	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1964	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1965	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1966	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1967	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1968	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1969	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1970	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1971	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1972	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1973	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1974	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1975	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1976	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1977	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1978	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1979	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1980	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1981	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1982	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1983	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1984	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1985	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1986	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1987	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1988	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1989	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1990	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1991	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1992	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1993	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1994	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1995	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1996	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%

1997	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1998	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
1999	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2000	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2001	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2002	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2003	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2004	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2005	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2006	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2007	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2008	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2009	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2010	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2011	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2012	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2013	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2014	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
2015	0.01520%	0.00400%	0.04000%	0.0001964%	0.0000357%	0.0003571%	0.00159%	0.00005%	0.00240%
Data sources and notes	A Market share of 0.08% is applied for solders, following personal communication with Brian O'Neill, Indium Manager for AIM Minor Metals (adivsed that 4 tons per month out of 5000 tons per month are indium-based solders). The average indium content within solders is dependent on the most common applications, with the 47 deg C melting point alloy the most commonly use (at 19.1% In). The minimum In content is assigned at 5% In and the maximum 50%, see Slattery (2010). No data as yet to make the content estimates dynamic. We know that there are no local manufacturers of indium solder, and hence the zero estimates are applied for exports.			Uncertain estimates. Based on In-Sn alloys reduced by a factor of 21 to account for the porportion of indium-based dental alloys in Updadhay et al (2006). No records of indium mentioned in Indian trade.			Pastes traded under 381010 and 381090, weith contents assumed to range between 0.2% to 8.9% within the paste. ~0.027% of items traded under this code represent indium-containing pastes according to Zauba.com (analysis in 2016).		

Table C6 (a): Content estimates for the coatings end use sector

	1			2			3			4		
Description	Laptop Computers + Tablets - Portable automatic data processing machines, weighing not more than 10 kg, consisting of at least a central processing unit, a keyboard and a display			Mobile Phones			LCD/LED Screens - PC			LCD/LED Screens - Other		
HS Code	847130			852520 (1988-2006) + 851712 (2007-)			847160 (1996-2006) + 852851 (2007-)			852821 (1996-2006) + 852859 (2007-)		
Period	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top
EXPORT												
1962	0.001012%	0.000506%	0.002024%									
1963	0.001012%	0.000506%	0.002024%									
1964	0.001012%	0.000506%	0.002024%									
1965	0.001012%	0.000506%	0.002024%									
1966	0.001012%	0.000506%	0.002024%									
1967	0.001012%	0.000506%	0.002024%									
1968	0.001012%	0.000506%	0.002024%									
1969	0.001012%	0.000506%	0.002024%									
1970	0.001012%	0.000506%	0.002024%									

1971	0.001012%	0.000506%	0.002024%									
1972	0.001012%	0.000506%	0.002024%									
1973	0.001012%	0.000506%	0.002024%									
1974	0.001012%	0.000506%	0.002024%									
1975	0.001012%	0.000506%	0.002024%									
1976	0.001012%	0.000506%	0.002024%									
1977	0.001012%	0.000506%	0.002024%									
1978	0.001012%	0.000506%	0.002024%									
1979	0.001012%	0.000506%	0.002024%									
1980	0.001012%	0.000506%	0.002024%									
1981	0.001012%	0.000506%	0.002024%									
1982	0.001012%	0.000506%	0.002024%									
1983	0.001012%	0.000506%	0.002024%									
1984	0.001012%	0.000506%	0.002024%									
1985	0.001012%	0.000506%	0.002024%									
1986	0.001012%	0.000506%	0.002024%									
1987	0.001012%	0.000506%	0.002024%									
1988	0.001012%	0.000506%	0.002024%									
1989	0.001012%	0.000506%	0.002024%									
1990	0.001012%	0.000506%	0.002024%									
1991	0.001012%	0.000506%	0.002024%									
1992	0.001012%	0.000506%	0.002024%									
1993	0.001012%	0.000506%	0.002024%									
1994	0.001012%	0.000506%	0.002024%									
1995	0.001012%	0.000506%	0.002024%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1996	0.001012%	0.000506%	0.002024%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1997	0.001012%	0.000506%	0.002024%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1998	0.001012%	0.000506%	0.002024%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1999	0.001012%	0.000506%	0.002024%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
2000	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
2001	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00022%	0.00013%	0.00033%	0.00022%	0.00013%	0.00033%
2002	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00032%	0.00020%	0.00050%	0.00032%	0.00020%	0.00050%
2003	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00043%	0.00027%	0.00067%	0.00043%	0.00027%	0.00067%
2004	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00054%	0.00033%	0.00084%	0.00054%	0.00033%	0.00084%
2005	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00065%	0.00040%	0.00100%	0.00065%	0.00040%	0.00100%
2006	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00076%	0.00047%	0.00117%	0.00076%	0.00047%	0.00117%
2007	0.001687%	0.000843%	0.003374%	0.00197%	0.00099%	0.00395%	0.00087%	0.00053%	0.00134%	0.00087%	0.00053%	0.00134%
2008	0.001687%	0.000843%	0.003374%	0.00201%	0.00100%	0.00401%	0.00097%	0.00060%	0.00150%	0.00097%	0.00060%	0.00150%
2009	0.001687%	0.000843%	0.003374%	0.00204%	0.00102%	0.00408%	0.00108%	0.00067%	0.00167%	0.00108%	0.00067%	0.00167%
2010	0.001687%	0.000843%	0.003374%	0.00217%	0.00109%	0.00434%	0.00119%	0.00074%	0.00184%	0.00119%	0.00074%	0.00184%
2011	0.001687%	0.000843%	0.003374%	0.00237%	0.00118%	0.00473%	0.00130%	0.00080%	0.00201%	0.00130%	0.00080%	0.00201%
2012	0.001687%	0.000843%	0.003374%	0.00253%	0.00127%	0.00506%	0.00141%	0.00087%	0.00217%	0.00141%	0.00087%	0.00217%

2013	0.001687%	0.000843%	0.003374%	0.00283%	0.00141%	0.00566%	0.00152%	0.00094%	0.00234%	0.00152%	0.00094%	0.00234%
2014	0.001687%	0.000843%	0.003374%	0.00329%	0.00164%	0.00658%	0.00162%	0.00100%	0.00251%	0.00162%	0.00100%	0.00251%
2015	0.001687%	0.000843%	0.003374%	0.00329%	0.00164%	0.00658%	0.00173%	0.00107%	0.00267%	0.00173%	0.00107%	0.00267%
IMPORT												
1962	0.0010122%	0.000506%	0.002024%									
1963	0.0010122%	0.000506%	0.002024%									
1964	0.0010122%	0.000506%	0.002024%									
1965	0.0010122%	0.000506%	0.002024%									
1966	0.0010122%	0.000506%	0.002024%									
1967	0.0010122%	0.000506%	0.002024%									
1968	0.0010122%	0.000506%	0.002024%									
1969	0.0010122%	0.000506%	0.002024%									
1970	0.0010122%	0.000506%	0.002024%									
1971	0.0010122%	0.000506%	0.002024%									
1972	0.0010122%	0.000506%	0.002024%									
1973	0.0010122%	0.000506%	0.002024%									
1974	0.0010122%	0.000506%	0.002024%									
1975	0.0010122%	0.000506%	0.002024%									
1976	0.0010122%	0.000506%	0.002024%									
1977	0.0010122%	0.000506%	0.002024%									
1978	0.0010122%	0.000506%	0.002024%									
1979	0.0010122%	0.000506%	0.002024%									
1980	0.0010122%	0.000506%	0.002024%									
1981	0.0010122%	0.000506%	0.002024%									
1982	0.0010122%	0.000506%	0.002024%									
1983	0.0010122%	0.000506%	0.002024%									
1984	0.0010122%	0.000506%	0.002024%									
1985	0.0010122%	0.000506%	0.002024%									
1986	0.0010122%	0.000506%	0.002024%									
1987	0.0010122%	0.000506%	0.002024%									
1988	0.0010122%	0.000506%	0.002024%									
1989	0.0010122%	0.000506%	0.002024%									
1990	0.0010122%	0.000506%	0.002024%									
1991	0.0010122%	0.000506%	0.002024%									
1992	0.0010122%	0.000506%	0.002024%									
1993	0.0010122%	0.000506%	0.002024%									
1994	0.0010122%	0.000506%	0.002024%									
1995	0.0010122%	0.000506%	0.002024%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1996	0.0010122%	0.000506%	0.002024%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1997	0.0010122%	0.000506%	0.002024%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
1998	0.0010122%	0.000506%	0.002024%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%

1999	0.0010122%	0.000506%	0.002024%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
2000	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00011%	0.00007%	0.00017%	0.00011%	0.00007%	0.00017%
2001	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00022%	0.00013%	0.00033%	0.00022%	0.00013%	0.00033%
2002	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00032%	0.00020%	0.00050%	0.00032%	0.00020%	0.00050%
2003	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00043%	0.00027%	0.00067%	0.00043%	0.00027%	0.00067%
2004	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00054%	0.00033%	0.00084%	0.00054%	0.00033%	0.00084%
2005	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00065%	0.00040%	0.00100%	0.00065%	0.00040%	0.00100%
2006	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00076%	0.00047%	0.00117%	0.00076%	0.00047%	0.00117%
2007	0.0016870%	0.000843%	0.003374%	0.00197%	0.00120%	0.00274%	0.00087%	0.00053%	0.00134%	0.00087%	0.00053%	0.00134%
2008	0.0016870%	0.000843%	0.003374%	0.00201%	0.00122%	0.00279%	0.00097%	0.00060%	0.00150%	0.00097%	0.00060%	0.00150%
2009	0.0016870%	0.000843%	0.003374%	0.00204%	0.00124%	0.00283%	0.00108%	0.00067%	0.00167%	0.00108%	0.00067%	0.00167%
2010	0.0033630%	0.002851%	0.004679%	0.00217%	0.00132%	0.00302%	0.00119%	0.00074%	0.00184%	0.00119%	0.00074%	0.00184%
2011	0.0033630%	0.002851%	0.004679%	0.00237%	0.00145%	0.00329%	0.00130%	0.00080%	0.00201%	0.00130%	0.00080%	0.00201%
2012	0.0033630%	0.002851%	0.004679%	0.00253%	0.00155%	0.00352%	0.00141%	0.00087%	0.00217%	0.00141%	0.00087%	0.00217%
2013	0.0033630%	0.002851%	0.004679%	0.00283%	0.00173%	0.00393%	0.00152%	0.00094%	0.00234%	0.00152%	0.00094%	0.00234%
2014	0.0033630%	0.002851%	0.004679%	0.00329%	0.00201%	0.00457%	0.00162%	0.00100%	0.00251%	0.00162%	0.00100%	0.00251%
2015	0.0033630%	0.002851%	0.004679%	0.00329%	0.00201%	0.00457%	0.00173%	0.00107%	0.00267%	0.00173%	0.00107%	0.00267%
Data Sources	Cuchiella et al. (2015), Buchert et al. (2012), Shingkikai (2011), Product Chart Database (Gibney 2016), von Gries & Wilts (2014), Zeng et al (2016). Content for Tablets added from 2010 onwards.			Barredo (2016), Cuchiella et al. (2015), Takahashi et al. (2009), MOE & METI (2010), von Gries & Wilts (2014), Product Chart Database (Gibney, 2016), Zeng et al (2016)			Cucchiella et al. (2015), Buchert et al. (2015), Zeng et al. (2016), von Gries & Wilts (2014).			Estimated with the same content as for PC LCD/LED Screens. - Therefore the same references were used.		

Table C6 (b): Content estimates for the coatings end use sector (continued)

	5			6			7			8		
Description	Smaller LCD Displays			Cameras			Flat Display Panel TVs (LCD, LED, Plasma)			Other Portable Electronics (MP3 Players, e-Readers, Car Navigation, etc)		
HS CODE	853120			852540 (1996-2006) + 852580 (2007-)			852812 (1996-2006) + 852872 (2007-)			8527-		
Period	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top
EXPORT												
1962	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1963	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1964	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1965	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1966	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1967	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1968	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1969	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1970	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1971	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1972	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1973	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1974	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%

1975	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1976	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1977	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1978	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1979	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1980	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1981	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1982	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1983	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1984	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1985	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1986	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1987	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1988	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1989	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1990	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1991	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1992	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1993	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1994	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1995	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1996	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1997	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1998	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.000129%	0.000074%	0.000298%	0.000005%	0.000003%	0.000008%
1999	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.000129%	0.000074%	0.000298%	0.000005%	0.000003%	0.000008%
2000	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.000129%	0.000074%	0.000298%	0.000005%	0.000003%	0.000008%
2001	0.00022%	0.00013%	0.00033%	0.000015%	0.000009%	0.000023%	0.000259%	0.000149%	0.000595%	0.000010%	0.000006%	0.000016%
2002	0.00032%	0.00020%	0.00050%	0.000023%	0.000014%	0.000035%	0.000527%	0.000298%	0.001190%	0.000016%	0.000010%	0.000024%
2003	0.00043%	0.00027%	0.00067%	0.000030%	0.000019%	0.000047%	0.000802%	0.000446%	0.001785%	0.000021%	0.000013%	0.000032%
2004	0.00054%	0.00033%	0.00084%	0.000038%	0.000023%	0.000059%	0.001090%	0.000595%	0.002380%	0.000026%	0.000016%	0.000040%
2005	0.00065%	0.00040%	0.00100%	0.000045%	0.000028%	0.000070%	0.001389%	0.000744%	0.002975%	0.000031%	0.000019%	0.000048%
2006	0.00076%	0.00047%	0.00117%	0.000053%	0.000033%	0.000082%	0.001702%	0.000893%	0.003570%	0.000036%	0.000022%	0.000056%
2007	0.00087%	0.00053%	0.00134%	0.000061%	0.000037%	0.000094%	0.002031%	0.001041%	0.004165%	0.000041%	0.000026%	0.000064%
2008	0.00097%	0.00060%	0.00150%	0.000068%	0.000042%	0.000105%	0.002373%	0.001190%	0.004760%	0.000047%	0.000029%	0.000072%
2009	0.00108%	0.00067%	0.00167%	0.000076%	0.000047%	0.000117%	0.002739%	0.001339%	0.005355%	0.000052%	0.000032%	0.000080%
2010	0.00119%	0.00074%	0.00184%	0.000083%	0.000051%	0.000129%	0.003130%	0.001488%	0.005950%	0.000057%	0.000035%	0.000088%
2011	0.00130%	0.00080%	0.00201%	0.000091%	0.000056%	0.000140%	0.003228%	0.001488%	0.005950%	0.000062%	0.000038%	0.000096%
2012	0.00141%	0.00087%	0.00217%	0.000098%	0.000061%	0.000152%	0.003331%	0.001488%	0.005950%	0.000067%	0.000042%	0.000104%
2013	0.00152%	0.00094%	0.00234%	0.000106%	0.000066%	0.000164%	0.003448%	0.001488%	0.005950%	0.000073%	0.000045%	0.000112%
2014	0.00162%	0.00100%	0.00251%	0.000114%	0.000070%	0.000176%	0.003581%	0.001488%	0.005950%	0.000078%	0.000048%	0.000120%
2015	0.00173%	0.00107%	0.00267%	0.000121%	0.000075%	0.000187%	0.003733%	0.001488%	0.005950%	0.000083%	0.000051%	0.000128%
IMPORT												
1962	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1963	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1964	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1965	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1966	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%

1967	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1968	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1969	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1970	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1971	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1972	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1973	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1974	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1975	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1976	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1977	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1978	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1979	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1980	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1981	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1982	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1983	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1984	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1985	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1986	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1987	0.00011%	0.00007%	0.00017%							0.000005%	0.000003%	0.000008%
1988	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1989	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1990	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1991	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1992	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1993	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1994	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1995	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1996	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1997	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%				0.000005%	0.000003%	0.000008%
1998	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.0001285%	0.0000746%	0.0001852%	0.000005%	0.000003%	0.000008%
1999	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.0001285%	0.0000746%	0.0001852%	0.000005%	0.000003%	0.000008%
2000	0.00011%	0.00007%	0.00017%	0.000008%	0.000005%	0.000012%	0.0001285%	0.0000746%	0.0001852%	0.000005%	0.000003%	0.000008%
2001	0.00022%	0.00013%	0.00033%	0.000015%	0.000009%	0.000023%	0.0002591%	0.0001493%	0.0003745%	0.000010%	0.000006%	0.000016%
2002	0.00032%	0.00020%	0.00050%	0.000023%	0.000014%	0.000035%	0.0005267%	0.0002987%	0.0007665%	0.000016%	0.000010%	0.000024%
2003	0.00043%	0.00027%	0.00067%	0.001065%	0.000019%	0.000047%	0.0008024%	0.0004481%	0.0011751%	0.000021%	0.000013%	0.000032%
2004	0.00054%	0.00033%	0.00084%	0.001069%	0.000023%	0.000059%	0.0010898%	0.0005975%	0.0016075%	0.000026%	0.000016%	0.000040%
2005	0.00065%	0.00040%	0.00100%	0.001073%	0.000028%	0.000070%	0.0013893%	0.0007470%	0.0020646%	0.000031%	0.000019%	0.000048%
2006	0.00076%	0.00047%	0.00117%	0.001077%	0.000033%	0.000082%	0.0017025%	0.0008966%	0.0025499%	0.000036%	0.000022%	0.000056%
2007	0.00087%	0.00053%	0.00134%	0.001080%	0.000037%	0.000094%	0.0020314%	0.0010463%	0.0030675%	0.000041%	0.000026%	0.000064%
2008	0.00097%	0.00060%	0.00150%	0.001084%	0.000042%	0.000105%	0.0023726%	0.0011960%	0.0036099%	0.000047%	0.000029%	0.000072%
2009	0.00108%	0.00067%	0.00167%	0.001088%	0.000047%	0.000117%	0.0027392%	0.0013459%	0.0042044%	0.000052%	0.000032%	0.000080%
2010	0.00119%	0.00074%	0.00184%	0.001092%	0.000051%	0.000129%	0.0031300%	0.0014958%	0.0048484%	0.000057%	0.000035%	0.000088%
2011	0.00130%	0.00080%	0.00201%	0.001095%	0.000056%	0.000140%	0.0032277%	0.0014963%	0.0050483%	0.000062%	0.000038%	0.000096%
2012	0.00141%	0.00087%	0.00217%	0.001099%	0.000061%	0.000152%	0.0033313%	0.0014969%	0.0052605%	0.000067%	0.000042%	0.000104%
2013	0.00152%	0.00094%	0.00234%	0.001103%	0.000066%	0.000164%	0.0034482%	0.0014974%	0.0054996%	0.000073%	0.000045%	0.000112%
2014	0.00162%	0.00100%	0.00251%	0.001107%	0.000070%	0.000176%	0.0035808%	0.0014981%	0.0057711%	0.000078%	0.000048%	0.000120%

2015	0.00173%	0.00107%	0.00267%	0.001111%	0.000075%	0.000187%	0.0037327%	0.0014989%	0.0060819%	0.000083%	0.000051%	0.000128%
Data Sources	It is assumed that the same weight ratio of In in smaller LCD displays also applies. This is a conservative estimation, as the weight of smaller LCD displays/LCD devices is also much smaller, suggesting a possible increase in content, however there is no guarantee that the same ratio of screen size also applies. In the absence of sufficient data to alter the other estimates, we apply the same ratios, but with larger uncertainty bounds. We believe this is a reasonable assumption as the content for Laptops, Mobiles and LCD Screens changes very little on a weight percentage basis, depiste their different weights and configurations.			Content scaled down to the relative weight of the cameras. It is assumed that these devices incorporate LCD displays for viewing. Digital formats for cameras were not introduced until 1988. von Gries & Wilts (2014) put In content at 0.0021 wt% In. This value is averaged with the other method for 2003 onwards, when digital cameras (incorporating LCD displays) became more prominent.			Nakajima et al. (2007), Buchert et al. (2012), Chancerel et al. (2015), Cucchiella et al. (2015), Shingkikai (2011), von Gires & Wilts (2014) and Zeng et al (2016). See Content- Screens spreadsheet. Max/Min values represent the cosest estimates to plus or minus one SD of the mean derived from the aforementioned studies.			Content scaled down by the relative average weight of these devices, from the content of 853120. Note that von Gries & Wilts (2014) put In content for Navigation systems at 0.01 wt% In.		

Table C6 (c): Content estimates for the coatings end use sector (continued)

	9			10			11		
Description	Cameras for film <16mm			Cameras for film >16mm			Motor vehicles for transport of persons (except buses)		
HS CODE	900711 (1988-2011) + 900710 (2011-)			900719			8703		
Period	**Average**	**Bottom**	**Top**	**Average**	**Bottom**	**Top**	**Average**	**Bottom**	**Top**
EXPORT									
1962									
1963									
1964									
1965									
1966									
1967									
1968									
1969									
1970									
1971									
1972									
1973									
1974									
1975									
1976									
1977									
1978									
1979									
1980									
1981									
1982									
1983									
1984									
1985									
1986									
1987									
1988									
1989							1.1219E-08	7.38095E-11	3.48381E-08
1990							1.1219E-08	7.38095E-11	3.48381E-08
1991							1.1219E-08	7.38095E-11	3.48381E-08
1992							1.1219E-08	7.38095E-11	3.48381E-08
1993							1.1219E-08	7.38095E-11	3.48381E-08
1994							1.1219E-08	7.38095E-11	3.48381E-08
1995							5.60952E-08	3.69047E-10	1.7419E-07
1996							5.60952E-08	3.69047E-10	1.7419E-07
1997							5.60952E-08	3.69047E-10	1.7419E-07
1998							5.60952E-08	3.69047E-10	1.7419E-07
1999							5.60952E-08	3.69047E-10	1.7419E-07
2000							5.60952E-08	3.69047E-10	1.7419E-07

2001							5.60952E-08	3.69047E-10	1.7419E-07
2002							5.60952E-08	3.69047E-10	1.7419E-07
2003							5.60952E-08	3.69047E-10	1.7419E-07
2004							5.60952E-08	3.69047E-10	1.7419E-07
2005							0.00001122%	7.38095E-10	3.48381E-07
2006							0.00001122%	7.38095E-10	3.48381E-07
2007							0.00001122%	7.38095E-10	3.48381E-07
2008							0.00001122%	7.38095E-10	3.48381E-07
2009							0.00001122%	7.38095E-10	3.48381E-07
2010							0.00001122%	7.38095E-10	3.48381E-07
2011							0.00001122%	7.38095E-10	3.48381E-07
2012							0.00001122%	7.38095E-10	3.48381E-07
2013							0.00001122%	7.38095E-10	3.48381E-07
2014							0.00001122%	7.38095E-10	3.48381E-07
2015							0.00001122%	7.38095E-10	3.48381E-07
IMPORT									
1962									
1963									
1964									
1965									
1966									
1967									
1968									
1969									
1970									
1971									
1972									
1973									
1974									
1975									
1976									
1977									
1978									
1979									
1980									
1981									
1982									
1983									
1984									
1985									
1986									
1987									
1988	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%			
1989	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1990	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1991	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1992	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1993	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1994	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	1.1219E-08	7.38095E-11	3.48381E-08
1995	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
1996	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
1997	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
1998	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
1999	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
2000	0.00001%	0.00000%	0.00001%	0.00001%	0.00000%	0.00001%	5.60952E-08	3.69047E-10	1.7419E-07
2001	0.00002%	0.00001%	0.00002%	0.00002%	0.00001%	0.00002%	5.60952E-08	3.69047E-10	1.7419E-07
2002	0.00002%	0.00001%	0.00004%	0.00002%	0.00001%	0.00004%	5.60952E-08	3.69047E-10	1.7419E-07
2003	0.00107%	0.00002%	0.00005%	0.00107%	0.00002%	0.00005%	5.60952E-08	3.69047E-10	1.7419E-07
2004	0.00107%	0.00002%	0.00006%	0.00107%	0.00002%	0.00006%	5.60952E-08	3.69047E-10	1.7419E-07
2005	0.00107%	0.00003%	0.00007%	0.00107%	0.00003%	0.00007%	0.00001122%	7.38095E-10	3.48381E-07
2006	0.00108%	0.00003%	0.00008%	0.00108%	0.00003%	0.00008%	0.00001122%	7.38095E-10	3.48381E-07
2007	0.00108%	0.00004%	0.00009%	0.00108%	0.00004%	0.00009%	0.00001122%	7.38095E-10	3.48381E-07
2008	0.00108%	0.00004%	0.00011%	0.00108%	0.00004%	0.00011%	0.00001122%	7.38095E-10	3.48381E-07
2009	0.00109%	0.00005%	0.00012%	0.00109%	0.00005%	0.00012%	0.00001122%	7.38095E-10	3.48381E-07
2010	0.00109%	0.00005%	0.00013%	0.00109%	0.00005%	0.00013%	0.00001122%	7.38095E-10	3.48381E-07

2011	0.00110%	0.00006%	0.00014%	0.00110%	0.00006%	0.00014%	0.00001122%	7.38095E-10	3.48381E-07
2012	0.00110%	0.00006%	0.00015%	0.00110%	0.00006%	0.00015%	0.00001122%	7.38095E-10	3.48381E-07
2013	0.00110%	0.00007%	0.00016%	0.00110%	0.00007%	0.00016%	0.00001122%	7.38095E-10	3.48381E-07
2014	0.00111%	0.00007%	0.00018%	0.00111%	0.00007%	0.00018%	0.00001122%	7.38095E-10	3.48381E-07
2015	**0.00111%**	**0.00007%**	**0.00019%**	**0.00111%**	**0.00007%**	**0.00019%**	0.00001122%	7.38095E-10	3.48381E-07
Data Sources	Same content applied as for other cameras. Scaled by relative weight of the product, assuming that the screen sizes remain consistent.			Same content applied as for other cameras. Scaled by relative weight of the product, assuming that the screen sizes remain consistent.			Average vehicle weight is 1354.84 kg / vehicle in the trade data. Content of In per passenger vehicle is determined from Min-Mean-Max values available in Du et al. (2015). Some scaling is applied on the assumption that LCDs have become more prominent in vehicles over time, however the extent of this increase is not known to the authors. Electric Vehicles considered to contain 0.03-0.05 g In per vehicle as per Table 5 in Grandell et al. (2016)		

Table C7 (a): Content estimates for the research and other end use sector

	1			2			3			4		
HS Description	Alkaline batteries			Indium coated glass for Laboratory purposes			Indium samples for test equipment - Microtomes, parts of scientific analysis equipment			Measuring or checking equipment nes - calibration equipment (e.g. "FIXED POINT CELL TIN CELL MINI METAL CASED INDIUM, 69 WITH CALIBRATION CERTIFICATION (1904-SN) C"		
HS CODE	850610 (1988-1995) + 850611 (1996-)			701790			902790 (90)			903180		
Period	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top
EXPORT												
1962	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1963	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1964	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1965	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1966	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1967	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1968	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1969	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1970	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1971	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1972	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1973	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1974	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1975	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1976	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1977	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1978	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1979	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1980	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1981	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1982	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1983	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%

1984	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1985	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1986	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1987	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1988	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1989	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001795%	0.000003611%
1990	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001719%	0.000003611%
1991	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000002139%	0.000003611%
1992	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000002002%	0.000003611%
1993	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001982%	0.000003611%
1994	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001750%	0.000003611%
1995	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001812%	0.000003611%
1996	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001813%	0.000003611%
1997	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001858%	0.000003611%
1998	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001682%	0.000003611%
1999	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001503%	0.000003611%
2000	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001572%	0.000003611%
2001	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001383%	0.000003611%
2002	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001358%	0.000003611%
2003	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001362%	0.000003611%
2004	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001533%	0.000003611%
2005	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001500%	0.000003611%
2006	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001421%	0.000003611%
2007	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001243%	0.000003611%
2008	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001555%	0.000003611%
2009	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001225%	0.000003611%
2010	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001207%	0.000003611%
2011	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001382%	0.000003611%
2012	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001521%	0.000003611%
2013	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001516%	0.000003611%
2014	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001286%	0.000003611%
2015	0.00050%	0.00000%	0.00100%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001166%	0.000003611%
IMPORT												
1962	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1963	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1964	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1965	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1966	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1967	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1968	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1969	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1970	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%

1971	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1972	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1973	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1974	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1975	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1976	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1977	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1978	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1979	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1980	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1981	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1982	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1983	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1984	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1985	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1986	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1987	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1988	0.00007%	0.00000%	0.00014%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001411%	0.000003611%
1989	0.00008%	0.00000%	0.00015%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001795%	0.000003611%
1990	0.00008%	0.00000%	0.00016%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001719%	0.000003611%
1991	0.00009%	0.00000%	0.00019%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000002139%	0.000003611%
1992	0.00009%	0.00000%	0.00019%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000002002%	0.000003611%
1993	0.00009%	0.00000%	0.00019%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001982%	0.000003611%
1994	0.00010%	0.00000%	0.00019%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001750%	0.000003611%
1995	0.00016%	0.00000%	0.00032%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001812%	0.000003611%
1996	0.00013%	0.00000%	0.00026%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001813%	0.000003611%
1997	0.00015%	0.00000%	0.00031%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001858%	0.000003611%
1998	0.00015%	0.00000%	0.00030%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001682%	0.000003611%
1999	0.00014%	0.00000%	0.00029%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001503%	0.000003611%
2000	0.00022%	0.00000%	0.00044%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001572%	0.000003611%
2001	0.00023%	0.00000%	0.00046%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001383%	0.000003611%
2002	0.00022%	0.00000%	0.00044%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001358%	0.000003611%
2003	0.00025%	0.00000%	0.00049%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001362%	0.000003611%
2004	0.00027%	0.00000%	0.00054%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001533%	0.000003611%
2005	0.00033%	0.00000%	0.00066%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001500%	0.000003611%
2006	0.00039%	0.00000%	0.00077%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001421%	0.000003611%
2007	0.00037%	0.00000%	0.00075%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001243%	0.000003611%
2008	0.00038%	0.00000%	0.00076%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001555%	0.000003611%
2009	0.00036%	0.00000%	0.00072%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001225%	0.000003611%
2010	0.00040%	0.00000%	0.00081%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001207%	0.000003611%
2011	0.00044%	0.00000%	0.00088%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001382%	0.000003611%

2012	0.00052%	0.00000%	0.00104%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001521%	0.000003611%
2013	0.00053%	0.00000%	0.00106%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001516%	0.000003611%
2014	0.00056%	0.00000%	0.00112%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001286%	0.000003611%
2015	0.00056%	0.00000%	0.00112%	0.001646%	0.000299%	0.00299%	0.00031553%	0.00021035%	0.00042070%	0.000001806%	0.000001166%	0.000003611%
Data Sources	Assumes a range of 100 to 1000 ppm per cell, and applies to between 0.1 amd 1% of the total battery market. Market share assumed based on the limited applications (e.g. reactor and high-temp processing, telemetry systems and military uses) of indium batteries. See Crompton (2000). More detailed market share data on indium batteries is needed. Scaled down in time according to total indium demand.			Based on 170 reports of indium-coated glass out for 4204 for 2014-2016 trade in India, we estimate 4.044% of the trade is indium-coated (ITO) glass. In content of ITO is at ~74%, and the weight of ITO vs. the weight of the glass itself is set at a maximum of 0.1% given that it is a thin film, however this is assumed as we do not have a reference for the relative weight of the film vs the weight of the glass.			Indium samples traded under this code from Indian records lead to = 103 records of / 24,482,737 = 0.00042070%. Samples generally described as pure indium for research purposes, however assumed 50% at minimum.			Uncertain estimate. Contents derived from the same proportions assigned to HS Code 903190 (relating to indium standards).		

Table C7 (b): Content estimates for the research and other end use sector (continued)

	5			6			7			8		
Description	Indium standard			Indium standard			In Trichloride			In Sulfate		
HS CODE	903190			382200			282739			283329		
Period	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top	Average	Bottom	Top
EXPORT												
1962	0.000001806%	0.000001411%	0.000003611%									
1963	0.000001806%	0.000001411%	0.000003611%									
1964	0.000001806%	0.000001411%	0.000003611%									
1965	0.000001806%	0.000001411%	0.000003611%									
1966	0.000001806%	0.000001411%	0.000003611%									
1967	0.000001806%	0.000001411%	0.000003611%									
1968	0.000001806%	0.000001411%	0.000003611%									
1969	0.000001806%	0.000001411%	0.000003611%									
1970	0.000001806%	0.000001411%	0.000003611%									
1971	0.000001806%	0.000001411%	0.000003611%									
1972	0.000001806%	0.000001411%	0.000003611%									
1973	0.000001806%	0.000001411%	0.000003611%									
1974	0.000001806%	0.000001411%	0.000003611%									
1975	0.000001806%	0.000001411%	0.000003611%									
1976	0.000001806%	0.000001411%	0.000003611%									
1977	0.000001806%	0.000001411%	0.000003611%									
1978	0.000001806%	0.000001411%	0.000003611%									
1979	0.000001806%	0.000001411%	0.000003611%									
1980	0.000001806%	0.000001411%	0.000003611%									
1981	0.000001806%	0.000001411%	0.000003611%									

1982	0.000001806%	0.000001411%	0.000003611%									
1983	0.000001806%	0.000001411%	0.000003611%									
1984	0.000001806%	0.000001411%	0.000003611%									
1985	0.000001806%	0.000001411%	0.000003611%									
1986	0.000001806%	0.000001411%	0.000003611%									
1987	0.000001806%	0.000001411%	0.000003611%									
1988	0.000001806%	0.000001411%	0.000003611%									
1989	0.000001806%	0.000001795%	0.000003611%									
1990	0.000001806%	0.000001719%	0.000003611%									
1991	0.000001806%	0.000002139%	0.000003611%									
1992	0.000001806%	0.000002002%	0.000003611%									
1993	0.000001806%	0.000001982%	0.000003611%									
1994	0.000001806%	0.000001750%	0.000003611%									
1995	0.000001806%	0.000001812%	0.000003611%									
1996	0.000001806%	0.000001813%	0.000003611%									
1997	0.000001806%	0.000001858%	0.000003611%									
1998	0.000001806%	0.000001682%	0.000003611%									
1999	0.000001806%	0.000001503%	0.000003611%									
2000	0.000001806%	0.000001572%	0.000003611%									
2001	0.000001806%	0.000001383%	0.000003611%									
2002	0.000001806%	0.000001358%	0.000003611%									
2003	0.000001806%	0.000001362%	0.000003611%									
2004	0.000001806%	0.000001533%	0.000003611%									
2005	0.000001806%	0.000001500%	0.000003611%									
2006	0.000001806%	0.000001421%	0.000003611%									
2007	0.000001806%	0.000001243%	0.000003611%									
2008	0.000001806%	0.000001555%	0.000003611%									
2009	0.000001806%	0.000001225%	0.000003611%									
2010	0.000001806%	0.000001207%	0.000003611%									
2011	0.000001806%	0.000001382%	0.000003611%									
2012	0.000001806%	0.000001521%	0.000003611%									
2013	0.000001806%	0.000001516%	0.000003611%									
2014	0.000001806%	0.000001286%	0.000003611%									
2015	0.000001806%	0.000001166%	0.000003611%									
IMPORT												
1962	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1963	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1964	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1965	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1966	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1967	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						

1968	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1969	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1970	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1971	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1972	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1973	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1974	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1975	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1976	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1977	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1978	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1979	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1980	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1981	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1982	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1983	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1984	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1985	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1986	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1987	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%						
1988	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000191%	0.000093%	0.000290%	0.000122%	0.000069%	0.000175%
1989	0.0000018%	0.0000018%	0.0000036%	0.0000017%	0.0%	0.0%	0.000236%	0.000127%	0.000345%	0.000145%	0.000049%	0.000241%
1990	0.0000018%	0.0000017%	0.0000036%	0.0000017%	0.0%	0.0%	0.000238%	0.000160%	0.000316%	0.000139%	0.000041%	0.000237%
1991	0.0000018%	0.0000021%	0.0000036%	0.0000017%	0.0%	0.0%	0.000339%	0.000366%	0.000313%	0.000193%	0.000036%	0.000350%
1992	0.0000018%	0.0000020%	0.0000036%	0.0000017%	0.0%	0.0%	0.000570%	0.000844%	0.000296%	0.000176%	0.000024%	0.000327%
1993	0.0000018%	0.0000020%	0.0000036%	0.0000017%	0.0%	0.0%	0.001273%	0.002267%	0.000279%	0.000172%	0.000019%	0.000324%
1994	0.0000018%	0.0000018%	0.0000036%	0.0000017%	0.0%	0.0%	0.000848%	0.001427%	0.000269%	0.000156%	0.000016%	0.000296%
1995	0.0000018%	0.0000018%	0.0000036%	0.0000017%	0.0%	0.0%	0.000334%	0.000392%	0.000276%	0.000267%	0.000028%	0.000506%
1996	0.0000018%	0.0000018%	0.0000036%	0.0000017%	0.0%	0.0%	0.000370%	0.000446%	0.000293%	0.000218%	0.000013%	0.000423%
1997	0.0000018%	0.0000019%	0.0000036%	0.0000017%	0.0%	0.0%	0.000507%	0.000696%	0.000318%	0.000258%	0.000013%	0.000504%
1998	0.0000018%	0.0000017%	0.0000036%	0.0000017%	0.0%	0.0%	0.000371%	0.000449%	0.000292%	0.000234%	0.000016%	0.000452%
1999	0.0000018%	0.0000015%	0.0000036%	0.0000017%	0.0%	0.0%	0.000349%	0.000422%	0.000277%	0.000197%	0.000012%	0.000381%
2000	0.0000018%	0.0000016%	0.0000036%	0.0000017%	0.0%	0.0%	0.000728%	0.001168%	0.000287%	0.000316%	0.000017%	0.000615%
2001	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000587%	0.000911%	0.000262%	0.000289%	0.000020%	0.000557%
2002	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000559%	0.000857%	0.000262%	0.000272%	0.000014%	0.000531%
2003	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000521%	0.000768%	0.000274%	0.000301%	0.000013%	0.000588%
2004	0.0000018%	0.0000015%	0.0000036%	0.0000017%	0.0%	0.0%	0.000541%	0.000765%	0.000316%	0.000369%	0.000012%	0.000725%
2005	0.0000018%	0.0000015%	0.0000036%	0.0000017%	0.0%	0.0%	0.000519%	0.000711%	0.000327%	0.000451%	0.000025%	0.000878%
2006	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000662%	0.001002%	0.000321%	0.000496%	0.000025%	0.000966%
2007	0.0000018%	0.0000012%	0.0000036%	0.0000017%	0.0%	0.0%	0.000462%	0.000604%	0.000320%	0.000415%	0.000013%	0.000818%
2008	0.0000018%	0.0000016%	0.0000036%	0.0000017%	0.0%	0.0%	0.000441%	0.000526%	0.000355%	0.000526%	0.000012%	0.001041%
2009	0.0000018%	0.0000012%	0.0000036%	0.0000017%	0.0%	0.0%	0.000273%	0.000217%	0.000329%	0.000401%	0.000020%	0.000781%

2010	0.0000018%	0.0000012%	0.0000036%	0.0000017%	0.0%	0.0%	0.000321%	0.000278%	0.000365%	0.000437%	0.000015%	0.000859%
2011	0.0000018%	0.0000014%	0.0000036%	0.0000017%	0.0%	0.0%	0.000306%	0.000216%	0.000396%	0.000542%	0.000015%	0.001068%
2012	0.0000018%	0.0000015%	0.0000036%	0.0000017%	0.0%	0.0%	0.000319%	0.000212%	0.000427%	0.000704%	0.000019%	0.001389%
2013	0.0000018%	0.0000015%	0.0000036%	0.0000017%	0.0%	0.0%	0.000308%	0.000196%	0.000420%	0.000716%	0.000018%	0.001414%
2014	0.0000018%	0.0000013%	0.0000036%	0.0000017%	0.0%	0.0%	0.000309%	0.000236%	0.000382%	0.000644%	0.000021%	0.001268%
2015	0.0000018%	0.0000012%	0.0000036%	0.0000017%	0.0%	0.0%	0.000290%	0.000201%	0.000378%	0.000523%	0.000017%	0.001028%
Data Sources	Determined from trade in India according to Zauba.com. Indium content of an individual standard assumed acording to a typical 1000 mg In/L solution. Bottom estimates scaled down to Australia by GDP, with top estimates arbitrarily set at twice the average.			Determined from trade in India according to Zauba.com. Indium content of an individual standard assumed acording to a typical 1000 mg In/L solution. Bottom estimates with bottom and top estimates arbitrarily set at half and twice the average.			Indium Trichloride consumed at a maximum rate of 10 tons/year in 2015/16 (pers. comm. O'Neill, 2016, Indium Manager at AIM Solder). Bottom estimate derived from this value scaled down by Total GDP to an Australian proportion, and changed yearly by the relative global consumption of indium compared to 2015. Top estimate derived from the Zauba.com trade of indium trichloride in india, scale down to Australia using relative GDP.			Bottom estimate derived from global demand for In Sulfate in 2015/16 (adivsed from pers. comm. O'neill, 2016, AIM Solder). Scaled down to Australia by relative total GDP. Content for earlier years is made proportional to total In demand. Top estimate is derived from the ratios observed for 2015/16 in India, scaled back to Australia by relative GDP between the two countries, and earlier year estaimtes are also scaled back according to global In demand for each year.		

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Appendix D: Additional studies published as a co-author during candidature

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Full length article

Modelling in-use stocks and spatial distributions of household electronic devices and their contained metals based on household survey data

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ABSTRACT

Waste electrical and electronic equipment (WEEE) contains a significant amount of critical and precious metals. Recovery of these metal resources is important for both environmental and economic reasons. However, the potential for metal recovery from the distributed resource of used electronic devices in households has not been well understood. This paper explores such potential through modelling in-use stocks and spatial distributions of metal resources in household electronic devices based on household survey data, using Australia as a case study. We focused on ten categories of electronic devices: smart mobile phones, plain mobile phones, tablets, laptops, desktops, flat screen TVs, CRT TVs, monitors, hand held music players and game devices. Regression models were built using demographic variables as predictors to estimate the amount of electronic devices currently in use in households, and the bottom-up approach was employed to estimate the stocks of forty three metals contained in the devices. A set of maps were produced to show the estimated distribution of the resource of in-use household electronic devices and specific metals of interest contained in these devices. We find that some metals such as Platinum-group elements have more stocks in Australian household devices than the potential stocks in Australian mineral deposits. There is some intrinsic resource value contained in Australian household electronic goods, and interest in recovery of these particular metals might come sooner than for others.

1. Introduction

A consequence of technological advancement in the consumer electronics industry is increasing levels of dependency on a greater quantity and variety of metals, exacerbated by ever increasing rates of consumption of high turn-over consumer products. This raises several concerns: 1) environmental concerns around existing practices of disposal in landfill and the problem of e-waste being exported to developing countries, 2) concern about the loss of valuable metal resources that are in demand and for which markets exist, 3) concern about the environmental consequences of continued reliance on mining, particularly of geologically scarce metals, and 4) concern around the need to secure supplies of critical metals like germanium, gallium and indium which are essential to the function of many popular electronic devices such as smart phones, TVs and laptops. While supplies of these metals are unlikely to be exhausted in the near future, there is much uncertainty in the mineral resource industry around the reporting of these metals, making it difficult to truly estimate what resources remain and where they are located (Mudd et al., 2016). Further, the substitution of these metals for others that may be more easily sourced is not always easily achieved (see Graedel et al., 2015). Often these and other factors contribute to metals used in high-tech applications being classified as ‘critical’. This means that they have been the subject of measurable concern over the risks of future primary supply restrictions, and/or that greater environmental harm may arise due to their extraction (Graedel et al., 2012; Helbig et al., 2016). Rare earth elements (REEs) are a classic example of critical metals in that they have wide ranging applications in modern technologies, e.g. in flat-screen TVs, batteries, cars and in the renewable energy sector, placing them in high demand globally, but their supply is vastly dominated by China, which has a history of imposing export restrictions (Sprecher et al., 2015; Weng et al., 2015). Consequently, governments and institutions of other countries have consistently

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rated these elements as among the most critical, highlighting the need for interventions to better secure their future supply (e.g. BGS, 2015; USDoE, 2011).

In Australia, the focus of this assessment, the main drivers for policy and management of waste electrical and electronic equipment (WEEE) are grounded in hazard reduction and in meeting Australia's obligations as a signatory to the Basel Convention on the Control of Transboundary Movements of Hazardous Wastes and their Disposal (UNEP, 1992). The Australian Government introduced a new Product Stewardship Act in 2009 and one of the first co-regulated product stewardship schemes introduced under this act was the National Computers and Television Recycling Scheme (NCTRS) (CoA, 2011). This scheme covered computers and televisions only. It involved calculation of the volume of used televisions and computers available for collection based on top-down methods that drew primarily on import and sales data. However a review of this scheme after its initial three years of operation revealed many problems including significant underestimates of the resource, and the lack of incentives for collecting organisations to collect it in rural and regional areas, where the resource was becoming a significant problem for local governments as it accumulated at waste transfer sites (CoA, 2014a; Lane et al., 2015). It is likely that this scheme, or something similar, will be extended to other categories of electronic products in the future and Australia's Product Stewardship legislation allows for this (CoA, 2011). For such schemes to be effective, more accurate information is required about the volume and distribution of the potential resource of used equipment. In particular, the widely dispersed resource located in households across the country represents the most significant logistic challenge for product stewardship collection schemes.

Thus far, the policy and management of metals contained in WEEE has not been driven by criticality concepts, however Australia presents an interesting case study for understanding the distribution of critical metals, as it is likely well-endowed with considerable mineral resources of critical metals (e.g. indium, Werner et al., 2016), yet possesses little to no domestic production capacity (primarily exporting critical metal laden concentrates for overseas processing), and simultaneously imports electronics containing critical metals on a relatively large scale. Skirrow et al. (2013) found that many metals present in electronic devices such as rare earth elements (REEs), gallium, indium and tungsten all rate quite highly in terms of their criticality to Australia, signalling the need for some concern over the security of future supply, however we are aware of no particular policies or targeted interventions to address these criticality concerns in Australia. Notwithstanding this inaction, there is still more to understand in terms of critical metals in Australia, for example whether recycled end-of-life goods might present a potential future resource, or more specifically, whether the scale and distribution of these metals in Australian households is in any way comparable to that of Australia's mineral resources. The recycling of electronic goods for critical metal recovery would need to be preceded by rather substantial changes in the price, supply dynamics and recovery technologies of many critical metals given that they are generally uneconomic to extract at present (Hagelüken, 2014), however in the event that such changes do take place, we explore the potential scale at which this could occur in Australia. Thus, while we are primarily concerned with the number of electronic devices in use in Australian households, the potential metal resource contained in them is presented as a secondary avenue of inquiry. To summarise, we have three objectives:

1 to estimate the in-use stocks of different types of electronic products in Australian households,
2 to assess the availability and amount of metal resources contained in these household products, and
3 to analyse geographical spread of the household electronic products and the metals they contain.

This provides an important data source for guiding future investments into recycling infrastructure and for policy and decision makers about the availability of metal resources from household electronic devices, and their geographical distributions.

While large organisations, including corporations and government agencies, generally contract the disposal of used electronic products to commercial businesses that may undertake a level of resource recovery, the potential stocks of electronic products accumulating in households can be considered an untapped resource that is currently poorly understood. There is strong evidence that households store used devices for some time before disposing of them (AMTA, 2015; IPSOS, 2015; Golev et al., 2016), which makes it difficult to assess the potential resource based solely on import or retail data. This means that if future governments or enterprises wanted to develop infrastructure to manage the recovery and reprocessing of discarded electronic goods, they could not determine the likely scale of operations, and Australia's recent experience with the NCTRS provides evidence of the problems caused when collection targets are based on flawed estimates.

Estimation and modelling of in-use electronic device stocks and their contained metals is often needed in lieu of direct measurements, and indeed the modelling of in-use stocks is now a well-established field with multiple approaches commonly adopted. The two most common methods of in-use stock analysis are the top-down and bottom-up approaches. The top-down approach effectively entails the collection of trade statistics into and out of a given system boundary (e.g. a country), and determining the difference between these flows over time as in-use stocks accumulated. The bottom-up approach entails the estimation of metal contents per unit (e.g. the amount of steel in a type of building) and multiplying this by the number of units within a system boundary (e.g. the number of buildings of that type). These and alternative methods for in-use stock estimation are described in depth in UNEP (2010) and Chen and Graedel (2015a, 2015b). While there are noted methodological uncertainties associated with all in-use stock approaches in the literature, perhaps the most pressing uncertainty arises from a lack of empirically collected and geographically relevant data sources, which is consistent among all of them. As noted in previous work (Zhu and Yu, 2016; Golev et al., 2016), it is often the goal of an in-use stock estimation study that data be synthesised to generate new knowledge around stocks, but not that the raw data be collected for the purposes of that study. This is problematic for in-use stock estimation (perhaps more so for bottom-up due to its geographical elements), as it cannot account for spatial variations between the system boundary and the source data.

This paper presents a methodology for estimating the number of electronic products in Australian households and the amount of valuable metals contained in them, and provides maps of their spatial distributions at various levels of geographical aggregation, based on a household survey. In the following sections, we first describe the household survey on electronic device ownership in Australian households. Following this, we link demographic variables to the number and type of electronic devices in people's homes. Through a review of literature on the metals typically contained in these devices, we then expand our analysis to include estimates of the metals contained in Australian household electronics and map their spatial distributions. While van Beers and Graedel (2006) used demographic proxies to estimate and map the in-use stocks of copper and zinc in buildings, infrastructure and consumer products across Australia, they did not estimate the copper and zinc in-use stocks in individual categories of consumer products. To our knowledge, no other studies have produced country wide maps of electronic devices and associated specialty metals currently held in domestic dwellings, providing a unique insight into the availability and distribution of above-ground metal resources, and offering useful geographical information to support future development of collection and recycling infrastructure and future policy interventions.

2. Methodology

2.1. Household survey

The data on the use and recycling of electronic devices in Australian households were drawn from an online survey conducted between February and March 2015 with a sample size of 1500 households. Unlike most surveys of consumer behaviour relating to these devices (e.g. IPSOS, 2015; AMTA, 2015; Deloitte, 2015), the unit of analysis was the household rather than the individual survey respondent and respondents were asked to respond to questions about ten categories of electronic devices for their household as a whole. The focus on households reflected our concern with the location of products in homes across Australia. It has some correspondence with the household survey on Energy Use and Conservation Survey conducted by the Australian Bureau of Statistics (ABS) in March 2014 as a supplement to the ABS monthly Labour Force Survey. The ten categories of household electronic devices were: smart phones, plain mobile phones, tablets, laptops, desktop computers, music players, monitors, flat screen TVs, CRT TVs, and game devices. The survey was hosted online using market research panels. The sample was representative of gender, age groups and of the population residing in different Australian states but included a subsample of 500 that oversampled households with members in the 15–24 years age group (30% of the 500) and for households in South Australia (14% of the 500). There were a priori reasons to assume that the 15–24 age group would be different from older age groups in terms of use of mobile phones and some other devices (AMTA, 2015; IPSOS, 2015) and a national waste audit conducted by the ABS in 2013 indicated disposal patterns for household waste in South Australia differed from those of other Australian States (ABS, 2013). Overall, the sample size is 1475. Among them 20% are in the 15–24 years age group, 33% in the age group of 25–44, 30% in the age group of 45–64 and 17% in the age group of 65 and above.

The survey included questions about patterns of acquisition, storage and disposal for the ten categories of electronic devices in respondents' households. It was designed to support a range of research enquires, however this paper focuses specifically on the assessment of stocks of these product types. A separate publication provides an analysis of flows of these products through households and addresses the issue of storage (Ward and Lane in process). The survey also collected key socio-demographic information relating to the type and composition of the household, along with the type and location (postcode) of the dwelling that matched with that collected by the ABS for the national census.

2.2. Estimating the number of devices

Our approach to estimating the number of electronic devices in a household is based on the statistical relationship between the number of electronic devices and socio-economic and demographical factors built upon the household survey data. Initially, we examined the following socio-economic and demographical factors:

(1) respondent's gender (male or female)
(2) respondent's employment status (employed or unemployed)
(3) family type (single and living with parents, or otherwise)
(4) education (secondary, diploma, bachelor and postgraduate)

Table 1
Estimating the number of electronic devices in Australian households (Sample size $n = 1475$.

Independent variable	Smart phone	Plain phone	Tablet	Music player	Laptop	Desktop	Monitor	Flat TV	Old TV	Game device
No of age <15	0.2102* (0.082)	0.0222 (0.054)	0.3224* (0.064)	0.2027* (0.065)	0.0427 (0.061)	0.0329 (0.054)	0.0741 (0.062)	0.2028* (0.070)	−0.0465* (0.025)	0.2578* (0.071)
No of age 15–24	0.9814* (0.076)	0.2200* (0.055)	0.3048* (0.042)	0.5643* (0.056)	0.7167* (0.056)	0.2693* (0.045)	0.3792* (0.054)	0.4169* (0.050)	0.0839* (0.027)	0.7926* (0.065)
No of age 25–44	1.0993* (0.062)	0.3991* (0.054)	0.4968* (0.050)	0.3702* (0.040)	0.7500* (0.041)	0.4071* (0.042)	0.4324* (0.042)	0.6969* (0.043)	0.1672* (0.027)	0.5372* (0.054)
No of age 45–64	0.9681* (0.054)	0.5572* (0.048)	0.4044* (0.040)	0.3684* (0.034)	0.7516* (0.038)	0.5258* (0.033)	0.5609* (0.036)	0.8896* (0.034)	0.1864* (0.021)	0.3503* (0.040)
No of age >65	0.4454* (0.048)	0.5997* (0.043)	0.2955* (0.032)	0.1457* (0.029)	0.5537* (0.042)	0.5452* (0.037)	0.6580* (0.056)	0.9529* (0.041)	0.1696* (0.029)	0.0335 (0.027)
F	349.61	140.54	193.69	143.7	366.19	218.23	196.63	523.22	52.36	146.75
p-value > F	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
R-squared	0.65	0.35	0.47	0.42	0.63	0.42	0.41	0.69	0.16	0.44

* $p < 0.01$; robust standard errors in parentheses.

(5) yearly household income (<$31,000, $31,000–62,000, $62,000–94,000, $94,000–13,0000, >$130,000)
(6) residence (first residence or not at the current address)
(7) location (urban or rural)
(8) language spoken at home (English or others)
(9) number of persons in the household in 5 age groups (<15, 15–24, 25–44, 45–64 and >65)

Linear regression models were built using all the above factors as independent variables to estimate the number of each type of electronic devices. The models indicate that about half of the socio-economic and demographical factors are not significant. These insignificant factors have weak or no prediction power. The R-squared value of each of these models is less than 0.3, which means less than 30% variations in the number of electronic devices can be explained by the regression models. However, the number of persons in each of the five age groups was mostly statistically significant at the significance level of 1% for the models. We used these age-related variables as independent variables to rebuild the regression models. Table 1 lists the coefficients, statistical significance and standard errors of these models. As indicated in Table 1, all the regression models have a *p*-value of less than 0.01, and therefore they are statistically significant at the significance level of 1%. The models were used to estimate the number of devices in a household according to the number of people in each of the age groups.

All regression models were built with Stata 13.1 (StataCorp, 2013).

2.3. Estimating the stocks of critical metals in household electronic devices

We took a bottom-up approach to estimate the stocks of critical metals in household electronic devices. This approach directly measures the stocks of a particular metal in all the devices and then sums them up, using the following equation (Zhu and Yu, 2016):

$$T = \sum_{i=1}^{N} (n_i m_i) \tag{1}$$

where n_i is the number of device i, m_i is the average or typical metal content of device i, N is the number of device categories. For this study, we compiled a table of average material content values for 43 critical metals in each type of device based on a literature review (Table 2). The number of devices in each category was estimated using the regression models in Table 1 based on the population data from the Australian Census 2011. As we were unable to locate assessments of average metal contents within music players, our estimates of critical metals are limited to the nine other categories.

2.4. Mapping the spatial distributions

Spatial distributions of household electronic devices and their contained metal stocks are represented in the form of maps. Mapping area units are defined according to the Australian Statistical Geography Standard (ABS, 2011), including statistical area level 1 (SA1), postcode areas (PA), suburbs, local government areas (LGA), greater capital city statistical areas (GCCSA) and states. SA1 is the smallest area unit for the census data release in Australia (ABS, 2011). There are about 55,000 SA1s covering the whole of Australia. An SA1 covers an area having an average population of about 400 persons. The regression models in Table 1 were applied to estimate the number of devices in every mapping area unit, resulting in maps showing estimated spatial distributions of each type of devices. Eq. (1) was then used to calculate the metal stocks in each mapping area unit, and the result is a map of the stock distribution of a specific metal in electronic devices.

The number of devices and metal stocks were estimated by implementing the regression models and Eq. (1) in ArcGIS 10.2.

3. Results

3.1. Number of household electronic devices

The number of household electronic devices of each category was estimated based on the Australian Census 2011. We estimated using our regression models that 19,607,741 smart phones, 8,574,275 plain phones, 9,131,518 music players, 8,674,873 tablets, 14,473,208 laptops, 8,606,898 desktops, 10,075,661 computer monitors, 15,096,129 flat screen TVs, 2,812,152 CRT TVs and 10,749,584 game devices were in use in Australia. Table 3 breaks down the estimated number of devices of each category by states. New South Wales had the largest number of devices in all categories, Victoria ranked the second and Queensland the third.

According to the Energy Use and Conservation Survey (EUCS) conducted by the Australian Bureau of Statistics (ABS) in March 2014 as a supplement to the ABS monthly Labour Force Survey, over 70% of Australian households had a smart phone with 47% of households having more than one smart phone; 51% of Australian households had a tablet, 44% of households had a desktop computer and 69% of households had a laptop computer with 10% of households having three or more laptops; nearly all Australian households (98%) had a TV with 43% of households having one TV, 35% of households having two TVs, and 20% having three or more TVs; and 72% of Australian households had a DVD/Blu-ray player, 53% of households had a stereo system, 33% of households had a mains powered games console, and 22% of households had a surround sound system (CoA, 2014b). Table 4 lists the number of households having each of these devices estimated from the ABS survey and the estimated number of devices in each category. The ABS survey did not include plain phones and computer monitors and did not differentiate between flat screen and CRT TVs.

Using the data in Table 4, it is estimated that in 2014 Australia had more than 12,137,400 smart phones, 6,720,800 tablets, 9,621,700 laptops, 4,675,400 desktops, 15,691,400 TVs, 6,479,200 music players (DVD/Blu-Ray players only) and 3,010,200 game devices (mains powered game consoles only). Comparing these numbers with the estimates from our regression models in Table 3, the total numbers of devices in each category based on the ABS survey data underestimate the number of devices in every category. There are four possible reasons. First, with the EUCS data, the number of devices of a particular type for a household having three or more devices of that type was simply counted as three, which directly led to the underestimates. Second, the ABS survey included mains powered game consoles only in the category of game devices, which caused a significant underestimate of the number of game devices. Third, there are uncertainties associated with the correspondence between sampling methods used in the ABS survey and our survey, which may have contribute to disparities between the results. Finally, our survey was conducted in 2015, one year later than EUCS.

3.2. Stocks of critical metals in household electronic devices

Using the estimates of the number of devices of each category calculated with the regression models, we estimated the amounts of the selected metals in household electronic devices in Australia as shown in Table 5. Table 6 lists the estimates by states.

According to Table 5, iron, copper, aluminium, lead and tin are the top five metals in the household electronic devices studied,

Table 2
Average values used for estimating metal contents within selected electronic goods.

Metals	Smart phone (g/unit)	Plain phone (g/unit)	Tablet (g/unit)	Laptop (g/unit)	Desktop (g/kg)	Monitor (g/unit)	lat TV (g/unit)	Old TV (g/unit)	Game device (g/kg)
Sb			0.154	0.770			0.710	14.000	
As			0.002	0.010					
Ba			0.490	2.500					5.100
In	0.005		0.008	0.101		0.074	0.591		
Al	3.648	12.000		30.570		149.600	720.000	306.500	40.000
Be	0.002			0.015	<0.001				
Cd								0.200	
Ce			<0.001	<0.001		<0.001	<0.001		
Cr			0.014	0.070				0.030	
Bi									0.260
Co	4.454	3.800	0.013	20.890	0.017		0.022		0.100
Cu	14.328	26.000	27.000	125.578		338.400	502.000	737.500	190.000
Dy			0.012	0.060					
Eu			<0.001	<0.001		<0.001	<0.001	0.149	
Fe	7.328			407.560		846.000	5400.000	2730.000	77.000
Ga	0.061			0.004	0.003	<0.001	0.005		0.016
Gd			<0.001	<0.001		<0.001	<0.001		
Ge	0.003								
Au	0.029	0.024	0.044	0.192	0.061	0.241	0.193	0.038	0.230
Pb	0.600	1.000	1.100	5.300		16.000		1319.000	13.000
Hg		1.000	<0.001	<0.001		<0.001			
La			<0.001	<0.001		<0.001	<0.001		
Li	<0.001						0.022		
Mo			0.008			0.633			
Nd	0.050		0.427	1.417		1.269			
Ni	1.500	1.000	0.722	3.600					
Pd	0.091	0.009	0.008	0.065	0.031	0.079	0.027	0.019	0.043
Pt	0.004			0.004					
Pr	0.010		0.055	0.274		<0.001	<0.001		
Sr									0.400
Si		5.000							
Ru				0.006	0.001				
Ag	0.182	1.000	0.050	0.428	0.376	0.965	0.447	0.535	0.740
Ta	0.166			1.607	0.073				0.083
Tb			<0.001	<0.001		<0.001	0.002		
Te	0.007				<0.001				
Ti						0.633			
Sn	1.114	1.000		4.88–14.08		24.000	18.000	32.000	26.000
W	0.215			0.038	0.020	0.633			
Y			<0.001			0.016	0.110	1.843	
Zn	1.000	4.000	<0.001						12.000
LREE	0.135–0.652			2.928–7.808			1.311		
HREE	0–0.068			0–0.697			0–3.933		

Collated and averaged from Buchert et al. (2012), Chancerel et al. (2015), Cucchiella et al. (2015), MoE and METI (2010), Nakajima et al. (2007), Oguchi et al. (2011), Shingkikai (2011), Takahashi et al. (2009), Zeng et al. (2016) and the Product Chart Database for Product (http://www.productchart.com/, courtesy of Marek Gibney).

Table 3
Number of household electronic devices.

State	Smart phone	Plain phone	Tablet	Music player	Laptop	Desktop	Monitor	Flat TV	Old TV	Game device
Australian Capital Territory	332,609	138,468	152,626	147,016	242,254	140,077	163,646	245,044	46,319	185,585
New South Wales	6,275,238	2,773,573	2,931,468	2,772,430	4,645,254	2,779,376	3,253,563	4,876,689	907,041	3,422,532
Northern Territory	202,600	77,679	93,404	91,639	143,809	79,677	93,560	140,507	25,967	118,274
Queensland	3,966,797	1,707,773	1,847,648	1,767,766	2,914,731	1,718,952	2,016,214	3,019,213	559,147	2,200,396
South Australia	1,438,495	652,399	670,145	632,584	1,074,375	651,157	761,417	1,139,686	213,026	772,987
Tasmania	444,338	203,027	207,847	196,751	332,352	202,409	237,012	354,842	65,671	238,951
Victoria	4,881,042	2,139,472	2,269,876	2,150,281	3,606,475	2,146,733	2,510,597	3,761,441	704,102	2,665,895
Western Australia	2,063,642	880,711	957,164	915,118	1,511,829	887,324	1,038,276	1,556,628	290,471	1,143,304
Other Territories	2,980	1,173	1,340	1,288	2,129	1,193	1,376	2,079	408	1,660
Total	19,607,741	8,574,275	9,131,518	8,674,873	14,473,208	8,606,898	10,075,661	15,096,129	2,812,152	10,749,584

Table 4
Number of households having electronic devices from the 2014 ABS Energy Use and Conservation Survey (Source: ABS).

No of devices	Smart phone	Tablet	Laptop	Desktop	TV	Music player	Game device
One	2,113,000	2,926,000	3,760,600	3,369,500	3,845,000		
Two	2,554,000	1,106,900	1,586,400	483,900	3,183,900		
Three or more	1,638,800	527,000	896,100	112,700	1,826,200		
None	2,703,700	4,450,600	2,776,000	5,046,900	156,300		
Total number of devices	12,137,400	6,720,800	9,621,700	4,675,400	15,691,400	6,479,200	3,010,200

Table 5
Estimated amounts of metals in household electronic devices in Australia (in kg).

	Smart phone	Plain phone	Tablet	Laptop	Desktop	Monitor	Flat TV	Old TV	Game device	Total
Sb			1,406	11,144			10,718	39,370		62,639
As			18	145						163
Ba			4,474	36,183					137,057	177,715
In	101		73	1,466		750	8,926			11,316
Al	71,537	102,891		442,404		1,507,319	10,869,213	861,925	1,074,958	14,930,247
Be	45			218	17					281
Cd								562		562
Ce			9	14		1	5			29
Cr			128	1,013				84		1,225
Bi									6,987	6,987
Co	87,329	32,582	119	302,341	1,457		326		2,687	426,841
Cu	280,939	222,931	246,551	1,817,510		3,409,604	7,578,257	2,073,962	5,106,052	20,735,806
Dy			110	868						978
Eu			9	14		1	61	420		506
Fe	143,685			5,898,727		8,524,009	81,519,097	7,677,175	2,069,295	105,831,987
Ga	1,204			53	222	30	75		430	2,014
Gd			9	14		11	23			57
Ge	62									62
Au	566	206	402	2,778	5,198	2,428	2,909	107	6,181	20,775
Pb	11,765	8,574	10,045	76,708		161,211		3,709,228	349,361	4,326,892
Hg		8,574	9	14		21				8,619
La			9	14		1	11			35
Li	13						326			339
Mo			73			6,378				6,451
Nd	980		3,899	20,506		12,786				38,171
Ni	29,412	8,574	6,593	52,104						96,682
Pd	1,783	77	73	938	2,624	793	414	54	1,156	7,913
Pt	78			58						136
Pr	196		502	3,966		1	2			4,666
Sr									10,750	10,750
Si		42,871								42,871
Ru				83	102					185
Ag	3,562	8,574	457	6,189	32,047	9,723	6,743	1,505	19,887	88,687
Ta	3,255			23,253	6,186				2,231	34,925
Tb			9	14		1	30			55
Te	145				17					162
Ti						6,378				6,378
Sn	21,839	8,574		137,206		241,816	271,730	89,989	698,723	1,469,877
W	4,220			549	1,696	6,378				12,842
Y			9			81	868	5,183		6,141
Zn	19,608	34,297	9						322,488	376,401
LREE	7,716			77,694			19,789			105,199
HREE	663			5,066			29,683			35,412

each having a stock of over one million kilograms in Australia. The national stock of iron in household electronic devices is over 105 million kg, copper over 20 million kg and aluminium over 14 million kg. The most critical metals, light rare earth element (LREE) and heavy rare earth elements (HREE) according to BGS (2015) have a stock of about 105,199 kg and about 35,412 kg respectively in the household electronic devices in Australia. Amongst other critical metals, antimony comes to be 62,639 kg, bismuth 6987 kg, germanium 62 kg, cobalt 426,841 kg, and indium 11,316 kg. Among the precious metals, silver amounts to 88,687 kg, gold 20,775 kg, palladium 7913 kg, praseodymium 4666 kg, and platinum 136 kg.

The distribution of the stocks of the metals in household electronic devices by states as shown in Table 6 indicates that the top three states, New South Wales, Victoria and Queensland, account for 77% of the national stocks for every category of metals.

3.3. *Spatial patterns of household electronic devices and metal stocks*

Our mapping indicates the spatial patterns of household electronic devices and their contained metal stocks in Australia at state and sub-state levels. At all the spatial scales, the distributions of metals are closely associated with those of household electronic devices. As expected, household electronic devices concentrate in the greater capital city statistical areas (GCCSA) (representing the socio-economic extent of each capital city), which are characterised by high population densities and large labour markets. Table 7 lists the number of household electronic devices in each GCCSA. According to Tables 3 and 7, about 66% of household electronic devices are concentrated in the GCCSAs.

The metals have a similar spatial pattern. For example, measured by the indium stocks (kg) per km^2, the Greater Melbourne city area has a stock density of 207.14 kg/km^2, the Greater Adelaide city area 200.93 kg/km^2, the Greater Sydney city area 187.7 kg/km^2, the Greater Perth city area 142.89 kg/km^2, the Australian Capital Territory 120.99 kg/km^2, the Greater Brisbane city area 69.24 kg/km^2, the Greater Hobart city area 62.72 kg/km^2, and the Greater Darwin city area 18.66 kg/km^2. The Rest of State regions have an indium stock density of less than 3.4 kg/km^2.

At the LGA level, the distributions of household electronic devices and their contained metal stocks have a consistent pattern across the range of the devices and metals, which exhibits a high concentration in the LGAs in the eastern and south-eastern coastal areas. It is because around 82% of the Australian population live within 50 km of the coast in these areas (Hugo et al., 2015) and most of them live in the capital cities as six of these are situated on the east and southeast coast. There are 577 LGAs in Australia. The City of Brisbane is the largest LGA by population, and has the largest stock for every device type and metal category. The second largest

Table 6
Estimated amounts of metals in household electronic devices by states (in kg).

	ACT	NSW	NT	QLD	SA	TAS	VIC	WA	Others
Sb	1,032.84	20,187.61	586.56	12,499.39	4,723.61	1,461.52	15,656.04	6,483.95	6.26
As	2.72	52.32	1.62	32.84	12.08	3.75	40.61	17.04	0.02
Ba	3,050.07	56,690.06	1,911.48	36,248.05	12,867.62	3,976.90	44,119.34	18,832.93	31.48
In	184.18	3,651.77	106.29	2,265.83	852.20	265.07	2,819.99	1,169.04	1.14
Al	243,579.18	4,815,895.35	140,908.94	2,990,919.74	1,123,109.50	349,119.32	3,720,371.36	1,543,605.93	1,607.67
Be	4.69	90.03	2.78	56.51	20.80	6.45	69.89	29.32	0.03
Cd	9.28	181.38	5.17	111.81	42.63	13.16	140.83	58.10	0.06
Ce	0.48	9.37	0.29	5.87	2.16	0.67	7.26	3.04	0.00
Cr	20.45	393.46	12.11	246.68	90.95	28.19	305.38	127.98	0.12
Bi	120.88	2,224.73	76.86	1,430.29	502.38	155.12	1,732.83	743.44	1.40
Co	7,135.76	137,006.49	4,236.51	85,978.72	31,659.97	9,812.67	106,349.33	44,610.00	51.69
Cu	343,393.39	6,667,940.06	202,864.64	4,172,591.23	1,544,797.98	479,184.67	5,163,559.00	2,158,249.85	2,612.28
Dy	16.33	313.93	9.71	197.06	72.48	22.47	243.64	102.23	0.10
Eu	8.33	163.08	4.68	100.68	38.27	11.83	126.52	52.26	0.05
Fe	1,722,121.88	34,157,293.51	990,670.48	21,176,351.54	7,978,051.02	2,480,529.58	26,380,017.48	10,927,349.51	10,804.50
Ga	34.11	644.75	20.70	407.57	147.95	45.78	501.02	211.65	0.36
Gd	0.93	18.31	0.55	11.41	4.25	1.32	14.16	5.89	0.01
Ge	1.05	19.77	0.64	12.50	4.53	1.40	15.38	6.50	0.01
Au	345.93	6,673.19	206.26	4,188.60	1,542.07	478.09	5,170.59	2,167.78	2.75
Pb	71,644.65	1,393,903.85	40,528.20	862,736.17	326,411.17	100,803.79	1,082,286.38	448,126.94	468.80
Hg	138.77	2,788.02	78.45	1,716.89	655.81	204.24	2,150.31	885.17	0.86
La	0.58	11.32	0.34	7.08	2.62	0.81	8.76	3.66	0.00
Li	5.50	109.46	3.17	67.83	25.57	7.96	84.48	34.98	0.04
Mo	104.54	2,082.87	60.06	1,290.99	487.53	151.55	1,607.53	664.58	0.65
Nd	631.36	12,276.29	371.79	7,675.47	2,846.31	883.26	9,509.65	3,971.58	3.92
Ni	1,618.27	31,027.07	964.04	19,485.37	7,161.32	2,219.55	24,083.05	10,110.44	12.61
Pd	131.66	2,542.78	77.94	1,592.72	588.80	182.51	1,970.49	824.70	1.09
Pt	2.30	43.68	1.38	27.53	10.05	3.11	33.95	14.30	0.02
Pr	77.99	1,497.75	46.43	940.46	345.70	107.17	1,162.53	487.93	0.49
Sr	185.97	3,422.67	118.25	2,200.44	772.90	238.64	2,665.90	1,143.76	2.15
Si	690.23	13,868.89	390.13	8,539.98	3,262.51	1,016.05	10,696.41	4,402.89	4.29
Ru	3.05	59.53	1.77	37.05	13.87	4.30	46.08	19.17	0.02
Ag	1,467.57	28,529.68	864.41	17,831.39	6,620.74	2,053.58	22,086.29	9,222.04	11.21
Ta	583.08	11,213.54	345.80	7,033.62	2,592.96	803.60	8,700.83	3,647.49	4.04
Tb	0.90	17.65	0.53	11.00	4.10	1.27	13.65	5.69	0.01
Te	2.74	51.94	1.65	32.76	11.94	3.69	40.37	17.02	0.03
Ti	103.32	2,059.42	59.31	1,276.22	482.17	149.88	1,589.37	656.92	0.64
Sn	24,692.82	471,151.46	14,951.14	297,411.13	108,292.43	33,540.42	365,558.99	154,078.48	219.04
W	211.86	4,133.45	123.98	2,579.03	960.98	298.14	3,199.33	1,333.02	1.71
Y	101.05	1,980.97	56.60	1,222.04	465.20	143.79	1,536.58	634.19	0.61
Zn	6,464.70	120,052.55	4,061.52	76,813.67	27,236.74	8,417.37	93,416.67	39,898.68	71.85
LREE	1,749.24	33,800.69	1,032.68	21,166.29	7,825.31	2,427.15	26,213.24	10,970.97	11.29
HREE	576.73	11,425.87	333.03	7,090.96	2,665.61	829.11	8,824.40	3,659.43	3.61

Table 7
Number of household electronic devices in each GCCSA.

GCCSA	Smart phone	Plain phone	Tablet	Music player	Laptop	Desktop	Monitor	Flat TV	Old TV	Game device
Australian Capital Territory	331,992	138,230	152,359	146,754	241,811	139,834	163,367	244,628	46,233	185,241
Greater Adelaide	1,109,034	497,155	514,723	487,350	825,877	497,118	581,175	869,420	163,162	598,773
Greater Brisbane	1,908,625	801,256	884,141	850,755	1,393,173	809,865	949,883	1,421,484	264,676	1,069,841
Greater Darwin	115,260	45,253	52,879	51,320	82,287	46,169	53,873	81,099	15,251	65,869
Greater Hobart	190,929	85,744	88,955	84,566	142,171	85,721	100,398	150,198	27,901	103,519
Greater Melbourne	3,675,680	1,581,815	1,703,573	1,615,315	2,700,853	1,591,558	1,859,286	2,787,384	524,715	2,019,104
Greater Perth	1,591,832	679,404	736,705	705,724	1,167,488	684,683	801,616	1,200,058	224,443	882,383
Greater Sydney	4,033,763	1,732,640	1,876,006	1,776,187	2,958,850	1,743,500	2,037,045	3,058,235	573,045	221,9459

LGA is Gold Coast. The Sydney LGA ranks 12 and the Melbourne LGA 59 in terms of device and metal stocks.

Household electronic devices and metal stocks mapped at SA1 can effectively show their spatial distributions at the neighbourhood level. Fig. 1 illustrates the distribution of tablets by SA1 in Sydney and surrounding areas. Fig. 2 is a map of the LREE stocks by SA1 around Melbourne CBD. Although there are no distinct patterns of spatial distribution at the SA1 level for both household electronic devices and metals, clusters of high concentrations can still be identified. For example, in Fig. 2, several pockets of SA1 areas with high LREE stocks are noticed in the north and central west of the mapped area. These correspond to high population, particularly within the age group of 25–44 years old which has the highest labour force participation rate in Australia (ABS, 2015). In general, areas with high concentrations are mostly located in districts where there are a great deal of development and the housing affordability is attractive to young families such as the western suburbs of Blacktown and Parramatta in Sydney, or in areas experiencing residential densification such as Indooroopilly and Kelvin Grove growth centres in Brisbane, or in the expanding urban fringe, such as Wyndham, Melton, Whittlesea and Casey in Melbourne.

4. Discussion and conclusions

By establishing the statistical relationships between the number of household electronic devices and age groups of the population based on a household survey and census data, we estimated the volumes of household electronic devices of ten categories at mul-

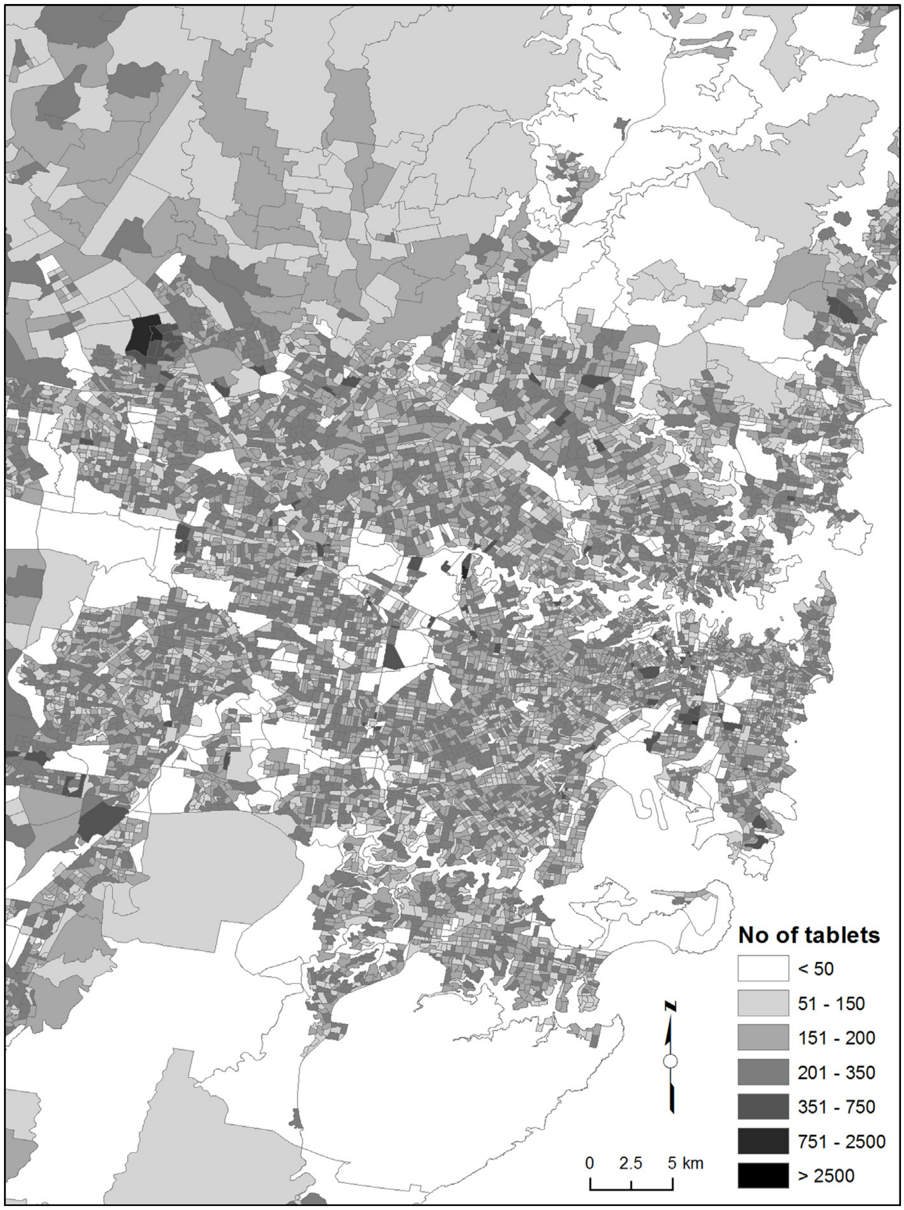

Fig. 1. Number of tablets by SA1 in Sydney and surrounding areas.

tiple levels of area aggregation in Australia. On the basis of this, we estimated the stocks of forty three metals in these products (except music players due to lack of material content data). The results have been presented in a series of maps contained in the online Australian Recyclable Resources Atlas (at http://wfw-atlas.monash.edu.au/atlas/build/).

Our simplified regression models employed age groups as independent variables. Although they are significant explanatory variables, a large percentage of the changes in the number of household electronic devices are unexplained by these variables and are the result of other factors, as indicated by the R-squared values listed in Table 1. We applied these variables only in order to use census data for estimation at various spatial scales from neighbourhoods to local government areas and states. The main purpose for multi-scale assessment is to reveal the spatial patterns or geographical variations of stocks of household electronic products and valuable metals contained in them to support decision and policy making by local, state and federal governments as well as recycling industries. The major policy implications are discussed below.

4.1. Collection systems in Australia

From the perspective of designing more effective collection schemes, different strategies are required for large less portable equipment than for smaller equipment so it is useful to differentiate the location of the different groups of products along these lines. The metal components also differ significantly between smaller and larger product types, with larger products containing a smaller proportion of critical and precious metals per kilogram. Estimating quantities and distributions by product type will therefore be important for any future investment in infrastructure to extract critical and precious metals in Australia.

While there is sufficient market value in late model mobile phones and tablets to support the business models of companies that expressly buy and sell used electronic devices for the purpose of reuse, only a small portion of the Australian population currently dispose of their equipment by selling it second hand (IPSOS, 2015; AMTA, 2015). Both small and large product types are likely

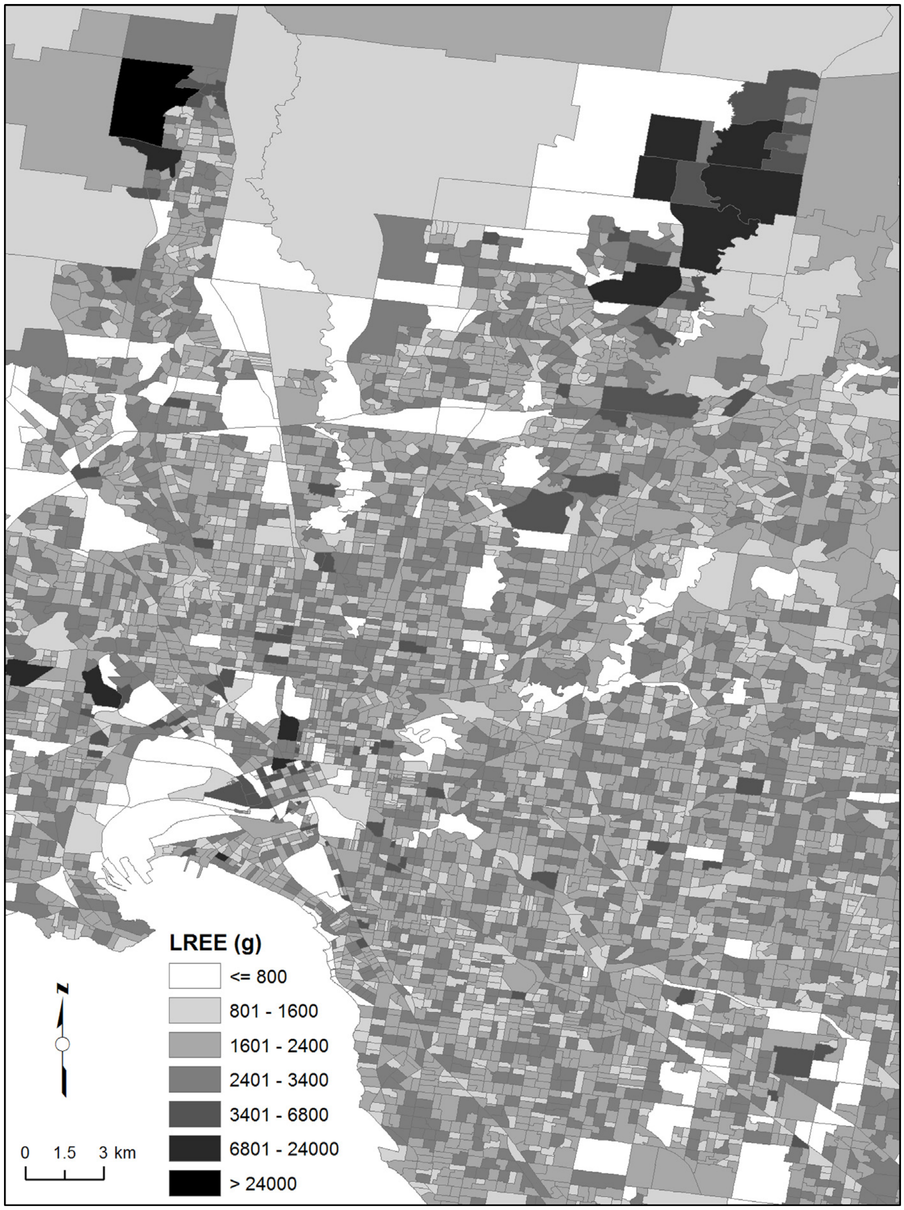

Fig. 2. LREE stocks by SA1 around Melbourne CBD.

to be stored for a period in residential dwellings prior to disposal in landfill.

According to Chen and Graedel (2015a, 2015b), in-use stocks of the products that are recently developed and introduced into markets to offer new services or functions (such as new models of mobile phones and tablets) and that are developed to replace products providing similar services or functions (e.g. the regulated change from analogue to digital TVs in Australia) are continuing to increase. Although our current analysis provides a one-time snapshot of in-use stocks of these products, a similar household survey will be conducted annually in Australia and the modelling framework we have developed in this paper will be applied to provide new estimates of their in-use stocks, which will lead to annual updates of the Australian Recyclable Resources Atlas and depiction of dynamic evolution of stocks over time.

Despite the rhetoric found in Australia's National Waste Policy of "managing waste as a resource to deliver economic, social and environmental benefits" (CoA, 2009: 2), policy initiatives around used electronics in Australia have so far been driven more by a logic of hazard than of resource values. In a study of mobile phone recycling in the US during the early 2000s, Geyer and Doctori Blass (2010) demonstrated that for mobile phones, which have the highest content of precious metals per volume of all the items considered here, the value of the contained metals was insufficient to drive collection systems without some form of subsidy to cover the costs of collection. As product design improves over time, the quantities of these metals used in each item decreases, so that the challenge remains the same even though there are larger quantities available for collection.

It should therefore be anticipated that a mixture of measures will be required to capture used electronics for the purpose of materials recovery in the future. These would likely include more convenient collection services, and both regulatory restrictions on sending used electronics to landfill and incentives for contributing to organised collections. While small items may be sent through parcel postage services, an option currently used by Mobile Muster, the industry product stewardship collection for mobile phones (AMTA, 2015), larger items must either be collected or dropped off at a convenient collection point. Once collected, there are likely to be different options for processing smaller product types containing higher portions of critical and precious metals than for larger items.

4.2. Critical metals in Australia

Critical metal recovery from end-of-life goods is in its infancy globally, with only a few commercial-scale examples to be found (e.g. Umicore operations in Belgium). We are presently aware of no recovery operations servicing populations as small as Australia's, suggesting that the recovery of critical metals from waste electronic products in Australia is unlikely in the short to medium term. It is however possible that in a future scenario where export restrictions are imposed, prices rapidly change and the technology for recovery becomes more widespread, our maps become a useful tool to target critical metal infrastructure investments. This is however speculative, and at least in the case of mobile phones, it has been established that copper and precious metals make up over 95% of the material value, with it being unlikely that profit margins for recovery will be increased by extracting additional metals with current technologies (Geyer and Doctori Blass, 2010). This suggests that at present, the mapping of more common and precious metals such as Cu, Ag and Au is a more near term contribution of this work. There are indeed some concerted efforts to establish domestic e-waste recycling capacity in Australia, with a focus on optimising for Cu and precious metals recovery, which could be supported by our mapping of metal contents.

In terms of the absolute scale of critical metals present in electronic devices, it is useful to compare to the quantities present in known mining operations to gain a greater sense of whether recycling would be preferred versus improved separation during primary mineral processing. This is however difficult to determine, as by-product/critical metals are often not reported by mining companies (Mudd et al., 2016), and as a result, there are only a few detailed studies of critical metal resources which cover Australia. Weng et al. (2015) examined global REE deposits, finding a total of 64,519 kt rare earth oxides reported for Australia. By comparison, we note that there are presently 140.6 t, or ~0.0002% of this mineral resource, present in smartphones, laptops and flat screen TV's in Australian households. A similar relationship is found for indium, where we estimate ~11 t In in household devices, again a minor fraction of the approximate 45,000 t In contained in Australian mineral deposits (Werner et al., 2016). Skirrow et al. (2013) have also examined critical metal supply potential for Australia, although with less detailed and reliable data on Australia's endowment of many critical metals. However at least in terms of absolute scale, this study still provides some useful indications that not all critical metal quantities are as dominated by the minerals sector. For example, by comparing our results with that of Skirrow et al. (2013), we find that some metals contained in household devices, such as copper and cobalt, are only less than one 10,000th of the Australian supply potential from mineral deposits, while others, such as Platinum-group elements, have more stocks in Australian household devices than the potential stocks in mineral deposits. For those metals with greater potential resources in households, we can at least conclude that there is indeed some intrinsic resource value contained in electronic goods, and we may suppose that interest in recovery of these particular metals might come sooner than for others.

Acknowledgements

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The world's lead-zinc mineral resources: Scarcity, data, issues and opportunities

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ABSTRACT

Lead and zinc keep humanity powered and sheltered, yet a comprehensive understanding of Pb-Zn resources in known mineral deposits has been lacking, leading to uncertainty over when we might expect the supply of these metals to face potential constraints. Addressing this, we compile an extensive database of the world's known Pb-Zn mineral deposits and provide in-depth analyses of their contained resources, ore-grades, economic value, by-products and geological settings. Our data indicate that at least 226.1 Mt Pb and 610.3 Mt Zn are present within 851 individual mineral deposits and mine waste projects from 67 countries (and one in international waters), at an average grade of 0.44 %Pb and 1.20 %Zn. The identified resources are dominantly present within sediment-hosted Pb-Zn deposits (490.6 Mt Pb + Zn + Cu), which contain the equivalent of VMS, Skarn, Porphyry, Epithermal and mixed sediment-hosted deposits combined, and 49% of these resources are reported in Australia, Russia, Peru and Canada alone. The reported Pb-Zn resources appear to be sufficient to meet global demand for both Pb and Zn until 2050, although this estimate is most certainly a minimum, as our case studies indicate a prevailing trend of deposits cumulatively producing well beyond their reported resources over time. Indeed, despite increasing historical production of Pb and Zn, estimated reserves and resources have also increased, and this is expected to continue.
We also present an analysis and review of additional aspects affecting the future sustainability of Pb-Zn resources, including an account of the history of Pb-Zn mining, case studies and trends in reporting, classifications of the dominant Pb-Zn mineral deposit types, analysis of reported by-product companion metals, review of tailings resource potential and case studies on the numerous challenges in environmental management historically faced for Pb-Zn mining. These analyses, alongside our comprehensive resource data, indicate that the future supply of Pb and Zn is likely to be governed by prevailing economic, social and environmental factors, much more so than sheer resource constraints.

1. Introduction

The extraction of metals from the Earth underpins a range of goods and services in modern society and, although this is a practice that has been fundamental to the survival of humankind since the Bronze Age, the production of an increasingly wide range of metals is fundamental to the provision and construction of numerous technologies and services (Graedel et al., 2015; Rankin, 2011). For example, nickel (Ni) is used extensively in stainless steel alloys, rhenium (Re) is critical for specialist alloys used in jet engines, copper (Cu) is used in piping and electrical and electronic equipment, gold (Au) is used in jewellery, has specialised uses as a conductor, and is a well-known financial store of value, indium (In[1]) and europium (Eu) are employed in colour flat screen displays, some rare earth elements (REEs) such as neodymium (Nd) and dysprosium (Dy) are crucial in enhancing the magnets used in sustainable energy technologies (e.g. wind turbines) while other REEs are used in phosphors in lighting, electronics and other technologies (especially military systems), silver (Ag) is used in solar photovoltaic panels, and so on. It is therefore reasonable to state that a wide range of metals are now indispensable to the modern world.

The mining of metals occurs all over the world from a variety of mineral deposit types with varying mining and ore processing technologies (Spitz and Trudinger, 2008). Some countries are endowed with major resources of particular metals, such as Chile and Cu, Brazil and niobium (Nb) or South Africa and Au and the platinum group elements (PGEs). In general, world metal production grew substantially over the 20th century – leading to the critical question of how long such continuing growth in production can be sustained? To answer this, it is fundamental to compile and synthesize known mineral deposits, as this forms the basis for future scenarios (e.g. Cu, see Mudd et al., 2013a; Northey et al., 2014).

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[1] Although the chemical symbol for most of the elements are used throughout this paper, we have elected to spell out indium to avoid confusion with the word "in" or the abbreviation of the natural logarithm, "ln".

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Long-term planning in the mining industry is done through the reporting of exploration results, mineral resources, ore reserves and mine production – and this is mainly achieved through formal industry or statutory codes. As mineral exploration continues, new deposits can be discovered and developed which add to reserves and resources. On the other hand, given that a specific deposit is a set size as determined by the geology of the system that formed the mineralization, ongoing mining over time will deplete the deposit, meaning there is a continuous requirement to discover new deposits to replace depleted reserves and resources. However, other factors can also influence whether a mineral deposit is worth mining or whether material once considered waste actually becomes ore (as a function of lowering cut-off grades for example) over the exact same time period of exploitation – such as new mining or ore processing technology, changing prices affecting the economics of a given mining project (for better or worse), and others.

Despite the growth in mined metal production, there has been a coincident general increase in reported world reserves or resources over many decades, with clear examples being Cu (Mudd et al., 2013a), Ni (Mudd and Jowitt, 2014), PGEs (Mudd, 2012a), REEs (Weng et al., 2015) or uranium (U) (Mudd, 2014). This suggests that resource depletion will not be the primary concern for the next few decades (say to 2050), despite what some commentators have speculated (e.g. Bardi, 2014; Cohen, 2007; Moyer, 2010). There are, however, a number of caveats to such a statement. Firstly, it depends on whether you examine a particular country or take a global view, as although some countries appear to have depleted their metallic mineral resources (e.g. zinc (Zn) in Germany, see later sections), many others continue to show strong increases in metallic mineral resources over time driven by exploration and mine development success combined with technological progress (e.g. larger haul trucks, improved flotation mills, finer grinding), favourable economic circumstances (e.g. growing demand, reasonable prices relative to mine production costs) (e.g. lead (Pb)-Zn in Australia), or for other reasons (e.g. improving security and political situations that have previously limited exploration and the discovery of mineral resources in well-endowed areas such as the Democratic Republic of the Congo). Secondly, it depends on the methodology used to develop estimates of metallic resources, namely whether the assessment strictly examines ore reserves or instead considers mineral resources. In general, ore reserves represent a shorter term mine plan (say 5–10 years) whereas mineral resources demonstrate the geological size and characteristics of a deposit, are typically somewhat similar to reserves and suggest a project's longer term potential (say 10–25 years or beyond). From this, compiled reserves estimates are, by their very nature, a limited estimate of future long-term metal supplies from mining (e.g. Cu, Mudd et al., 2013a; Co, Mudd et al., 2013b). Claims that the world is in imminent danger of running out of reserves are often repeated over the years in various circles and forms (e.g. Bardi, 2014; Cohen, 2007; Moyer, 2010) but fail to recognize additional mineral resources which are well understood and have a good likelihood of being bought into production. Third, it fails to account for the potential to discover major new mineral deposits which will increase both reserves and resources. For example, major discoveries of world importance in the past 25 years include Voisey's Bay in Canada (Ni-Cu-Co), Oyu Tolgoi (Cu-Au-Ag) in Mongolia, Cannington (Pb-Zn-Ag) in Australia, Detour Lake in Canada (Au) and Turfspruit in South Africa (PGEs-Ni-Cu-Au). Overall, understanding the amount of known reserves and resources within either country or world assessments of future mining is fundamentally important, but it is equally important to understand the assumptions, data sources and methodological approach used in these assessments (see Mudd et al., in press).

In this paper, we compile and synthesize a detailed global assessment of Pb-Zn mineral resources, an assessment that is critical for a variety of reasons. Firstly, such an assessment has not been published to date. Secondly, such deposits are increasingly important for a range of by-product metals extracted at metal refineries that are not of economic significance in the mining-milling stages – here termed companion metals. For example, indium, germanium (Ge), gallium (Ga) and cadmium (Cd) typically report to Zn concentrates and can be extracted at Zn refineries, but given their low relative value to Zn, they are rarely credited in the value paid for Zn concentrates. This means that having a more complete assessment of global Pb-Zn mineral resources will facilitate more detailed assessments of a variety of companion metals (see Werner et al., under review-a,-b) that are increasingly critical to our modern way of life (Harper et al., 2014). Thirdly, the Pb-Zn sector is an area of the global mining industry where there has not been a major discovery for over 20 years (see Schodde, 2013; Schodde, pers. comm.), leading to major concerns over resource depletion occurring more rapidly than for other metals such as Cu, Au or Ni (amongst others). Finally, although Pb has a high global recycling rate, many uses of Zn are dissipative – meaning Zn is not as readily recyclable as Pb (Ciacci et al., 2015).

All of this means that it is important to assess current Pb-Zn mineral resources not only in their own right but also for their potential to support companion metals production. This paper presents the results of a global assessment of Pb-Zn mineral resources and discusses the implications of the size of these resources for future supply of these base metals as well as a range of other associated critical companion metals.

2. Background to the present day lead-zinc mining industry

2.1. Brief historical review

Both Pb and Zn each have a long history of mining going back millennia, although with important differences. We briefly review this history to provide context to understanding reserves-resources in mining over time, as well as to highlight the significant historical features of each metal; the following sub-sections draw from Sinclair (2009) for Pb and Sinclair (2005) for Zn.

2.1.1. History of lead mining

The mining and use of Pb goes back several millennia, with the ancient mines known across the Phoenician, Roman and similar empires evolving to the wide extent of Pb mining that occurred and occurs across the world in more recent centuries. The main uses for Pb in ancient times were to extract Ag and Au, as well as for water pipes, a weatherproof seal for buildings, as Pb sheets, in decorative (stained) glass windows, paints or even in cosmetics. In the past few centuries, Pb use expanded to early Gutenberg-era printing presses, munitions in weapons, and more recently as a store of electrical energy in Pb-acid batteries, as an additive in petrol to enhance engine performance, a stability additive in plastics (resistance to heat and ultraviolet (UV) radiation, especially UV from sunlight), in older cathode ray tube-type televisions, and more recently in radiation shielding for nuclear equipment or facilities. As can be seen, Pb has, historically at least, had a surprising array of uses.

Beginning in about the 1960s, there was a growing recognition of the toxic effects of exposure to Pb, especially for young children who are more sensitive to Pb. This led to many dissipative uses being phased out from the 1970s onwards (e.g. Pb in petrol or paints), often in line with new materials discovered to replace the need for Pb, as well as uses being focussed on applications where capture and recycling makes good sense (such as Pb-acid batteries). Due to these major changes in the Pb industry, Pb is now one of the most recycled metals in the world with secondary supplies meeting more than 50% of world demand (UNEP, 2013).

The production of Pb is typically through a pyrometallurgical route, with modern mines generally producing a Pb sulfide concentrate (containing around 50% Pb). The concentrate is smelted to produce Pb bullion, which is further refined to a pure metal to meet customer specifications. Given the increasingly dominant role of secondary Pb supplies, refineries are often designed to handle both primary bullion as well as the recycling of Pb scrap – especially Pb-acid batteries.

The historical global production of Pb by country is given in Fig. 1, showing the rise and fall of some countries (e.g. Spain, Canada), whilst

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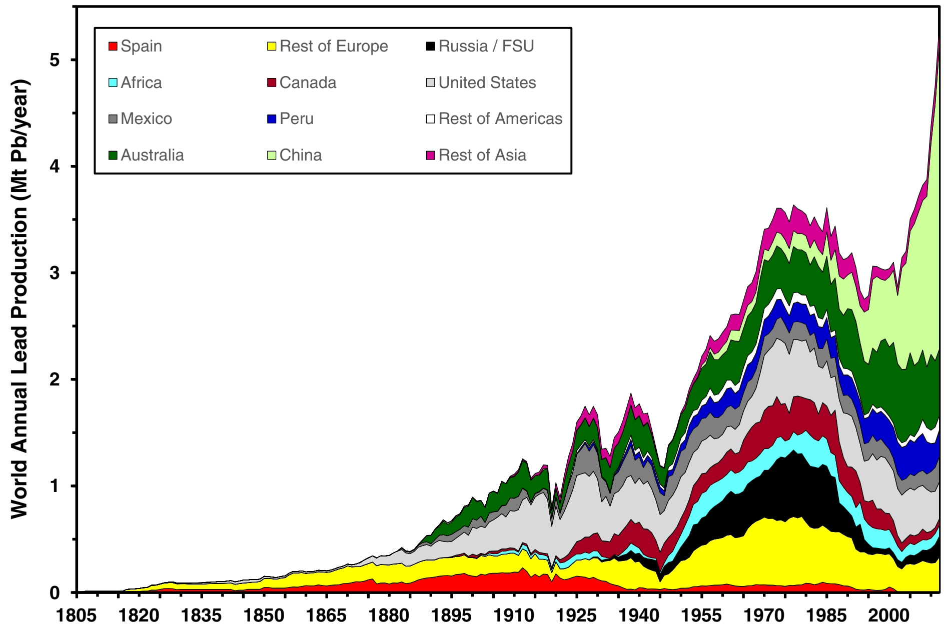

Fig. 1. Historical Pb production by country or region (data sources: ABARE, var.; BREE, var.; Mitchell, 1981, 1982, 1983; OCE, var.; Schmitz, 1979; USBoM, var.; USGS, var.)

others continue to maintain or grow significant levels of production (e.g. Australia, Peru, USA). Perhaps the most remarkable country in this regard is the exceptional growth in Pb production from China, growing from 0.16 Mt Pb in 1980 to 2.84 Mt Pb in 2012 – moving from 4.5% to 53.3% of world Pb mining over the same time. Furthermore, although world Pb production peaked in 1977 at 3.67 Mt Pb and then declined (corresponding to the period when Pb uses were affected by environmental and public health concerns), China's rapid growth has fuelled global Pb production to a record 5.32 Mt Pb in 2012.

2.1.2. History of zinc mining

The predominant use and forms of Zn for over two millennia was almost entirely as brass, an alloy of Cu and Zn. After discovering the secrets of refining pure Zn from Asia in the 17th century, Zn mining and production began to grow in Europe from the mid-18th century, followed closely by a ramping up of production in the USA. The main use at this time remained brass production but also involved the production of diecast alloys, with more recent uses in the 20th century emerging for protective galvanised coatings for various alloys (especially steel), rubber manufacture, use as sacrificial anodes on ships, as a trace nutrient in fertilizers and minor pharmaceutical uses (e.g. sunscreens for skin protection).

By the late 19th century, Zn mining continued to grow, dominated by Germany until World War I – after which time it failed to regain its position and was overtaken by production in the USA and later in Australia and Canada. The historical global production of Zn by country is given in Fig. 2 and is similar to Pb in that it charts the rise and fall of some countries (e.g. Germany, Canada), whereas others continue to

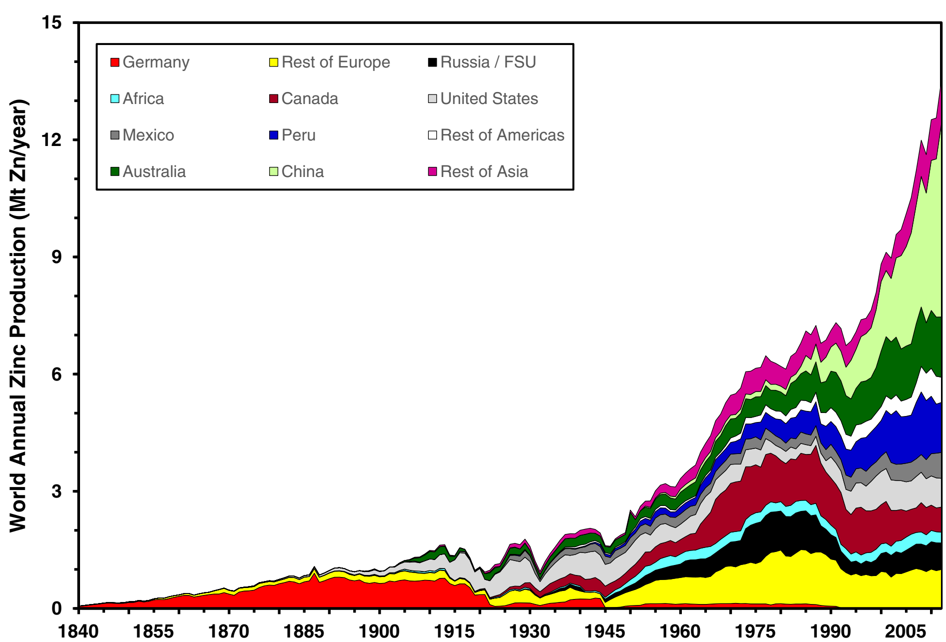

Fig. 2. Historical Zn production by country or region (data sources: sources: ABARE, var.; BREE, var.; Mitchell, 1981, 1982, 1983; OCE, var.; Schmitz, 1979; USBoM, var.; USGS, var.)

maintain or grow significant levels of production (e.g. Australia, USA). Similarly to Pb, China's Zn production has grown quite rapidly in recent decades, from 0.15 Mt Zn in 1980 to 4.93 Mt Zn in 2012 – moving from 2.4% to 36.5% of world Zn mining over the same time, and (again like Pb) driving up world Zn production levels.

2.2. Trends in reported reserves-resources by country

In recent decades, many countries have compiled regular assessments of their reserves or resources of many minerals and metals, such as Australia's Identified Mineral Resources (GA, var.), the Indian Minerals Yearbook (IBM, var.), the (former) United States Bureau of Mines' Minerals Yearbook and continued by the United States Geological Survey (USGS) (USBoM, var.; USGS, var.), the Canadian Minerals Yearbook (NRC, var.) or more recently China's Mineral Resources (e.g. Wang, 2015). However, it is critical to differentiate reserves from resources, both of which have strict definitions in the mining industry (and discussed in depth by Mudd et al., in press). In concept, proved and probable ore reserves are derived from short term mine plans and are economic at present, whereas measured, indicated and inferred mineral resources are well studied geologically and in other ways (e.g. metallurgy), and although they are not necessarily economic at present they have strong prospects for eventual economic extraction. In practice, ore reserves can be presented as a subset of measured and indicated resources, or mineral resources can be reported separately to ore reserves. In either way, by definition, this means that ore reserves are always smaller than mineral resources.

The country data for Pb-Zn reserves or resources are presented in Fig. 3, with a more detailed compilation given in Table 1, including cumulative production. For Australia, there is a strong and consistent increase over more than a century in Pb-Zn resources, with more recent evidence of such growth in India and especially China. The United States and South Africa show somewhat stagnant or declining Pb-Zn resources, although how much this is a reflection of cycles in metals prices is unclear – that is, the more recent decline may primarily be a function of lower prices causing reclassification of reserves to resources rather than 'mining depletion'. Canada shows a strongly consistent downward trend in its reported reserves, and, although Pb is not a dominant feature of current Canadian mines and deposits, Zn still remains critical to a range of operating mines and potential future projects. As noted above, the evidence on reserves-resources over time is variable and it depends on the methods and sources used as well as the extent of data over time.

Table 1
Global estimates of Pb-Zn reserves and/or resources by country.

Country	USGS reserves (2014)			National reserves or resources			Cumulative production (~1800–2013)		
	Mt Pb	Mt Zn	kt Ag	Mt Pb	Mt Zn	kt Ag	Mt Pb	Mt Zn	kt Ag
Australia	35	62	85	58.7	91.2	125.0	41.4	54.8	≫93.3
China	14	43	43	73.85	144.86	237.0	27.8	53.4	–
Russia	9.2	–	–	–	–	–	20.9	38.6	–
Peru	7	29	98.9	–	–	–	13.4	37.7	–
Mexico	5.6	16	37	–	–	–	19.0	23.8	–
United States	5	10	25	–	–	–	50.0	56.7	≫169.5
India	2.6	11	–	11.9	36.7	–	1.54	8.66	–
Poland	1.7	–	85	–	–	–	3.53	12.8	–
Bolivia	1.6	4.5	22	–	–	–	1.85	6.23	–
Sweden	1.1	–	–	–	–	–	4.88	10.5	–
Ireland	0.6	1.1	–	–	–	–	2.21	9.69	–
South Africa	0.3	–	–	0.3	14	–	2.43	2.61	–
Canada	0.247	5.9	7	0.40	4.133	6.916	19.4	61.9	–
Chile	–	–	77	–	–	–	0.13	0.90	–
Kazakhstan	–	10	–	–	–	–	–	–	–
Brazil	–	–	–	nd	2.079	3.91	0.87	5.22	–
Germany	–	–	–	–	–	–	8.07	47.9	–
Spain	–	–	–	–	–	–	14.8	10.3	–
Rest of the world	3	42	50	–	–	–	73.0	75.1	–
Totals	**87**	**230**	**530**	–	–	–	**282.3**	**516.2**	–

Notes: nd – no data; Country reserves for 2014 (USGS, 2015) and cumulative production (Kelly et al., 2016); Australia's total mineral resources as at Dec. 2013 (GA, 2014); China reserves as of 2014 (Wang, 2015); Indian reserves as of 2012 (IBM, 2015); South African reserves as of 2012 (CMSA, 2014); Canadian reserves as of 2012 (Drake, 2012); Brazil reserves as of 2012 (DNPM, 2012); . Additional historical data from (ABARE, var.; BREE, var.; Mitchell, 1981, 1982, 1983; OCE, var., Schmitz, 1979; USBoM, var.; USGS, var.); all historical production is approximate and best estimates based on the various sources cited (most sources appear reasonable and correlate well in any case).

There can, however, be inconsistencies in these assessments and the level of available reporting of these reserves and resources. For example, the USGS report Australian Zn reserves as 62 Mt Zn (USGS, 2015), although GA report Australia's economic Zn resources as 62.3 Mt Zn and note code-compliant reserves as 28.9 Mt Zn (GA, 2014). Although this is largely a reflection of the economic focus of an assessment, such large differences add high levels of uncertainty to our understanding of the long-term potential for ongoing mining and supply of these metals. For example, Australia's Zn production from mines in 2013 was ~1.52 Mt Zn (OCE, 2014), meaning the difference between 28.9

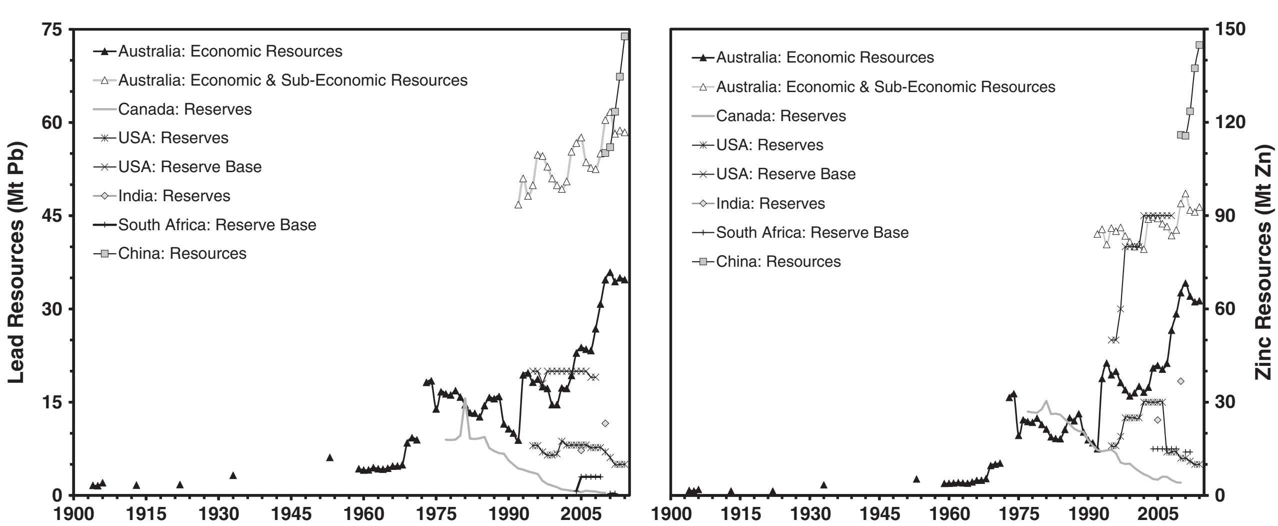

Fig. 3. Reported reserves or resources of Pb-Zn by country.

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and 62.3 Mt Zn represents some 22 years of future supply (assuming constant production). In South Africa, the only Pb-Zn project with reported reserves is the Black Mountain group, now owned by Indian miner Vedanta Resources plc. The 2014 reported ore reserves for Black Mountain (VR, 2014) contain a total of 0.40–0.32 Mt Pb-Zn (i.e. 0.40 Mt Pb and 0.32 Mt Zn), with additional mineral resources of 2.49–14.89 Mt Pb-Zn – while the South African Chamber of Mines reports national reserves[2] of 0.3–14 Mt Pb-Zn (CMSA, 2014).

Fig. 3 also shows that some countries have resource and reserve data that stays the same (or is very similar) over many years, often despite significant production over that time and varying economic price cycles that should also affect the compilation of reported reserves and resources for these metals. As such, it can be hard to have full confidence in this data – despite the fact that there is (for the most part) excellent public reporting by the mining industry on reserves and resources by projects around the world. It remains important, therefore, to be careful in reviewing or using such data as it can be easy to infer or extrapolate unreasonably (e.g. confusing reserves versus resources). These issues are further explored and discussed extensively later in the paper (see also Mudd et al., in press).

2.3. Resource-reserve-cumulative production case studies in Pb-Zn mining

There are numerous fields around the world which have had a long life and provide useful case studies to assess the factors which underpin reserves and resources in the Pb-Zn sector, as well as the conversion through to actual production. Only a select few mines are presented here in detail, with a detailed compilation of resource-reserve-cumulative production statistics for numerous projects given in Table 2. The mines we focus on are mainly located in Australia, Canada and the USA as these countries provide the most readily available data (predominantly from corporate annual and quarterly reporting). Overall, these case studies indicate that mineral resources are often upgraded to ore reserves before then being converted to mine production, demonstrating confidence in the process of resource-reserve-production reporting. Critically, these data show that mineral resources are a good guide to long-term production, as cumulative production for many projects often exceeds initial ore reserves or mineral resources. This is not to argue that all mining projects will demonstrate such behaviour, but simply that mineral resources are a reliable source of data for assessing such issues over the long-term, as exemplified by recent studies on global Cu (Mudd et al., 2013a), Ni (Mudd and Jowitt, 2014), PGEs (Mudd, 2012a) or U (Mudd, 2014) resources.

2.3.1. Broken Hill, Australia

The discovery of the large Pb-Zn-Ag deposit in 1883 at Broken Hill, Australia, has been pivotal not only to Australian mining but also to world mining, as the field was pivotal in the rise of some of the world's largest mining companies, namely BHP Billiton plc/Ltd as well as Rio Tinto plc/Ltd (see Mudd, 2009b, for a concise history). The early years of Broken Hill exploited rich Ag-dominant ores from the oxidised (or weathered) materials at or near the surface – and rich profits were made and established the Broken Hill Proprietary Company Ltd (or BHP) as a major mining house, along with numerous other operators along the increasingly world famous 'line of lode'; a photographic panorama is shown in Fig. 4.

Within a decade, the oxidised ores were within sight of depletion and attention turned to the more abundant but difficult to treat Pb-Ag sulfide ores, with commercial processing beginning in 1895. The next major challenge was the 'Zn problem' – separating the Pb from Zn to allow more efficient smelting of these metals. By 1904, the accumulated tailings alone at Broken Hill were estimated to contain some 6.69 Mt at 6% Pb, 19% Zn and 184 g/t Ag (Woodward, 1965). Engineers and metallurgists set to work, including a brewer from Melbourne, and the process of froth flotation was invented, finally allowing the production of separate Pb and Zn concentrates for subsequent smelting or refining (Bear et al., 2001; Lynch et al., 2010; Raggatt, 1968) – and the tailings were then profitably treated for their Zn content with additional Pb-Ag production. The main company involved in the early years of tailings reprocessing was Consolidated Zinc Ltd – one of the forerunner companies to Conzinc Riotinto Australia Ltd (or CRA), the effective Australian sister company of UK mining house Rio Tinto Zinc plc (which together merged to form Rio Tinto plc/Ltd in 1995). Despite two world wars and varying economic cycles, the Broken Hill field remains in production in 2015 and is now operated as a single project by Perilya Ltd (which from Dec. 2013 is now a wholly owned subsidiary of China's Pb-Zn mining group Shenzhen Zhongjin Lingnan Nonfemet Co Ltd).

As noted in Table 2, early ore reserves at Broken Hill were reported to be modest in comparison to the known size of the geological zones of mineralisation, and despite continued mining, exploration and development work has been able to maintain and increase reported reserves and resources for this world-class Pb-Zn field. The Broken Hill field has now been in operation for 132 years and, at present reported mineral resources (see Table 2) and ore processing of ~1.7 Mt ore/year, remarkably still has more than 20 years mine life remaining. The historical trends in ore grades and ore milled are shown in Fig. 5, showing the increasing emphasis on Zn over Pb-Ag, mainly in response to greater demand and higher prices for Zn.

2.3.2. Sullivan, Canada

The discovery of metallic sulfides in the Sullivan-Kimberley area of British Columbia can be traced to 1892, although mine production did not start until 1900. Here we summarize the history of this field from BCGS (2015).

After initial difficulties in treating the ores and developing a commercial project in the 1900s to 1910s, the international developments in differential flotation technology finally allowed the separation of Pb and Zn and economic development of the large Sullivan-Kimberley deposit from 1920. Large scale production continued until late December 2001, giving almost exactly a century's worth of operations; cumulative production[3] was 150.45 Mt ore grading ~6.4% Pb, ~6.0% Zn and ~71 g/t Ag to produce 8.412–7.944 Mt Pb-Zn and 9283 t Ag, as well as minor production of 175 kg Au, 9.7 kt Sn, 5.1 kt Cu, ≫135 t indium, 3.1 kt Cd, 22 t Bi and 414 t Sb.

In 1979 Sullivan reported ~49 Mt of mineral resources, although mining from 1980 to closure in late December 2001 only extracted ~38.6 Mt of ore. The historical trends in ore grades and ore milled are shown in Fig. 6 (including some estimated data), again showing the increasing emphasis on Zn over Pb-Ag, mainly in response to greater demand and higher prices for Zn.

2.3.3. Cannington, Australia

The Ag-rich Cannington Pb-Zn-Ag deposit was discovered by BHP Ltd in 1990 during a regional exploration campaign looking for Broken Hill-style deposits near Mt Isa in western Queensland, Australia. This case study is synthesized from Bailey (1998), BHP (var.), BHPB (var.) and S32 (2015).

Follow-up drilling of anomalies defined from aerial magnetic surveys led to the discovery hole containing 20 m at 12.1% Pb, 0.6% Zn and 870 g/t Ag – making it one of the potentially richest Pb-Zn discoveries in recent times. Further exploration work went on to prove a substantial, high grade deposit and BHP quickly sought and won approvals for development, with construction starting in 1996. The project was a conventional underground mine with a flotation mill to produce Pb and Zn concentrates, with first production in late 1997. In 2010

[2] It should also be noted that the caption to Table 3 of CMSA (2014) refers to 'reserve base' but the column is labelled 'reserves'.

[3] It should be noted that assayed ore grades were generally only reported from the 1970s onwards, and all pre-1973 data assumes 80–90% recovery for Pb, Zn and Ag, based on typical metallurgical recovery rates (e.g. Mudd, 2009a, 2009b).

Table 2
Ore reserves, additional mineral resources and cumulative production for selected Pb-Zn projects.

Project	Deposit type	Ore reserves							Additional mineral resources							Cumulative production							
		Year	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Year	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Year	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Mt WR
Broken Hill field, Australia	SH Pb-Zn	1905	9.4	16.6	15.9	327	–	–	–	–	–	–	–	–	–	1883–2012[g]	215.1	~10.1	~9.9	~154	~0.1	~0.2	≫5
		Dec. 2012	14.9	4.8	6.2	47	–	–	Dec. 2012	23.3	7.1	9.3	116	–	–								
Cannington, Australia	SH Pb-Zn	June 1997	4	13.9	6.2	645	–	–	May 1997[a]	39.8[a]	11.4[a]	4.2[a]	528[a]	–	–	1997–Sept. 2015	47.05	9.5	3.4	432	–	–	–
		June 2015	21	5.90	3.82	225	–	–	June 2015	71	4.1	2.7	127	–	–								
Century, Australia	SH Pb-Zn	June 1998	98.5	1.7	11.6	43	–	–	June 1998	6.6	1.7	19.6	91	–	–	2000–Sept. 2015	86.32	1.6	11.1	39	–	–	≫200
		June 2014	7.9	1.6	8.2	40	–	–	June 2014	2.1	2.2	12.1	51	–	–								
McArthur River, Australia	SH Pb-Zn	-	–	–	–	–	–	–	1990	227	4.1	9.2	44	0.2	–	1996–2014[g]	~32.75	4.0	12.0	50	–	–	–
		June 1995	26	6.3	14.0	63	–	–	June 1995	78	6.2	13.7	61	–	–								
		Dec. 2014	103.6	4.7	10.2	46	–	–	Dec. 2014	83.7	3.9	9.3	40	–	–								
Woodcutters, Australia	SH Pb-Zn	1984	1.1	7.7	17.9	140	–	–	–	–	–	–	–	–	–	1985–1999	4.72	6.0	12.8	80	–	–	–
		June 1990	3.75	5.6	12.6	116	–	–	June 1990	2.41	5.6	11.9	115	–	–								
Polaris (Little Cornwall), Canada	SH Pb-Zn	1975	~22.7	4.3	14.1	34	–	–	–	–	–	–	–	–	–	1981–2002	20.18	3.6	13.4	–	–	–	–
		-	–	–	–	–	–	–	~2005[b]	~5.9[b]	4.0[b]	15.6[b]	–	–	–								
Nanisivik (Baffin Island), Canada	SH Pb-Zn	1975	6.3	1.4	14.1	61	–	–	–	–	–	–	–	–	–	1976–2002[g]	17.86	~0.65	9.03	41	–	–	–
		-	–	–	–	–	–	–	~2005[b]	~7.3[b]	0.3[b]	2.8[b]	35[b]	–	–								
Red Dog, USA	SH Pb-Zn	1986	77	5.0	17.1	82	–	–	–	–	–	–	–	–	–	1989–2014	70.04	5.4	19.9	97	–	–	≫50
		Dec. 2014	52.8	4.3	16.5	80	–	–	Dec. 2014	0.3	3.4	10.2	68	–	–								
Pine Point, Canada	SH Pb-Zn	1964	~15.9	4.8	7.4	–	–	–	–	–	–	–	–	–	–	1965–1988	63.98	3.0	7.0	–	–		–
		-	–	–	–	–	–	–	2014	25.38	0.88	2.57	-	–	–								
Lisheen, Ireland	SH Pb-Zn	1998	13.4	2.4	13.3	–	–	–	–	–	–	–	–	–	–	1999–2014[g]	~21.1	~2.1	~12.1	–	–		–
		March 2015	0.7	1.29	9.46	–	–	–	March 2015	1.4	2.51	14.0	-	–	–								
Tara-Navan, Ireland	SH Pb-Zn	1974	69.9	2.6	10.1	–	–	–	–	–	–	–	–	–	–	1977–2014	85.5	1.9	7.9	–	–	–	–
		Dec. 2014	15.3	1.5	6.6	–	–	–	Dec. 2014	12.6	2.1	6.2	-	–	–								
Rosebery, Australia	VMS	1919	2.60	7.4	27.3	294	~0.3	3.89	–	–	–	–	–	––	–	1913–Sept. 2015	36.11	4.41	13.1	144	0.48	2.31	–
		June 2014	5.4	3.4	9.7	115	0.3	1.4	June 2014	18.8	3.1	8.6	141	0.35	2.3								
Hudson Bay field, Canada[c]	VMS	1918[dd]	17.96	–	3.49	35.7	1.69	2.54	–	–	–	–	–	–	–	1930–2014[c]	197.48	~0.1	3.44	24.0	2.00	1.65	–
		Jan. 2015	23.551	–	5.07	23.8	1.20	2.13	Jan. 2015	10.877	–	3.20	27.7	1.12	3.56								
Kidd Creek, Canada	VMS	-	–	–	–	–	–	–	~1964	>50[d]	–	–	–	–	–	1967–2014	~151.2	~0.2	6.04	~82	2.29	-	–
		Dec. 2014	11.6	–	4.3	52	2.0	-	Dec. 2014	8.0	–	5.9	53	2.0	–								
Myra Falls, Canada	VMS	1966	1.9[e]	–	–	–	–	–	–	–	–	–	–	–	–	1967–2014	30.20	~0.4	5.57	49.4	1.62	1.90	–
		Dec. 2014	5.87	0.61	5.92	77.1	0.85	1.78	Dec. 2014	3.57	0.77	7.05	99.4	0.98	2.31								
Duck Pond, Canada	VMS	Dec. 2006	4.08	–	5.68	59.3	3.29	0.86	Dec. 2006	0.54	–	17.9	165	7.69	1.75	2007–2014	4.54	–	4.54	–	2.85	–	–
		Dec. 2014	0.4	–	3.8	–	3.01	–	–	–	–	–	–	–	–								
Skorpion, Namibia	VMS	1998	–	–	–	–	–	–	1998	19.5	–	10.1	–	–	–	2003–2014[g]	~15.8	–	~11.2	–	–	–	–
		March 2015	5.1	–	8.77	–	–	–	March 2015	3.1	–	9.67	–	–	–								
Perseverance, Canada	VMS	June 2008	5.0	–	13.6	30	1.0	0.3	–	–	–	-	–	–	–	2008–2013[g]	~5.1	–	~13.3	–	~1.2	-	–
Antamina, Peru	Skarn	Dec. 2001	552.4	–	1.03	13.7	1.23	0.03[f]	Dec. 2001	100.2	–	0.55	8	0.59	0.02[f]	2001–2014	471.17	–	0.92	11	0.97	~0.02[f]	1541
		June 2015	614	–	1.00	11	0.95	0.023[f]	June 2015	1762	–	0.67	11	0.87	0.018[f]								

Notes: WR – waste rock. All mine production and reserves-resources data mostly from company corporate annual and/or quarterly reporting, with some data from government reports (e.g. Freeman et al., 2015; NRC, var.) or updated previous studies (e.g. Mudd, 2007, 2009a, 2010a).

[a] Estimated from total mineral resources (Bailey, 1998) minus reported ore reserves.
[b] Based on total mineral resources from Paradis et al. (2007) minus cumulative production data (and hence are approximate only and not code-compliant estimates).
[c] Hudson Bay production data includes all mines in the field (i.e. Flin Flon, 777, Chisel North, Callinan, Ruttan, Lalor-Snow Lake, Reed, etc.).
[d] From Hannington and Barrie (1999) (but no grade data);
[e] From Chong et al. (2005) (but no grade data).
[f] data is %Mo not g/t Au.
[g] Includes some estimated data

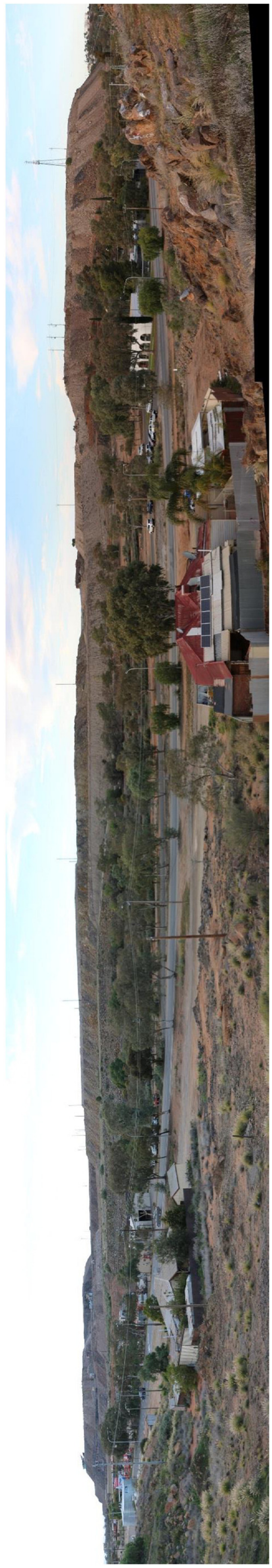

Fig. 4. The famous line of lode at Broken Hill after 131 years of mining (July 2014).

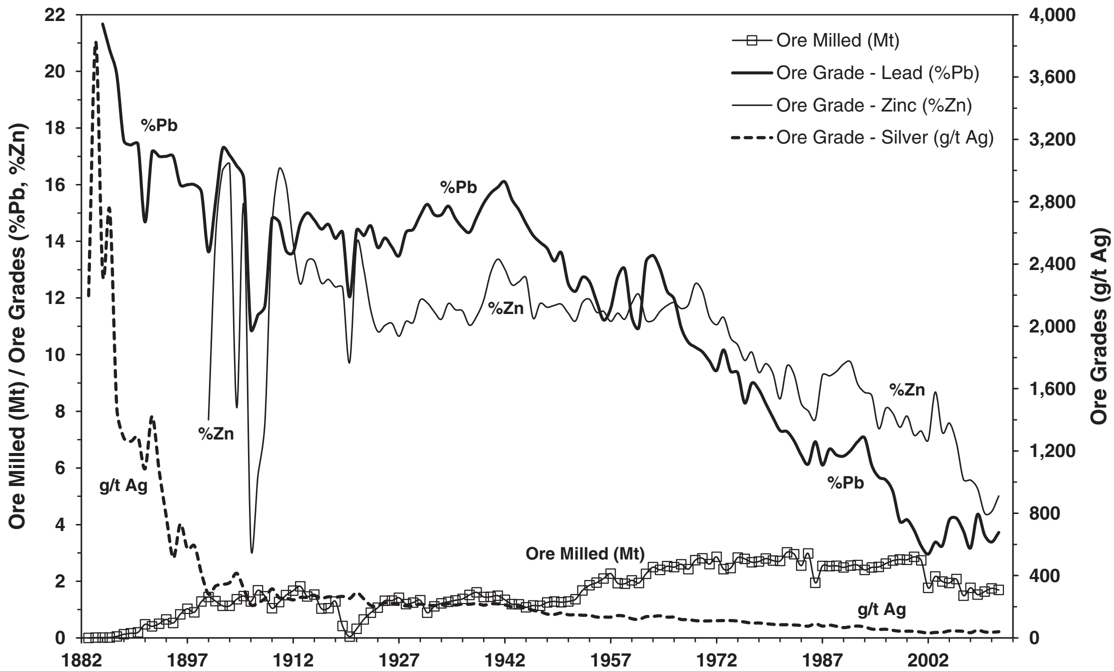

Fig. 5. Historical trends in ore grades and ore milled for the Broken Hill field, Australia (data updated from Mudd, 2007, 2009b).

BHP Billiton (formerly BHP) won approvals to begin the conversion to a combined underground and open cut mine as well as a mill expansion.

Curiously, although the May 1997 total mineral resources was 43.8 Mt at 11.6% Pb, 4.4% Zn and 538 g/t Ag, in their 1997 annual report BHP only report an ore reserve of just 4 Mt at 13.8% Pb, 6.2% Zn and 640 g/t Ag. By September 2015, total mine production was 47.05 Mt at 9.5% Pb, 3.4% Zn and 432 g/t Ag – greater than the original 1997 mineral resource. Perhaps just as importantly, however, is that remaining total mineral resources (as of June 2015) are still 91.9 Mt at 4.48% Pb, 2.98% Zn and 149 g/t Ag – including ore reserves of 21.3 Mt at 5.90% Pb, 3.82% Zn and 225 g/t Ag. The latest event in the history of the project was the inclusion of Cannington in the 2015 spin-out of South32 from BHP Billiton.

The trends in ore grades for ore milled, ore reserves and additional mineral resources are shown in Fig. 7. For comparison, typical ore grades at BHP's namesake Broken Hill mine over the same time period (i.e. the past ~20 years) have averaged 3.2–5.8% Pb, 4.4–8.7% Zn and 30–49 g/t Ag – showing the high grade nature of Cannington and its significance in global Pb-Zn-Ag mining.

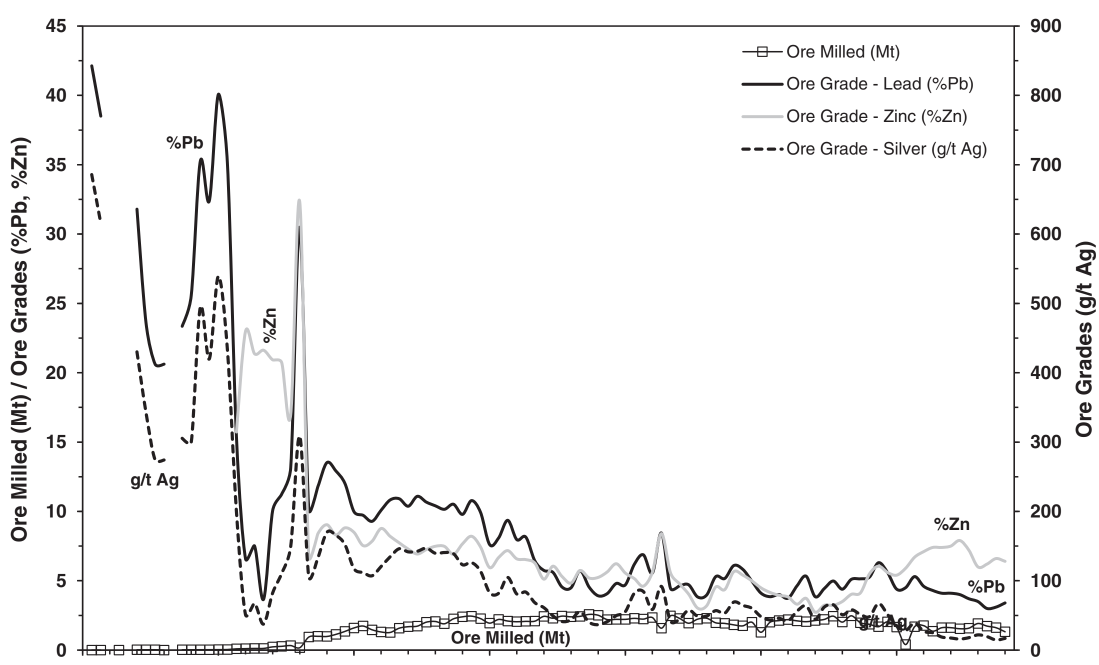

Fig. 6. Historical trends in ore grades and ore milled for the Sullivan-Kimberley field, Canada (data updated from BCGS, 2015; Mudd, 2009a; NRC, var.).

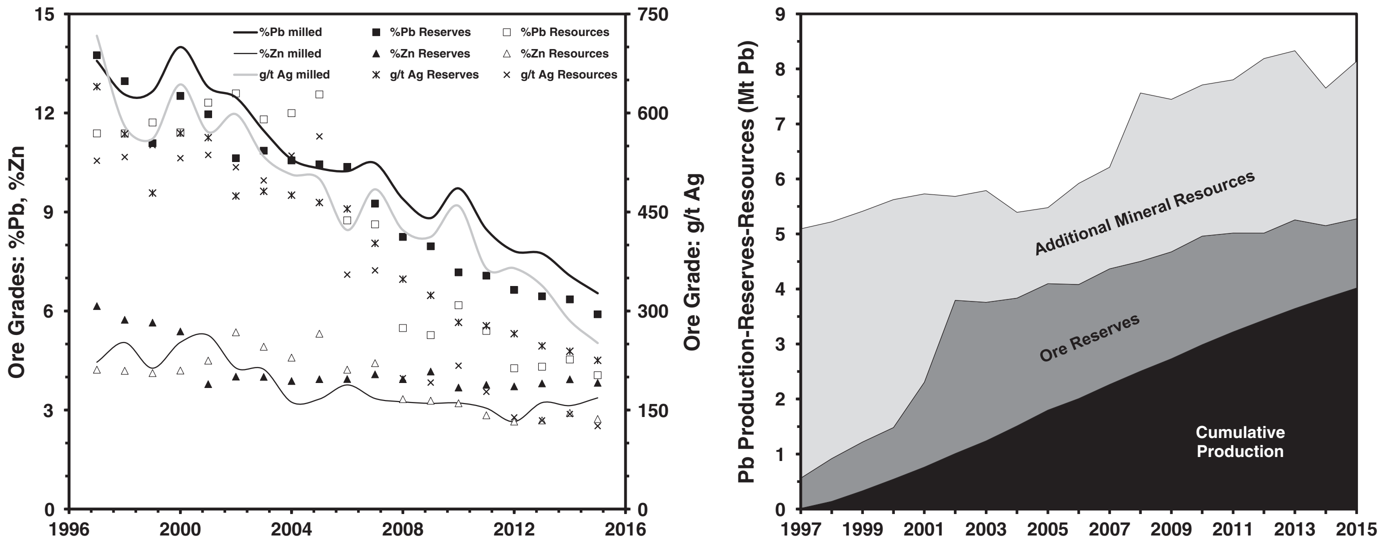

Fig. 7. Trends in ore grades for ore milled, ore reserves and additional mineral resources (left) and cumulative Pb production plus remaining reserves-resources (right) for the Cannington mine (data compiled from BHP, var.; BHPB, var.; S32, 2015).

2.4. Pb-Zn mineral deposit types

A wide range of mineral deposit types host significant Pb-Zn reserves and resources, and the resources in our database are classified by individual mineral deposit types according to the information available. Here, we provide a brief overview of the main Pb-Zn deposits types and discuss the controls on the concentrations of Pb and Zn within deposits that are generally exploited for other commodities (e.g. porphyry Cu deposits).

2.4.1. Sediment-hosted Pb-Zn deposits

Sediment-hosted Pb-Zn deposits are not genetically related to igneous activity, are sediment-hosted, and have Pb and/or Zn (rather than e.g. Cu) as their primary commodity (Leach et al., 2005). These deposits represent the world's most important source of Pb and Zn, are also an important source of other metals, such as Ag, Cu, As, Ba, Bi, Ge, Hg, Mn, Ni, P, Sb and Tl, and are becoming an increasingly important source of critical metals, such as indium (e.g. Werner et al., 2015, under review-a,-b,). The two primary subsidiary classifications of sediment-hosted Pb-Zn deposits are sedimentary exhalative (SEDEX) deposits and Mississippi Valley-type (MVT) deposits, both of which form at lower temperatures than, for example, Volcanogenic Massive Sulfide deposits that are directly related to magmatic activity (see Leach et al., 2005). SEDEX deposits form as a result of the seafloor venting of hydrothermal fluids of the replacing of existing sediments by secondary minerals as a result of hydrothermal activity (e.g. Leach et al., 2005; Robb, 2005). In comparison, MVT deposits form from low temperature and high-salinity basinal or connate fluids during the diagenesis of sediments within sedimentary basins (e.g. Robb, 2005). These distinctions between SEDEX and MVT deposits can at times be unclear or subjective, especially for deposits where little information is available (e.g. Leach et al., 2005), meaning that we focus on the primary classification of these types of deposits rather than SEDEX or MVT sub-classifications, or the further subdivision of these deposits into, for example, Broken Hill- or Irish-type deposits.

2.4.2. Volcanogenic Massive Sulfide (VMS) deposits

Volcanogenic Massive Sulfide (VMS) or Volcanic-Hosted Massive Sulfide (VHMS) deposits form at or near the seafloor in submarine volcanic environments (e.g. Galley and Koski, 1999). These deposits form in a wide variety of settings, have formed throughout geological history from the Archean to the modern day, are located across the globe, and range in size from insignificant to giant, as exemplified by the Kidd Creek VMS deposit in Ontario, Canada (current resources of 21.5 Mt ore grading 4.93% Zn, 2.04% Cu and 53 g/t Ag, with past production of ~151.2 Mt ore grading ~0.2% Pb, 6.04% Zn, 2.29% Cu and ~82 g/t Ag; see later Tables). However, although VMS deposits formed in any setting can contain significant amounts of Zn, VMS deposits containing significant concentrations (i.e. potentially economic amounts) of Pb typically only form in bimodal-felsic magmatic environments within supra-subduction epicontinental arcs or in siliciclastic-felsic settings within mature epicontinental back-arcs within environments dominated by continent-derived sedimentary and volcaniclastic units (Franklin et al., 2005). This difference relates to the source of the metals within these systems, with the frequently basaltic source of metals within bimodal-mafic, mafic, and pelite-mafic deposits containing minimal amounts of Pb but significant amounts of Cu and Zn, resulting in the Cu-Zn (rather than Cu-Zn-Pb) deposits that form in these settings (e.g. Franklin et al., 2005; Jowitt et al., 2012).

2.4.3. Skarn deposits

Skarn deposits form as a result of the interaction between magmato-hydrothermal fluids derived from intrusions and associated deeper magma chamber systems with surrounding country rocks, most notably limestones (Einaudi et al., 1981; Meinert et al., 2005). These deposits form during regional or contact metamorphism by a variety of differing metasomatic processes that can involve a variety of metamorphic, magmatic, marine and/or meteoric fluids (Meinert, 1992; Meinert et al., 2005). Skarns can contain a wide range of commodities that include Pb and Zn, as well as W, Sn, Mo, Cu, Fe, Pb, Zn, Au, Ag, Bi, Te, and As, with the commodities present given in an individual skarn dependent on differences in composition, oxidation state and the metallogenic affinity of the pluton (e.g. Einaudi et al., 1981). Skarns are commonly associated with and can form adjacent to or even within magmatic plutons, meaning that this type of mineralization may be genetically related to porphyry and epithermal deposits (Meinert et al., 2005).

2.4.4. Porphyry deposits

Porphyry deposits are bulk-tonnage, low- to medium-grade mineral deposits that are genetically linked with intrusive magmatism (e.g. Kirkham, 1971; Sillitoe, 2010; Sinclair, 2007), and are the world's most important source of Cu as well as important sources of Mo, Au, and Ag (e.g. Mudd et al., 2013a). These deposits, as mentioned elsewhere, are often associated with other mineral deposit types that may be more important in terms of Pb and Zn resources, such as epithermal, skarn, and manto types of mineralization (e.g. Hedenquist et al., 1998; Sillitoe, 2010; Sinclair, 2007). The magmatic systems that form these deposits are commonly associated with felsic to intermediate arc-type calc-alkaline magmas (Richards, 2003; Sillitoe, 1972; Sinclair, 2007) and can contain a diverse range of commodities, including Cu, Mo, Au, Ag, Re, PGEs, W, Sn, Bi, Zn, indium, and Pb (Kirkham and Sinclair, 1996). Typical porphyry Cu, Cu-Au, and Cu-Mo deposits are not well known for significant Pb or Zn endowments, although this is not always the case, as exemplified by the porphyry-type W-Mo subclass of deposits that may also contain significant amounts of Pb and Zn.

2.4.5. Epithermal deposits

Epithermal mineral deposits are generated by generally subaerial hydrothermal circulatory systems associated with magmatic heat sources, and the fluids generated by these systems form mineralization at temperatures <300 °C and <1.5 km below the local water table (Simmons et al., 2005; Tosdal et al., 2009). Low- to intermediate-sulfidation epithermal subtypes are most commonly associated with Pb-Zn mineralisation, although it should be noted that some of the Ag-Pb-Zn deposits classified as epithermal by mining companies (and hence in our database) most likely form deeper and at higher temperatures than is typical for epithermal systems (e.g. Sillitoe et al., 1998; Simmons et al., 2005). Epithermal mineralization forms within shallow sections of hydrothermal systems that typically form in volcanic arc settings (Simmons et al., 2005) and may be related to deeper, cogenetic, porphyry and skarn systems (e.g. Jowitt et al., 2013). The majority of epithermal mineral resources are dominated by Au and Ag, but may also contain Cu, Pb, Zn, As, Sb, and Sn (Simmons et al., 2005).

2.4.6. Sediment-hosted mixed deposits

This class of deposits refers to deposits such as Viken, Talvivaara and Woodlawn, which either have an unclear origin or are considered to be a mixture of some of the above classes (e.g. between VMS and Sedex end-members). The unclear nature of these resources means we use this separate category to avoid mixing mineral deposit types with potentially differing genetic origins.

2.4.7. Manto deposits

Manto-type deposits are blanket-like polymetallic replacement deposits formed of stratabound disseminated mineralization that are also frequently associated with hydrothermal brecciation (e.g. Sillitoe, 2003). These blankets of mineralisation can also be associated with discordant sulfide chimneys and the grades within the deposits appear to be controlled by permeability contrasts, with faults, breccias, intrusive contacts and flow tops all providing preferential sites for mineralization (Sillitoe, 2003). The term is generally used in South and Central America, and is associated with Cu and Pb-Zn-Ag subtypes, with the latter being the most important for this study, although the genesis of these deposits remains controversial with both magmato-hydrothermal (e.g. Wolf et al., 1990), hydrothermal (e.g. Kojima et al., 2009) and metamorphic (e.g. Sato, 1984; Wilson et al., 2003) fluid origins being proposed for this style of mineralisation and deposits. The latter is supported by the similarities between these deposits and stratiform sediment-hosted Cu mineralisation of the Kupferschiefer and the Zambian Copper Belt (e.g. Kirkham, 1996), although manto deposits are also associated with plutonic complexes, suggesting that magmatism may have been crucial in generating the fluid circulation that formed these deposits (e.g. Maksaev and Zentilli, 2002; Sillitoe, 2003). A link with IOCG mineralisation has also been suggested, although this again remains controversial (e.g. Groves et al., 2010; Williams et al., 2005).

2.4.8. Iron oxide copper-gold (IOCG) deposits

The iron-oxide copper-gold (IOCG) deposit category was first defined after the discovery of the giant Olympic Dam Cu-U-Au-Ag deposit but now covers a somewhat loosely grouped set of mineral deposit types (e.g. Groves et al., 2010). Strictly (according to Groves et al., 2010), IOCG deposits host structurally controlled magmato-hydrothermal mineralization that contains economic abundances of Au and Cu, are commonly brecciated and are associated with Na-Ca alteration, are light rare earth element (LREE) enriched, and contain high abundances of low-Ti Fe oxides that are associated with low-S sulfide mineralization. A range of commodities are present within IOCG deposits, including Fe, Cu, and Au, but often with significant or by-product Ag, U, REE, Bi, Co, Nb, and P, amongst others, with Pb and Zn infrequently present in economic amounts (e.g. Corriveau, 2007), making this deposit category less significant for global Pb-Zn resources than some of the others described here.

2.4.9. Orogenic gold and mesothermal vein deposits

Orogenic Au deposits form from low salinity, near neutral pH, mixed H_2O-CO_2 metamorphic fluids (e.g. Goldfarb et al., 2005; Robb, 2005), are known to have formed throughout Earth's history (e.g. Goldfarb et al., 2010) and are hosted by regionally metamorphosed terranes that developed as a result of compressional tectonism at convergent plate margins (i.e. orogenesis). Here, we have grouped these orogenic Au deposits with deposits classified as mesothermal vein deposits within our database, primarily as both of these deposit types most likely formed from the same processes and in the same settings, although insufficient data was provided by the available descriptions for the mesothermal deposits in our database to confirm these similarities. Both deposit types generally host significant amounts of Au mineralization, with Pb and Zn produced as by-products, although this category of deposit does include orogenic or mesothermal deposits that are more base metal rich than is typical for these Au-dominated classes of deposit.

2.4.10. Mine wastes (tailings and slags)

Mine wastes of various types are increasingly being viewed as economic opportunities rather than environmental issues and can often contain significant (if not currently economic) concentrations of Pb and Zn. These wastes include tailings, which contain the fine-grained (as a result of crushing and separation) bulk of the uneconomic or gangue fraction of ore that is discarded during mineral processing; these wastes are generally produced in large volumes during milling and often contain sulfide minerals that can be the source of environmental contamination when exposed to oxidation or weathering, but can also be of economic interest if present in sufficient concentrations. In comparison, slags are generated during the cooling of molten solutions of oxides produced during base metal smelting and refining. This means that although tailings and slags do not occur naturally, we consider these to be relevant classes of anthropogenic mineral deposit types that may be an important future resource of base and critical metals, including Pb and Zn. It should be noted that the metallurgical processes that produce these wastes are optimised on a project by project basis to suit the specific conditions present at each facility, meaning that the physical composition of both slags and tailings can be highly variable. In addition, there remains a number of technical challenges to processing these materials (e.g. Lottermoser, 2010; Shen and Forssberg, 2003); see also our later discussion sub-section on tailings), meaning these resources may be more theoretical than the mineral deposit-hosted resources described elsewhere in this paper.

2.4.11. Miscellaneous

This category incorporates all other deposit types where insufficient information was available to classify these deposits with any degree of certainty; this classification therefore avoids the introduction of any more uncertainty or misclassification into our database.

2.5. Our hybrid resource accounting approach

The global mining industry typically uses formal codes for the reporting of exploration results, mineral resource and ore reserves, such as the Joint Ore Reserves Committee (JORC) Code in Australia

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(AusIMM, MCA and AIG, 2012), the South African Code for Reporting of Mineral Resources and Mineral Reserves (SAMREC, 2009), Canada's National Instrument 43-101 ('NI43-101') (OSC, 2011) and associated CIM Code (CIM, 2014), PERC in Europe (PERC, 2013), and others in Russia, China, Chile, etc. In the United States, the requirements from the Securities & Exchange Commission (SEC) are somewhat different (see SEC, 1992), and only allow the reporting of ore reserves unless required by foreign law – meaning that estimates of mineral resources are typically reported by such companies as additional mineralized material (e.g. see reporting by the Coeur d'Alene Mines Corporation). The international harmonisation of resource-reserve reporting is being undertaken through CRIRSCO, the Committee for Mineral Reserves International Reporting Standards. These codes have, in the main, proven robust in ensuring reasonable estimates of mineral deposits (as noted earlier and shown by Table 2).

In this study, we adopt formally reported mineral resources as the basis to assess the long-term future of Pb-Zn mining but also expand our scope to allow the inclusion of non-code basis reporting that is not compliant with current code requirements. This non-code data can come from historical estimates of resources or reserves, other countries' reporting standards, or technical and research reports on particular mining projects that estimate reserves but not to code-compliant standards. Although historical estimates are not code-compliant, there are an increasing number of examples where modern exploration work has confirmed such historical estimates to a modern code-compliant resource or even increased them, such as Berg Aukus in Namibia (China Africa Resources plc), Toral in Spain (Portex Minerals Inc) and Jubilee in Canada (Merrex Gold Inc.). Our compiled resources data are therefore presented as code or non-code based to indicate its potential reliability and to give the reader a measure of confidence in the data we present. In addition, all of the metals reported as part of a mineral resource are included in our database in order to assess and understand the relative importance of Pb-Zn as primary metals versus other co/by-product metals.

In general, resource data are compiled as they are reported by individual deposit, although sometimes these data are aggregated to a single resource as this is how the project is operated. For some fields, however, this distinction is sometimes unclear; for example, the Mt Isa field in western Queensland, Australia, is listed as separate major deposits despite being operated as a single project with one central mill and Pb smelter. In addition, some deposits have reported most recent resources that exclude Pb-Zn as a result of mill design changes that mean these projects no longer produce Pb or Zn concentrates (e.g. La Granja, Peru; Viscaria, Sweden; Jabal Sayid, Saudi Arabia). However, we have included these older resources if they are similar in size and scope to the newer resource as this allows a more complete assessment of world Pb-Zn resources (any economic assessment of projects is outside the scope of this paper). When a resource is reported at varying cutoff grades, we adopt the lowest cutoff grade for inclusion in our dataset, primarily as we focus on total mineral resources that may be available for mining over the longer term. Given the long-term decline in base metal ore grades (see later discussion), and the fact that low grade ore stockpiles can be considered a future mineral resource (see results later), we believe this is reasonable to ensure a robust minimum estimate of the world's Pb-Zn resources.

There are also numerous deposits where Pb or Zn occur but are not reported in code-based estimates of mineral resources. Although this is simply the outcome of a companies' technical and economic assessment, it does not mean the Pb/Zn are never recoverable – simply that it is uneconomic at the time of the assessment. Examples where Pb-Zn are known to occur but are not reported include Cape Ray, Perevalnoye, Pueblo Viejo and Ok Tedi, amongst others; as such, we have excluded these deposits as there is insufficient data to include for reasonable comparison to the primary data sources adopted, although the Pb-Zn potential of these deposits are worth noting.

There have been some global compilations of Pb-Zn resources published, often focussing on specific mineral deposit types such as skarns (Meinert et al., 2005) or sediment-hosted Pb-Zn (Leach et al., 2005). We have consulted these publications and cross-checked with other information sources to confirm mined production versus remaining resources (or depletion and site closure), and where possible, we have incorporated data from such publications when we are confident there are remaining resources and labelled such resources as non-code based.

To address the vexing issue of whether Pb-Zn are primary products or by-products, we assign prices to all reported metals and group the resources data based on whether the proportional value of Pb-Zn-Ag is greater (or lower) than 70% – thereby allowing a view of the differences in ore grades and contained metals between Pb-Zn as primary products or by-products.

Finally, all deposits are classified by primary mineral deposit type (as described above) based upon the best available geological evidence and by the dominant mineral deposit type within a given project (e.g. Jowitt et al., 2013; Mudd and Jowitt, 2014). Where there is a lack of specific information, other information on regional geology is used to assign a deposit type.

3. Results: World resources of Pb-Zn

3.1. Total Pb-Zn resources

Our compiled mineral resources data (Table 3) includes 851 deposits with 51,030.0 Mt of mineral resources at average overall grades of 0.44% Pb, 1.20% Zn, 16.4 g/t Ag, 0.28% Cu and 0.12 g/t Au. This contains an estimated 226.1 Mt Pb, 610.3 Mt Zn, 837,128 t Ag, 144.4 Mt Cu and 6132 t Au, or 980.8 Mt Pb + Zn + Cu, along with a wide variety of possible co/by-products from a number of projects (as explored in a later sub-section). Of these 851 deposits, 294 and 557 are non-code and code-based, respectively. Of these, 17 resources are reported for mine wastes, of which two are for stockpiles (i.e. low grade ore), one for a smelter slag heap and 14 are for tailings resources.

Sediment-hosted Pb-Zn deposits dominate contained Pb-Zn metal resources with 141.6–345.1–3.9 Mt Pb-Zn-Cu, followed by Volcanogenic Massive Sulfide deposits with 25.6–106.9–44.5 Mt Pb-Zn-Cu, and lesser amounts from the remaining deposit types. For comparison, the mine wastes contain a mere 0.83–2.01–0.19 Mt Pb-Zn-Cu – almost insignificant in contrast to the dominant deposits.

Although mesothermal vein deposits have the highest average combined base metal grades at 3.49–2.77–0.03 %Pb-Zn-Cu (total 6.28%), there are only 13 such deposits compared to the 276 sediment-hosted Pb-Zn deposits which average 1.81–4.40–0.05 %Pb-Zn-Cu (total 6.26%) and the 270 Volcanogenic Massive Sulfide deposits which average 0.60–2.52–1.05 %Pb-Zn-Cu (total 4.17%) – showing the importance of high grades combined with large mineral resources in yielding higher contained metal totals.

The results (Table 3) for the proportional value of Pb-Zn-Ag (greater or lower than 70%) shows some interesting outcomes. Firstly, just over half (527) of the deposits show Pb-Zn-Ag >70% of value, with average grades of 0.75% Pb, 1.77% Zn and 23.2 g/t Ag (i.e. higher by factors of ~6× for Pb, ~3× for Zn and ~2.5× for Ag). Secondly, despite being similar in resource tonnage, the vast majority of contained Pb-Zn is in deposits with Pb-Zn-Ag value >70% – specifically 197.8–470.1 Mt Pb-Zn, representing some 87.5% and 77.0% of total contained Pb and Zn, respectively. This suggests that Pb-Zn can be targeted as primary metals with a higher recoverability than if they were by-products. Finally, the dominant role of Cu is clear for deposits where Pb-Zn-Ag value <70%, along with the importance of Au, with ore grades and contained metals being ~10× higher than deposits with Pb-Zn-Ag value >70%.

With respect to code or non-code based resources data, a clear majority is contained in code-based resources, specifically 159.3–458.4 Mt Pb-Zn out of the total of 226.1–610.3 Mt Pb-Zn. This gives

Table 3
Global Pb-Zn mineral resources by deposit type, proportional Pb-Zn-Ag value and reporting basis.

Deposit type	%PbZnAg value	No.	Basis	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Mt Pb	Mt Zn	t Ag	Mt Cu	t Au	Mt Pb + Zn + Cu	%Pb + Zn + Cu
Sediment-hosted Pb-Zn (SH Pb-Zn)	>**70**	124	C	5336.2	1.77	4.41	30.70	0.01	0.01	94.44	235.11	163,845	0.52	48.0	330.06	6.19
	<**70**	9	C	88.0	0.95	1.54	21.83	0.47	0.63	0.83	1.35	1,920	0.41	55.3	2.60	2.95
	>**70**	129	NC	2290.2	1.97	4.61	14.25	0.06	0.004	45.23	105.68	32,635	1.41	9.1	152.32	6.65
	<**70**	13	NC	124.8	0.84	2.37	0.78	1.26	0.11	1.05	2.96	97	1.57	14.2	5.58	4.47
	>**70**	1	S NC	0.12	3.5	24	150	–	–	0.004	0.029	19	–	–	0.03	27.5
		Sub-total	*7839.3*	*1.81*	*4.40*	*25.3*	*0.05*	*0.02*	*141.55*	*345.13*	*198,516*	*3.91*	*126.5*	*490.59*	*6.26*	
Volcanogenic Massive Sulfides (VMS)	>**70**	54	C	661.5	1.76	4.24	74.08	0.27	0.40	11.61	28.08	49,009	1.77	262.8	41.46	6.27
	<**70**	133[a]	C[a]	2097.1	0.45	2.52	29.51	1.11	0.79	9.38	52.78	61,893	23.24	1650.4	85.40	4.07
	>**70**	22	NC	129.5	2.70	3.59	46.97	0.04	0.04	3.50	4.64	6,082	0.05	4.7	8.19	6.33
	<**70**	60	NC	1355.6	0.08	1.55	10.46	1.43	0.59	1.15	20.97	14,185	19.40	805.1	41.52	3.06
	>**70**	1	S C	2.0	–	22	–	–	–	–	0.44	–	–	–	0.44	22.00
		Sub-total	*4245.7*	*0.60*	*2.52*	*30.9*	*1.05*	*0.64*	*25.64*	*106.92*	*131,169*	*44.47*	*2,723.0*	*177.02*	*4.17*	
Skarn	>**70**	47[b]	C[b]	1294.4	0.80	2.47	40.47	0.08	0.04	10.33	32.03	52,381	1.07	46.2	43.44	3.36
	<**70**	25[c]	C[c]	4259.2	0.10	0.72	13.86	0.53	0.21	4.35	30.67	59,021	22.72	904.2	57.73	1.36
	>**70**	13[d]	NC[d]	140.8	2.50	2.89	69.48	0.01	–	3.52	4.07	9,780	0.01	–	7.60	5.40
	<**70**	7	NC	213.1	0.34	0.62	3.89	0.32	0.22	0.71	1.32	830	0.68	46.3	2.71	1.27
		Sub-total	*5907.5*	*0.32*	*1.15*	*20.7*	*0.41*	*0.17*	*18.92*	*68.09*	*122,013*	*24.49*	*996.6*	*111.49*	*1.89*	
Porphyry	>**70**	2[e]	C[e]	940.7	0.25	0.46	17.63	–	0.05	2.34	4.37	16,581	–	45.1	6.70	0.71
	<**70**	11	C	13,562.7	0.01	0.06	1.78	0.43	0.06	1.24	8.36	24,086	57.75	750.6	67.36	0.50
	>**70**	2	NC	1.3	0.48	0.35	321.10	–	–	0.006	0.005	408	–	–	0.01	0.84
	<**70**	1	NC	160.0	3.20	3.60	180.00	1.50	0.60	5.12	5.76	28,800	2.40	96.0	13.28	8.30
		Sub-total	*14,664.7*	*0.06*	*0.13*	*4.8*	*0.41*	*0.06*	*8.70*	*18.50*	*69,876*	*60.15*	*891.8*	*87.35*	*0.60*	
Epithermal	>**70**	71[f]	C[f]	1974.2	0.78	1.62	103.63	0.07	0.09	15.34	31.95	204,582	1.31	177.6	48.60	2.46
	<**70**	28	C	1109.5	0.19	0.72	20.01	0.31	0.80	2.13	8.01	22,205	3.47	884.0	13.61	1.23
	>**70**	25	NC	990.7	0.58	0.49	53.79	0.002	0.005	5.71	4.89	53,288	0.02	4.6	10.62	1.07
	<**70**	14[g]	NC[g]	57.9	1.27	2.02	64.73	0.68	1.90	0.74	1.17	3748	0.39	109.8	2.30	3.97
		Sub-total	*4132.3*	*0.58*	*1.11*	*68.7*	*0.13*	*0.28*	*23.91*	*46.02*	*283,822*	*5.19*	*1176.1*	*75.12*	*1.82*	
Sediment-hosted mixed[h]	>**70**	1	C	10.1	4.03	10.15	84.45	1.78	0.55	0.41	1.03	853	0.18	5.6	1.61	15.96
	<**70**	7		12,571.1	0.01	0.11	0.05	0.03	-	1.69	14.14	691	3.51	–	19.33	0.15
	>**70**	1	NC	0.02	26.70	7.30	1,027	–	–	0.01	0.0015	21	–	–	0.01	34.00
		Sub-total	*12,581.3*	*0.02*	*0.12*	*0.1*	*0.03*	*<0.001*	*2.10*	*15.16*	*1564*	*3.69*	*5.6*	*20.95*	*0.17*	
Mesothermal vein	>**70**	12	C	98.9	3.50	2.76	160.9	0.03	0.03	3.46	2.73	15,908	0.03	3.0	6.22	6.29
	<**70**	1	NC	1.3	2.50	3.50	242.2	–	5.33	0.03	0.04	306	–	6.7	0.08	6.00
		Sub-total	*100.2*	*3.49*	*2.77*	*161.9*	*0.03*	*0.10*	*3.49*	*2.77*	*16,215*	*0.03*	*9.7*	*6.29*	*6.28*	
Iron oxide copper-gold (IOCG)	>**70**	1	C	8.3	0.35	0.02	836.46	0.26	–	0.03	0.002	6,917	0.02	–	0.05	0.63
	<**70**	5	C	317.3	0.02	0.59	3.23	0.50	0.05	0.08	1.88	1,024	1.58	15.0	3.53	1.11
		Sub-total	325.6	0.03	0.58	24.4	0.49	0.05	0.11	1.88	7,941	1.60	15.0	3.58	1.10	

Miscellaneous[i]	<70	3	C	971.7	0.01	0.23	–	0.01	–	0.07	2.26	0	0.08	–	2.40	0.25
	>70	1	NC	0.1	3.70	–	316.0	–	–	0.01	–	44	–	–	0.01	3.70
		Sub-total	*971.8*	*0.01*	*0.23*	*0.0*	*0.01*	–	*0.07*	*2.26*	*44*	*0.08*	–	*2.41*	*0.25*	
Mine wastes (tailings & slags)[j]	>70	6	T C	16.2	0.25	2.65	36.41	–	–	0.04	0.43	589	–	–	0.47	2.91
	<70	6	T C	129.0	0.57	0.80	15.86	0.15	0.71	0.74	1.03	2,046	0.19	91.2	1.96	1.52
	>70	2	T NC	0.6	6.79	9.79	5.99	0.005	0.04	0.04	0.06	4	0.00003	0.03	0.11	16.58
	<70	1	T NC	1.0	–	2.30	21.00	0.40	–	–	0.02	21	0.004	–	0.03	2.70
		Sub-total	*146.8*	*0.56*	*1.05*	*18.1*	*0.13*	*0.62*	*0.82*	*1.55*	*2660*	*0.19*	*91.2*	*2.56*	*1.74*	
Manto[k]	>70	4	C	25.1	1.02	3.87	81.4	0.05	0.52	0.26	0.97	2,042	0.01	13.0	1.24	4.94
Orogenic gold	>70	2	C	21.2	1.09	1.96	10.38	0.10	0.002	0.23	0.42	220	0.02	0.04	0.67	3.15
	<70	5	C	37.5	0.72	0.89	18.80	0.25	1.56	0.27	0.33	705	0.09	58.3	0.70	1.86
	<70	1	NC	31.0		0.90	11.00	1.60	0.80	–	0.28	341	0.50	24.8	0.78	2.50
		Sub-total	*89.7*	*0.56*	*1.15*	*14.1*	*0.68*	*0.93*	*0.50*	*1.03*	*1,266*	*0.61*	*83.2*	*2.14*	*2.39*	
Code (C)	>70	*324*	*C*	22,931.7	0.61	1.53	22.33	0.04	0.03	139.72	349.80	512,175	8.27	595.7	497.79	2.17
	<70	*220*	*C*	22,453.0	0.08	0.48	7.65	0.49	0.19	18.75	106.67	171,707	109.52	4323.5	234.94	1.05
Non-code (NC)	>70	*193*	NC	3552.6	1.63	3.36	28.78	0.04	0.01	57.97	119.30	102,259	1.49	18.3	178.76	5.03
	<70	*97*	NC	1943.7	0.45	1.67	24.85	1.28	0.57	8.80	32.49	48,308	24.95	1,102.9	66.24	3.41
Mine wastes (C + NC)	>70	*10*	C + NC	19.0	0.47	5.07	32.31	<0.001	0.001	0.09	0.96	613	<0.001	0.0	1.05	5.54
	<70	*7*	C + NC	130.0	0.57	0.81	15.90	0.15	0.70	0.74	1.05	2,067	0.19	91.2	1.98	1.53
By dominant value	>70	527	–	26,503.3	0.75	1.77	23.21	0.04	0.02	197.78	470.05	615,046	9.76	614.0	677.59	2.56
	<70	324	–	24,526.7	0.12	0.57	9.05	0.55	0.22	28.29	140.22	222,082	134.66	5,517.6	303.17	1.24
		851		**51,030.0**	**0.44**	**1.20**	**16.4**	**0.28**	**0.12**	**226.1**	**610.3**	**837,128**	**144.4**	**6,132**	**980.8**	**1.92**

Notes: No. – number of deposits; C – code-based, NC – non-code based, S C – stockpile code based, S NC – stockpile non-code based, T C – tailings code based, T NC – tailings non-code based. Some values may not add directly due to rounding.

[a] Includes two VMS-Epithermal deposits.

[b] Includes one skarn-manto and one skarn-epithermal.

[c] includes one each of skarn (Greisen), skarn-epithermal and skarn-manto deposits.

[d] includes two skarn-manto deposits.

[e] Includes one porphyry-VMS and one porphyry-skarn deposit.

[f] Includes one epithermal-skarn deposit.

[g] Includes one epithermal-mesothermal deposit.

[h] Includes two sediment-hosted Pb-Zn (shale-hosted), three sediment-hosted Pb-Zn/VMS, one sediment-hosted polymetallic, one shale-hosted and one sediment-hosted U deposits; [i]Includes one deposit each of granite-related, magmatic alkaline intrusive, magmatic sulfide and intrusion-related.

[j] Includes original ores from VMS, epithermal, sediment-hosted Pb-Zn, mesothermal vein and skarn deposits.

[k] Includes one manto/skarn/epithermal and one manto-Cu deposits.

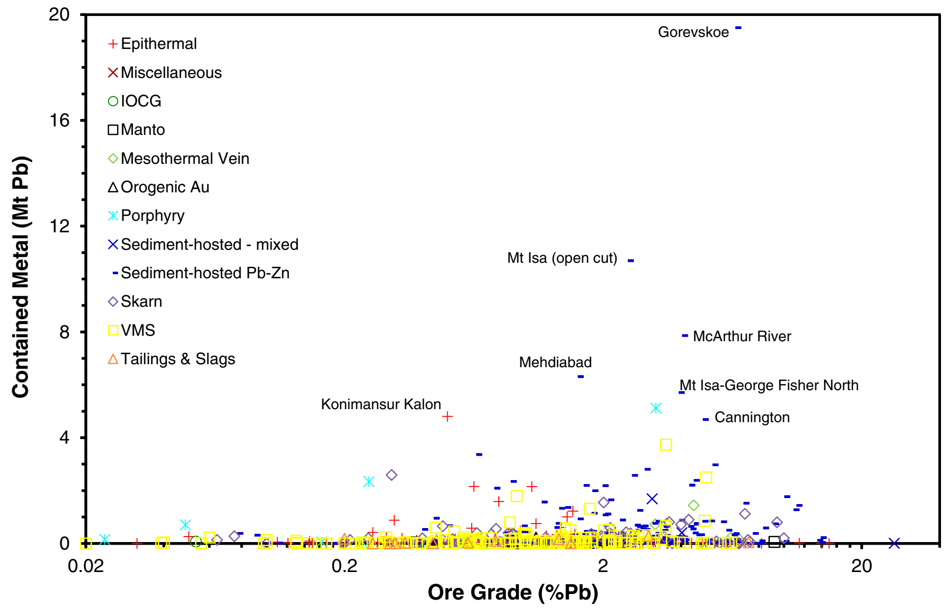

Fig. 8. Contained Pb versus ore grade by deposit types.

good confidence in our compiled resources data, but also suggests there are opportunities to convert non-code based resources into code based data (as highlighted earlier for historically reported resources at some projects).

The relationship between ore grades and contained metals by deposit type is shown in Figs. 8 to 10. All graphs show a wide variation of ore grades as well as a broad normal distribution with a negative skew towards lower grades. The Pb resources which stand out as exceptional include the Mt Isa field, Gorevskoe, Cannington, Mehdiabad, Konmansur Kalon and McArthur River, with exceptional Zn resources including Kholodninskoe, McArthur River, Mehdiabad, Rampura Agucha, Antamina and Talvivaara, with the latter two being unusually large for their respective deposit types. All of these graphs indicate that sediment-hosted Pb-Zn deposits typically have higher amounts of contained metals and higher ore grades, whereas Volcanogenic Massive Sulfide deposits are probably the most variable of the deposits in our database. Although Cu is important in many deposits, adding Cu in the graphs does not appear to materially affect the overall pattern of contained metals versus ore grades.

The country data given in Table 4 indicates that the country with the largest Pb-Zn resources is Australia at 59.5–95.4 Mt Pb-Zn, followed by Russia with 28.0–62.9 Mt Pb-Zn, Canada with 22.8–76.8 Mt Pb-Zn and Peru at 11.7–52.7 Mt Pb-Zn, although Peru also has substantial Cu within their Pb-Zn resources at 39.7 Mt Cu (Canada and Australia have just 6.9 and 4.4 Mt Cu, respectively). Another interesting aspect of the country data is the high variability of Ag resources, with Mexico's Ag resources approximately double that of Australia or Peru, primarily as a result of the importance of the Ag-rich epithermal deposits to the Pb-Zn resources in Mexico. When examining Table 4, it is noteworthy that Australia and Canada both have larger remaining resources than cumulative production to date (see data in Table 2), with most other countries having considerably lower remaining resources. This again

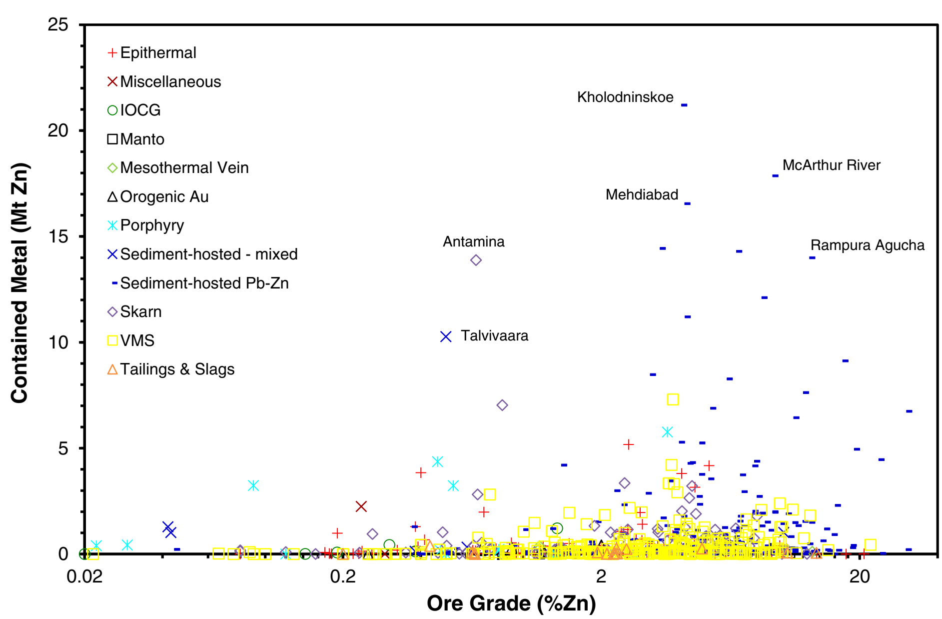

Fig. 9. Contained Zn versus ore grade by deposit types.

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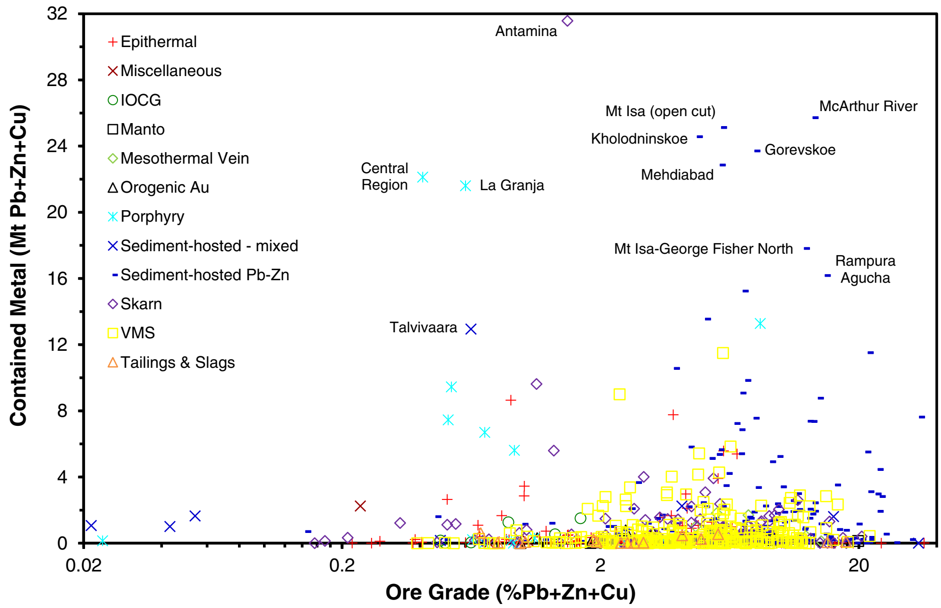

Fig. 10. Contained metals versus combined base metal ore grade by deposit types.

shows the variability in apparent Pb-Zn resource depletion and the critical importance of maintaining a global view when considering metal resources.

It is also important to remember that our compiled data does not include deposits in countries which do not publicly report ore reserves and mineral resources by project or company. In this manner, we are substantially under-estimating the resources of several countries – but especially China. For 2014, China's national Pb-Zn resources were estimated at 73.85–144.86 Mt Pb-Zn (Wang, 2015), whereas our data only contains 24 deposits with 2340 Mt of mineral resources grading 0.36–0.86% Pb-Zn and containing 8.7–20.5 Mt Pb-Zn – showing that we are still missing the vast majority of the Chinese Pb-Zn sector. This is likely to be similar for many countries which have minimal to no public reporting of resources-reserves (e.g. Iran, Pakistan, among others). A detailed review of China's sediment-hosted Pb-Zn deposits is given by Wang et al. (2014), including tonnage-grade data for 35 deposits accounting for 17.9–69.8 Mt Pb-Zn, although it is unclear whether their data is remaining resources or cumulative production plus remaining resources. As such, we have noted but not included these data, primarily as a result of their uncertain provenance.

The largest 25 deposits by contained base metals (Mt Pb + Zn + Cu) or combined base metal ore grades (%Pb + Zn + Cu) are shown in Tables 5 and 6, respectively. Unsurprisingly Antamina is the world's largest single resource, primarily as a result of the contained Cu within the deposit, whilst McArthur River is a close second. Furthermore, the Mt Isa field has two deposits in the top 25 largest, and if all of Mt Isa's 5 deposits are combined this would make Mt Isa the biggest Pb-Zn resource by a considerable margin (that is, a total of 20.33–32.51 Mt Pb-Zn). Similarly, if the numerous Howard's Pass deposits were aggregated, they would reach 6.59–20.29 Mt Pb-Zn. A curious point to note is that all deposits except Cannington contain more Zn than Pb. From a geological perspective, Zn is more geochemically abundant than Pb in the earth's crust (67 versus 17 mg/kg; Rudnick and Gao, 2014), and given their similar geochemical behaviour during ore forming processes it is therefore to be expected that Zn shows higher grades than Pb. In reality, however, over the past half a century Zn has typically averaged higher prices than Pb and has also seen considerable growth in demand compared to Pb (which has been dampened by environmental and public health concerns). It is debateable whether the mining industry, both during exploration and mining, has favoured Zn over Pb over the past half a century or so, as exploration cannot target one metal over the other given their similar geochemistry, but mining can certainly choose to mine Zn-rich ores if this suits their mine planning and economic needs (as noted earlier for Broken Hill and Sullivan).

3.2. Relative economic value of all metals and deposits

In order to understand the relative role of individual metals, especially potential co/by-products, we have determined the economic value of all of the deposits within our database. Price data was sourced from OCE (2014) and USGS (2015), and was generally based on international benchmark sources (such as a metals trading exchange). All ore grades were converted to 2013 average market prices and contained metals re-plotted against total unit ore value (US$/t ore), as shown in Fig. 11. Curiously, this effectively smooths the distribution and reduces the skewness of the data, as well as emphasizing the wide range in unit ore value.

One perhaps unexpected result in Fig. 11 is the unusual Buckton-Buckton South deposit in Canada that has extremely low grades of Zn-Cu but also contains the richly prized scandium (Sc) – which is generally viewed as being as valuable as Au,[4] albeit within an extremely small Sc market of ~10 t Sc/year) and adds very substantially to the unit ore value of this deposit. If Sc is removed, however, unit ore and total value for Buckton-Buckton South drops to just US$13.61/t ore and US$68.2 billion, respectively – both considerably lower but clearly showing that the best potential value for this project lies in its Sc potential. It must be recognised that the present unrealistically high Sc value is largely a function of low demand and supply (i.e. an extremely niche market controlled by a small number of companies in Russia, China, Kazakhstan and Ukraine; USGS, 2016) – and it is therefore highly contentious as to what the future could hold for Sc. However, given the various uses with strong potential future demand profiles if a reliable and economic supply can be established (e.g. solid oxide fuel cells, specialty alloys, optical and electronics technologies; see Wang et al., 2011), Sc clearly remains a metal worth examining. Thus, although a recent preliminary economic assessment of the potential development of Buckton

[4] According to USGS (var.-a), prices in the USA for Sc between 2007 to 2014 have ranged from US$0.7/g to US$235/g, depending on the degree of purity and chemical form, such as oxide, acetate, fluoride, chloride, iodide, Sc-aluminium (Al) alloy, or dendritic metal. We chose the 2014 value of US$134/g for Sc metal as this appears to be the most representative of Sc value.

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Table 4
Global Pb-Zn mineral resources by country.

Country	No.	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Mt Pb	Mt Zn	t Ag	Mt Cu	t Au	Mt Pb + Zn + Cu
Australia	132	2427.2	2.45	3.93	41.0	0.18	0.09	59.49	95.42	99,595	4.39	207.2	159.31
Russia	40	2561.9	1.09	2.46	13.6	0.75	0.33	27.98	62.94	34,948	19.31	852.3	110.23
Canada	229	7599.6	0.30	1.01	7.9	0.09	0.11	22.75	76.82	60,287	6.88	844.3	106.45
Peru	48	7537.0	0.16	0.70	13.9	0.53	0.04	11.73	52.68	105,048	39.71	282.0	104.12
Mexico	65	4966.0	0.30	0.93	38.1	0.15	0.22	14.73	46.08	189,163	7.44	1,084.4	68.25
Kazakhstan	17	6075.4	0.13	0.33	3.3	0.41	0.13	7.60	19.87	20,075	24.61	796.2	52.09
India	38	681.7	1.62	5.12	43.3	0.09	0.02	11.02	34.92	29,511	0.58	16.2	46.52
China	24	2379.7	0.36	0.86	13.1	0.35	0.13	8.65	20.50	31,173	8.25	315.6	37.40
Iran	10	449.6	1.88	5.61	37.1	–	–	8.45	25.23	16,693	–	–	33.68
USA	25	440.8	1.44	4.94	71.8	0.34	0.32	6.36	21.78	31,647	1.49	140.6	29.64
North Korea	2	426.0	1.75	3.98	67.6	0.56	0.23	7.46	16.96	28,800	2.40	96.0	26.82
South Africa	8	676.6	0.51	2.58	1.3	0.03	0.03	3.46	17.44	888	0.23	22.3	21.14
Finland	13	2155.6	0.001	0.50	0.1	0.14	0.01	0.02	10.88	305	3.07	11.3	13.97
Argentina	5	2088.8	0.11	0.09	16.6	0.43	0.06	2.36	1.83	34,634	8.96	122.3	13.15
Portugal	3	214.7	1.05	3.84	54.0	1.15	0.03	2.25	8.25	11,597	2.46	6.7	12.97
Sweden	13	5590.8	0.04	0.16	2.5	0.02	0.01	2.36	9.08	13,906	1.36	71.9	12.80
Spain	6	271.0	1.06	2.17	32.3	0.74	0.62	2.87	5.89	8761	2.02	168.0	10.77
Brazil	13	143.6	1.61	5.69	18.4	0.11	0.07	2.31	8.18	2646	0.16	10.7	10.64
Ireland	14	115.1	1.37	6.38	7.4	0.04	–	1.58	7.34	851	0.04	–	8.96
Greenland	7	1095.0	0.07	0.74	0.1	<0.001	–	0.73	8.09	79	0.0015	–	8.81
Tajikistan	1	961.0	0.50	0.40	52.0	–	–	4.81	3.84	49,972	–	–	8.65
Algeria	7	145.4	1.60	4.16	7.5	0.08	0.07	2.33	6.05	1093	0.12	10.7	8.50
Georgia	11	127.2	0.65	3.65	8.2	1.66	1.40	0.83	4.64	1048	2.12	178.4	7.59
Dem. Rep. Congo	2	31.3	–	16.65	–	1.81	–	–	5.21	–	0.57	–	5.77
Azerbaijan	1	95.0	1.43	3.63	44.2	0.59	–	1.36	3.45	4199	0.56	–	5.37
Namibia	8	59.9	1.97	6.16	17.3	0.13	–	1.18	3.69	1033	0.08	–	4.95
Eritrea	7	134.5	–	2.12	18.4	1.00	0.46	–	2.85	2474	1.34	62.1	4.20
Indonesia	3	25.1	6.00	10.16	4.3	–	–	1.51	2.55	109	–	–	4.06
Morocco	3	38.7	3.96	4.36	72.9	0.25	0.36	1.53	1.69	2824	0.10	13.8	3.32
Bolivia	10	542.3	0.15	0.34	35.3	0.02	–	0.80	1.86	19,123	0.11	–	2.77
Pakistan	2	24.4	2.59	8.07	11.3	–	–	0.63	1.97	277	–	–	2.60
Saudi Arabia	4	58.2	0.03	3.53	13.2	0.55	0.27	0.02	2.05	768	0.32	16.0	2.39
Greece	2	22.8	4.52	5.88	132.6	–	7.68	1.03	1.34	3018	–	174.8	2.37
Sudan	2	120.7	–	0.79	2.1	1.10	1.18	–	0.96	259	1.33	142.1	2.28
International Waters	1	89.5	–	2.06	38.4	0.45	–	–	1.84	3437	0.40	–	2.25
Armenia	5	50.8	0.88	2.08	45.6	0.97	1.64	0.45	1.06	2319	0.49	83.5	1.99
Turkey	6	56.4	0.72	1.87	21.6	0.85	0.75	0.40	1.06	1215	0.48	42.3	1.94
Kyrgyzstan	2	19.4	3.72	3.95	16.0	1.17	1.77	0.72	0.77	312	0.23	34.4	1.72
Uzbekistan	1	14.4	3.50	7.24	134.0	0.86	0.38	0.50	1.04	1931	0.12	5.5	1.67
Poland	1	21.2	1.54	5.88	–	–	–	0.33	1.25	–	–	–	1.57
Chile	5	60.6	0.32	2.03	31.3	0.03	1.35	0.19	1.23	1898	0.02	81.7	1.44
Cuba	3	27.3	1.54	1.33	18.9	2.38	–	0.42	0.36	516	0.65	–	1.43
Yemen	2	12.7	1.22	9.04	68.9	–	–	0.16	1.15	876	–	–	1.31
Burkina Faso	1	13.0	0.14	9.82	56.3	–	–	0.02	1.27	730	–	–	1.29
Romania	2	29.3	1.47	2.10	31.9	0.68	1.63	0.43	0.61	935	0.20	47.9	1.24
Serbia	3	18.2	2.17	3.32	46.4	0.60	0.77	0.39	0.60	845	0.11	14.0	1.11
Guatemala	2	33.9	0.77	1.56	373.0	0.42	–	0.26	0.53	12,635	0.14	–	0.93
Dominican Rep.	4	69.0	–	0.52	7.5	0.63	1.34	–	0.36	516	0.44	92.5	0.80
Tunisia	6	17.2	1.70	2.78	3.8	–	–	0.29	0.48	65	–	–	0.77
Botswana	2	25.3	1.04	1.77	3.9	–	–	0.26	0.45	100	–	–	0.71
Wales	1	25.8	0.57	1.10	13.3	0.85	0.15	0.15	0.28	344	0.22	3.9	0.65
South Korea	1	7.2	3.45	4.80	–	–	–	0.25	0.35	–	–	–	0.59
Honduras	1	6.4	2.27	5.57	62.7	–	–	0.15	0.36	403	–	–	0.50
PNG	3	41.2	0.002	0.13	3.8	1.09	0.65	0.001	0.05	158	0.45	26.6	0.50
Ecuador	1	10.0	0.23	2.72	50.8	2.03	2.62	0.02	0.27	506	0.20	26.1	0.50
Montenegro	3	9.2	1.24	3.78	2.9	0.29	–	0.11	0.35	27	0.03	–	0.49
Nepal	2	2.4	3.01	14.68	23.5	–	–	0.07	0.35	56	–	–	0.42
Zambia	2	1.7	4.39	13.41	–	–	–	0.07	0.23	–	–	–	0.30
Germany	2	43.8	–	0.67	–	–	–	–	0.29	–	–	–	0.29
France	2	4.1	3.47	3.54	0.3	–	–	0.14	0.14	1	–	–	0.28
Philippines	3	8.8	0.32	1.95	25.6	0.87	1.53	0.03	0.17	226	0.08	13.5	0.28
Laos	2	3.0	1.41	7.74	–	–	–	0.04	0.23	–	–	–	0.27
Columbia	1	8.7	–	2.81	3.4	–	–	–	0.24	29	–	–	0.24
Egypt	2	4.4	0.44	4.51	19.3	0.51	1.15	0.02	0.20	85	0.02	5.1	0.24
Tanzania	1	6.1	–	1.57	17.5	1.25	1.30	–	0.09	106	0.08	7.9	0.17
Thailand	1	1.6	–	10.30	–	–	–	–	0.17	–	–	–	0.17
Kenya	1	1.6	–	4.50	33.8	2.08	0.56	–	0.07	53	0.03	0.9	0.10
UK	1	3.5	–	0.29	–	0.45	–	–	0.01	–	0.02	–	0.03
World total	**851**	**51,030.0**	**0.44**	**1.20**	**16.4**	**0.28**	**0.12**	**226.1**	**610.3**	**837,128**	**144.41**	**6,132**	**980.8**

Notes: No. – number of deposits.

excluded Sc (and thorium) due to the lack of markets, it noted that future production should be considered if significant demands arise – especially since Sc uptake and use have been hampered by low supply reliability (pages 13 or 211; Puritch et al., 2014). We have therefore decided to keep a value for Sc but recognise that this is contentious – although in reality, even if Buckton was removed as a Zn resource from

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Table 5
Top 25 Pb-Zn-Cu mineral resources by contained base metals.

Deposit/Project	Deposit type	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Other	Mt Pb	Mt Zn	t Ag	Mt Cu	t Au	Pb+Zn+Cu Mt	Pb+Zn+Cu %	Past production (to 2014)
Antamina, Peru	Skarn	2121		0.65	10.3	0.83		0.017% Mo	–	13.88	21,840	17.69	–	31.57	1.49	0.04–3.47–4.71 Mt Pb-Zn-Cu, 4189 t Ag, 55.3 kt Mo
McArthur River, Australia	SH Pb-Zn	194	4.0	9.2	41.0				7.86	17.86	7948	–	–	25.72	13.26	0.68–2.96 Mt Pb-Zn, 809 t Ag
Mt Isa (open cut), Australia	SH Pb-Zn	427.8	2.5	3.4	51.7				10.69	14.43	22,107	–	–	25.12	5.87	8.81–9.81 Mt Pb-Zn, 20,897 t Ag
Kholodninskoe, Russia	SH Pb-Zn	519	0.65	4.08	28			In-Cd-Se-Te	3.36	21.20	14,532	–	–	24.56	4.73	–
Gorevskoe, Russia	SH Pb-Zn	300	6.5	1.4					19.5	4.2	–	–	–	23.7	7.9	–
Mehdiabad, Iran	SH Pb-Zn	394	1.6	4.2	36				6.30	16.55	14,184	–	–	22.85	5.80	–
Central Region, Kazakhstan[a]	Porphyry & VMS	5397		0.01	1.39	0.40	0.08	0.01% Mo	–	0.46	7526	21.67	440.0	22.13	0.41	Insufficient data
La Granja, Peru	Porphyry	3600		0.09[b]		0.51			–	3.24	–	18.36	–	21.60	0.60	–
Mt Isa-George Fisher North, Australia	SH Pb-Zn	145.1	3.9	8.3	62				5.70	12.11	9067	–	–	17.81	12.27	(See Mt Isa open cut above)
Rampura Agucha, India	SH Pb-Zn	109.4	2.0	12.8	59				2.19	13.99	6,467	–	–	16.17	14.78	≫0.5–5.5 Mt Pb-Zn[c]
Black Mountain-Gamsberg, South Africa	SH Pb-Zn	214.3	0.44	6.67					0.95	14.29	-	–	–	15.24	7.11	≫0.6–0.4–0.05 Mt Pb-Zn-Cu[d]
Goemdok (Gumdock), North Korea	SH Pb-Zn	266	1.88	4.21					2.34	11.2	-	–	–	13.54	5.09	Insufficient data
Hysean, North Korea	Porphyry	160	3.2	3.6	180	1.5	0.6		5.12	5.76	28,800	2.4	96.0	13.28	8.30	Insufficient data
Talvivaara, Finland	SH Pb-Zn/VMS	2053		0.50		0.13		0.22% Ni, 0.02% Co, 0.0017% U_3O_8	–	10.27	–	2.67	–	12.94	0.63	~49 kt Ni, 104 kt Zn, 1.1 kt Co[e]
Red Dog, USA	SH Pb-Zn	53.1	4.5	17.2	81.7				2.39	9.12	4337	–	–	11.51	21.67	2.13–11.42 Mt Pb-Zn, 4125 t Ag
Neves-Corvo, Portugal	VMS	193.32	0.9	3.8	53	1.2			1.79	7.30	10,306	2.41	–	11.50	5.95	4.7–217–747 kt Pb-Zn-Cu, 286 t Ag[f]
Shalkiya, Kazakhstan	SH Pb-Zn	274	0.8	3.1					2.09	8.47	-	–	–	10.56	3.85	–
Ozernoe (Ozerny), Russia	SH Pb-Zn	135.1	1.16	6.12	36.7			0.016% Cd	1.57	8.27	4960	–	–	9.83	7.28	–
Peñasquito (mill), Mexico	Skarn	851.4	0.30	0.83	30.7		0.48		2.59	7.04	26,098	–	411.5	9.63	1.13	Insufficient data
Los Azules, Argentina	Porphyry	1786	0.01	0.02	1.9	0.50	0.06		0.18	0.40	3355	–	111.1	9.45	0.53	–
Howard's Pass-Don Group, Canada	SH Pb-Zn	130.2	1.69	5.29					2.19	6.88	-	–	–	9.07	6.97	–
Gai, Russia	VMS	380	0.06	0.74	6.3	1.57	0.9		0.23	2.81	2394	5.97	342.0	9.01	2.37	–
Dugald River, Australia	SH Pb-Zn	63	1.8	12.1	32				1.14	7.62	1986	–	–	8.76	13.90	–
Konimansur Kalon, Tajikistan	Epithermal	961	0.5	0.4	52				4.81	3.84	49,972	–	–	8.65	0.9	–
Cerro de Pasco, Peru	Epithermal	203.1	1.06	2.55	100.2	0.21			2.15	5.18	20,356	0.44	–	7.76	3.82	Insufficient data

[a] Formerly Karaganda.
[b] Rio Tinto no longer report Zn grades for La Granja but it is included in this paper as it remains a globally significant Zn resource and may in future be economically viable (%Zn was reported up until 2010, after which the development excluded Zn recovery).
[c] Production data 2006–2014 only.
[d] Production data 2001–2014 only.
[e] Production data 2009–2013 only.
[f] Production data 2006-2014 only (excludes prior Cu-Sn mining).

our database, it would not materially affect any of the other results, findings or implications from this study.

The average unit value per tonne of ore by deposit type is shown in Table 7, including the split between primary metals (Pb-Zn-Ag-Cu-Au) and possible companion metals (e.g. indium, Fe, Ni, W, Mo, etc.). In general, the dominant value is from primary metals, except for unconventional deposit types (e.g. shale-hosted) where companion metals are the dominant value.

The top 15 projects by unit ore value are shown in Table 8, showing the exceptional value of Virginia Silver at US$1984/t ore (due to its rich Ag grade), Buckton-Buckton South at US$1474/t ore, Val at US$1293/t ore and Caledonia at US$1042/t ore – although only Buckton-Buckton South is of major size (5 Gt) with the others being very small deposits (<1 Mt). The remaining deposits are valuable due to their high base metal and precious metal ore grades, although it is worth noting that almost all of them have not yet been developed into operating projects (probably due to their typically small size or, in the case of Buckton, low grades and complex metallurgy). Finally, the top 15 deposits by total economic value are shown in Table 9, with Buckton being by far the most valuable deposit at a staggering US$7,384 billion, due almost entirely to Sc contained within this deposit (although the price of Sc would undoubtedly be reduced in the event of significant production from this very large deposit, as discussed above). In comparison, the next two deposits, Kvanefjeld and Central Region, are worth US$230 and US$189 billion, respectively. As mentioned above, the value for the top two of these deposits is driven by a rare metal (or group) – Sc for Buckton or REEs for Kvanefjeld. Buckton is truly a giant shale-hosted deposit at 5 Gt of mineral resources, and although it has very low grades (e.g. 0.017% Zn, 0.007% Ni[5]), it does have an unusually wide array of

[5] These Zn and Ni grades are only marginally above crustal geochemical abundance of 67 and 47 mg/kg, respectively, for Zn and Ni (Rudnick and Gao, 2014).

Table 6
Top 25 Pb-Zn-Cu mineral resources by combined base metal ore grades.

Deposit/Project	Deposit Type	Mt	%Pb	%Zn	g/t Ag	Other	Mt Pb	Mt Zn	t Ag	Other	Pb + Zn + Cu	
											Mt	%
Holliday, Canada	Epithermal?	0.036	14.95	20.78	427.2		0.005	0.008	15.5	–	0.01	35.73
Angouran, Iran	Sed.-hosted Pb-Zn	22.23	4.0	30.3	90		0.88	6.74	2,005	–	7.62	34.26
Val, Canada	Sediment-hosted U	0.020	26.7	7.3	1,027		0.005	0.001	20.5	–	0.01	34.00
Flinders Group, Australia	Sed.-hosted Pb-Zn	0.694	1.4	30.2			0.010	0.21	–	–	0.22	31.60
Jabali (stockpile), Yemen	Sed.-hosted Pb-Zn	0.12	3.5	24	160		0.004	0.029	19.2	–	0.03	27.50
Red Bird, Canada	Sed.-hosted Pb-Zn	2.177	6.5	18.5	68.5		0.14	0.40	149.1	–	0.54	25.00
Hudson Bay Mountain-Silver Lake 2, Canada	Epithermal	0.03	6.7	17.7	449.13		0.002	0.005	13.5	–	0.01	24.40
Prairie Creek, Canada	Sed.-hosted Pb-Zn	11.670	10.9	12.8	197	0.45% Cu	1.27	1.49	2297	52 kt Cu	2.82	24.14
Vazante	Sed.-hosted Pb-Zn	18.8	–	23.7	–		–	4.46	–	–	4.46	23.70
Dairi-Anjing Hitam, Indonesia	Sed.-hosted Pb-Zn	8.11	9.1	14.6	12		0.74	1.18	97.3	–	1.92	23.70
Sullivan, Canada[a]	Sed.-hosted Pb-Zn	12.80[a]	11.2[a]	12.0[a]	129[a]		1.43[a]	1.54[a]	1647[a]	–	2.97[a]	23.25[a]
Shaimerden (stockpile), Kazakhstan	VMS?	2.0		22			–	0.44	–	–	0.44	22.00
Kherzet Youcef, Algeria	Sed.-hosted Pb-Zn	1.6	3.6	18.4			0.058	0.29	–	–	0.35	22.00
Craig, Canada	Sed.-hosted Pb-Zn	0.875	13.5	8.5	123		0.12	0.074	107.6	–	0.19	22.00
Lady Loretta, Australia	Sed.-hosted Pb-Zn	14.27	5.7	16.1	97		0.82	2.29	1391	–	3.11	21.80
Red Dog, USA	Sed.-hosted Pb-Zn	53.1	4.5	17.2	81.7		2.39	9.12	4337	–	11.51	21.67
Menghu, China	Sed.-hosted Pb-Zn	0.233	13.8	7.8			0.032	0.018	-	–	0.05	21.60
Kipushi, DRC	Sed.-hosted Pb-Zn	25.975		19.04		2.18% Cu	–	4.95	–	567 kt Cu	5.51	21.23
Kootenay King, Canada	Sed.-hosted Pb-Zn	0.01	5.36	15.6	66.5		0.001	0.002	0.7	-	0.002	20.96
Mengya, China	Skarn	2.0	10	10	-	0.5% Cu	0.2	0.2	–	10 kt Cu	0.41	20.50
Ganesh Himal-Suple, Nepal	Sed.-hosted Pb-Zn	1.1	4.05	16.25	19.43		0.045	0.18	21.4	–	0.22	20.30
Platosa, Mexico	Manto/Skarn/Epithermal?	0.706	9.18	10.66	854		0.065	0.075	602.6	–	0.14	19.84
Polaris, Canada[a]	Sed.-hosted Pb-Zn	5.9[a]	4.0[a]	15.6[a]			0.24[a]	0.92[a]	-	–	1.16[a]	19.64[a]
Stemwinder, Canada	Sed.-hosted Pb-Zn	0.025	3.7	15.6	76.3		0.001	0.004	1.9	–	0.00	19.30
Silver King, Australia	Sed.-hosted Pb-Zn	1.6	13.9	5.1	157		0.22	0.082	251.2	–	0.30	19.00

[a] Estimated from total mineral resources minus cumulative production.

metals – but especially high value Sc. Even if a lower Sc price of US$7/g is used, this still leaves the deposit valued at ~US$450 billion and double the value of the Kvanefjeld deposit. For other deposits, significant value can be attributed to Ni, U, vanadium (V) or molybdenum (Mo). This highlights the potential opportunities from considering all possible metals in a deposit rather than the traditional metals, as when present, they can add substantial economic value to a deposit.

In addition, Pb-Zn deposits are often associated with a wide variety of metal co/by-products, although these co/by-products may not be reported and are generally only of interest to smelters or refineries and not the mines themselves since they receive little to no revenue for these metals. The extent of reported data for these other metals is compiled in Table 10 – showing the paucity of companion metals data reported in resource estimates. In further analysing the value split between metals, the following stands out:

- 521 deposits have Pb-Zn-Ag >70% of economic value;
- 398 deposits have Zn as >40% of economic value;
- 304 deposits have Cu-Au as >30% of economic value;
- 130 deposits have Au as >20% of economic value;
- 29 deposits have all other metals as >30% of economic value.

This demonstrates that the primary focus of the economics of these projects remains on the key Pb-Zn-Ag-Cu-Au commodities, with low potential recognised for adding value from the full spectrum of companion metals which could be present in a given deposit.

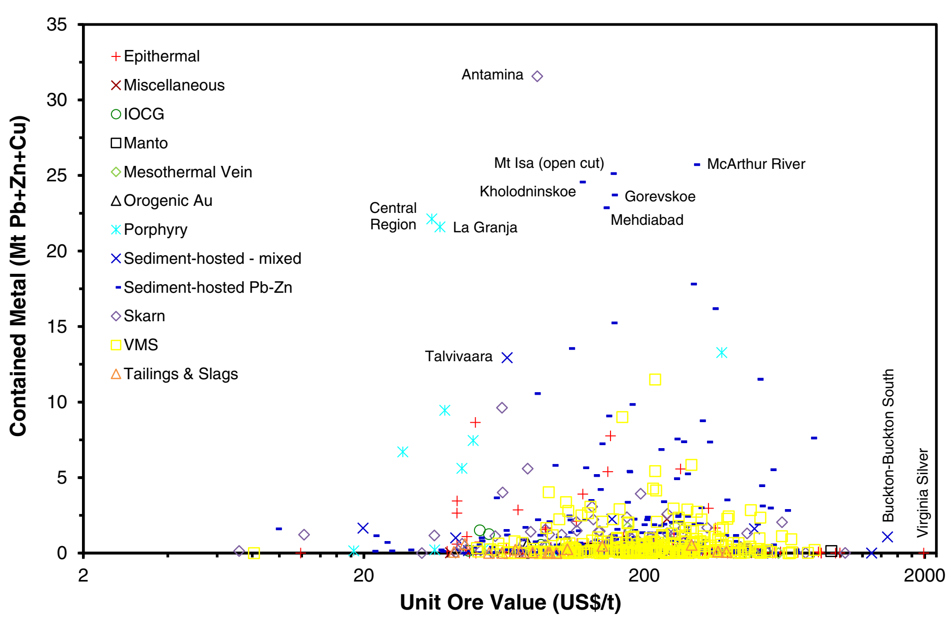

Fig. 11. Contained metals versus unit ore value by deposit types.

Table 7
Average Pb-Zn-Cu mineral resources by deposit type and unit value (US$/t ore).

Deposit type	US$/t ore		
	Pb-Zn-Ag-Cu-Au only	Other Metals	Total
Sediment-hosted Pb-Zn	149.45	1.16	150.61
Volcanogenic Massive Sulfide	183.96	0.68	184.64
Skarn	79.41	7.74	87.15
Porphyry	37.75	2.88	40.63
Epithermal	97.53	0.18	97.71
Sediment-hosted mixed[a]	5.06[a]	606.71[a]	611.77[a]
Mesothermal vein	231.29	2.19	233.48
Iron oxide Cu-Au	64.06	14.21	78.27
Miscellaneous[a]	5.76[a]	232.87[a]	238.62[a]
Mine wastes	78.99	0.10	79.09
Manto	177.86	–	177.86
Orogenic Au	129.39	–	129.39
Overall average	68.97	156.22	225.19

[a] The unit value for these deposit types are often dominated by other metals (e.g. Sc, Mo, Ni, U, etc., such as Viken, Buckton, and others; see Supplementary Information for individual deposit details).

4. Discussion and implications

Our methodology and results lead to a numbering of interesting questions and issues, revolving around the accuracy of the resource data, the role of continuing mineral exploration, trends in ore grades and deposit types, the increasingly refractory nature of Pb-Zn ores, the future role of additional value from the wide spectrum of possible companion metals, the potential for tailings to be a future mineral resource and last but not least the environmental issues facing the Pb-Zn mining sector.

4.1. Size of the remaining Pb-Zn resource endowment?

Our compiled mineral resources data, 51,030 Mt at 0.44–1.20% Pb-Zn for 226.1–610.3 Mt Pb-Zn, significantly exceeds the 2014 USGS world reserves estimate of 87–230 Mt Pb-Zn (Table 1). In addition, our data only has 4.7–11.4 Mt Pb-Zn in China compared to China's 2011 national estimate of 55.1–116.0 Mt Pb-Zn, meaning an additional 50.4–104.6 Mt Pb-Zn in China alone. These data also exclude (not by choice!) numerous deposits for which we have not been able to find reliable resource data in the publicly available literature, a problem that is exacerbated in countries that do not practice public reporting through formal codes. This means that our estimate is clearly an underestimate, but by how much is difficult to ascertain, of course, without knowing the data for these other deposits.

A critical case study is Canada – which for 2010 reported national reserves of 0.4–4.13 Mt Pb-Zn (see Table 1), compared to our compiled data which has 22.3–76.8 Mt Pb-Zn in mineral resources. Despite Canada's reserves appearing to diminish significantly in recent years, there are clearly known mineral resources which have the potential to be upgraded to reserves status and potentially mined, depending on the usual factors which must be considered by the mining industry in doing so.

As noted earlier, it is critical to distinguish between the formal reporting of ore reserves and mineral resources. In the mining industry, longer term mine planning is based on mineral resources, and so with our data it is possible to make some basic future production projections and test whether our resources can meet such expectations. Here, we use linear regression models based on historical production as these are reasonably conservative and are not likely to over-estimate future production growth; further exploration of other regression options are discussed in the supplementary information. The linear regressions are derived from production from 1885 to 2012 for Pb and 1932 to 2012 for Zn. Estimated Pb-Zn production in 2050 and 2100 and correlation coefficients are given in Table 11, with Pb-Zn production in Fig. 12. By 2050, our Pb-Zn resources data are capable of meeting this cumulative demand – but only just for both Pb and Zn (albeit excluding the additional Pb-Zn resources reported by China but including Chinese production, as discussed above).

4.2. Exploration results – Past 25 years and limited success?

The continued discovery of new mineral deposits is fundamental to the ongoing ability of the mining industry to continue to increase production around the world. New discoveries can come from extensions to operating mines (aka 'brownfields' exploration) or in new areas which have not been mined before (or 'greenfields'). In the mineral exploration sector, there appears to have been very little success in discovering any large greenfields Pb-Zn deposits for about 25 years. For example, the discovery years (mostly from Laznicka, 2014) for some of the various large projects in production in 2015 included (production data from relevant company):

- Rampura Agucha (India) – discovered in 1977, 2013 production 57.0–652.7 kt Pb-Zn;
- Antamina (Peru) – long-known for Cu-Pb-Ag, modern exploration proved a significant orebody by the 1960–70s and later a giant resource in the late 1990s (e.g. Petersen, 1965; Redwood, 2004), 2013 production 3.8–219.5–427.3 kt Pb-Zn-Cu;
- Red Dog (USA) – discovered in 1968, 2013 production 96.7-551.3 kt Pb-Zn;

Table 8
Top 15 Pb-Zn-Cu mineral resources by unit ore value.

Deposit/Project	Mt	US$/t ore	Proportional value (%)					
			Pb	Zn	Ag	Cu	Au	Others
Virginia Silver, Canada	0.02	1983.9	4.20	2.41	90.94	0	2.45	–
Buckton-Buckton South, Canada	5008.7	1474.2	0	0.03	0	0.02	0	0.08% Ni, 0.03% Co, 0.03% Mo, 0.05% U_3O_8, 0.30% V, 0.35% TREO + Y, 99.08% Sc, 0.03% Li
Val, Canada	0.020	1293.4	39.14	12.26	48.61	0	0	–
Caledonia, Canada	0.068	1041.6	1.09	15.53	41.37	40.67	1.33	–
Holliday, Canada	0.036	996.1	28.46	45.30	26.24	0	0	–
Peak, Canada	0.003	962.2	27.59	0	26.14	36.09	10.19	–
Platosa, Mexico	0.706	927.9	18.77	24.95	56.29	0	0	–
Hudson Bay Mountain-Silver Lake 2, Canada	0.03	856.0	14.84	44.90	32.10	0	8.16	–
Kokanee (Scranton/Sunset), Canada	0.018	854.2	18.20	20.34	17.19	0	44.27	–
Tambomayo, Peru	1.6	835.4	4.77	7.80	20.96	0	66.48	–
Solwara 1, Papua New Guinea	2.57	809.2	0	1.88	2.24	66.42	29.47	–
Angouran, Iran	22.23	788.4	9.50	83.50	7.00	0	0	–
Solwara 12, Papua New Guinea	0.23	766.4	0	10.20	4.47	66.15	19.18	–
Tintina (Eagle), Canada	0.091	751.6	15.14	28.89	55.97	0	0	–
Topia, Mexico	0.429	706.4	16.06	12.41	71.52	0	0	–

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Table 9
Top 15 Pb-Zn-Cu mineral resources by total value (US$billion).

Deposit/Project	Mt	US$billion	Proportional value (%)					Others
			Pb	Zn	Ag	Cu	Au	
Buckton-Buckton South, Canada	5008.7	7384.0	0	0.03	0	0.02	0	0.08% Ni, 0.03% Co, 0.03% Mo, 0.05% U_3O_8, 0.30% V, 0.35% TREO + Y, 99.08% Sc, 0.03% Li
Kvanefjeld, Greenland	956	230.4	0	2.12	0	0	0	8.24% U_3O_8, 89.64% TREO + Y
Central Region, Kazakhstan[a]	5396.8	188.6	0	0.53	2.44	79.80	9.53	7.70% Mo
Antamina, Peru	2121	176.3	0	17.10	7.58	69.68	0	5.65% Mo
La Granja, Peru	3600	134.5	0	5.23	0	94.77	0	–
Talvivaara (Kolmisoppi/Kuusilampi)	2052.8	133.1	0	16.75	0	13.92	0	57.91% Ni, 9.52% Co, 1.91% U_3O_8
Häggån, Sweden	2350	100.0	0	2.20	0	0	0	12.53% Ni, 13.09% Mo, 26.51% U_3O_8, 45.67% V
Jiama, China	1449.0	71.1	1.87	1.29	6.22	61.81	8.43	20.36% Mo
Los Azules, Argentina	1786	69.4	0.49	1.24	2.96	88.79	6.53	–
Mt Isa (open cut), Australia	427.8	65.1	31.12	48.11	20.77	0	0	–
Gai, Russia	380	63.4	0.68	9.63	2.31	65.35	22.03	–
Kholodninskoe, Russia	519	61.3	10.39	75.10	14.51	0	0	–
Viken, Sweden	3062	60.8	0	4.59	0	4.19	0	28.86% Ni, 62.36% U_3O_8
Hysean, North Korea	160	60.4	16.07	20.70	29.16	27.58	6.49	–
McArthur River, Australia	194	58.5	25.44	66.25	8.31	0	0	–

[a] Formerly Karaganda.

- Mt Isa Group (Australia) – discovered in 1923, 2013 production 167.8–405.1 kt Pb-Zn;
- Century (Australia) – discovered in 1990, 2013 production 54.2–488.2 kt Pb-Zn (although Century closed in late 2015);
- Cannington (Australia) – discovered in 1990 (Bailey, 1998), 2013 production 209.1–63.4 kt Pb-Zn;
- McArthur River (Australia) – discovered in 1955, 2013 production 45.8–203.3 kt Pb-Zn;
- Garpenberg (Sweden) – discovered ~1200 CE (i.e. almost a millennia of mining), 2013 production 70–162 kt Pb-Zn.

The relative lack of success in the discovery of new Pb-Zn deposits (i.e. deposits with Pb and/or Zn as the main focus rather than as co- or by-products) is the result of two linked factors; namely the low prices of these metals and a lack of exploration for these type of deposits, probably as a result of the poor economics of these metals. This is exacerbated by the fact that the top six Zn mines (outside of China) alone produce some 2.5 Mt of Zn per year out of an annual production of ~13 Mt Zn/year (OCE, 2014). If these mines were to close or to otherwise be affected (e.g. in a more major sense than the temporary closure of the Bingham Canyon mine in April 2013 had on the Cu market), this could have a significant effect on the global Zn market. This scenario feeds the perceptions of a Zn market crunch, where an excess of demand combines with insufficient supply to drive Zn prices up. Mining companies are starting to respond to this, with an increase in Zn exploration expenditure recorded in Australia at least in 2015 (OCE, 2015), albeit after year-on-year decreases in Zn exploration expenditure between 2011 and 2014 (AME, 2015).

In early 2016, exploration company Rox Resources Ltd announced the discovery and maiden mineral resource for the Teena deposit, near the McArthur River mine – at 58 Mt grading 1.6% Pb–11.1% Zn (see supplementary data), this makes it of global significance and could perhaps be seen as the first major such discovery in recent years.

4.3. Declining ore grades and more refractory ores

The past century (or even longer) has been characterized by a gradual decline in the grades of Pb-Zn ores being processed, as exemplified by Australia (Mudd, 2007, 2009b) and by Pb in Canada (Mudd, 2009a). Mining statistical data published in CDBS (var.a,b) and GNHSC (var.) indicate that Canadian Pb-Zn mining in the early 20th century focused on ore grades much higher than those prevailing in the latter part of this century (e.g. Sullivan, as discussed earlier). The Canadian Pb-Zn mining data used here has been partially updated from Mudd (2009a), and represents the majority of Canada's historical Pb-Zn mining. The trends in ore grades for both Australia and Canada are shown in Fig. 13, showing the rich grades from early mining in the late 1800s often associated with small projects that exploited rich veins or oxide ores, followed by a long, gradual decline in all three metals throughout the 20th century. The especially low Canadian Pb ore grades in the most recent decades (reaching just 0.04% Pb in 2014) are a reflection of the closure of mines (closure year in brackets) with reasonable Pb ore

Table 10
Extent of reported data for other metals by deposit type (based on reported ore grades).

Deposit type	No. deps	Pb	Zn	Ag	Cu	Au	Ba	Ni	Co	Sn	In	Mo	Fe	As	V	Other
Sediment-hosted Pb-Zn	276	250	270	135	41	17	8	–	1	–	2	–	1	–	1	Cd 4, Ga 1, CaF_2 4
Volcanogenic Massive Sulfide	270	156	269	220	241	187	–	7	10	-	-	1	-	3	-	Cd 1, Pt 1
Skarn	92	73	91	76	48	30	–	–	2	6	6	2	4	–	1	Ga 1, W 1, Cd 2, Mn 1, Bi 1, TiO_2 1
Porphyry	16	12	15	14	11	11	–	–	–	1	1	5	–	1	-	W 1, Bi 2
Epithermal	138	130	125	126	49	69	1	–	–	–	2	1	–	–	-	Cd 2, Hg 1, Sb 1, CaF_2 1, Sr 1, Bi 1, Te 1, Se 1
Sediment-hosted mixed	9	3	9	4	6	1	–	7	3	–	–	3	–	–	3	REEs 2, U 5, Li 1, Re 1, P_2O_5 1, CaF_2 1, Sc 1
Mesothermal vein	13	12	12	11	1	5	–	–	–	–	–	–	–	–	–	Ge 1
Iron oxide Cu-Au	6	3	6	4	6	3	–	1	2	–	–	2	–	–	–	Re 2
Miscellaneous	4	2	2	1	2	–	–	1	1	1	–	–	–	–	–	REEs 1, U 1
Mine wastes	15	12	15	12	8	6	–	–	–	–	1	–	–	–	–	–
Manto	4	4	4	4	1	1	–	–	–	–	–	–	–	–	–	–
Orogenic Au	8	6	8	8	6	7	–	–	–	–	–	–	–	–	–	–
Totals	***851***	***663***	***826***	***615***	***420***	***337***	***9***	***16***	***19***	***8***	***12***	***14***	***5***	***4***	***5***	Various

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Table 11
Linear regressions of future world Pb-Zn production.

	Pb	Zn
Years based on	1885–2012	1932–2012
Correlation coefficient	0.884	0.935
Production in 2050	5.02 Mt Pb	15.5 Mt Zn
Production in 2100	6.45 Mt Pb	22.0 Mt Zn
Cumulative production 2013–2050	170.6 Mt Pb	499.9 Mt Zn
Cumulative production 2013–2100	458.0 Mt Pb	1,442.4 Mt Zn
Global resources this study	226.1 Mt Pb	610.3 Mt Zn

grades such as Pine Point (1988), Faro (1997), Sullivan (2001), Polaris (2002), Baffin Island-Nanisivik (2002), and Brunswick (2013), as well as a change to Cu-Zn-dominant deposits that have either very low grade Pb or do not report Pb, as exemplified by VMS deposits such as the Hudson Bay field, Kidd Creek, Langlois and Myra Falls, among others. In contrast, Australia's Pb ore grades, although declining steadily, are still reasonable due to the dominance of sediment-hosted deposits such as Mt Isa, Cannington and Broken Hill. In 2014, average Canadian ore grades were about 0.04% Pb, 3.61% Zn and 31.7 g/t Ag, which compares to Canada's average resource grades (see Table 4) of 0.30% Pb, 1.01% Zn and 8.0 g/t Ag, suggesting that current data for Pb in ore milled is under-reported, meaning current Pb ore grades are probably similar to this grade. Overall, in comparing current ore processing grades to our resources data suggests that long-term ore grades are inevitably going to continue to decline.

For Australia, long-term data on reserves and/or resources are available for most of the major fields and mines, allowing over a century's view of average ore grades in reported reserves-resources (Fig. 14). As can be observed, the degree of decline in Pb-Zn-Ag ore grades is very significant – and based on current reported mineral resources in Australia, it is highly unlikely that ore grades could ever increase. For example, even though Cannington was a significant high grade discovery, it has not helped Australia return average ore grades seen up to the 1960s (that is, from the late 1990s onwards Ag grades still ranged from ~90 down to ~45 g/t Ag, compared to grades >150 g/t Ag up to ~1972).

A potential outcome of declining ore grades is that recovery rates during milling could decline, since lower ore grades are often associated with poorer recoveries (e.g. U, Mudd, 2014). This depends on mill configuration (especially differential flotation) and production costs versus market prices and demand – that is, it can sometimes be more profitable for a mine to lower operating costs by reducing recovery but they can still return a profit given prevailing prices. It is commonly possible to operate a mill at higher recovery but this entails higher costs and is not always economic. Based on studies of various base metal mining sectors across the world (e.g. Mudd, 2007, 2009a, 2010b; Mudd and

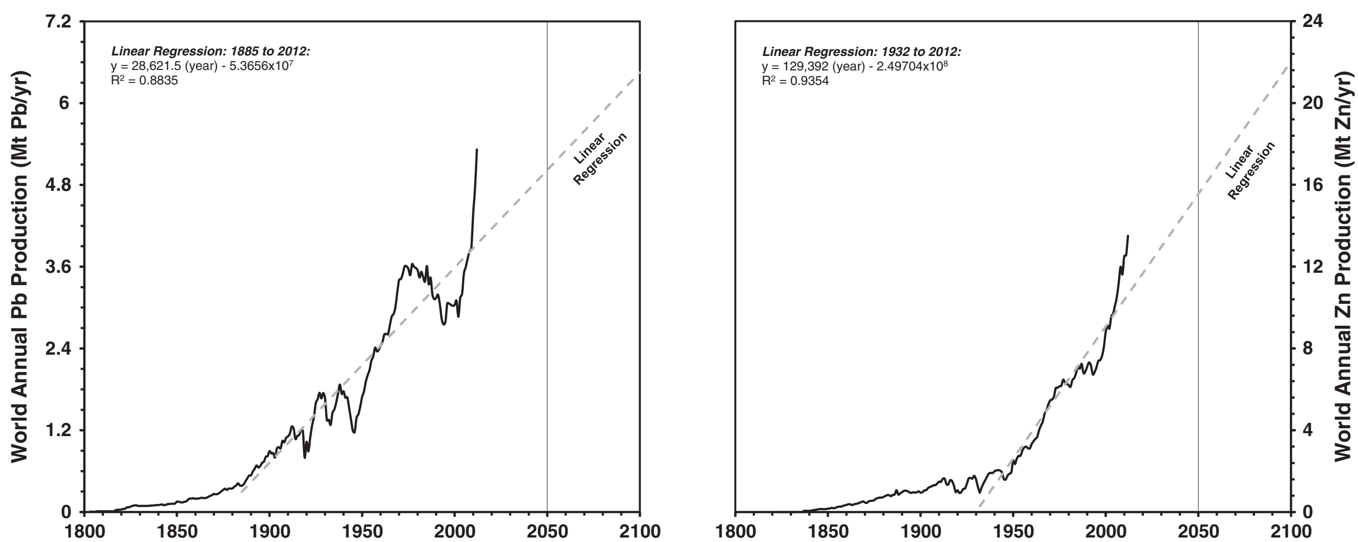

Fig. 12. Annual world production of Pb (left) and Zn (right) and linear regressions of future production (years 1885 to 2012 for Pb, years 1932-2012 for Zn; see Supplementary Information for more discussion).

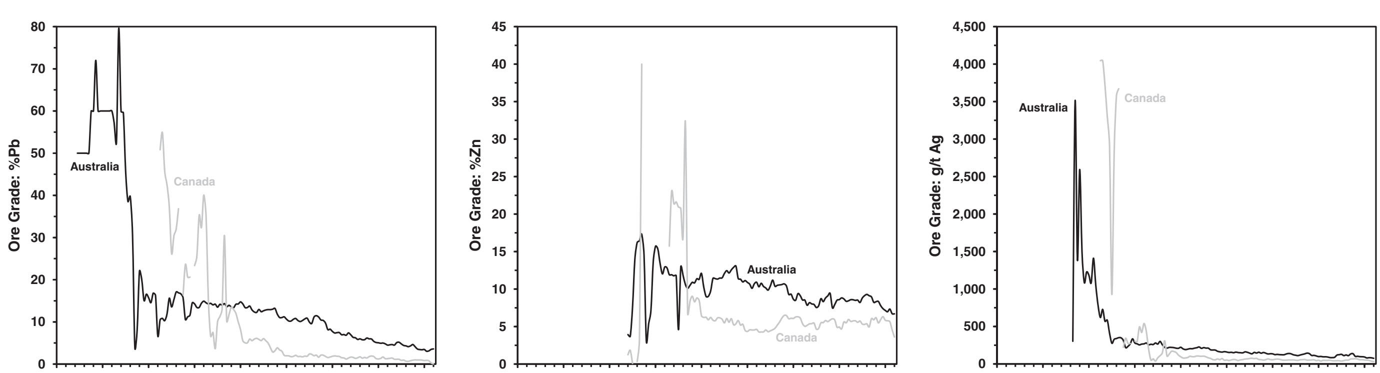

Fig. 13. Long-term historical trends in Pb-Zn-Ag grades of ore milled for Australia and Canada (data updated from Mudd, 2007, 2009a, 2009b, 2010a) (note: prior to 1880 in Australia, the data is mostly from selective hand-sorting of ore at the Northampton field in Western Australia, hence the approximate Pb concentrate grades).

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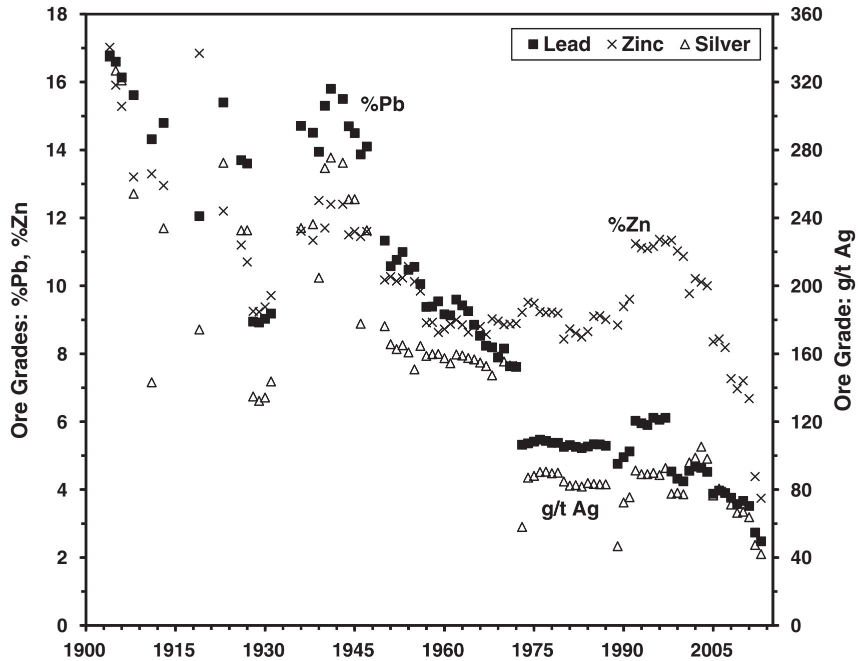

Fig. 14. Long-term historical trends in Australian Pb-Zn-Ag grades of reserves and/or resources (as reported at the time) (data updated from Mudd, 2007, 2009b, 2010a).

Jowitt, 2014; Mudd et al., 2013a,b), it is possible to find selected examples of high recovery at low ore grades – but detailed analysis of this issue is outside the scope of this paper.

A critical issue that is not captured by ore grades alone is the ease with which an ore is processed using existing mining-milling technology – in other words, how refractory the ore is (e.g. Bulatovic, 2007, 2010; Schlesinger et al., 2011; Schlitt, 2011). Some ores are very difficult to separate Pb from Zn using standard flotation technology; for example, the McArthur River deposit, despite still being one of the largest and highest grade Pb-Zn deposits in the world, is highly refractory. It was discovered in 1955 but developments in milling technology did not allow the project to be developed until the mid-1990s (specifically the new IsaMill grinding technology; Rossberg and Pafumi, 2013) – even then a bulk Pb-Zn concentrate was produced which can only be sent to very few smelters around the world (namely facilities using Imperial Smelting Furnace technology). The main problem at McArthur River is that the ore and economic minerals are extremely fine-grained, presenting a significant challenge in grinding and flotation to allow economic recovery.

Another major challenge is the issue of impurities in Pb or Zn concentrates. Although some impurities can be considered a potential resource, such as indium, Cd or Ga, others can be considered a toxic risk or 'penalty element', such as As, Sb, Hg, or Ge, that negatively affect worker's occupational exposure levels, cause environmental pollution, or create difficulties in generating concentrates that meet processing restrictions present at many smelters or refineries, all of which can be very difficult to address. Some metal impurities, however, can be considered both as potential resources as well as toxic risks (e.g. Cd, Ge, Sb, Ni). Increasingly strict regulations are evolving all over the world, such as the REACH legislation in Europe,[6] which are forcing smelters and refineries to continually improve both their sourcing of lower impurity concentrates as well as the nature and scope of processing and pollution control technology in use at specific facilities. Typical specifications for Pb and Zn concentrates are given in Table 12, showing the typical range of concentrations expected. Fundamentally, whether a metal can be viewed as a resource or a problem is dependant fundamentally on recovery costs, market prices and demand versus the costs of pollution control and waste residue management.

4.4. Companion metals and possible co/by-products

The production of a variety of companion metals is typically done at a smelter or refinery; for example, a Pb smelter can typically extract Au, Ag, Cu, Sb and convert S to sulfuric acid (H_2SO_4), whereas a Zn refinery can typically extract indium, Ge, Ga as well as also converting S to H_2SO_4. Although many other metals may be present in concentrates (see typical ranges in Pb-Zn concentrates in Table 12), the production (or not) of these other metals is mainly a function of demand and economics rather than limiting technology. This means that if world markets demand more of a particular metal at a price that is below the cost a smelter or refinery can produce at, then it is worth investing

Table 12
Typical commercial specifications for Pb and Zn concentrates (preferred maxima in brackets) (Sinclair, 2005, 2009).

	Pb conc.	Zn conc.		Pb conc.	Zn conc.
Zn	3–15% (10%)	48–56%	Ag	100–2000 g/t	20–200 g/t
Pb	55–75%	1.0–3.0% (3.5%)	Au	0–5 g/t	0–2 g/t
Cd	0.005–0.20%	0.15–0.30%	Ni	5–50 g/t	10–100 g/t (100 g/t)
Cu	0.005–1.0%	0.10–1.5% (2.0%)	Co	0–20 g/t	10–200 g/t (200 g/t)
Fe	2–12%	1.5–10% (12%)	Hg	5–50 g/t (10 g/t)	10–100 g/t (60 g/t)
S	14–25%	30.5–32.5%	Tl	–	0–50 g/t (50 g/t)
Mg	–	0.05–0.2% (0.3%)	In	–	50–500 g/t
SiO_2	2–10%	0.2-2.5% (3.0%)	Ge	–	0–100 g/t (40 g/t)
Ca	0.05–1.5%	0.2–1.0% (2.0%)	Ga	–	0–200 g/t
Mn	–	0.02–0.8% (0.4%)	Se	0–30 g/t	0–50 g/t (50 g/t)
As	0.02–0.5% (0.2%)	0.01–0.5% (0.5%)	Ba	–	100–500 g/t
Sb	0.01–0.3% (0.2%)	0.01–0.1% (0.2%)	Bi	5–1000 g/t (10 g/t)	10–500 g/t
Na	–	0–0.2% (0.01%)	F	5–500 g/t (100 g/t)	0–200 g/t (100 g/t)
K	–	0–0.1% (0.01%)	Cl	10–1000 g/t (500 g/t)	10-1000 g/t (500 g/t)
Al	0.2%	–	C	–	0–1000 g/t (1.0%)
			Sn	2–50 g/t	–

[6] See REACH website, echa.europa.eu/regulations/reach (accessed 3 February 2016)

capital in the extra process steps to extract that metal. This is exemplified by the considerable investment in greater indium capacity at Zn refineries to meet strongly growing world demand for indium in electronic consumer goods (especially those with flat screen displays; e.g. Werner et al., 2015, 2016).

A number of the Pb-Zn deposits in our compiled data contain numerous 'unusual' metals reported, such as Sc within the Buckton deposit or the REEs within the Kvanefjeld deposit. As shown earlier, these metals can provide substantial additional value for a mineral deposit and individual mining projects. There are, however, some important caveats to consider before considering the potential for companion metals to add value to a deposit.

Some of the largest projects, either by tonnage or value, which report a variety of companion metals are typically shale-related (e.g. Buckton, Talvivaara, Viken or Häggån) or contain a particularly valuable metal such as Sc or REEs (e.g. Buckton or Kvanefjeld, respectively). These somewhat unusual polymetallic projects remain very technically challenging, primarily as milling technologies such as froth flotation are typically geared towards the separation of conventional minerals such as Pb, Zn or Cu sulfides at grain sizes that may be larger than those typically expected in fine-grained shales, among other potential processing issues. This means that the process flow sheet adopted for a particular project is critical in terms of which metals are recoverable and at what rate, as this governs economics and technical performance. Some examples are crucial to highlight in this regard:

- *Viken (Continental Precious Minerals Inc), Sweden* – the initial development plan focussed on U-Mo-V and excluded Ni-Cu-Zn, but now the focus is on U-Ni-Cu-Zn whilst excluding Mo-V;
- *Talvivaara (Talvivaara plc), Finland* – developed after many years of research, Talvivaara began construction in early 2007 with commercial production in early 2009 using a bioleaching technology (e.g. Jowitt and Keays, 2011). Although a built at a large scale (~9.8 Mt ore/year) and focussed on Ni-Zn-Co-U, the project has faced numerous operational setbacks, environmental incidents, cost overruns, and in November 2014 the mine was officially declared bankrupt and placed into care and maintenance mode.

A common issue faced by mining companies in determining how to proceed with a particular polymetallic deposit is the lack of specific guidance in mineral resource codes on a major/primary product versus a by-product. Given that the economic value of many companion metals commonly accrues at a smelter or refinery and that the value of many companion metals is so low to a mine, this exacerbates the recognition of value throughout the whole supply chain. This issue and related problems are addressed extensively by Mudd et al. (in press).

These cases highlight the challenge of very low grade ores and 'unconventional' deposits and process configurations, and especially the challenges involved in delivering on economic value from a complex mix of primary and companion metals.

4.5. Tailings as possible future mineral resources?

In general, tailings are not considered a valuable resource in the mining industry and are typically viewed as waste that costs financial and technical resources to manage appropriately. There have been exceptions, such as the Zn-rich tailings processed in the early years at Broken Hill (after the developments in flotation allowed economic recovery), the potential for certain types of tailings to be used during CO_2 sequestration and adding value in the form of carbon credits (e.g. Wilson et al., 2009), or within the Au sector (especially South Africa, where old tailings are commonly reported as current mineral resources for Au and/or U as well as some former tailings dams being reprocessed to extract more Au but also undertake remediation and ensure better long-term environmental and social outcomes). In general, however, tailings reprocessing remains extremely rare in the base metals sector, primarily as the original plant was designed to treat the ore in a particular manner and the residual metals in tailings were unrecoverable at the time – meaning the remaining mineralogy is often refractory or expensive to recover (otherwise it would have been extracted originally).

In our compiled data there are 17 Pb-Zn resources associated with mine wastes – 14 of which are tailings, 2 ore stockpiles and 1 of which is a slag dump. Curiously, many of these 'wastes' typically have slightly higher Pb-Zn-Ag ore grades than the world average (see Table 3), although they are much smaller in scale than existing mines and reported resources. This leads to the interesting confluence of a lack of discovery for major new deposits and the growing potential for companion metals to add value to Pb-Zn projects. In other words, the gradual mining and depletion of mineral resources means that tailings could potentially become a more attractive resource in the future – both for the production of primary metals such as Pb-Zn-Ag $\pm$ Cu $\pm$ Au as well as a variety of potential companion metals.

As many of these companion metals were not produced on an industrial scale until the latter part of the 20th century, there has been a significant period of time where Pb-Zn ore has been processed without the recovery of companion metals, and hence we would expect the associated tailings to be particularly enriched in these metals. Even today, despite considerable improvements in the processing efficiency of indium in recent decades, it is estimated that on average, ~35% of the indium present in Zn concentrates is refined, suggesting the that bulk of indium extracted from Pb-Zn deposits still accumulates in mine wastes (see Werner et al., 2015).

Given past production data, it is possible to make a reasonable estimate of tailings for many projects, using ore processed, ore grades, metals extracted and recovery rates. Assuming Pb-Zn concentrates are each ~50% Pb or Zn, with Cu concentrates at 20% Cu (typical grades for by-product Cu concentrates), this allows an estimate of the remaining tailings and approximate grades. That is, ore processed minus concentrates extracted approximates remaining tailings (assuming no tailings are used as part of underground mine backfill), with grades estimated based on recovery rate and original ore grade. For a variety of projects in Australia, Peru and Canada with complete historical production data, estimates of tailings quantities and grades are compiled in Table 13 (the full data used to derive these estimates is presented in the supplementary information). As can be seen, just these projects alone are estimated to contain 7.0–12.7 Mt Pb-Zn, with modest additional Ag-Cu-Au. For the Pb-Zn companion metals, studies of elemental deportment can be used in conjunction with historical production data to estimate the extent to which companions have accumulated in the calculated tailings volumes, although as discussed in Mudd et al. (in press), these studies are rare, particularly for those deposits with available historical production data. Case studies of indium accumulation in tailings are presented for Heath Steele and Brunswick 6–12, Canada in Werner et al. (2015) and for Broken Hill, Australia in Werner et al. (2016), highlighting that several years' worth of global supply for this critical metal have accumulated in these three deposits alone. This strongly suggests that if individual tailings assessments were to be conducted for companion metals across all deposits listed in our Pb-Zn database, quantities would be calculated which rival that of the global mineral resources of these metals.

The former Hellyer mine in Australia, which operated from 1985 to 1999, is the most recent example of reprocessing Pb-Zn-Ag-Cu-Au tailings after mine closure. The original ore was fine-grained and difficult to process through conventional flotation methods at the time, with metallurgical test work taking considerable effort to achieve a viable process flow sheet (Richmond and Lai, 1988). Our estimate of tailings quantity and grades is shown in Table 13, along with the code-based resource published in 2001 – which is in very close agreement to our estimate (suggesting our estimates for other sites are reasonable despite their 'conceptual' nature). The Hellyer project was bought by metallurgical technology company Intec Ltd in 2003, who then developed a

Table 13
Estimates of tailings as potential future mineral resources for selected Pb-Zn projects in Australia, Canada and Peru.

Project	Years	Mt	%Pb	%Zn	g/t Ag	%Cu	g/t Au	Mt Pb	Mt Zn	t Ag
Broken Hill, Australia	1883–2014[c]	138.1	0.63	0.83	12	–	–	0.87	1.15	1667
Rosebery, Australia	1936–2014[c]	24.0	0.79	1.36	27	0.13	0.57	0.19	0.33	643
Century, Australia	2000–2014[a,c]	64.8	0.67	2.52	17	–	–	0.43	1.63	1121
Cannington, Australia	1997–2014[c]	34.9	1.38	1.34	64	–	–	0.48	0.47	2243
Mt Isa, Australia	1931–2014[c]	181.5	1.38	2.13	33	–	–	2.51	3.86	6010
Antamina, Peru	2001–2014[c]	445.2	-	0.33	-	0.16	–	–	1.46	–
Sullivan, Canada	1900-2001	117.7	0.84	0.69	10	–	–	0.99	0.81	1119
Kidd Creek, Canada	1966-2014[c]	121.5	0.09	0.71	16	0.10	–	0.11	0.87	1897
Myra Falls, Canada	1967–2014[c]	25.4	0.15	0.64	11	0.20	0.84	0.04	0.16	276
Polaris (Little Cornwall), Canada	1981–2002	13.7	0.28	0.70	–	–	–	0.04	0.10	
Nanisivik (Baffin Island), Canada	1976–2002	14.6	0.12	0.44	–	–	–	0.02	0.06	133
Pine Point, Canada	1965–1988	51.8	0.06	0.43	–	–	–	0.03	0.22	
Brunswick, Canada	1964-2013	120.0	1.07	1.34	42	0.09	–	1.28	1.60	5048
Sub-total		***1353.2***	***0.52***	***0.94***	***15***	***~0.04***	***~0.03***	***6.98***	***12.72***	***20,157***
Hellyer – our estimate	1985–1999	11.0	2.7	2.7	93	~0.11	-	0.29	0.29	1017
Hellyer – reported	Rep. 2001	10.88	3.0	2.8	88	0.16	2.6	0.33	0.30	957
Hellyer Tailings Reprocessing	2006–2008	~1.9	*0.49*[b]	*1.62*[b]	*8.2*[b]	nd	nd	9 kt	30 kt	15
Hellyer – reported	Rep. 2011	9.5	3.0	2.5	104	0.2	2.6	0.29	0.24	988
Woodlawn – reported	Rep. 2013	11.65	1.35	2.29	32	0.50	0.29	0.16	0.27	371
Woodlawn – our estimate	1978–1999	11.1	1.4	2.2	34	0.46	0.43	0.16	0.25	382

Notes: nd – no data.
[a] Century closed at the end of 2015.
[b] Yield only and not true assayed ore grade.
[c] Still operating at end of 2014.

proprietary processing technology to treat the Hellyer tailings. The former Hellyer mill was refurbished to allow it to treat the tailings in a project that was in operation from late 2006 to mid-2009, when it was closed due to the collapse in world metals prices at this time. Production data for this period is shown in Table 13, along with the subsequent updated tailings resource estimate from 2011. The assayed ore grades of reprocessed tailings was not publicly reported, but comparing the 2001 ore grades (3.0% Zn) versus the yields achieved during tailings reprocessing (0.49% Zn) shows that recovery rates were low, and were probably a strong contributing factor in the poor technical and economic performance of the project. This is probably related to the fine-grained nature of the original ore, the original process flow sheet leaving only more refractory minerals in the tailings as well the oxidation and weathering of the principal economic Pb-Zn minerals in the tailings dam, especially galena (see Bott and Lumsden, 2009).

The former Woodlawn mine in Australia, which operated from 1978 to 1998, also has reported tailings-based mineral resources. Our estimate of tailings derived from historical production data is given in Table 13, and is in close agreement with the reported 2013 mineral resource. There is also an additional mineral resource reported for the former Woodlawn underground mine, with recent plans as of mid-2015 looking to process tailings concurrently with fresh underground ore (see Ebbels, 2015). Although the operating company, TriAusMin Ltd,[7] had spent about a decade of effort on a new Woodlawn project and received statutory approvals in July 2013, it has still not proceeded as of late 2015. This is probably a reflection of the current world metals markets and project economics as much as the technical challenges involved in developing a tailings reprocessing project.

A rather different and perhaps unexpected example of tailings is the Zn associated with tailings from the Ok Tedi Cu-Au-Ag mine in Papua New Guinea. Ok Tedi disposes of tailings into the adjacent Ok Tedi-Fly River system, along with considerable erosion of waste rock (see Bolton, 2008). Curiously, for the years 2003 and 2004, Inmet Ltd, the minority (18%) partner in Ok Tedi, reported in their sustainability disclosures that discharges of metals to water were some 95.8/21.2 and 87.7/19.4 kt Cu/Zn in 2003 and 2004, respectively (Inmet, 2005). Based on Ok Tedi's production data around this time (~27 Mt ore/year at ~0.83% Cu for ~184.5 kt Cu, plus Au-Ag), this equates to an approximate Zn ore grade of ~0.1% Zn – suggesting that porphyry-skarn deposits like Ok Tedi (Jowitt et al., 2013) may also host potentially important Zn mineralisation (e.g. Agua Rica and Los Azules porphyry projects in Argentina) – as well as the need to understand such large metal pollution loads from mining projects where riverine tailings and waste rock disposal is approved by government for use.

Overall, although tailings can be a potentially valuable resource, in reality the remaining Pb-Zn is often quite refractory in nature and, despite cheaper mining (say a dredge in a tailings dam versus open cut or underground mine), this often leads to complex and expensive processing that may in turn lead to low metal recoveries Therefore, at present, tailings probably remain a higher cost resource unless very specific and unique circumstances facilitate a clear economic proposition (especially in the face of current world metals markets and low prices).

4.6. Environmental issues facing the Pb-Zn sector

One of the major challenges facing the world's Pb-Zn sector is managing environmental pollution issues within statutory guidelines amid ever increasing community expectations. Historically, Pb in particular is renowned for negative health impacts on workers as well as users, especially where direct contact was made (e.g. mining or cosmetics; see Finkelstein et al., 2014) – or Pb was used with indirect contact (e.g. Pb pipes for water were used extensively in the Roman empire; e.g. Nriagu, 2011). Furthermore, Pb exposure represents a significant risk to infants and young children, as Pb accumulating in blood will affect their intellectual development, especially their mental and cognitive functions (Canfield and Jusko, 2008).

In general, guidelines and regulations for Pb use and levels in the environment have become very strict in recent decades. Here we very briefly review the cases of Mt Isa, McArthur River and Magellan in Australia, and discuss some other key environmental aspects of Pb-Zn mining. Although this section is intended to be succinct and not exhaustive, it is sufficient to demonstrate the breadth and depth of environmental issues and challenges facing the Pb and Pb-Zn mining and processing sectors.

4.6.1. Mt Isa Pb-Zn-Ag and Cu field, Australia

The Mt Isa Pb-Zn-Ag field in western Queensland, Australia, was discovered in 1923 and entered commercial production in 1931 with a mine, mill and Pb smelter. About a decade later large Cu deposits were

[7] TriAusMin Ltd merged with Heron Resources Ltd in mid-2014.

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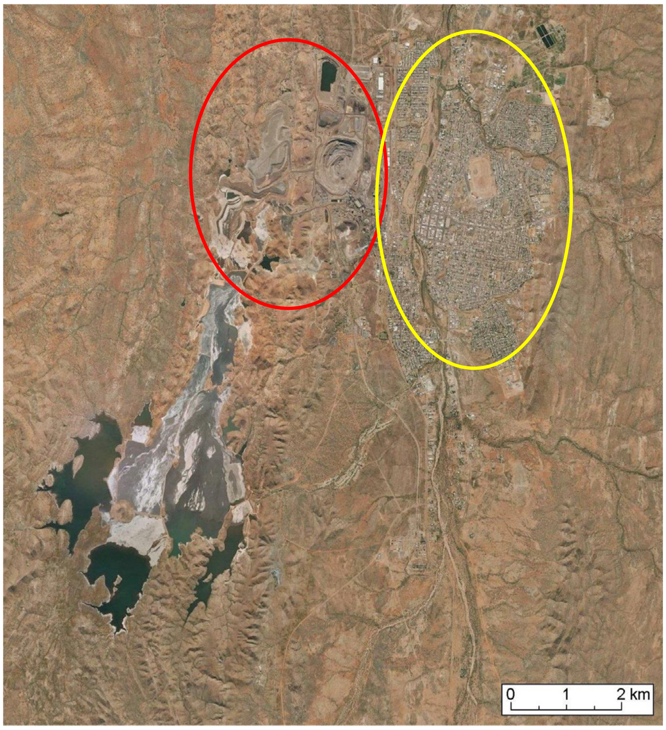

Fig. 15. Aerial view of the Mt Isa mining-smelting complex (red oval) and township (yellow oval) (the northern operations, George Fisher North/South (Hilton) and Handlebar Hill, are ~15 km north of this image) (adapted from GE, 2016); approximate centre co-ordinates 20° 42′ S, 139° 29′ E).

also discovered in this region, with commercial production beginning in 1953 and also including a mine, mill and Cu smelter. The field maintains large scale Pb-Zn-Ag and Cu operations with a very large mineral resource base for both (see Tables 5 and 9; for Cu, see Mudd et al., 2013a). The town of Mt Isa was developed for and has grown along with mining (as well as supporting the development of agriculture in the region), and in 2012 was home to about 22,600 people (ABS, 2015). The Leichardt River splits the town to the east from the main operations to the west (Fig. 15), with a view of the smelters from the central township shown in Fig. 16.

In 1994, high blood Pb levels (BLLs) in children were first confirmed at Mt Isa and partial pollution controls were put in place in 2000 as well as follow-up health and environmental studies (Munksgaard et al., 2010). There were two primary lines of thought put forward – first, part of the problem was 'natural' in that the Mt Isa region was 'rich in Pb' (i.e. natural contamination, although even if natural this is not the same as 'safe'); and second, that the significant BLLs were a result of historical mining-milling-smelting activities, and since the operations had installed some pollution control technologies and improved environmental management, they were no longer a significant Pb source.

Recent research has shown these arguments to be factually incorrect (see Mackay, 2011; Mackay et al., 2013; Taylor et al., 2014, 2015, 2010). Firstly, the mineralisation containing Pb-Zn only crops out on the western side of the Leichardt River (where the mining and smelting operations are), with differing geology and soils free of Pb-Zn mineralization on the eastern side of the river where most of the Mt Isa township is located; in other words, there are no naturally Pb-enriched soils in the vast majority of the township area. Secondly, the top 5 cm of surface soils are strongly enriched in Pb-Zn-Cu and have a clear Pb isotopic signature that is indicative of Pb derived from Mt Isa ores (and is not from the local use of tetraethyl lead containing Mt Isa derived Pb). The only realistic way to explain the elevated concentrations and isotopic fingerprint is ongoing emissions from the Mt Isa mining-milling-smelting complex and airborne dispersion – especially since there are no naturally elevated metals in the top 5 cm in the first place.

According to 2013/14 data from Australia's National Pollutant Inventory (NPI, 2016), total Pb emissions to the environment were estimated at 997.4 t Pb, of which 979.1 t Pb was to the atmosphere. For the total Pb emissions, some 669.6 t Pb is from diffuse sources such as vehicles, bushfires, or others, which leaves 327.8 t Pb from direct sources. The Pb emissions for the Mt Isa complex were some 80.6 t Pb, which is much higher than other facilities such as the Port Pirie Pb smelter (53.3 t Pb), and other processing sites such as Broken Hill (30.7 t Pb), Olympic Dam Cu-U-Au-Ag (18.3 t Pb) and Cannington (13.2 t Pb). Throughout the 2000s, NPI-reported Pb emissions for the Mt Isa complex generally ranged from 225 to 413 t Pb, including one year at 544 t Pb in 2000 – suggesting that recent emissions have declined significantly. This clearly highlights that Mt Isa was and remains a strong source of Pb (and other) emissions to the local environment.

Fig. 16. View of the Mt Isa Cu (left) and Pb (right) smelters from the centre of Mt Isa township (April 2010).

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Fig. 17. Smoke plume due to oxidation of sulfides contained in waste rock at McArthur River (photo: Jane Bardon, Australian Broadcasting Corporation, date 26 March 2015; used with permission).

4.6.2. McArthur River Pb-Zn-Ag project, Australia

This case study of the McArthur River Pb-Zn-Ag project is summarised from EES (2009, 2010, 2011, 2012), ERIAS (2014, 2015), MetServe (2012) and URS (2005, 2006). The original 1996 development of the McArthur River Pb-Zn-Ag project (Northern Territory (NT), Australia) involved the initial development of an underground mine (1.6 Mt/year) with crushing, grinding and flotation to produce a bulk Zn-dominant concentrate with moderate Pb-Ag levels. Ongoing technical challenges and difficult economics meant that a variety of options were investigated in the early 2000s to expand the project and improve profitability. Although an onsite Zn refinery with a 350 MW power station and a large open cut scale (4.8 Mt/year) was initially considered, the final option by 2005 was to convert from underground to open cut mining, including a 6 km long diversion of the McArthur River itself (since the orebody goes underneath the project's namesake river) as well as slightly expanding milling capacity (to 1.8 Mt/year) with improved processing technology. The project proved highly controversial, especially for local indigenous communities, and after various court cases and an extended environmental approvals process, the conversion to open cut mining (to ~2.5 Mt/year) and the diversion of the McArthur River was completed by early 2009. Subsequently, in 2012, a further expansion was proposed to processing rate of 5–5.5 Mt/year, although this has been quickly overtaken by the increasingly difficult and complex environmental issues being faced by the project.

One of the outcomes of the 2005 to 2006 approvals process was an expert independent monitor (IM), appointed by the NT Government, to assess the environmental performance and compliance with various conditions of approvals – and although this a unique and rare scenario in the mining industry (given that statutory environmental monitoring reports are, for the most part, never made publicly available), it is clearly a valuable approach in helping to transparently address community concerns over perceived and actual environmental impacts and risks. The first annual report for 2008 documented broad compliance but also some procedural non-compliances and raised varying levels of concern regarding tailings seepage, dredge spoil management, weeds, dust, fencing maintenance, inadequate analyses of laboratory versus field results and the performance of clay liners for waste rock dumps. From 2009 to 2014, despite broad compliance, almost all of these aspects continued to remain significant issues of concern to the IM but with an increasingly critical tone given the growing scale of the risks being identified – especially the growing problem of acid, saline and metalliferous drainage from sulfidic mine wastes (see later sub-section). In particular, by 2015, the problem of sulfide oxidation in waste rock was proving to be an extremely critical issue due to the large plumes of smoke being generated from such dumps (caused by the heat of the process; Fig. 17) as well as presenting major challenges for seepage, water quality and site water management – so much so that a new environmental approvals process has been instituted specifically to focus on waste rock and overburden management as well as a major increase in the financial bond for rehabilitation being held by the NT Government. Furthermore, new issues of concern have developed, including (i) the mobilisation of metals into the environment and the ecological food chain, with concerns raised about the potential for elevated metals in fish and cattle especially[8]; (ii) the changes to geochemical classification of waste rock leading to a major reduction in the quantity of benign or non-acid forming mine wastes which can be used as liners or covers in managing sulfidic wastes and associated seepage risks; and (iii) construction quality issues in parts of the tailings storage facility and waste rock dumps.

[8] In August 2015, it was reported that up to 400 cattle had gained access to a mine in the region, with 5 cattle culled for testing and one of these returned a Pb concentration above the human health limit (ABC, 2015). Cattle Shot and Tested Over Mine Site Contamination Fears on Northern Territory Stations. ABC Rural, Australian Broadcasting Corporation (ABC), http://www.abc.net.au/news/2015-08-21/contaminated-mine-cattle-nt/6715438 (Published 21 August 2015; Accessed 28 January 2016).; see also the independent monitor reporting.

The increasingly critical nature and scale of the environmental risks being faced by the McArthur River project are a salutary lesson in ensuring that pre-development assessments and approvals processes are as thorough as possible in identifying risks and proposing effective management strategies – but especially outlines that these processes are not always accurate in predicting the large scale of the environmental problems that can emerge at mining projects. Although the project has a current mine life of more than 20 years, the significant environmental, technical and economic challenges in this area means that the future of this mine remains difficult to predict.

4.6.3. Magellan Pb oxide project, Australia

This case study of the Magellan project, which was initially developed by Canadian miner Ivernia West Inc in 2004 and began commissioning in January 2005, is summarised from ESHC (2007), IWI (var.) and Ljung et al. (2010). The project is based on an unusual Pb oxide deposit with no economic Zn or Ag that only produces a Pb concentrate through a flotation plant. The project is located in the very centre of Western Australia, ~900 km north-east of Perth and just west of Wiluna, with the Pb concentrate transported south to the small coastal town of Esperance. By December 2006, the local Esperance community had begun to observe birds falling from the sky, and by January 2007, over 20 such incidents had been reported with thousands of bird deaths estimated. Although initial thoughts were a viral or bacterial cause, testing showed elevated Pb derived from the Magellan concentrates (as evidenced by isotopic fingerprinting). The transport of Pb concentrates was banned from the port and the project was forced to shut down in March 2007. Testing of 600 young children was also undertaken, showing 81 with blood Pb levels above the international standard of 5 μg/dL and a clear contribution of Magellan-derived Pb between 30 to 87% for all results. For the children with blood Pb levels above 3 μg/dL, 84% had at least 50% of the Pb derived from Magellan.

At the heart of the problem was the form in which the Pb concentrates were transported to, stored and handled at the port. Although there was a regulatory expectation that the concentrates would be agglomerated into a granular or pellet form and dried at the mine before transport (to minimise dust risks), the concentrates were left in a moist, fine-grained form – making them very prone to dusting as they dried out during transport, storage and handling. The project was re-opened again in early 2010, after a range of engineering improvements, but this time transporting concentrates under strict conditions through the port of Fremantle near Perth. Following the discovery of external Pb contamination on the containers during transport, which may or may not have been due to Pb concentrate sourced from Magellan, the project was closed again in April 2011. The project was re-opened yet again by Ivernia in April 2013, this time with a major emphasis on community engagement, responsible Pb mining and product stewardship, but poor market conditions forced the project back into care and maintenance again in January 2015 – by which time the project was renamed 'Paroo Station' along with Ivernia renaming their corporate image as Lead*FX* Inc.

Despite well documented practices and standards for transporting Pb concentrates, the Magellan mine and Esperance port clearly failed to achieve good outcomes and significant environmental pollution and public health impacts were the result – outcomes which were entirely preventable but also cost Ivernia dearly in financial and reputation terms. The Pb contamination has now been remediated at the Esperance port (Kohlrusch, 2012).

4.6.4. Other examples of environmental and public health impacts

The environmental and public health challenges faced by the Mt Isa, McArthur River or Magellan projects are not unique and effectively apply to all Pb-Zn operations around the world. Although many of the environmental pollution and public health problems can be traced to historical legacies from practices that did not consider such long-term problems, it is clear that not only these legacies remain in many parts of the world but that the issues remain ongoing. Select examples, amongst many others, of significant pollution issues at Pb-Zn mining or smelting/refining sites include:

- Kabwe Pb-Zn-Ag mine and smelter, Zambia (Tembo et al., 2006);
- Broken Hill Pb-Zn-Ag mine, Australia (Dong et al., 2015);

Fig. 18. Select Australian examples of acid mine drainage at former Pb-Zn mines. Left - Severely AMD-impacted stream immediately downstream of the historic Sunny Corner mine and smelter site, NSW, Australia (14 July 2013) ; Middle - AMD-affected stormwater drain in the Pb-Zn mining town of Zeehan, Tasmania, Australia (4 February 2014); Right - AMD entering a drainage channel from the rehabilitated tailings dam at Captain's Flat, NSW, Australia (3 July 2015).

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- Port Pirie Pb smelter, Australia (Taylor et al., 2013);
- La Oroya smelter, Peru (Fraser, 2009; Reuer et al., 2012).

4.6.5. Acid mine drainage

The vast majority of Pb-Zn deposit types contain resources dominated by sulfides that are also associated with sulfide-bearing gangue, which commonly contains pyrite, pyrrhotite, and other non-economic sulfides. The mining and exposure of these sulfides to the surface environment facilitates weathering and oxidation to form sulfuric acid, which in turn leaches out numerous salts and heavy metals. This process is commonly known as acid and metalliferous drainage or 'AMD' (aka acid mine drainage) and remains a widespread and difficult challenge for mine waste and the environmental management of mining (see Da Rosa et al., 1997; Dold, 2008; Lottermoser, 2010; Spitz and Trudinger, 2008; Taylor and Pape, 2007), amongst others. Some select examples of significant AMD issues include (some photographs are also given in Fig. 18):

- Rosebery Pb-Zn-Ag-Cu-Au mine, Tasmania, Australia (Evans, 2009);
- Zeehan Pb-Zn-Ag field and overall mining-related AMD contributing to the Pieman River, Tasmania, Australia (e.g. Evans et al., 2003);
- Captain's Flat, New South Wales, Australia, ongoing AMD after mine rehabilitation (Brooks, 1997; Jacobson and Sparksman, 1988);
- Iberian Pyrite belt, southern Spain/Portugal (e.g. Nieto et al., 2007);
- Thalanga, Queensland, Australia (Thienenkamp and Lottermoser, 2003)

Sulfide oxidation can be a natural geological process, with the discovery of the Red Dog deposit credited to signs of visible sulfide oxidation (i.e. orange staining of surface rocks; see Moore et al., 1986). However, the scale of AMD risks and issues posed by modern Pb-Zn mining are clearly far in excess of anything that could be ascribed as natural in origin, and such risks and impacts to water resources remains a fundamental area of community concern regarding existing and potential new mines.

The mining industry is, in theory, now better equipped to assess and manage AMD risks, especially given the recent textbooks (Lottermoser, 2010; Rankin, 2011; Spitz and Trudinger, 2008) or technical handbooks (Parker and Robertson, 1999; Taylor and Pape, 2007) that focus on this subject – but there remains much to be done to document AMD risks and issues and especially demonstrating good long-term environmental management outcomes after rehabilitation of such wastes.

4.6.6. Environmental management

Although there are clearly a variety of significant environmental risks inherent in Pb-Zn mining, it is important to recognise that the mining industry has improved considerably in this area in the past few decades. Part of this is more rigorous environmental impact assessment before approvals, but also greater expectations from the broader community that mining simply cannot cause environmental pollution and then walk away as they were allowed to do in the past – modern mining must implement sound environmental management practices throughout the whole mine life cycle, from exploration through development to operations and closure and rehabilitation (Allen et al., 2013; Rankin, 2011; Spitz and Trudinger, 2008). Mining companies are also being increasingly judged on how they integrate environmental aspects into broader sustainability performance and reporting, with transparency a key test for many in the community (see Fonseca, 2010; Mudd, 2010a, 2012b).

Given the substantial resource quantities identified in this study, it is likely that primary production of Pb and Zn will continue to dominate the future supply of these metals, however it is worth recognising that changes in societal expectations around the sustainable sourcing of metals may encourage a shift towards recovery and recycling of Pb and Zn through secondary production pipelines, thereby limiting the requirement for continued production from, and discovery success of, Pb-Zn mineral deposits. Future requirements to reduce the embodied environmental and social impacts of Pb and Zn extraction (as well as that of the extraction of their companions) may lead to increased recovery from tailings, and/or greater scrutiny on the trade of metals sourced from particular mines, similar to the sourcing of 'fair trade' foods. While a comprehensive assessment of the environmental impacts of each of the identified Pb-Zn deposits is beyond the scope of this paper, it is believed that a compilation of known potential sources and recognition of their unique environmental and social challenges is an important first step in characterising the sustainability of future Pb and Zn supply.

5. Conclusions

This paper has presented a detailed appraisal of the world's known Pb-Zn resources resulting from an extensive compilation of individual deposits reporting Pb-Zn mineralisation, each classified according to deposit type, and noted for their economic value and reported by-products. Our data show that at least 226.1 Mt Pb and 610.3 Mt Zn are present within 851 individual mineral deposits and mine waste projects from 67 countries and one in international oceanic waters, at an average grade of 0.44% Pb and 1.20% Zn. The identified resources are dominantly present within sediment-hosted Pb-Zn deposits (490.6 Mt Pb + Zn + Cu), which contain the equivalent of VMS, Skarn, Porphyry, Epithermal and mixed sediment-hosted deposits combined, and over 49% of these resources are reported in Australia, Peru, Russia and Canada alone. We have presented multiple case studies where reported reserves and resources for a given year are largely outstripped by future cumulative production. This strongly suggests that the true extractable quantities of both Pb and Zn are far beyond the quantities reported in our database and that future estimates are likely to increase, particularly given the increasing value of some by-products – although the lack of global exploration success remains of concern. Of course, there are still many challenges facing the Pb-Zn mining industry, which we have highlighted through numerous individual case studies, relating to declining ore grades, waste management, and many other socio-environmental factors. These kinds of challenges are far more likely to place restrictions on the supply of Pb and Zn in the future than the availability of mineral resources.

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Quantifying the potential for recoverable resources of gallium, germanium and antimony as companion metals in Australia

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ABSTRACT

Although critical to newly evolving and increasingly essential technologies, antimony (Sb), gallium (Ga), and germanium (Ge) are generally recovered as byproducts or 'companion metals' of other metal ores. The stage at which companion metals are extracted depends on metallurgical processes by which the host ore mineral is extracted and processed; many companion metals are recovered late during this processing. Therefore, the current and future supply of companion metals relies not only on production of major commodities, but also on the efficient recovery of these metals during processing that recovers the primary commodity.

National geological surveys, particularly the USGS, publish annual estimates of global reserves for a variety of primary metals, but generally not for companion metals. This study provides estimates for the geogenic stocks (in waste rock piles, tailings, smelting, and refining) of Ga, Ge, and Sb as companion metals. These elements are mined in Australia but may be recovered outside of Australia, but their life cycles have not yet been well understood.

Based on the methodology adapted, this paper estimates a minimum of 970–1230 kt of Ga, 30–10,000 kt of Ge and 70–1000 kt of Sb in current Australian lead-zinc-silver, gold, copper, iron ore, coal, bauxite, and bauxite residue (red mud) resources. The large range of estimated stocks stems from the variable range of ore grades reported by companies and the considerable uncertainty that exists among the grade estimates presented. However, these estimates are reflective of best practice in mineral resource estimation of Ga, Ge, and Sb, and provide a basis for determining similar recoverable resource estimates of other companion metals, such as indium, rhenium, and selenium, all of which are of increasing importance in modern-day life.

1. Introduction

Metals such as gallium (Ga), germanium (Ge), and antimony (Sb) are generally recovered from the production of other metal ores and are critical to varying extents for the efficient operation of modern technologies. They are often referred to as 'companion metals' in the increasingly popular field of industrial ecology (Mudd et al., 2014), but other studies have referred to them as byproducts (Bleiwas, 2010; Mudd et al., 2013), 'hitch-hikers' (Ayres and Peiro, 2013; Hunt et al., 2013), or coproducts.

The three elements (Ga, Ge, and Sb) are considered 'critical' because they are either geologically scarce or are subject to potential supply constraints and needed for an economically important purpose where substitution is difficult or does not give the same level of performance (see Graedel et al., 2012; BGS, 2015; Harper et al., 2014; Panousi et al., 2015). An increasing body of scholarly literature around the world is studying the issue of scarce elemental sustainability and resource criticality (Graedel et al., 2011; Hunt et al., 2013). The level of criticality of a commodity reflects the combination of 'risk of supply' and the 'importance of the particular commodity'. The issue of criticality has been among the prominent discussions at government level around the world and has led to commissioning a number of studies and initiatives relating to raw material supply and criticality. The seminal and most important among such studies is the report entitled '*Minerals, Critical Minerals, and the US Economy*' prepared by the US National Research Council and published in 2008 (NRC, 2008). Several other countries, including both suppliers and users of

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raw materials, have commenced studies and initiatives to develop national strategies for securing a stable supply of raw materials, linked to the most important materials for their economy (BGS, 2012; Buchert et al., 2009; Chapman, 2013; EU, 2014; Skirrow et al., 2013). More than fifty materials were analysed in various studies, and Table 1 summarises the criticality of the three elements of interest in this study. Although the variation is wide among the studies, Ga, Ge, and Sb are generally ranked high in criticality, and in some cases very high.

The concentrations of Ga, Ge, and Sb are rarely reported in many host metal ore mineral deposits (or unknown in other deposits), resulting in uncertainty in determining the exact resource estimates for Australia and/or rest of the world. The U.S. Geological Survey (USGS) and Geoscience Australia (GA) publish annual estimates of resources for major commodities (*e.g.*, Cu, Pb, Zn, Fe, Au, etc.), but do not provide such data for companion metals. In many instances the non-availability of such information may cause technical difficulties during the smelting of concentrates of the major economic metal (*e.g.*, Sb in Cu and so on). It is therefore useful to prepare estimates of what might be termed as the 'recoverable resource' (i.e. the amount that if required, could be extracted and put into use over the next several decades, see Mudd et al., 2013) for a particular companion metal. In this study, we present an estimate for the potentially recoverable resources of Ga, Ge, and Sb in Australian primary ore deposits and geogenic stocks (such as waste rock piles, tailings, smelting, and refining).

2. Methodology and data sources

2.1. Production

Global production data cover 1970–2012 for gallium; 1955–2012 for Ge, and 1900–2012 for Sb. The data are primarily sourced from the various United States Geological Survey (USGS, 1994–2015) and British Geological Survey (BGS, 1990–2014) reports. Throughout the paper data are expressed in metric units unless otherwise stated.

2.2. Apparent use

United States domestic apparent use data are sourced from various USGS reports (USGS, 1994–2015), mainly from the Mineral Resources and Minerals Yearbooks. The actual sources may include either the mineral statistics publications of the U.S. Bureau of Mines and the U.S. Geological Survey—Minerals Yearbook (MYB); Mineral Commodity Summaries (MCS) and its predecessor, Commodity Data Summaries (CDS); and Metal Prices in the United States through 1998 or combination thereof. The apparent use estimate is reported in two significant figures only.

Data for 1955–70 are published under the category "Use," for the year 1971 "Actual use," and for 1972 to the most recent year "Apparent Use." These data are reported in the 'Commodity Data Summaries' and the 'Mineral Commodity Summaries' and are estimated using the following equation:

$$\begin{aligned}\text{Apparent Use} = (&\text{Domestic mine production}\\ &+\text{Secondary production from old scrap}\\ &+\text{Net import reliance}).\end{aligned}$$

2.3. Reserves and resources

In this paper we have used both USGS and JORC code (AusIMM et al., 2004) classifications in defining reserves. The USGS 'reserve' figures loosely correspond to what is defined as 'proved & probable ore reserves' under the JORC code standard, and the USGS's 'reserve base' figure includes 'measured & indicated resources' but excludes inferred resources. This is equivalent to the 'economically demonstrated resource' (EDR) used by GA in reporting resource data from Australia.

2.4. End use

Here we define end use as the use of the mineral commodity in a particular industrial sector or product, as per the classification used by the USGS. This is distinct from end use by the consumer which, in turn, accounts for the trade of final production containing a metal of interest (e.g. as per (Meylan and Reck, 2016)). End-use data are sourced from various USGS reports and the estimates and are derived by applying the reported percentages of end-use use to the calculated apparent use; actual use may be greater. Data are rounded to no more than two significant digits and thus data may not add up to totals shown in many cases.

Specific sources include:

• Mineralogical	(Anderson, 2012; Barthelmy, 2012; IMA, 2014; Jaques et al., 2002)
• Geological information	(Anderson, 2012; Berger et al., 2008; Cook et al., 2009; Ehrig et al., 2013; Eilu and Groves, 2001; Hart, 2005; Höll et al., 2007; Jaques et al., 2002)
• Reserves and resources	(BGS, 1990–2014; GA, 1990–2014; Mudd et al., 2014; USGS, 1994–2015)
• Recycling	(Khaliq et al., 2014; UNEP, 2011, 2013)
• Mineral production and use	(BGS, 1990–2014; GA, 1990–2014; USGS, 1994–2015)
• GDP and metal prices	(ABS, 1990–2014; BREE, 2014; UN, 2014; USGS, 2013)

3. Overview of gallium, germanium and antimony

3.1. Characteristics

Table 2 summarises the chemical and geochemical properties of Ga, Ge and Sb, along with their sources and metallurgical processes by which they are recovered. In the following discussion, features

Table 1
Summary of rankings of critical and other commodities from recent global studies of materials [Data source: (Skirrow et al., 2013)].

Element	Order of priority of raw material criticality as identified by different studies							
	USA DoE & DoD*	Japan	Wills & Chapman	EU		BGS, UK	South Korea	Australia
				Low C Econ	EU			
Gallium	6 (11)*	9 (17)	2 (27)	3 (13)	5 (14)	17 (41)	1 (24)	2 (43)
Germanium	5 (13)#	NA	11 (27)	10 (13)	6 (14)	11 (41)	NA	23 (43)
Antimony	NA	16 (17)	9 (27)	NA	1 (14)	3 (41)	12 (24)	10 (43)

* DoE & DOD – Dept. of Energy & Defense.
# Numbers in parenthesis are the total number of elements considered in study.

Table 2
Important characteristics and uses of Ga, Ge and Sb.

Characteristics	Gallium (Ga)	Germanium (Ge)	Antimony (Sb)
Element type	Metal	Semi-conducting metal	Semimetal
Principal sources of extraction in the world	Gallium is produced mainly during the processing of bauxite, where it is recovered by electrolysis of Al hydroxide solution. A secondary Ga source is recovered by leaching of iron residue produced during Zn extraction from sphalerite concentrates	Germanium is be recovered by leaching of iron residue produced during Zn recovery from sphalerite concentrates. Ge can also be recovered from of coal ash	The main Sb source is the stibnite, although it can be recovered from other Sb-bearing sulfides (e.g. tetrahedrite). Following flotation, Sb is recovered pyrometallurgically using volatilisation, smelting or liquation/iron precipitation. Antimony is mined either as a principal product or as a byproduct of the smelting of Sn and Au ores in nine countries
Extraction in Australia	By-product of Al production and Zn smelting	By-product of Zn smelting	Main product and co-product (with Au)
Estimated crustal abundance (ppm)	16–19	1.3–1.5	0.2
Recycling	Recycled from scrap generated in the manufacture of Ga-As based devices	Ge used in the optics industry is routinely recycled from new scrap. Worldwide 30% of Ge is generated from production residues	Small amounts are recycled from Pb-acid batteries
References	Moskalyk (2003)	Alfantazi and Moskalyk (2003)	Anderson (2012)

of these elements pertinent to determining potentially recoverable resources are identified.

3.1.1. Gallium

Gallium is a metallic element that is chemically similar to aluminium. It occurs naturally as a trace element in bauxite, sphalerite, and coal. Although there are eleven minerals listed in the Mineralogy Database that contain Ga (Barthelmy, 2012; IMA, 2014), these minerals are very uncommon, and currently, all Ga is produced as a companion metal mainly during alumina production from bauxite but also during zinc residue processing (Table 2). An excellent review of gallium's mineralogy can be found in Moskalyk (2003).

3.1.2. Germanium

Although nearly 30 minerals containing Ge exist in the Mineralogy Database (Barthelmy, 2012), it is also a highly dispersed element. Currently, Ge is recovered as a by-product from sphalerite ores, especially from sediment-hosted, massive Zn–Pb–Cu(–Ba) deposits and carbonate-hosted Zn–Pb deposits, from polymetallic Kipushi-type deposits, and lignite and coal deposits in China and Russia. Figures for worldwide Ge reserves are not available. A more detailed description and review about Ge can be found in Moskalyk (2004).

3.1.3. Antimony

Despite a relatively low crustal abundance, Sb-minerals are relatively common in many types of ore deposits. Primary Sb minerals include sulfides, oxides, and mixed sulfides-oxides, and the mineralogy database (Barthelmy, 2012; IMA, 2014) identified approximately 264 minerals that contain Sb. Of these, antimonite, stibnite, valentinite, as well as antimonides and sulpho-antimonides of metals like lead, copper, and silver are the most important ore minerals from which Sb is being recovered. Antimony is also recovered as a byproduct of Au, Ag, Pb, and Zn concentrates. The world's resources of Sb are primarily located in Bolivia, China, Mexico, Russia, and South Africa, with China containing the largest portion (60%). Although Australia has historically produced significant quantities of Sb from the Blue Spec (Western Australia, Hillgrove (New South Wales) and Costerfield (Victoria) deposits, at present there are no active Sb mines in Australia. As of December 2015, Australia's EDR for Sb was 159 kt (GA, 1990–2015). This EDR is from existing deposits where Sb is the main commodity of interest and does not include deposits in which Sb could be produced as a companion metal. A more detailed description and review about Sb can be found in Anderson (2012).

4. World production and use of gallium, germanium and antimony

4.1. Historical world production of gallium, germanium and antimony

The world production of Ga, Ge and Sb has seen mixed trends over time, having in common a strong growth in the past two decades (Fig. 1). The production of a companion metal (Ga, Ge, and Sb) is typically not dictated by changes in its demand, but rather depends on its host's production or demand. If the host demand is modest, companion supply could be constrained. Conversely, if host demand is large, companion extraction may exceed companion demand.

The increasing production trends are largely attributable to economic progress in developing countries and to population growth coupled with industrial development in the world. The demand is also a result of the increased use of companion metals in emerging technologies, such as electronic and solar energy applications (Ga and Ge) and as alloying elements in high-temperature applications (Sb). Although there is a large potential for recovery of Ga, Ge, and Sb from their host metals, the recovered quantities reflect the metal demand in a specific year which, in turn, affects the price.

Gallium can be derived from bauxite and zinc ores (and zinc refinery residues), from industrial sources, Ga scrap materials, and coal fly ash. Today, most of the world's primary Ga is extracted from bauxite during the bauxite-alumina refining, mostly during the Bayer process through accumulation in the caustic liquor and also from the flue dust collected in plants producing aluminium through the electrolytic process. According to the USGS estimate for 2014, the world's primary Ga production was 440 tonnes and the leading producers of impure Ga are China, Germany, and Ukraine. They comprise roughly 50% of world primary production. The leading purchasers of refined Ga are China, Japan, and the US. Some of the impure Ga product is refined onsite, but much of it is sold and processed into Ga metal of differing purities elsewhere.

Most of the Ga contained in bauxite ore is actually disposed of as bauxite residue (as a component of red mud) from the processing of ore to alumina. An increase in Ga demand beyond current capacity could be easily satisfied with the addition or expansion of recovery circuits at existing alumina facilities. It is unlikely that the future supply of Ga will be constrained because of limited Al production. Currently, very little Ga is recovered from recycled

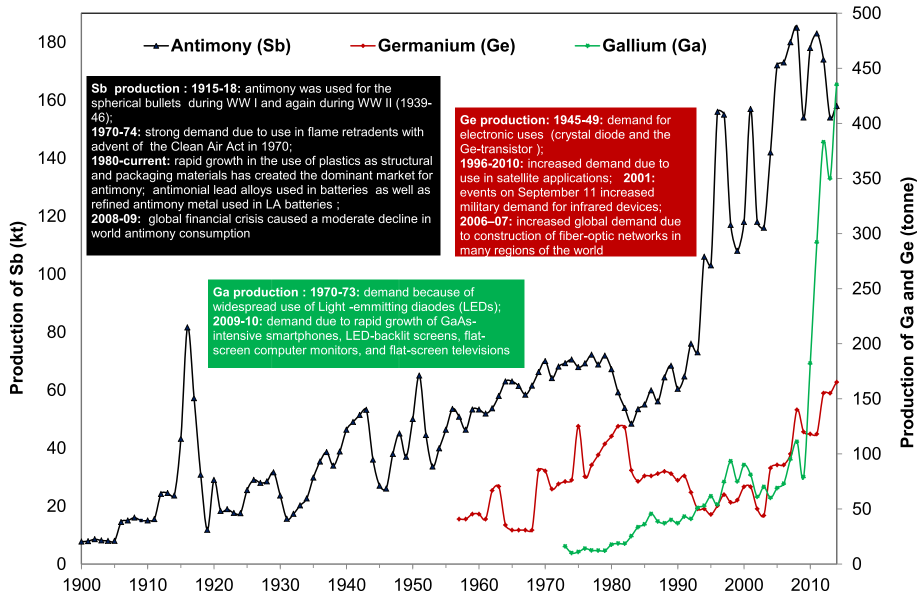

Fig. 1. Historical world production with significant events that were responsible in demand fluctuations.

obsolete materials owing to its oversupply in the market from primary sources, low prices, and a missing recycling infrastructure to capture the wide applications of Ga in electronics (Ciacci et al., 2015).

Approximately 165 tonnes of primary Ge metal in various forms was produced in 2014, of which roughly 70% was recovered from residues generated during the production of refined zinc derived from ores, and about 30% originated from leaching fly ash (USGS, 1994–2015). The vast majority of Ge production came from China and Canada, and other countries with notable Ge production include Belgium, Russia, and the United States. In 2007, Ge metal was in short supply as falling base metal prices had mining companies reduce zinc production, even though there was increased demand for Ge from the fiber optics and infrared technologies industries (Guberman, 2007). More Ge could potentially become available with an improvement of the zinc market. The amount of Ge potentially recoverable from coal fly ash is essentially unlimited, but the commercial recovery of the metal is not currently viable to the extent that it has replaced Ge recovered from Zn concentrates. The amount of Ge recovered from recycling efforts is not a significant contributor to the world supply of Ge.

Antimony was mined as a principal product as well as a companion of base metal ore smelting operations around the world in approximately ten countries in 2012. While most of the world's primary Sb was mined in China (83%), Canada and Russia (4% each) were the next two leading producers. The other producers include Bolivia and the Republic of South Africa. Approximately 160,000 tonnes of primary Sb was produced in 2014, 80% of it in China. The world primary production of Sb grew by 2.8% per year between 2006 and 2010. Secondary Sb accounts for nearly 20% of total supply, mostly as antimonial Pb recovered from and reused in lead-acid batteries (Guberman, 2016).

4.2. Historical prices of gallium, germanium and antimony

Gallium, Ge, and Sb are not traded on international metal exchanges, and prices are commonly directly negotiated between the producers and consumers. Although producer-quoted prices should not be regarded as actual selling prices, they provide an indication on the trend of Ga prices. Prices since 2002 are based on the average value of U.S. imports as provided by the U.S. Census Bureau. Fig. 2 presents a time series trends in prices of these elements, both in nominal and real dollar terms. While Ga and Ge prices have declined over time (in real terms), the trend for Sb is mixed.

Overall, the Ga price appears to have decreased significantly since the 1960s (Fig. 2). It has fluctuated in recent years, with rising prices indicating growing demand, and falling prices indicating a supply response to the previously high price. The 2000–2001 price run-up was primarily due to inventory stocking by the cell-phone supply chain fearful of a shortage (Jaskula, 2014).

Germanium prices started to decline progressively in the late 1950s, and by 1966 bottomed out at $175 per kilogram of metal, the lowest price ever quoted (Fig. 2). This price rose in 1979 because of increased use and tight supply. In 1998, Ge prices increased despite an oversupply that resulted from slight decreases in world demand for optical fibres and polyethylene terephthalate and an increase in total supply owing to greater amounts of recycling and continued releases of Ge from national stockpiles. There was a perceived shortage of Ge in 2007 which was reflected in its price, which increased by nearly 30% to $1240 per kilogram for zone-refined Ge at year end 2007 from about $950 per kilogram at the beginning of the year. In 2008, the price of the metal continued to rise, and by year end 2008, was $1490 per kilogram.

During the past five decades, Sb has seen a few extreme price swings. Changes in Chinese government policies have been the most important factors affecting Sb prices since the early 1990s, as well as spikes or declines in the American and (or) foreign demand for Sb and changes in the pattern of world production. For example, in 1994, Sb prices surged strongly; this was attributed initially to shipment delays from the world's largest producers by extensive flooding in the regions of China's major Sb mines and smelters. Between 1995 and mid-2002 prices followed the declining trend but rose considerably during the third and fourth quarters of 2002. In recent years prices have fluctuated, increasing in times of growing demand, and dropping in a supply response to the high price. Fig. 2 shows the price of Sb, averaged over the decades, going back to 1900. Price spikes were very much linked to wars and supply disruptions from China, but often the price impact was short lived.

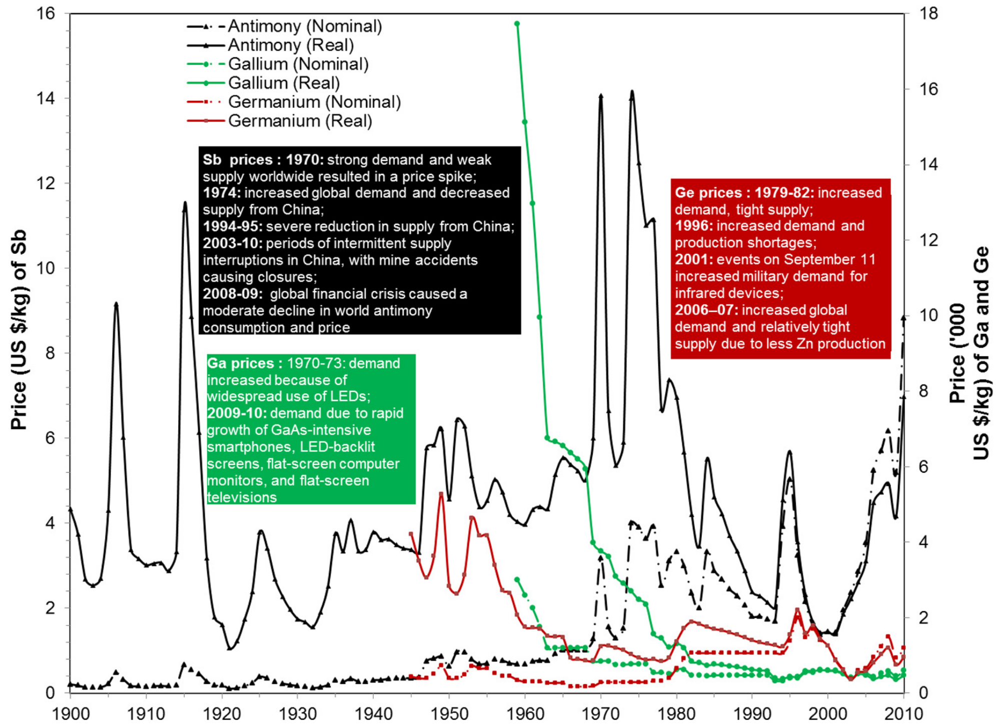

Fig. 2. Historical prices of Ga (99.9999% purity), Ge and Sb (real US dollars normalised the year 2000).

4.3. Historical use of Ga, germanium and antimony

4.3.1. Gallium uses and rates of use

Gallium end-use categories and typical lifetimes (numbers in parenthesis) are provided in Fig. 3 for the US and the world. The US use is different from that of the rest of the world in that in 2012 more than 99% of the contained Ga was used in the form of GaAs to manufacture optoelectronic devices, whereas integrated circuits dominated the world use. In 2012, U.S. use of Ga for use in ICs was 7% less than in 2011 due to a decreased rate of growth of GaAs-rich smartphones (Jaskula, 2014). These applications of Ga (Fig. 3) are important for aerospace, telecommunications industries as well as in the components that are used in high-performance computers and smartphones (cellular telephones that have advanced personal computer-like functionality).

Gallium is widely used in microelectronic components in a wide variety of products, including GaAs, GaN, and GaP direct band-gap semiconductors. These compounds also have the property to change electricity directly into laser light and are used in the manufacture of light emitting diodes (LEDs). Gallium has a potential application in photovoltaics, where it can be used as a component of thin-film coating Cu-In-Ga selenide (CIGS) on solar panels. Due to better performance of power transistors made with GaN (over those with GaAs), the type and number of products that use

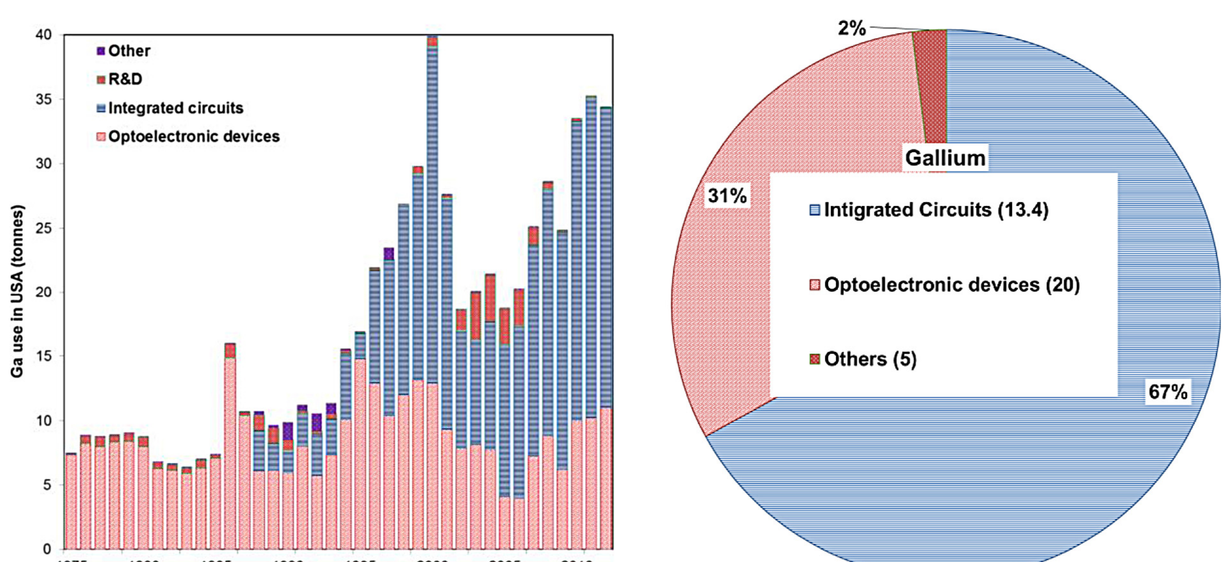

Fig. 3. Sectoral use of Ga in the US (left) and the world average for 2012 (right) [numbers in parenthesis are the average lifetimes in years of the in-use products containing Ga].

advanced GaN-based transistors are expected to increase (USGS, 1994-2015). GaAs is well suited for photovoltaic cells but its application in consumer goods is limited by the high cost of producing GaAs crystals, a necessary component of the cell.

4.3.2. Germanium uses and rates of use

The various end-use categories and their typical lifetimes (numbers in parenthesis) for Ge are illustrated in Fig. 4. The use of Ge is slightly different in the US than in the rest of the world in that more Ge was used in infrared optic systems (50%) to see in the dark or through fog or smoke. This property has led to Ge's widespread military use for surveillance and weapon sighting operations. Global Ge use is likely to increase during the next several years owing to growth in Ge-based optical blanks and windows incorporated in infrared devices used by military and law enforcement agencies.

The largest use of Ge is in the semiconductor industry and in various electronic devices. Other applications include fiber optics communications and polymerization catalysts. When doped with small amounts of As, Ga, In, Sb or phosphorus, Ge is used to make transistors for use in electronic devices. Germanium is also used to create alloys and as a phosphor in fluorescent lamps. Some Ge compounds seem to be effective in killing certain types of bacteria, and are currently being studied for use in chemotherapy.

4.3.3. Antimony uses and rates of use

The various end-use categories of Sb (flame retardants and transportation including batteries), ceramics and glass, chemicals, and other uses (ammunitions, cable coverings, fireworks, metal castings, paper, pigments, rubber products, sheet, pipe, and type metal) and their typical lifetimes (number of years in parenthesis) are illustrated in Fig. 5. Antimony trioxide has been used to enhance the flame-retardant properties of plastics, rubber and textiles, and other combustibles. Lead-Sb alloys are being used in ammunition, antifriction bearings, cable sheaths, corrosion-resistant pumps and pipes, roof sheet solder, and tank lining. Antimony is also used as a decolorizing and refining agent in the manufacture of some forms of glass, such as optical glass.

Growing demand for Sb applies to both its metallurgical (automotive production, replacement and construction) and non-metallurgical applications (e.g., polymers, PETs [polyethylene terephthalate], PVC, CRT and solar glass). The use of Sb in flame retardants is expected to remain its principal use, globally as well as in the United States (Carlin, 2014; USGS, 1994–2015).

Use of Sb rose 3.1% per year between 2000 and 2010, reaching 199,500 tonnes in 2010, due to its use in lead alloys, flame retardants, and as a catalyst and heat stabiliser in polymers. The demand for lead-acid batteries in automotive applications and fabricated lead products for construction in emerging economies has increased Sb use worldwide. Future demand for Sb is likely to grow due to its demand in the flame retardants sector and in the lead acid battery sector, which together accounted for 80% of Sb use worldwide in 2010.

5. Geological stocks of gallium, germanium and antimony in Australia

As discussed earlier in this contribution, many critical commodities are produced as companion metals during the extraction of major commodities such as Cu, Zn, Pb and Ni. Skirrow et al. (2013) discussed the geological distribution of a range of critical commodities in ore systems, finding that many critical commodities, including Ga, Ge and Sb, are concentrated in the ores of major commodities and are or can be extracted during processing of these ores. Subsequent to the publication of Skirrow et al. (2013), a large dataset of mineralised samples, including many ore samples, has become available as part of the OSNACA (Ore Samples Normalised to Average Crustal Abundance) project at the University of Western Australia (Brauhart et al., in review; http://www.cet.edu.au/research-projects/special-projects/projects/osnaca-ore-samples-normalised-to-average-crustal-abundance). Analysis of the OSNACA sample set was undertaken at the Bureau Veritas Australia lab in Perth. Although this dataset is not comprehensive, it does provide a guide to the concentrations of critical commodities in a range of Australian ores. The OSNACA data have been supplemented by analyses of Certified Reference Material (CRM) ore standards from Ore Research Pty Ltd (www.oresearch.com.au; hereafter referred to as OREAS samples). Where certified analyses of critical commodities were not provided for CRMs, additional analyses were commissioned from the Analabs Pty Ltd lab in Brisbane. For Ga, Ge and Sb, concentrations were determined using ICP-MS analysis following four acid dissolution for Ga and peroxide fusion for Ge and Sb. The complete OSNACA/OREAS dataset

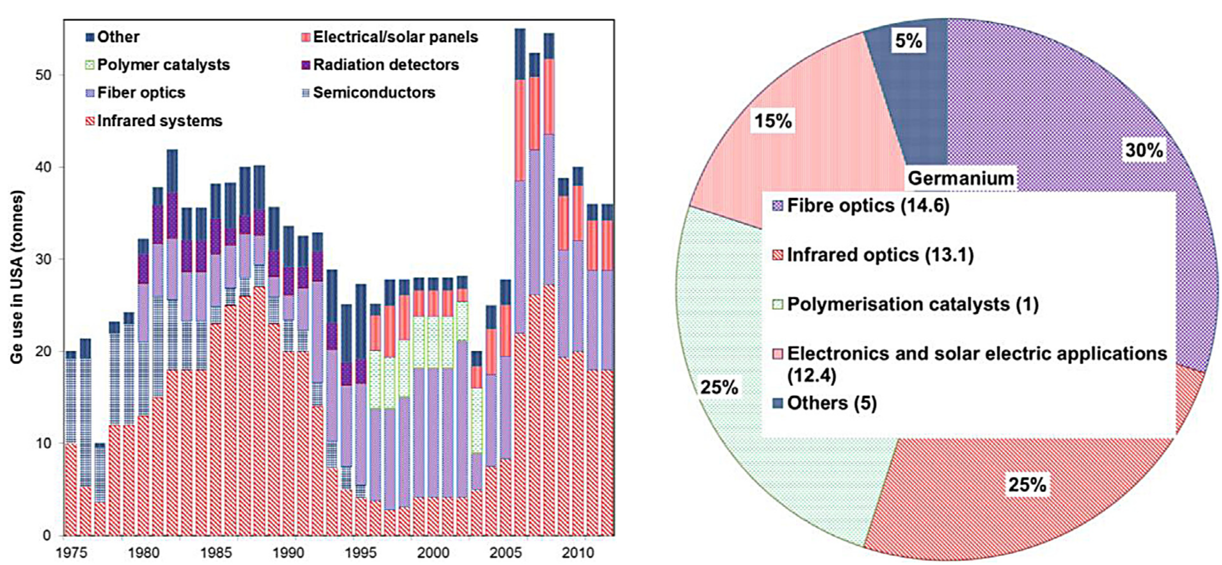

Fig. 4. Sectoral use of Ge in the US (left) and rest of the world (average 1998–2012) (right) [numbers in parenthesis are the average lifetimes of the in-use products containing Ge].

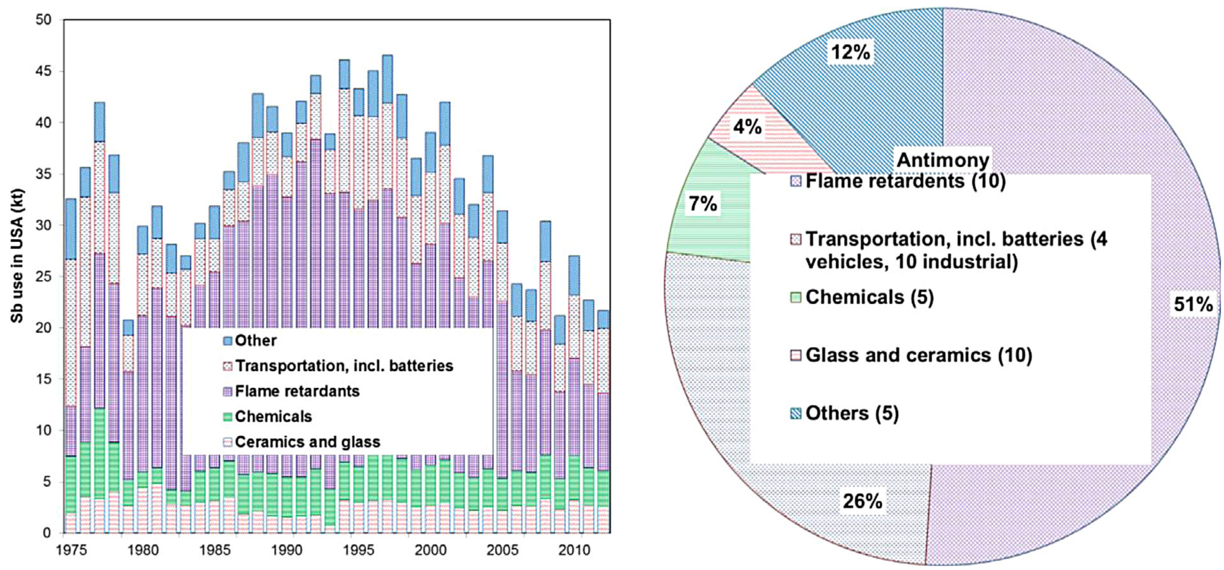

Fig. 5. Sectoral use of Sb in the US (left) and rest of world for 2012 (right) [numbers in parenthesis are the average lifetimes of the in-use products containing Sb].

contains 322 analyses of moderately to strongly mineralised samples from of wide range of deposit types. Analyses from this dataset of Ga, Ge and Sb, combined with analyses of major commodities are available as an electronic data package.

5.1. Methods of geological resource estimates of gallium, germanium and antimony

The combined OSNACA/OREAS dataset and trace metal data from other sources were combined with EDR data (current at 31 December 2015) extracted from Geoscience Australia's OZMIN database and other databases to estimate geological stocks of Ga, Ge and Sb in Australia. Prior to this analysis, however, the general Ga, Ge and Sb grades of various deposits and deposit classes were combined with restrictions posed by the metallurgical processes used to extract these metals to limit the deposit classes considered in the analysis to those most likely to have viable production. Fig. 6 illustrates grade data from the OSNACA/OREAS dataset for Ga, Ge and Sb according to the major metal assemblage. Deposits dominated by the Zn-Pb-Ag major metal assemblage have the most consistent enrichment of Ga, and, in particular, Ge and Sb. This is consistent with metallurgical practices that extract Ga and Ge from Zn (sphalerite) concentrates (Table 2). In addition to sphalerite concentrates, Ga and Ge are currently extracted from the processing of bauxite for Al, and coal fly ash, respectively. The metallurgy of Sb differs from Ga and Ge in that most Sb is currently extracted as a major commodity from stibnite ores, primarily in China. However, Sb has also been produced as a coproduct of Ag production from the Sunshine operations in the United States (Anderson, 2012). During roasting of sulphide concentrates, Sb can be recovered as Sb_2O_3 from flues, condensing pipes, baghouses and Cottrell precipitators (Anderson, 2012), raising the possibility of recovery during the processing of sulphide concentrates. Hence, in addition to considering EDR data from stibnite ores, we also consider the presence of Sb in polymetallic, particularly Zn-Pb-Ag, ores.

As all three metals of interest could possibly be recovered from Zn-Pb-Ag-rich ores and concentrates, estimates of geological stocks of Ga, Ge and Sb have used the geochemical relationships of each metal to Zn, and the EDR of Zn. To make these estimates, the ratio of Zn to the metal of interest (e.g., Zn/Ga, Zn/Ge and Zn/Sb) was calculated for each sample from the OSNACA/OREAS dataset. Using samples in which Zn content exceeded 1%, the Zn/Ga, Zn/Ge and Zn/Sb characteristics were determined for each deposit (arithmetic mean) and deposit class (geometric mean: deposits were classed according to Huston et al., 2016). These data were used with Zn EDR data to provide the best estimate of geological stock of the metals of interest. In this analysis, the geometric mean of the relevant deposit class was used for deposits where no compositional data are available. Where data were available, the deposit arithmetic average was used to determine geological stocks. For some deposits (Rosebery, Mount Isa, Broken Hill, Cannington and Dry River South) data from outside of the OSNACA/OREAS dataset allowed a better estimate for some commodities. In addition to these "best" estimates, minimum and maximum estimates for each deposit class were made using values of Zn/Ga, Zn/Ge, and Zn/Sb two standard deviations above and below the geometric mean, respectively.

In addition to the above estimates, "best" estimates of geological stocks of Ga from bauxite and "red slime" were made using EDR and geometric mean concentration data for both of these materials. Minimum and maximum stocks were estimated using concentration values two standard deviations below and above the mean, respectively. Similarly, "best", minimum and maximum stocks of Ge were estimated using concentration data of Ge in coal fly ash. Economically demonstrated resource data for stibnite mines from the OZMIN dataset (at 31 December 2016) were also included in the geological stock estimates for Sb.

5.2. Estimate of geological stocks of gallium

As shown in Fig. 6, unlike many other commodities, the concentration of recoverable Ga is only a few times that of its crustal abundance. Recovery of Ga is highly dependent upon recovery of major commodities; in this case Al and Zn. Zinc-Pb-Ag deposits are not particularly enriched in Ga relative to other deposits or the average crustal abundance. In fact, the highest concentrations of Ga in the OSNAC/OREAS dataset are from sn-W-Ta deposits, in particular sn-Ta pegmatite deposits. The other Ga-rich sample is from the Mount Weld carbonatite deposit, suggesting that this and related deposits could be potential Ga sources, although the technology for commercial extraction does not exist.

As shown in Table 3, which indicates total Australian Ga stocks of 477.4 kt, the largest portion of Australian Ga stocks resides in bauxite deposits, bauxite residue (red mud) and Zn-rich deposits

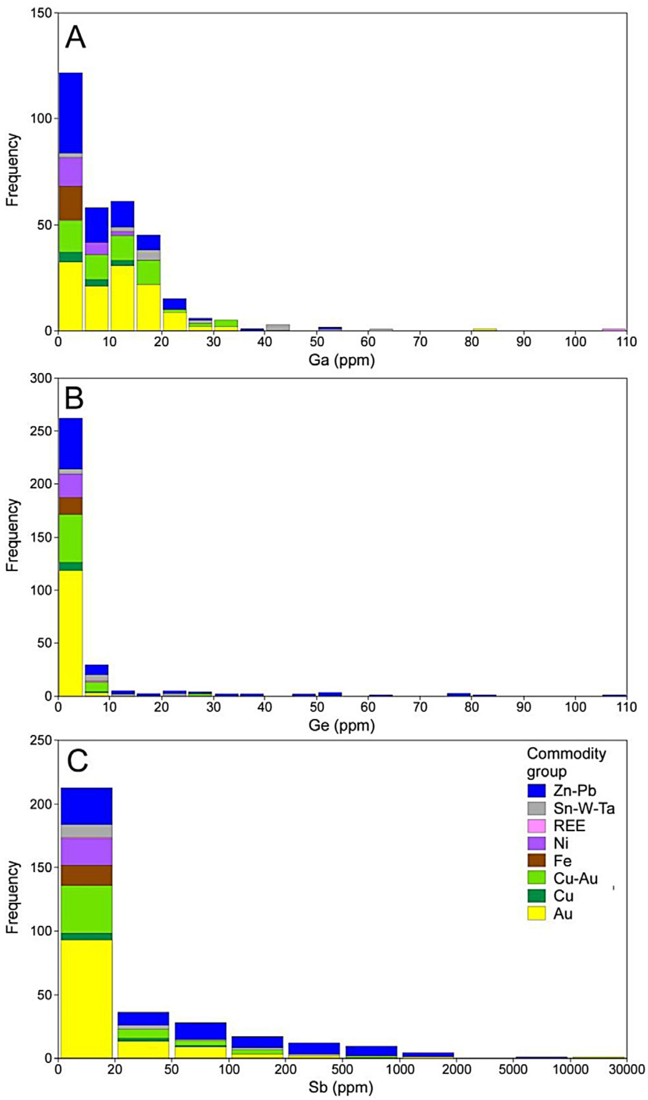

Fig. 6. Histograms showing the concentrations of Ga (A), Ge (B) and Sb (C) according to deposit class (classes after Huston et al., 2016) from the OSNACA/OREAS dataset.

account for a relatively small (2.4 kt) part of these stocks. Due to large quantities, bulk commodities such as iron ore and coal, are large repositories of Ga and Ge (Table 3), but this Ga is not likely to be recovered from these ores due to low concentrations and the lack of recovery methods. Hence, recoverable Australian and global Ga stocks are hosted by bauxite and, to a lesser extent, Zn-Pb-Ag deposits, and Ga production will be highly dependent upon Al and Zn production into the foreseeable future.

5.3. Estimate of geological stocks of germanium

Fig. 6B shows that virtually all analyses with Ge concentrations above 10 ppm are from Zn-Pb-Ag deposits, with a highest value of 105 ppm. As Ge is currently recovered from sphalerite concentrates, in terms of grade and extractive metallurgy, Zn-Pb-Ag deposits are likely the only sources of Ge from metallic ore deposits. In detail, however, there appear to be systematic differences in Ge tenor between different classes of deposits. Fig. 7 shows that samples from deposits formed from low temperature (<200 °C), oxidised fluids (those coloured brown: Mississippi Valley-type) and siliciclastic-carbonate (McArthur-type of Cooke et al., 2000) tend to have significantly higher Ge concentrations than samples from deposits formed from higher temperature (>200 °C), reduced ore fluids (green symbols: siliciclastic-mafic (Selwyn-type of Cooke et al., 2000), volcanic-hosted massive sulphide, intrusion-related (e.g., skarn) and orogenic base metal (Cobar-style)). This probably relates to the hydrothermal geochemistry of Ge, and probably suggests that it is more readily transported in and/or deposited from low temperature and/or oxidised fluids. The most important consequence of this relationship is that Australia's geological stocks of Ge are dominated by the giant Zn-Pb-Ag deposits of the North Australian zinc belt, which contribute over 95% of Australia's non-coal Ge stocks.

We have identified coals of NSW and Queensland as hosting the elements of interest in this study based on their inorganic elemental associations (major and potential companion elements). Ge reserves and resources in brown coal have been considered by Frenzel et al. (2014), suggesting considerable potential for Australia given the scale of coal deposits across eastern Australia. Germanium can also be extracted from coal fly ash, although at present this is not done in Australia. This process represents a second potential source of Ge, with a "best" estimate of geological stocks of 17.5 kt. The Ge concentration of fly ash has been reported by Swaine (1965) which presents a great variability (i.e., 6–2000 ppm) in the reported concentrations and thereby estimation of exact value is very difficult.

5.4. Estimated geological stocks of antimony

Unlike Ga and Ge, the main geological stocks of Sb are not as a companion metal, but are from deposits in which Sb would, if mined, be extracted as a major commodity. Australia's current EDR for Sb is 159.3 kt, most of which is from deposits at Costerfield, Victoria and Hillgrove, New South Wales (Geoscience Australia's OZMIN database). Based upon our analysis, the potential Sb stock as a companion metal totals 77.0 kt with the largest potential stocks at Cannington (27.8 kt), Mount Isa (16.8 kt) and McArthur River (HYC: 10.9 kt). Although these estimates are uncertain, they do indicate potential production Sb from these sources, particularly Cannington, depending upon if the Sb can be recovered during roasting of the concentrates.

Although Sb is most consistently concentrated in Zn-Pb-Ag deposits, other types of deposits can also be enriched in Sb. The highest Sb grade (>3% Sb) in the OSNACA/OREAS database is from a stibnite-rich sample from Wiluna orogenic gold deposit in Western Australia. Many orogenic gold deposits contain significant stibnite, locally attaining ore grades for Sb. If sufficient grades and tonnages are demonstrated in such deposits, they can be mined for Sb as a major commodity, with Au as a co- or by-product (e.g., Hillgrove and Costerfield). In addition to these orogenic gold deposits, some sn-W deposits and Cu-Au-rich volcanic-hosted massive sulphide deposits can have locally high concentrations of Sb. Given the metallurgy of these ores and the patchy character of the high Sb concentrations, it is unlikely that such deposits will contain viable Sb stocks. Hence, like Ga and Ge, potential Sb stocks as a companion metal are restricted to Zn-Pb-Ag deposits.

There appears also to be increased code-based reporting of Sb in Australian mineral deposits, for example at Hillgrove, NSW (0.349 Mt, 1.6% Sb) and Blue Spec, WA (0.646Mt, 1.2% Sb), suggesting greater recognition of Sb as an extractable commodity in Australia. The ability to scale up Sb production in the future appears more limited than for Ga and Ge, given that the majority of our calculated maximum is already being met. Our data may still be used to infer the extent to which certain host commodities or deposit types will be important for consideration, should Australia pursue greater Sb production.

Table 3
Australian estimates of Ga, Ge and Sb resources by deposit type according to OSNACA/OREAS dataset.

Deposit type	Zn EDR (Mt)	Ga best estimate (kt)	Zn/ Ga – 2SD	Ga maximum (kt)	Zn/Ga + 2SD	Ga minimum (kt)
Intrusion-related base metal	0.66	0.416		0.416		0.416
Orogenic base metal	0.28	0.011	4799	0.059	87296	0.003
Sediment-hosted – clastic dominated – siliciclastic-carbonate	49.43	1.722	6470	7.640	187194	0.264
Sediment-hosted – clastic dominated – siliciclastic-mafic	4.81	0.053	767	6.272	10132228	0.000
Sediment-hosted – Mississippi Valley-type	0.54	0.012	7388	0.074	1104814	0.000
Volcanic-hosted	5.38	0.194	415	12.982	6921372	0.001
Sub-Total	61.11	2.408		27.442		0.685
	EDR (Mt)	Ga best estimate (kt)		Ga maximum (kt)		Ga minimum (kt)
Bauxite	6464	416	–	639		–
Bauxite Residue (red-mud or red slime)	788	59	–	70		–
Grand Total	–	477	–	736		0.685
Deposit type	Zn EDR (Mt)	Ge best estimate (kt)	Zn/ Ge – 2SD	Ge maximum (kt)	Zn/Ge + 2SD	Ge minimum (kt)
Intrusion-related base metal	0.66	0.188		0.188		0.188
Orogenic base metal	0.28	0.002	24686	0.011	627276	0.000
Sediment-hosted – clastic dominated – siliciclastic-carbonate	49.43	10.576	768	64.356	34437	1.435
Sediment-hosted – clastic dominated – siliciclastic-mafic	4.81	0.043	9364	0.514	1960203	0.002
Sediment-hosted – Mississippi Valley-type	0.54	0.032	1149	0.474	86980	0.006
Volcanic-hosted	5.38	0.086	6654	0.809	342258	0.016
Sub-Total	61.11	10.927		66.351		1.648
Deposit type	Zn EDR (Mt)	Sb best estimate (kt)	Zn/ Sb – 2SD	Sb maximum (kt)	Zn/Sb + 2SD	Sb minimum (kt)
Intrusion-related base metal	0.66	6.780		6.780		6.780
Orogenic base metal	0.28	0.211	281	0.999	6333	0.044
Sediment-hosted – clastic dominated – siliciclastic-carbonate	49.43	32.831	–	177.817	44014	1.123
Sediment-hosted – clastic dominated – siliciclastic-mafic	4.81	30.094	12	396.964	504092	0.010
Sediment-hosted – Mississippi Valley-type	0.54	0.147	427	1.276	2365628	0.000
Volcanic-hosted	5.38	6.941	45	119.445	19403	0.277
Sub-Total	61.11	77.004		703.281		8.234
Deposit type	Sb EDR (kt)	Ga best estimate (kt)		Sb maximum (kt)		Sb minimum (kt)
Sb-Au deposit or stibnite (Hillgrove, Wild Cattle Creek, Blue Spec, etc.)	159.3	2.17	–	–	–	–
Grand Total	–	79.174	–	–	–	–

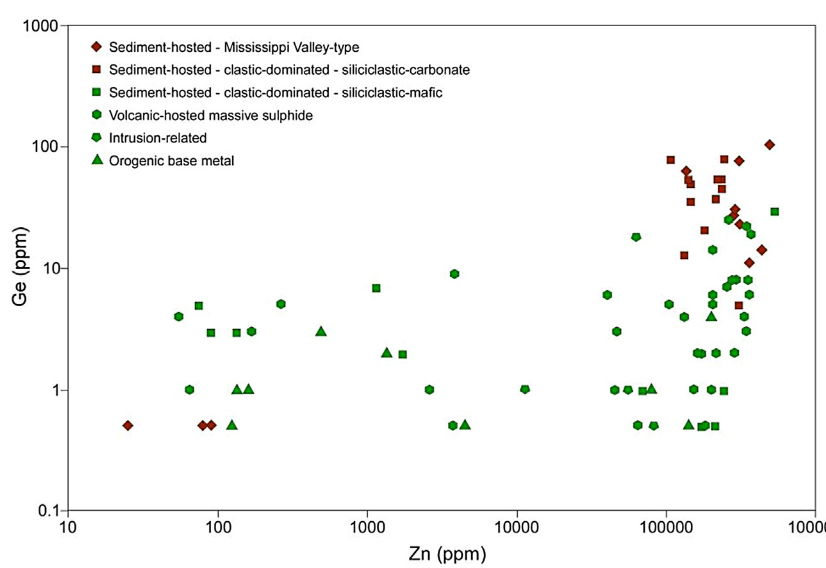

Fig. 7. Scattergram showing Zn and Ge concentrations from Zn-Pb-Ag deposits from the OSNACA/OREAS dataset. Brown symbols indicate analyses from low temperature, oxidised deposits, and green symbols indicate analyses from high temperature, reduced deposits. (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

5.5. Australian Economic Demonstrated Resources (EDRs) and production of major host ore or metals

Fig. 8(a)–(f) show recent trends of production, use, and reported mineral resources for host metals of Ga, Ge, and Sb in Australia. Of particular note is that exploration success, technological improvement, and adaptive market conditions have ensured stable or growing resource estimates across the board, despite unprecedented levels of production. The strong growth in demonstrated resources suggests growth also in Australia's endowment of extractable companion metals, and increased host metal production suggests greater processing capacity for companions. A distinction is made between currently economic quantities (i.e., reserves) and stocks that are likely to become economic in the future but are not economic under current prices or cut-off grades (resources). The ratio of reserves/resources has tended to be stable or to increase for most metals, suggesting favourable economic conditions for recovery of these host metals. Iron ore is an exception, where resource growth has significantly outstripped growth in iron ore reserves.

We see also in Fig. 8 that domestic demand has not kept pace with production, reflecting Australia's increased role as a mineral export nation. Given the high-tech applications for Ga and Ge (Figs. 3–5), it seems likely that Australia's domestic use of these critical metals may have originated from Australian deposits, even though they have been exported overseas for processing, to later be imported again in finished products. Further analyses of critical metal material flows would provide greater clarity on this, and are expected to be the subject of future research by the authors.

Although there is greater potential for Ga production in Australia, there is no production due to low prices of Ga in the international market. Historically, Rhodia Pinjarra operated a facility 5 km south of Alcoa's Pinjarra alumina refinery to extract Ga. The production capacity at that time was about 30 tpa of gallium. GEO Gallium, a wholly-owned subsidiary of GEO Specialty Chemicals, Inc. (Germany), also had plans to produce Ga but has announced that it had deferred investing in its Pinjarra extraction facility and had no plans to restart it as their principals went bankrupt. On the other hand, Nyrstar's Port Pirie operations receives ores containing only 0.5–2 tpa of Ga production potential (Nyrstar, 2014) and hence it was not viable for Nyrstar to invest in extracting gallium.

Considering Nyrstar's recent announcement of upgrades to its Hobart operations (Nyrstar, 2014), it is expected that some production of Ge (as well as In) will take place in Australia. The Ge comes through parageothite as an impurity, approximately 10–25 t annually. Germanium originates from century zinc concentrate, treated in Nyrstar Hobart, and all Ge ends up in the blast furnace slag. The recovery through the site is approximately 70%. Nyrstar have done tests to remove the Ge in concentrate (~1%) by co-precipitation with iron. Our estimates suggest that there is a very high production potential of Ge in Australia every year. The considerable variation in this estimate mostly comes from the uncertainty of Ge concentrations in Australian black coal, given grade estimates in the range of 6–2000 ppm Ge for black coal ash.

The data suggest that a similar story can also be told for Sb. However, Australia's potential in regards to Sb appears more realised than for Ga and Ge, as Geoscience Australia reports annual production of around 16 kt Sb ore, contributing approximately 10% of world production (Fig. 1). There appears also to be increased code-based reporting of Sb in Australian mineral deposits, for example at Hillgrove, NSW (0.349 Mt, 1.6% Sb) and Blue Spec, WA (0.646 Mt, 1.2% Sb), suggesting greater recognition of Sb as an extractable commodity in Australia.

Fountain (2013) conducted a review on the behaviour of 'companion metals' or 'byproduct metals' or 'minor elements' in smelters and refineries, explaining the deleterious effects of each of them. According to this study, the issues caused by the minor

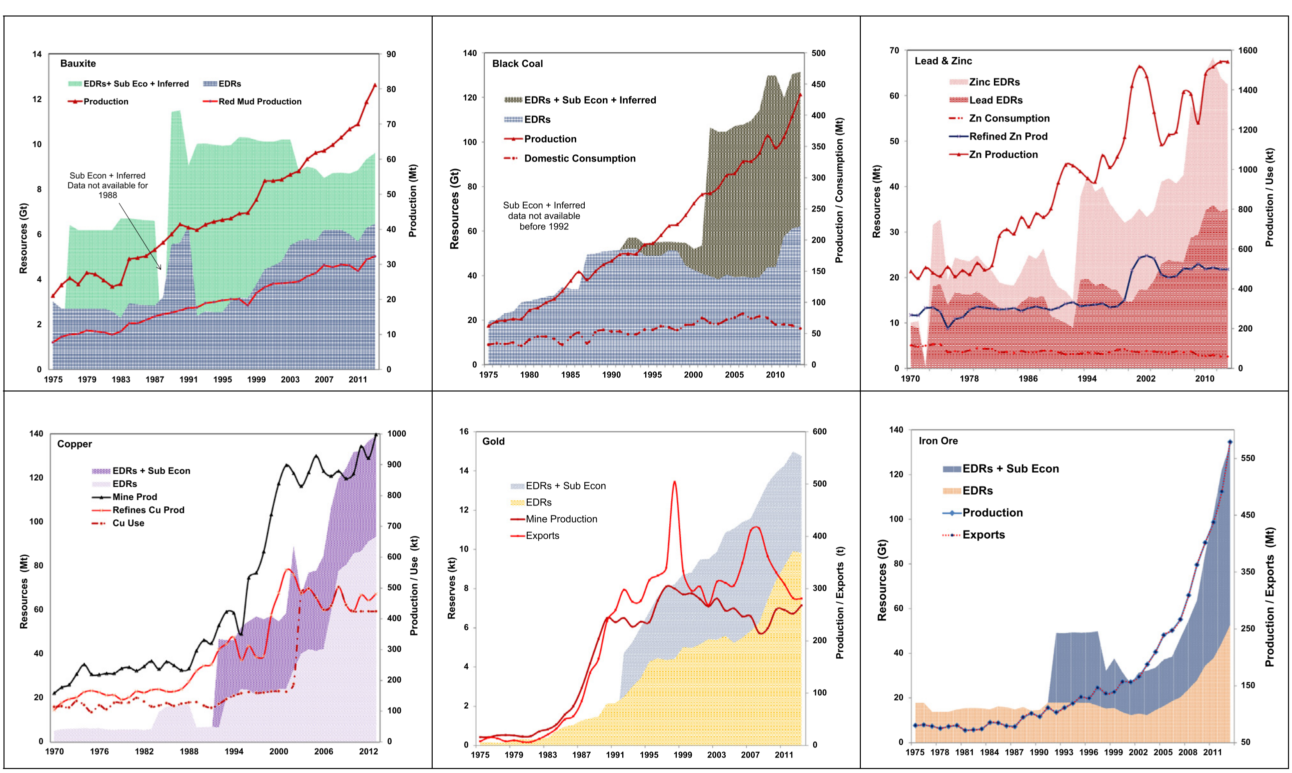

Fig. 8. Australian stocks of host metal (mineral) resources (EDRs + Sub-economic + Inferred resource).

elements in ore concentrates include effects on the occupational health of smelter workers, increased environmental disposal costs, increased operating costs and reduced cathode copper quality. For example, copper smelters include penalties if the concentrations of elements, such as arsenic, mercury and fluorine, exceed stipulated limits. It is less commonly known that the behaviour of these elements in the smelters can depend on the processes used to treat the concentrate. The principle behind the penalties is to compensate the smelters and refineries for additional costs caused by the presence of the penalty elements, but these costs will vary depending on the process used and the location of the smelter.

6. Conclusions

In this paper we have carried out estimates for the amounts of Ga, Ge, and Sb as companion metals in Australian ores. This paper has compiled comprehensive and first-of-its-kind data sets of reported host and companion metal resources in Australia for 2014 based on formally reported minerals. It is clear that there is considerable potential to expand the resource base of existing deposits that could contain companion metals such as Ga, Ge, and Sb.

We also demonstrated the approach that can be taken to convert those estimates into potential quantities of companion metals that can be produced from those deposits. Much of the necessary information needed to validate these estimates and to compare them with global stocks and flows of these metals is not commonly available; possible explanations include the lack of recognition of potential value, the focus by companies on host metals, lack of sufficient data, or the smaller market size and/or prices for companion metals. Nonetheless, the results presented above indicate that the value of the companion metals in Australian ores could be substantial, that it is not being fully taken advantage of, and that opportunities may exist for a wider scope of natural resource extraction than has traditionally been assumed.

Appendix A. Supplementary data

Supplementary data associated with this article can be found, in the online version, at http://dx.doi.org/10.1016/j.oregeorev.2016.11.020.

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The exposure of global base metal resources to water criticality, scarcity and climate change

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ABSTRACT

Mining operations are vital to sustaining our modern way of life and are often located in areas that have limited water supplies or are at an increased risk of the effects of climate change. However, few studies have considered the interactions between the mining industry and water resources on a global scale. These interactions are often complex and site specific, and so an understanding of the local water contexts of individual mining projects is required before associated risks can be adequately assessed. Here, we address this important issue by providing the first quantitative assessment of the contextual water risks facing the global base metal mining industry, focusing on the location of known copper, lead, zinc and nickel resources.

The relative exposure of copper, lead-zinc and nickel resources to water risks were assessed by considering a variety of spatial water indices, with each providing a different perspective of contextual water risks. Provincial data was considered for water criticality (CRIT), supply risk (SR), vulnerability to supply restrictions (VSR) and the environmental implications (EI) of water use. Additionally, watershed or sub-basin scale data for blue water scarcity (BWS), the water stress index (WSI), the available water remaining (AWaRe), basin internal evaporation recycling (BIER) ratios and the water depletion index (WDI) were also considered, as these have particular relevance for life cycle assessment and water footprint studies. All of the indices indicate that global copper resources are more exposed to water risks than lead-zinc or nickel resources, in part due to the large copper endowment of countries such as Chile and Peru that experience high water criticality, stress and scarcity. Copper resources are located in regions where water consumption is more likely to contribute to long-term decreases in water availability and also where evaporation is less likely to re-precipitate in the same drainage basin to cause surface-runoff or groundwater recharge.

The global resource datasets were also assessed against regional Köppen-Geiger climate classifications for the observed period 1951–2000 and changes to 2100 using the Intergovernmental Panel on Climate Change's A1FI, A2, B1 and B2 emission scenarios. The results indicate that regions containing copper resources are also more exposed to likely changes in climate than those containing lead-zinc or nickel resources. Overall, regions containing 27–32% (473–574 Mt Cu) of copper, 17–29% (139–241 Mt Pb + Zn) of lead-zinc and 6–13% (19–39 Mt Ni) of nickel resources may have a major climate re-classification as a result of anthropogenic climate change. A further 15–23% (262–412 Mt) of copper, 23–32% (195–270 Mt) of lead-zinc and 29–32% (84–94 Mt) of nickel are exposed to regional precipitation or temperature sub-classification changes. These climate changes are likely to alter the water balance, water quality and infrastructure risks at mining and mineral processing operations. Effective management of long-term changes to mine site water and climate risks requires the further adoption of anticipatory risk management strategies.

1. Introduction

The mining industry spans all hydrological contexts and climate regions, with these contexts influencing the water risks facing mining operations and the potential for the industry to impact surrounding ecosystems, industries and communities. Access to water is a potential

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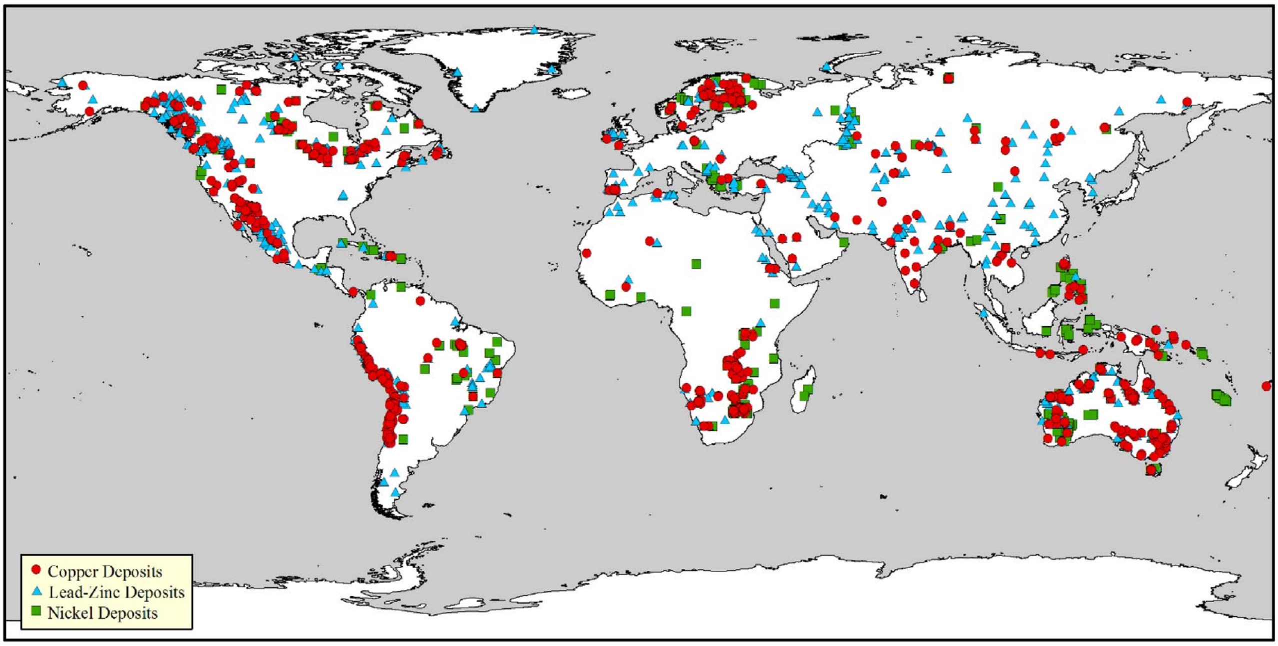

Fig. 1. Location of copper, lead-zinc and nickel resources considered in this study. Maps showing mineral deposit types and operating status are shown in electronic supplementary Figs. S.1–S.5. Raw data provided in supplementary Tables S.24–S.26 and in Google Earth format (.kmz).

constraining factor on mineral resource development, regardless of the climate and absolute water scarcity of a region. The presence of other competing water users, such as agriculture, may limit the ability to allocate water resources to the mining industry (e.g. Shang et al., 2016). Concerns over water may also change community support and reduce a mine's perceived social license to operate (Wessman et al., 2014). Assessing these risks requires the use of systems approaches that can integrate mine site water balances, catchment hydrology and the water use requirements of regions.

Mine sites utilise water in a range of processes, such as mineral processing and dust suppression, and the overall water requirements are highly variable due to factors such as: the local climate, ore mineralogy and grade, the scale of infrastructure and ore processing, and the extent of tailings dewatering and water recycling (Gunson et al., 2012; Mudd, 2008; Northey et al., 2013, 2014a, 2016). The local nature of mine site water use impacts has impeded the ability to produce global scale assessments of the water risks associated with the industry. Previous research has outlined global estimates of the water withdrawals associated with non-fuel mining (Gunson, 2013), however drawing meaningful conclusions requires understanding where this water use occurs. Improving the outcomes of these studies requires knowledge of how the spatial distribution of the mining industry relates to local contexts and environmental pressures. Global assessments have been conducted to assess the distribution of the mining industry in relation to biodiversity and conservation areas (Durán et al., 2013; Murguía et al., 2016). However, to date there have been no quantitative global assessments of the contextual water risks facing the mining industry.

This article presents a detailed assessment of the spatial distribution of known base metal resources in relation to a variety of water risk and impact indices. The assessment focuses on copper, lead-zinc and nickel resources as these metals are vital for modern infrastructure and are expected to have continued or growing demand into the future (Daigo et al., 2014; Elshkaki et al., 2016; Kleijn et al., 2011). The exposure of regions containing these resources to climate change has also been assessed by considering regional data for Köppen-Geiger climate classifications and how these may evolve with climate change (Kottek et al., 2006; Rubel and Kottek, 2010). The information and data provided by this study may form a basis for further assessment of the global base metals industry to understand climate change adaptation requirements, water footprint or life cycle impacts, and expected changes to mine site water balance, water quality and infrastructure risks.

2. Methods and data sources

2.1. Copper, lead-zinc and nickel resource datasets

This study utilises datasets for individual copper (Mudd et al., 2013), lead-zinc (Mudd et al., 2017) and nickel (Mudd and Jowitt, 2014) resources that were developed over several years and are primarily based upon the mineral resource reporting of individual exploration and mining companies. Typically these resource disclosures are made as part of a company's statutory or financial reporting obligations. The copper dataset includes resource data for 730 deposits containing 1781 million tonnes (Mt) of copper (363,270 Mt ore @ 0.49% Cu; Mudd et al., 2013). The lead-zinc dataset includes resource data for 852 deposits representing a combined resource of 837 Mt lead-zinc (50,882 Mt ore @ 1.64% Pb + Zn; Mudd et al., 2017). While, the nickel dataset includes data for 476 deposits containing 293 Mt of nickel (61,365 Mt ore @ 0.48% Ni; Mudd and Jowitt, 2014). Individual deposits in these datasets have been classified according to primary and/or dominant mineral deposit types (e.g. Jowitt et al., 2013). Individual resources in the datasets have also been classified as being either an undeveloped deposit or a recently operating mine-site, based upon the status of the deposit for the year the dataset was compiled (Copper: 2010; Lead-Zinc: 2013; Nickel: 2011). The datasets provide minimum estimates of known resources and so the results presented represent the minimum exposed resource to the various water and climate risks.

Coordinate data (latitude and longitude) for individual deposits within the datasets have been added and crosschecked from a range of sources, including government geological organisations (the United States Geological Survey, the British Columbia Geological Survey, Geoscience Australia, Geological Survey of Finland, etc.), online resources (company websites, mindat.org, dmgeode.com, etc.), consultant databases (e.g. SNL database), scholarly literature (journals, conference proceedings, books, etc.), and company technical reports

Table 1
Regional water indices considered by the study.

Abbreviation	Name	Description	Spatial Data Coverage: Contained Cu, Pb + Zn, Ni
CRIT	Water Criticality [1]	Composite water risk indicator that is a function of the SR, VSR and EI indices described below. Measured between 0 and 100.	100%, 96%, 94%
SR	Supply Risk [1]	An index of physical water supply risks, as well as water governance and upstream geopolitical risks. Measured between 0 and 100.	100%, 97%, 95%
VSR	Vulnerability to Supply Restrictions [1]	An index combining the economic importance of water, the ability to compensate for supply restrictions and the general susceptibility of the region. Measured between 0 and 100.	100%, 96%, 94%
EI	Environmental Implications [1]	The potential environmental impacts associated with utilising water in a region – based upon life cycle impact assessment procedures. Measured between 0 and 100.	100%, 96%, 94%
AWaRe	Available Water Remaining [2][3]	The inverse of regional availability minus demand associated with environmental flow requirements and human consumption. AWaRe data specific to non-agricultural water use have been used. Values are normalised between 0.01 and 100, relative to a global average of 1.	100%, 100%, 100%
BWS	Blue water scarcity [4]	The ratio of the domestic consumption of blue water to the availability of blue water within a region.	55%, 68%, 47%
BIER	Basin Internal Evaporation Recycling [5]	The ratio of evaporation that is re-precipitated elsewhere within the same water basin.	100%, 100%, 100%
BIER-h	Hydrologically Effective Basin Internal Evaporation Recycling [5]	The ratio of evaporation that is re-precipitated and causes surface runoff or groundwater recharge elsewhere within the same water basin.	100%, 100%, 100%
WDI	Water Depletion Index [5]	The vulnerability of a basin to freshwater depletion. Accounts for consumption-to-availability ratios, surface and groundwater stocks and the overall aridity of the region. Normalised between 0.01 and 1.	100%, 100%, 100%
WSI	Water Stress Index [6]	A function of a region's water withdrawals, long-term water availability, and inter- and intra-annual precipitation variability. Measured between 0.01 and 1.	100%, 100%, 99%
WTA	Withdrawal-to-Availability ratio [7]	The ratio of water withdrawals in a region to the region's long-term water availability.	100%, 100%, 100%

Data sources: [1] Sonderegger et al. (2015); [2] WULCA (2016); [3] Boulay et al. (2016a, 2016b); [4] Hoekstra et al. (2012); [5] Berger et al. (2014); [6] Pfister et al. (2009); [7] Alcamo et al. (2003).

(Canadian National Instrument 43–101 reporting in the SEDAR database, JORC reporting, etc.). Indian copper deposits were aggregated at the state level during original data compilation and so coordinates for these was assigned based upon the approximate mid-point of each state. Fig. 1 shows the spatial distribution of copper, lead-zinc and nickel deposits considered in this study. More detailed maps showing mineral deposit types and operating status are shown in electronic supplementary Figs. S.1–S.5.

The contribution of copper, lead-zinc and nickel to the total economic value of the individual mineral resources was estimated based upon average metal price data for the period 2006–2010 (USGS, 2013). This also provides a coarse indication of potential metal co-products that may be extracted when developing these resources and could potentially provide a basis for economic allocation within water footprint or life cycle assessment studies.

2.2. Regional water indices

Exposure to water risks was assessed by considering regional data for a range of water risk and impact indices (Table 1 and Fig. 2), with further descriptions of these indices provided in the Supplementary information. Consideration of these indices alongside each other provides a richer perspective of regional water contexts than could be achieved by considering each index in isolation. The reader is encouraged to refer to the associated references for detailed information on the conceptualisation and data underpinnings of each index.

Values for each index were assigned to individual deposits or resources using GIS software. Weighted average index values were then determined for the copper, lead-zinc and nickel resources at several different scales – globally, for individual countries and for primary mineral deposit types. These were calculated according to Eq. (1), where I represents the regional index value and R represents the contained copper, nickel or combined lead-zinc metal tonnage associated with each mineral deposit/resource i. Weighted averages calculated using mineralised ore tonnages in place of contained metal tonnages are presented in the electronic Supplementary information for comparison.

$$ResourceWeightedAverage = \frac{\sum_i I_i \times R_i}{\sum_i R_i} \tag{1}$$

There are some limitations regarding the extent of spatial coverage for several of the indices (see Table 1). Where data has not been available, resources in that region have been excluded from subsequent calculations of statistics for that index.

2.3. Köppen-Geiger climate classifications

The Köppen-Geiger climate classification scheme was first described by Köppen (1900) before being modified by Geiger (1954), and more recently global climate maps based upon modern temperature and precipitation monitoring data have been developed (e.g. Kottek et al., 2006; Peel et al., 2007). Under the Köppen-Geiger climate classification scheme, regions are assigned major classifications based upon temperature ranges, or for the case of arid regions, precipitation levels. Precipitation and temperature sub-classifications are also assigned to provide further detail and seasonal information of the local climate. The scheme utilises five major climate classifications, six precipitation sub-classifications and eight temperature sub-classifications – which are often abbreviated using a short-hand lettering scheme. The temperature and precipitation thresholds used to assign regional climate classifications are shown in Table 2.

The resource datasets were assessed against global maps of historic and future Köppen-Geiger climate classifications. The basis of this analysis is the global climate classification dataset for 1951 to 2000 developed by Kottek et al. (2006) on a 0.5° by 0.5° latitude-longitude grid. The exposure of base metal resources to climate change was assessed for the IPCC emissions scenarios A1FI, A2, B1 and B2 (IPCC, 2000). These emission scenarios correspond to a greater than 66% probability of global temperature increases above pre-industrial levels of 4.1–6.2 °C (A1FI), 3.5–5.2 °C (A2), 2.0–3.2 °C (B1) and 2.6–3.7 °C (B2) respectively by 2100 (Rogelj et al., 2012). Rubel and Kottek (2010) provide spatial data for these scenarios for the periods 2001–2025,

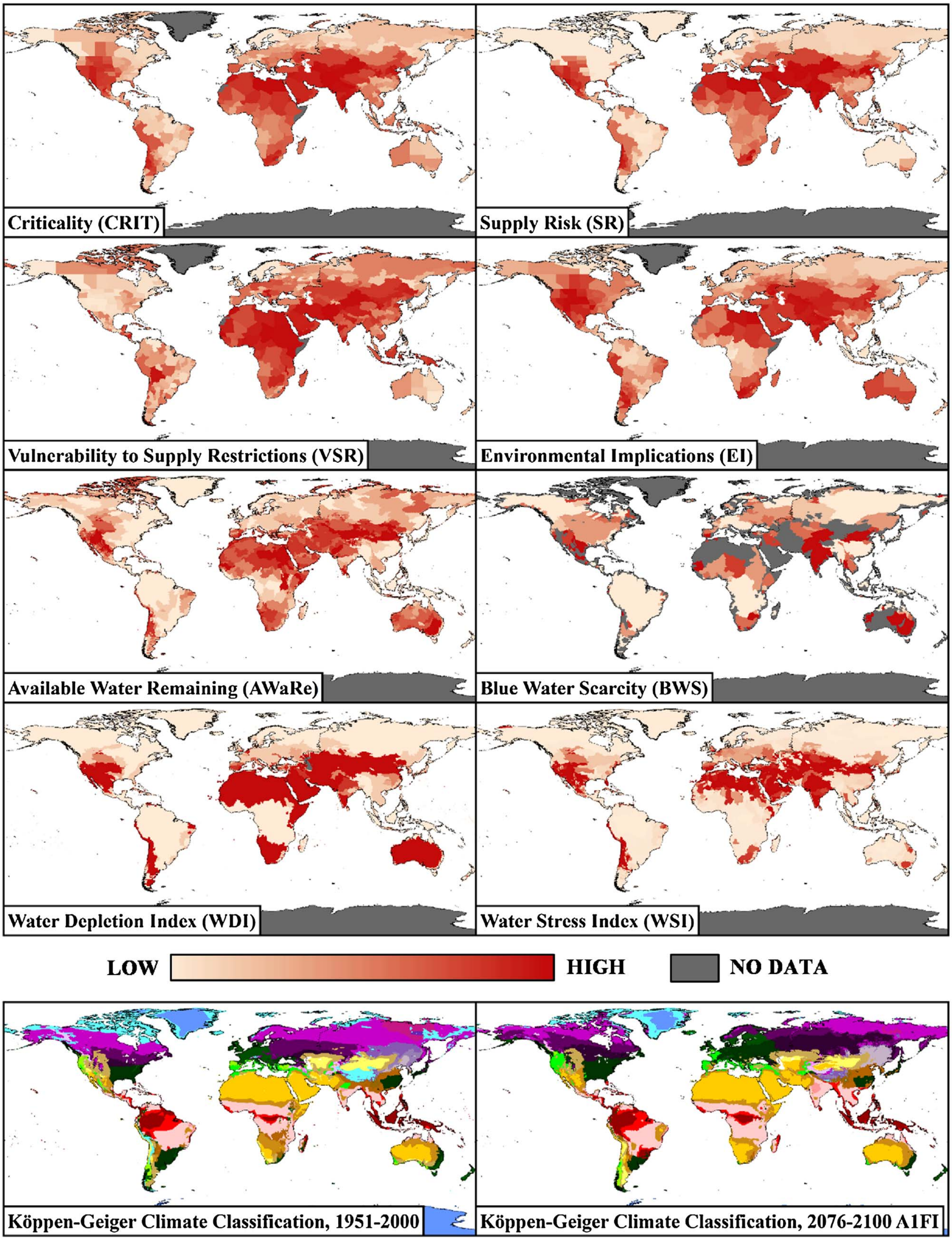

Fig. 2. Water indices and climate classifications used to understand the contextual water risks in regions containing copper, lead-zinc and nickel resources. Data sources: CRIT, SR, VSR, EI (Sonderegger et al., 2015), AWaRe (Boulay et al., 2016a, 2016b; WULCA, 2016), BWS (Hoekstra et al., 2012), WDI (Berger et al., 2014), WSI (Pfister et al., 2009), Köppen-Geiger climate classifications for the observed period 1951–2000 (Rubel and Kottek, 2010) and the IPCC scenario A1FI for the period 2076–2100 (Rubel and Kottek, 2010). BIER and BIER-h are not shown, but are presented in Berger et al. (2014).

Table 2
Temperature and precipitation thresholds for the version of the Köppen-Geiger climate classification scheme utilised by Kottek et al. (2006) and Rubel and Kottek (2010). Regional climates should be assessed against the criteria for E followed by B and then subsequently A, C or D.

Major Climate Classifications					
Precipitation Sub-classifications			Temperature Sub-classifications		
A: Equatorial ($T_{min} \geq +18\,°C$)					
f	Fully humid	$P_{min} \geq 60$ mm			
m	Monsoon	$P_{ann} \geq 25(100\text{-}P_{min})$			
s	Dry summer	$P_{min} < 60$ mm in summer			
w	Dry winter	$P_{min} < 60$ mm in winter			
B: Arid ($P_{ann} < 10\,P_{th}$)					
S	Steppe	$P_{ann} > 5\,P_{th}$	**h**	Hot	$T_{ann} \geq +18\,°C$
W	Desert	$P_{ann} \leq 5\,P_{th}$	**k**	Cold	$T_{ann} < +18\,°C$
C: Warm Temperate ($-3\,°C < T_{min} < +18\,°C$)					
s	Dry summer	$P_{smin} < P_{wmin}$, $P_{wmax} > 3\,P_{smin}$ and $P_{smin} < 40$ mm	**a**	Hot summer	$T_{max} \geq +22\,°C$
w	Dry winter	$P_{wmin} < P_{smin}$ and $P_{smax} > 10\,P_{wmin}$	**b**	Warm summer	Not (**a**) and at least 4 $T_{mon} \geq +10\,°C$
f	Fully humid	Neither **Cs** nor **Cw**	**c**	Cool summer and cold winter	Not (**b**) and $T_{min} > -38\,°C$
D: Snow ($T_{min} \leq -3\,°C$)					
s	Dry summer	$P_{smin} < P_{wmin}$, $P_{wmax} > 3\,P_{smin}$ and $P_{smin} < 40$ mm	**a**	Hot summer	$T_{max} \geq +22\,°C$
w	Drys winter	$P_{wmin} < P_{smin}$ and $P_{smax} > 10\,P_{wmin}$	**b**	Warm summer	Not (**a**) and at least 4 $T_{mon} \geq +10\,°C$
f	Fully humid	Neither **Ds** nor **Dw**	**c**	Cool summer and cold winter	Not (**b**) and $T_{min} > -38\,°C$
			d	Extremely continental	Like (**c**) but $T_{min} \leq -38\,°C$
E: Polar ($T_{max} < +10\,°C$)					
			F	Frost	$0\,°C \leq T_{max} < +10\,°C$
			T	Tundra	$T_{max} < 0\,°C$

Nomenclature: T_{ann} – Annual mean near-surface temperature; T_{min} – Coldest month mean temperature; T_{max} – Warmest month mean temperature; T_{mon} – Monthly mean temperature; P_{ann} – Annual Precipitation; P_{min} – Lowest monthly precipitation; P_{smin} – Lowest monthly precipitation in the summer half-year; P_{wmin} – Lowest monthly precipitation in the winter half-year; P_{smax} – Highest monthly precipitation in the summer half-year; P_{wmax} – Highest monthly precipitation in the winter half-year; P_{th} – Dryness threshold (function of annual temperature and precipitation seasonality, see Kottek et al., 2006).

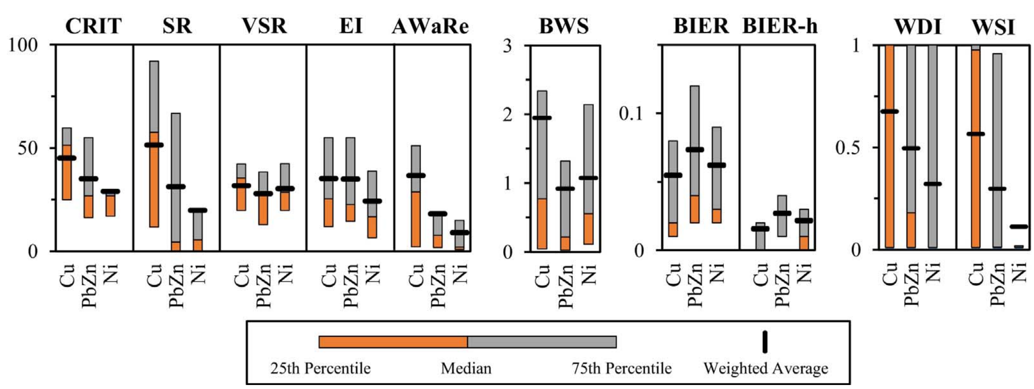

Fig. 3. Weighted average, median and interquartile range for Water Criticality (CRIT), Supply Risk (SR), Vulnerability to Supply Restrictions (VSR), Environmental Implications (EI), Available Water Remaining (AWaRe), Blue Water Scarcity (BWS), Basin Internal Evaporation Recycling (BIER), hydrologically effective Basin Internal Evaporation Recycling (BIER-h), Water Depletion Index (WDI) and the Water Stress Index (WSI). Statistics determined based upon contained metal tonnages of copper, lead-zinc and nickel resources.

2026–2050, 2051–2075, and 2076–2100–based upon averaging the monthly ensemble mean of 5 general circulation models (CGCM2, CSIRO2, HadCM3, PCM, ECHam4) that are available in the TYN SC 2.03 dataset (Mitchell et al., 2004).[1] Changes in the climate classification of regions containing base metal resources for these time periods and scenarios relative to the observed period 1951–2000.

Considerable uncertainty exists when modelling Köppen-Geiger classifications at the global scale, particularly at the level of temperature and precipitation sub-classifications. McMahon et al. (2015) assessed the average accuracy of GCMs to reproduce historical climate classifications (from 1950 to 1999). The proportion of grid cells across 46 GCM runs that were correctly assigned was 77% for a single letter classification, 57% for a two letter classification and 48% for a three letter classification. Additionally, temporal shifts in the boundaries of climate zones are poorly modelled by GCMs (Zhang and Yan, 2016). Therefore considerable care should be taken when interpreting the results presented by this study, particularly the detailed results for individual deposits and countries that are inherently more uncertain than the average results for each commodity.

3. Results

Summary results describing the spatial distribution of copper, lead-zinc and nickel resources in relation to the various water indices and climate classifications are provided in the following sections. Further results, figures and detailed datasets are provided in the electronic Supplementary information.

[1] Mitchell et al. (2004) describes the development of the TYN SC 2.00 dataset that only included four general circulation models (GCM). The ECHam4 GCM was subsequently added in version 2.03 of this dataset.

Table 3
Summary of results for major deposit types and countries with large resource endowments. More detailed results for all countries and weighted averages calculated on an ore tonnage basis are available in the Supplementary information. PGM refers to platinum group metals (Platinum, Palladium, Rhodium and Rhenium).

	Copper	No.	Mt Ore	% Cu	Mt Cu	Contained Value (%) Cu	Other metals >5%	Weighted Averages, Metal Basis CRIT	SR	VSR	EI	AWaRe	BWS	BIER	BIER-h	WDI	WSI
	Total	730	363,270	0.49	1,781	53	Au, PGM, Ni	45	51	32	35	37	1.95	0.05	0.02	0.68	0.56
	Undeveloped Deposits	468	136,945	0.43	586	48	Au, PGM, Mo, Ni	36	31	29	34	25	0.82	0.08	0.03	0.48	0.35
	Recently Operating	262	226,325	0.53	1,195	55	PGM, Au, Ni	49	61	33	36	42	0.75	0.04	0.01	0.77	0.67
	Deposit Type																
1	Porphyry	200	291,016	0.45	1,317	70	Au, Mo	51	64	32	39	45	2.69	0.04	0.01	0.78	0.72
2	Sediment-hosted Cu	62	10,874	1.52	165	74	Co	27	14	41	11	6	0.19	0.18	0.04	0.08	0.04
3	Iron Oxide Copper Gold	51	17,730	0.71	125	53	Au, U3O8	26	10	19	33	27	1.30	0.03	0.01	0.88	0.15
4	Magmatic Sulfide	133	26,543	0.29	76	9	PGM, Ni, Au	25	14	24	23	6	0.78	0.07	0.03	0.16	0.09
5	Skarn	39	4,901	0.70	34	71	Zn, Mo	39	25	35	50	7	0.23	0.13	0.03	0.07	0.08
6	Volcanogenic Massive Sulfide	144	4,042	0.78	32	50	Zn, Au	30	25	23	30	19	1.26	0.05	0.02	0.47	0.25
7	Other/Miscellaneous	50	2,701	0.59	16	54	Ni, Au	65	81	38	60	43	2.44	0.05	0.02	0.74	0.74
8	Sediment-hosted Pb-Zn	21	2,746	0.39	11	34	Ni, Zn, Co	16	3	12	23	11	0.31	0.05	0.01	0.79	0.03
9	Epithermal	30	2,717	0.18	5	40	Zn, Au, Pb	39	36	29	44	21	0.47	0.07	0.02	0.58	0.57
	Country																
1	Chile	51	122,768	0.54	658	89	Au	61	91	37	34	57	4.00	0.01	0.00	1.00	0.99
2	USA	56	49,427	0.34	170	53	Au, Mo, Ni, PGM	48	51	7	64	57	1.66	0.07	0.01	0.64	0.62
3	Peru	52	35,088	0.48	168	78	Mo, Au	40	25	35	55	38	0.01	0.10	0.02	0.56	0.54
4	Australia	149	20,292	0.63	127	50	Au, U3O8	22	1	15	35	33	2.24	0.02	0.00	0.97	0.04
5	Russia	14	5,516	1.07	59	35	Ni, PGM, Au	21	3	34	12	3	0.25	0.10	0.06	0.01	0.01
	Rest of World	408	130,177	0.46	598	36	PGM, Au, Ni	34	31	35	25	13	0.63	0.09	0.03	0.37	0.26

	Lead-Zinc	No.	Mt Ore	% Pb+Zn	Mt Pb+Zn	Contained Value (%) Pb+Zn	Other Metals >5%	Weighted Averages, Metal Basis CRIT	SR	VSR	EI	AWaRe	BWS	BIER	BIER-h	WDI	WSI
	Total	852	50,882	1.64	837	30	Mo, Cu, Ag	35	32	28	35	18	0.91	0.07	0.03	0.50	0.30
	Undeveloped Deposits	620	31,467	1.33	419	21	Mo, Cu	33	29	29	30	19	0.70	0.08	0.03	0.44	0.29
	Recently Operating	232	19,415	2.15	418	48	Cu, Ag, Ni	37	34	26	39	18	1.11	0.07	0.02	0.55	0.31
	Deposit Type																
1	Sediment-hosted Pb-Zn	279	7,875	6.22	490	90	Ag	33	28	28	32	17	0.82	0.07	0.03	0.55	0.25
2	Volcanogenic Massive Sulfide	269	4,156	3.15	131	42	Cu, Au, Ag	29	23	27	27	16	1.31	0.05	0.02	0.30	0.23
3	Skarn	92	5,884	1.48	87	36	Cu, Fe, Ag	48	48	33	53	22	0.62	0.07	0.02	0.55	0.50
4	Epithermal	138	4,082	1.70	69	45	Ag, Cu, Au	49	53	30	50	28	1.13	0.11	0.02	0.64	0.51
5	Porphyry	16	14,646	0.18	27	11	Cu, Mo, Ag	51	65	30	44	32	1.59	0.08	0.02	0.41	0.42
6	Sediment-hosted Mixed	9	12,581	0.14	17	1	Mo, Ni	9	1	8	12	1	0.37	0.07	0.03	0.01	0.13
7	Mesothermal Vein	13	100	6.26	6	65	Ag	35	20	21	44	14	0.68	0.10	0.06	0.22	0.19
8	Iron Oxide Copper Gold	6	326	0.61	2	19	Mo, Cu, Ag, Ni	39	52	21	30	65	4.85	0.08	0.01	0.88	0.62
9	Miscellaneous	4	972	0.24	2	14	U3O8	15	0	17	19	0	-	0.01	0.00	0.01	0.01
10	Mine Wastes	14	146	1.55	2	52	Au, Cu, Ag	30	24	27	32	7	0.39	0.07	0.01	0.16	0.10
11	Orogenic Au	8	90	1.71	2	35	Cu, Au, Ag	22	8	14	32	49	1.57	0.04	0.00	0.65	0.43
12	Manto	4	25	4.89	1	68	Ag, Au	22	14	24	19	1	0.08	0.08	0.02	0.12	0.12
	Country																
1	Australia	132	2,427	6.38	155	78	Ag, Cu	17	1	15	25	17	1.10	0.03	0.01	0.88	0.04
2	Canada	230	7,574	1.32	100	58	Cu, V, Ag, Au, Mo	14	0	18	14	2	0.21	0.12	0.07	0.02	0.01
4	Peru	48	7,537	0.85	64	32	Cu, Ag	40	25	35	55	5	0.01	0.15	0.03	0.27	0.29
3	Russia	40	2,562	3.55	91	49	Cu, Fe, Au	22	3	35	13	6	0.35	0.13	0.04	0.02	0.04
5	Mexico	65	4,966	1.22	61	45	Ag, Cu, Au	56	76	24	50	41	2.74	0.05	0.01	0.90	0.75
	Rest of World	337	25,816	1.42	366	19	Mo, Cu	48	56	34	45	24	1.31	0.06	0.02	0.55	0.48

	Nickel	No.	Mt Ore	% Ni	Mt Ni	Contained Value (%) Ni	Other Metals >5%	Weighted Averages, Metal Basis CRIT	SR	VSR	EI	AWaRe	BWS	BIER	BIER-h	WDI	WSI
	Total	476	61,365	0.48	292.5	82	Cu, Co	29	20	31	25	9	1.07	0.06	0.02	0.33	0.11
	Laterite	224	15,838	1.12	178.1	91	Co	25	12	32	17	8	0.83	0.06	0.02	0.23	0.05
	Undeveloped Deposits	173	11,955	1.08	129.7	90	Co	26	12	33	18	9	0.99	0.06	0.02	0.26	0.05
	Recently Operating	51	3,883	1.25	48.5	93	Co	21	10	29	14	5	0.11	0.05	0.02	0.16	0.03
	Sulfide	252	45,526	0.25	114.3	71	Cu, PGM	35	31	28	35	10	1.20	0.07	0.02	0.45	0.21
	Undeveloped Deposits	166	27,500	0.20	54.3	75	Cu, PGM	34	31	24	36	12	1.19	0.08	0.03	0.45	0.18
	Recently Operating	86	18,027	0.33	60.0	69	Cu, PGM	35	32	31	33	9	1.21	0.06	0.02	0.44	0.24
	Deposit Type - Laterite																
1	Oxide	112	10,750	1.04	112.3	88	Co	26	15	32	19	9	1.28	0.04	0.01	0.26	0.06
2	Hydrous Mg silicate	74	3,435	1.49	51.2	97		22	6	34	8	4	0.39	0.08	0.03	0.06	0.02
3	Clay silicate	38	1,653	0.88	14.6	90	Co	23	2	25	28	10	0.27	0.08	0.03	0.66	0.02
	Deposit Type - Sulfide																
1	Magmatic Sulfide	224	37,720	0.28	106.9	72	Cu, PGM	37	34	29	36	11	1.26	0.07	0.02	0.48	0.23
2	Hydrothermal Ni	27	6,523	0.09	6.1	61	Ni, Co	8	1	9	9	2	0.40	0.08	0.03	0.03	0.03
3	Fe-Ni alloy	1	1,284	0.11	1.4	100		7	0	5	11	1	0.01	0.12	0.08	0.01	0.01
	Country - Laterite																
1	Indonesia	16	2,071	1.61	33.3	97		26	5	45	1	1	-	0.05	0.03	0.01	0.01
2	Australia	69	4,182	0.75	31.5	88	Co	26	1	24	38	23	4.19	0.02	0.00	0.99	0.06
3	Phillipines	25	1,566	1.15	18.0	94	Co	16	13	21	10	3	1.68	0.02	0.01	0.04	0.02
4	Cuba	10	1,340	1.21	16.2	86	Co	29	31	29	26	5	-	0.01	0.00	0.17	0.05
5	New Caledonia	23	808	1.86	15.0	94	Co	-	-	-	-	1	-	0.00	0.00	0.01	0.01
	Rest of World	81	5,872	1.09	64.1	90	Co	24	15	33	14	8	0.33	0.11	0.04	0.10	0.07
	Country - Sulfide																
1	South Africa	45	18,362	0.18	33.2	89	Cu	66	85	38	63	19	2.08	0.08	0.01	0.91	0.49
2	Canada	54	6,377	0.34	21.9	79	Cu, Co	12	0	11	17	1	0.45	0.09	0.05	0.01	0.02
3	Russia	14	4,493	0.46	20.5	61	Cu, PGM	17	2	28	7	2	0.26	0.05	0.03	0.01	0.01
4	Australia	55	2,034	0.58	11.9	92		26	0	25	38	15	5.95	0.02	0.00	0.97	0.02
5	China	3	435	1.38	6.0	82	Cu	71	87	57	65	11	2.05	0.02	0.00	1.00	1.00
	Rest of World	81	13,826	0.15	20.9	51	Cu, PGM, Co	22	10	23	24	11	0.68	0.11	0.02	0.14	0.06

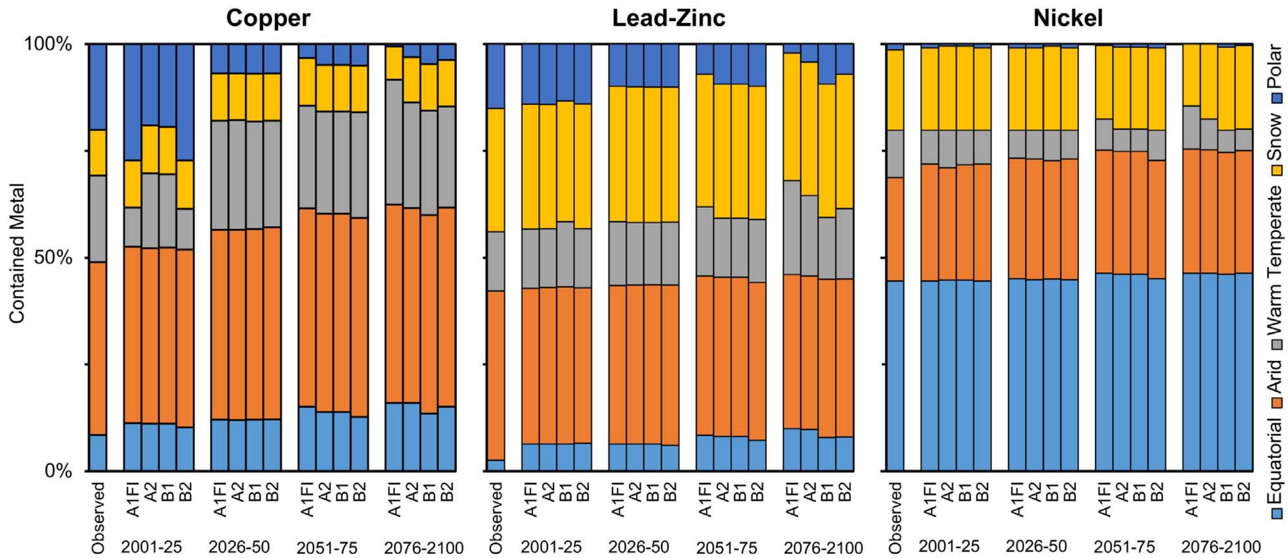

Fig. 4. Proportion of copper, lead-zinc and nickel resources contained in regions with each major Köppen-Geiger climate classification. Data shown for the observed period 1951–2000 (Kottek et al., 2006) and IPCC emissions scenarios A1FI, A2, B1 and B2 until 2100 (Rubel and Kottek, 2010). Resource data sources: copper (Mudd et al., 2013), lead-zinc (Mudd et al., 2017), nickel (Mudd and Jowitt, 2014), resource locations (see Section 2.1).

3.1. Regional water indices

The key results showing the exposure of copper, lead-zinc and nickel resources to the regional water indices are shown in Fig. 3 and Table 3. The results indicate that on average global copper resources are more exposed to these contextual water risks than global lead-zinc or nickel resources across all indices considered (note that a higher BIER or BIER-h may be preferable). Additionally, global lead-zinc resources appear more exposed than nickel resources for all indices barring the BWS, BIER and BIER-h. The aggregate results for the various indices are largely consistent, independent of the individual index. However, the results for specific countries or deposit types are highly variable.

The average water criticality in regions containing copper deposits is higher than those containing lead-zinc or nickel deposits. This is largely due to the exceptionally high SR regions in Chile and southern Peru that contain significant amounts of contained copper in porphyry deposits. The average VSR is similar for all three commodity groups, although some regions containing large lead-zinc and nickel resources, such as China, Kazakhstan or Iran, may be highly susceptible to water supply restrictions. Regions containing sediment-hosted Cu deposits also appear more acutely vulnerable to supply restrictions than regions containing porphyry copper deposits.

The general exposure to water risks can also be assessed by considering the proportion of contained resources that are located in regions experiencing water stress or scarcity above global averages. Some 55% (951 Mt) of current copper, 27% (230 Mt) of current lead-zinc and 4% (11 Mt) of current nickel resources are located in regions with a WSI exceeding 0.602, the global average WSI associated with fresh water consumption (Ridoutt and Pfister, 2013). In comparison, some 57% (1018 Mt) of current copper, 25% (208 Mt) of current lead-zinc and 15% (45 Mt) of current nickel resources are located in regions that have AWaRe values higher than the global average for non-agricultural water use (20; WULCA, 2016). The proportion of contained metal in regions with a BWS exceeding 1–indicating regions where water is being overexploited – is 39% (387 Mt) for copper, 36% (206 Mt) for lead-zinc and 19% (55 Mt) for Ni (note that these values for BWS are highly uncertain due to data limitations – see Table 1).

The BIER data indicate that some 5%, 7%, and 6% of evaporation in regions containing copper, lead-zinc and nickel resources respectively is re-precipitated within the same drainage basin. However only 2%, 3% and 2% respectively of the evaporation will re-precipitate and replenish blue water stores through surface runoff and groundwater recharge (indicated by BIER-h). As with the other indices, there is high variability for the BIER and BIER-h data between individual deposit types and countries. For instance, the weighted average BIER for copper in sediment-hosted Cu deposits and in Skarn deposits is 18% and 13% respectively, considerably higher than the average for all copper deposits of 5%.

The results for recently operating mine sites can also be compared with those for undeveloped deposits. Undeveloped copper resources are located in less water scarce or stressed regions than resources that have been recently mined, primarily as a result of the high proportion of overall copper resources that are contained within the large-scale copper operations in water scarce regions of Chile and Peru. Our data for lead-zinc resources also indicates that undeveloped deposits are generally located in less water stressed regions than recently operated mines, although with less consistency between individual indicators. The averages for CRIT, SR, EI, BWS, WDI, WSI were significantly higher for recently operated lead-zinc resources, whereas VSR, AWaRe, BIER, BIER-h were marginally lower for these resource. There was no significant difference between the overall results for undeveloped and operating nickel sulfide resources. However, undeveloped nickel laterite deposits are located in more water scarce or critical regions than recently operated nickel laterite mines, although it should be noted that the nickel laterite data are more uncertain than data for the other commodities and resource types as a result of the limited spatial indicator data available for regions containing these resources (e.g. New Caledonia; see Tables 1 and 2).

Further detailed results for individual countries or resource types are provided in Supplementary Tables S.1–S.8.

3.2. Köppen-Geiger climate classifications

The proportion of copper, lead-zinc and nickel contained within different climate zones are shown in Fig. 4, with more detailed information for precipitation and temperature sub-classifications given in Table 4.

A large proportion of copper deposits (40%, 721 Mt Cu) are located in regions with arid climates, including 43% (572 Mt Cu) of copper contained in porphyry deposits (largely within southern Peru and Chile) and 85% (106 Mt Cu) of copper in iron oxide copper gold deposits (largely in Australia, Peru and Chile). Of the copper contained within porphyry deposits, 25% (327 Mt Cu) is within regions classified as having a polar climate (mostly in the alpine tundra sections of the

Table 4
Base metal resources contained in Köppen-Geiger climate zones, based upon the observed period 1951–2000 (Kottek et al., 2006). Results for deposit types, other time periods and climate scenarios are available in the electronic Supplementary information. Resource data sources: copper (), lead-zinc (Mudd et al., 2017), nickel (Mudd and Jowitt, 2014), resource locations (see Section 2.1).

Köppen-Geiger Climate Classification		1951-2000, Observed (Kottek et al., 2006)											
		Copper				Lead-Zinc				Nickel			
		No.	Mt Ore	% Cu	Mt Cu	No.	Mt Ore	%Pb+Zn	Mt Pb+Zn	No.	Mt Ore	% Ni	Mt Ni
	Equatorial	**74**	**31,735**	**0.48**	**151**	**51**	**437**	**4.98**	**22**	**129**	**10,133**	**1.29**	**130**
Af	Fully humid	24	17,297	0.55	95	11	86	5.23	4	47	4,186	1.41	59
Am	Monsoonal	12	9,584	0.38	36	3	49	2.24	1	29	1,639	1.16	19
As	Summer dry	2	9	2.45	0	3	4	2.55	0	4	63	1.96	1
Aw	Winter dry	36	4,845	0.41	20	34	298	5.39	16	49	4,245	1.20	51
	Arid	**255**	**158,801**	**0.45**	**721**	**237**	**13,387**	**2.48**	**331**	**149**	**18,433**	**0.38**	**71**
BSh	Steppe, hot arid	85	16,564	0.42	70	90	4,833	3.73	180	53	13,216	0.24	31
BSk	Steppe, cold arid	49	42,196	0.35	149	60	5,996	0.94	57	22	95	1.50	1
BWh	Desert, hot arid	68	21,940	0.63	137	74	1,887	3.19	60	73	4,690	0.68	32
BWk	Desert, cold arid	53	78,101	0.47	365	13	671	5.09	34	1	432	1.39	6
	Warm Temperate	**169**	**63,455**	**0.57**	**361**	**173**	**4,448**	**2.60**	**116**	**71**	**10,922**	**0.30**	**32**
Cfa	Fully humid, hot summer	22	5,245	0.34	18	20	308	6.61	20	7	577	0.86	5
Cfb	Fully humid, warm summer	37	4,710	1.04	49	70	657	4.28	28	10	382	0.82	3
Cfc	Fully humid, cool summer	1	10	1.17	0	5	39	5.26	2	-	-	-	-
Csa	Summer dry, hot summer	17	4,258	0.36	16	34	2,314	1.76	41	5	411	1.05	4
Csb	Summer dry, warm summer	13	27,377	0.55	151	14	328	1.66	5	7	619	0.77	5
Csc	Summer dry, cool summer	-	-	-	-	-	-	-	-	-	-	-	-
Cwa	Winter dry, hot summer	65	15,813	0.70	111	17	152	7.30	11	32	6,361	0.15	10
Cwb	Winter dry, warm summer	14	6,043	0.28	17	13	650	1.23	8	10	2,573	0.22	6
Cwc	Winter dry, cool summer	-	-	-	-	-	-	-	-	-	-	-	-
	Snow	**171**	**46,466**	**0.41**	**190**	**312**	**25,683**	**0.94**	**241**	**122**	**21,446**	**0.26**	**55**
Dfa	Fully humid, hot summer	-	-	-	-	4	54	1.38	1	1	74	0.86	1
Dfb	Fully humid, warm summer	53	7,865	0.55	43	99	7,839	0.95	74	44	5,638	0.30	17
Dfc	Fully humid, cool summer	101	28,141	0.37	104	165	15,260	0.72	110	71	15,073	0.23	35
Dfd	Fully humid, extremely continental	-	-	-	-	1	18	13.28	2	-	-	-	-
Dsa	Summer dry, hot summer	-	-	-	-	2	32	26.20	8	-	-	-	-
Dsb	Summer dry, warm summer	6	6,221	0.11	7	6	59	5.14	3	-	-	-	-
Dsc	Summer dry, cool summer	3	434	0.24	1	23	180	6.95	13	-	-	-	-
Dsd	Summer dry, extremely continental	-	-	-	-	-	-	-	-	-	-	-	-
Dwa	Desert, hot summer	-	-	-	-	-	-	-	-	-	-	-	-
Dwb	Desert, warm summer	-	-	-	-	2	315	5.00	16	-	-	-	-
Dwc	Desert, cool summer	8	3,804	0.91	35	10	1,924	0.75	14	6	661	0.41	3
Dwd	Desert, extremely continental	-	-	-	-	-	-	-	-	-	-	-	-
	Polar	**61**	**62,813**	**0.57**	**358**	**79**	**6,928**	**1.82**	**126**	**5**	**431**	**0.92**	**4**
EF	Frost	-	-	-	-	-	-	-	-	-	-	-	-
ET	Tundra	61	62,813	0.57	358	79	6,928	1.82	126	5	431	0.92	4

Andean porphyry copper belt through Chile, Peru and Argentina), 7% (86 Mt Cu) is in snow climates, 10% (132 Mt Cu) is in equatorial climates, and 15% (200 Mt Cu) is located in warm temperate climates (mostly within Chile and Argentina). Warm temperate climates also contain 78% (129 Mt Cu) of the copper in sediment-hosted Cu deposits, divided between the fully humid-warm summer climate (Cfb) of Poland (19%, 31 Mt Cu) and the winter dry-hot summer climate (Cwa) of the Central African copper belt (59%, 98 Mt Cu) running through Zambia and the Democratic Republic of Congo. For iron oxide copper gold deposits, 83% (104 Mt Cu) of the contained copper is located within desert regions of Australia, Chile, Peru and Mauritania (BWh or BWk). The majority of copper in magmatic sulfide or skarn deposits are in regions with snow or polar climates.

Lead-zinc deposits are primarily located in the arid regions of Australia, Peru and Mexico as well as in the polar and snow climate regions of Canada and Russia. Some 50% (245 Mt Pb + Zn) of the lead-zinc resources contained in sediment-hosted Pb-Zn deposits are located in arid climate regions – with roughly two-thirds of this (153 Mt Pb + Zn) being in hot arid steppe climate (BSh) regions (predominantly in Australia and India). A further 25% (123 Mt Pb + Zn) of the lead-zinc contained in sediment hosted Pb-Zn deposits are located in snow climate regions (mostly within Russia and Canada). Regions with snow climates also contain 55% (72 Mt Pb + Zn) of the lead-zinc in volcanic massive sulfide deposits.

Nickel laterite deposits are formed through tropical weathering processes and so 71% (127 Mt Ni) of nickel laterite resources are contained in countries with equatorial climates such as New Caledonia, Cuba, Indonesia, and the Philippines. Nickel sulfide resources display a very different geographic distribution, with only 3.5% (4 Mt Ni) being contained in equatorial climate areas. Regions with snow climates (such as parts of Canada, Russia, the USA and Finland) contain 44% (50 Mt Ni) of nickel sulfide resources and a further 38% (43 Mt Ni) is contained in regions with arid climates (primarily within South Africa and Australia).

The proportion of contained resource in each major climate classification for each time period to 2100 under the various IPCC emissions scenarios (Rubel and Kottek, 2010) are shown in Fig. 4. Through time the proportion of contained resource in regions with polar or snow climates is expected to decline due to reclassification to arid or warm temperate climates. Concurrently, many regions with arid or warm temperate climates are likely to be reclassified to equatorial climates (Rubel and Kottek, 2010). Overall, 27–32% (473–574 Mt Cu) of copper, 17–29% (139–241 Mt Pb + Zn) of lead-zinc and 6–13% (19–39 Mt Ni) of nickel is contained in regions that may have a major climate reclassification. A further 15–23% (262–412 Mt Cu) of copper, 23–32% (195–270 Mt Pb + Zn) of lead-zinc and 29–32% (84–94 Mt Ni)

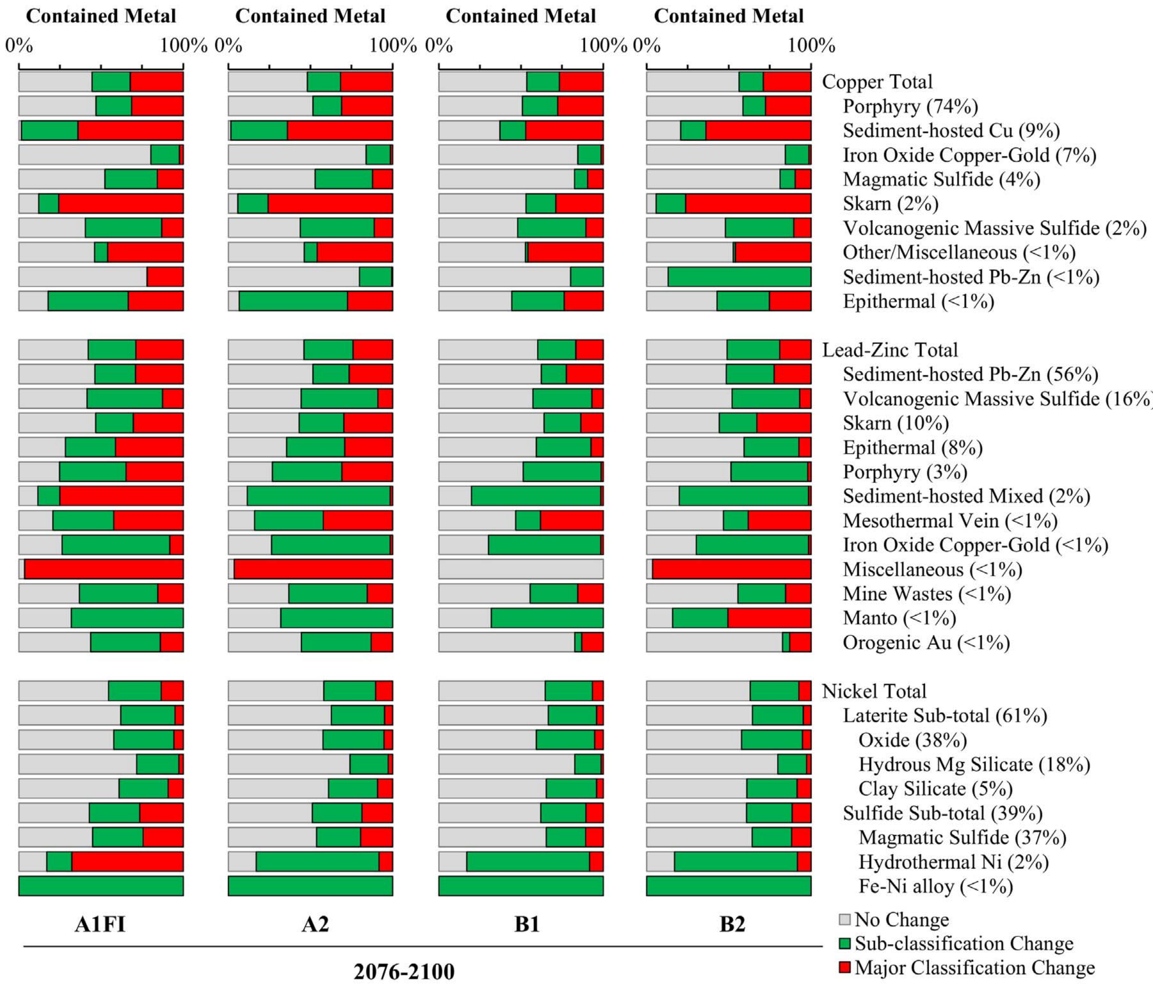

Fig. 5. Proportion of copper, lead-zinc and nickel resources located in regions with a changing Köppen-Geiger climate classification under the A1FI, A2, B1 and B2 IPCC scenarios. Data shown is for the period 2076–2100 (Rubel and Kottek, 2010) and is relative to the observed period 1951–2000 (Kottek et al., 2006). The distribution of copper, lead-zinc and nickel contained in individual deposit types is shown as a percentage in brackets. Figures showing other time periods are available in the electronic Supplementary information. Resource data sources: copper (Mudd et al., 2013), lead-zinc (Mudd et al., 2017), nickel (Mudd and Jowitt, 2014), resource locations (see Section 2.1).

of nickel is contained in regions that may have a precipitation or temperature sub-classification change. The exposure of individual deposit types to major climate classification changes, or precipitation and temperature sub-classification changes, is shown in Fig. 5. In addition, the flow of contained resources between these climate classifications are summarised in Fig. 6 for the IPCC's A1FI and the B1 emission scenarios. Equivalent figures for other climate scenarios or time periods are shown in Supplementary Figs. S.7–S.12.

Regions containing sediment-hosted Cu deposits (large in warm temperature climates) appear to be highly exposed to the impact of future climate change. Regions containing 42–59% (70–98 Mt Cu) of copper in sediment-hosted Cu deposits may be re-classified by the end of the century from having warm temperate climates with dry winters and hot summers (Cwa) to having equatorial climates with dry winters (Aw). A further 19% (31 Mt Cu) of copper in sediment-hosted Cu deposits may also be reclassified from being in fully humid-warm summer temperate climate (Cfb) to being in a fully humid-hot summer climate (Cfa). For porphyry deposits, 73–180 Mt of copper contained in arid climates may experience climate reclassifications, mostly from changes in temperature sub-classification from cold arid to hot arid.

Polar climates contain 15% (126 Mt Pb + Zn) of lead-zinc resources. Of this, 37–52% (47–65 Mt Pb + Zn) and 2–34% (3–43 Mt Pb + Zn) may be reclassified to being in snow and warm temperate climates respectively. Regions with 13–19% (106–161 Mt Pb + Zn) of contained lead-zinc resources are in snow climates that may have temperature or precipitation sub-classification changes by 2100. This manifests as either a shift of precipitation sub-classification from fully-humid to summer dry or desert, or as a change in temperature sub-classification associated with increasing summer temperatures. A further 8–14% (65–121 Mt Pb + Zn) of lead-zinc resources are located in arid regions that may have sub-classification changes.

Regions containing nickel resources are less likely to have major climate reclassifications by 2100, however they are still moderately exposed to climate sub-classification changes. Monsoonal (Am), fully humid (Af) and summer dry (As) equatorial climate sub-classifications are likely to have a net increase in contained nickel laterite resource by 2100, whereas equatorial-winter dry climates (Aw) will likely have a net decrease in contained nickel laterite resource. Nickel laterite resources contained in warm temperate climates may decline by 4–4.5% (7–8 Mt Ni) by 2100, due to reclassification to arid or equatorial climates. Also up to 14% (16 Mt Ni) of nickel contained in sulfide deposits in snow climates may be reclassified to warm temperate climates.

4. Discussion

4.1. Water availability and hydrological variability risks

A mining operation's water balance can be categorised as positive during periods where water accumulates on-site, or as negative during

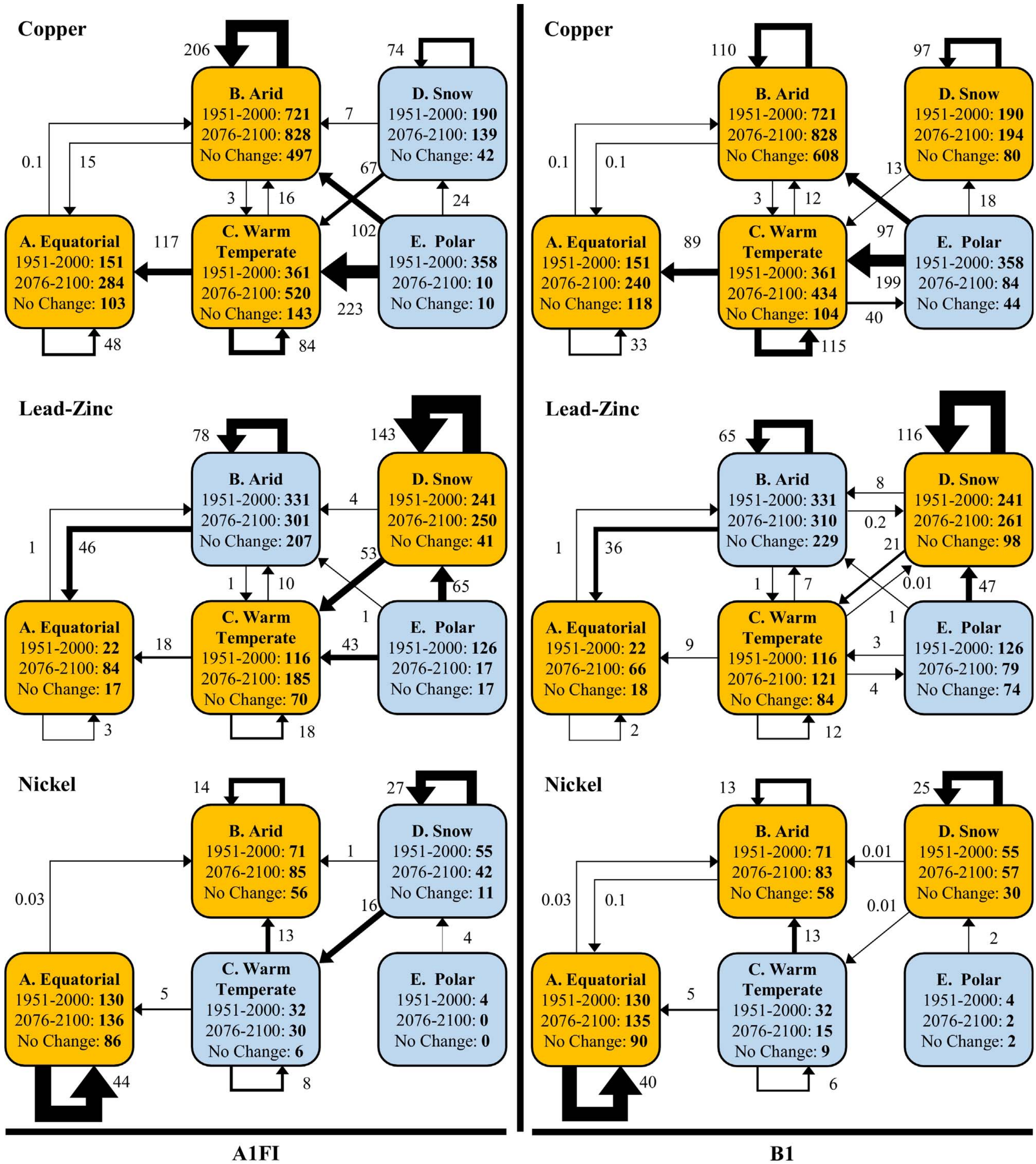

Fig. 6. Changes in the Köppen-Geiger climate classification of regions containing base metals from 1951 to 2000 (Kottek et al., 2006) to 2076–2100 for the IPCC emissions scenario A1FI and B1 (Rubel and Kottek, 2010). Figures for other scenarios are shown in the Supplementary information. Values indicate million tonnes of contained metal. Flow width is in proportion to the global resource for the individual metal(s). Flows returning to the same climate classification represent a change in precipitation or temperature sub-classification. Yellow and blue shading indicates a respective net increase or decrease in contained resource. Resource data sources: copper (Mudd et al., 2013), lead-zinc (Mudd et al., 2017), nickel (Mudd and Jowitt, 2014), resource locations (see Section 2.1). (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

periods where a net loss of water occurs. Water positive operations may at some stage require active discharge of water, whereas a water negative operation may require continual imports of water to meet the requirements of ore processing facilities and to achieve other site objectives, such as dust suppression or the maintenance of tailings storage facility wet covers. The data for Köppen-Geiger climate classifications provides an initial basis to assess the average water balance when developing a base metal deposit (e.g. a mine located in an arid region will often have a negative water balance). However, further analysis is required to assess the implications of the various temperature and precipitation sub-classifications on average mine site water balances.

Sourcing sufficient water to meet the requirements of mining operations may be challenging or controversial in some circumstances.

This is particularly the case in regions where access to water resources are also fully allocated for other purposes, such as agriculture, forestry or environmental flows. The SR, BWS, WSI and AWaRe data provide an indication of regions where access to water resources and competition with other users may potentially constrain the development of base metal resources.

Regions containing fully or over-allocated water resources can lead to strong incentives for the mining industry to develop alternative sources of water. As an example, Chile accounts for one third of global mined copper production and also contains the largest known copper resources (Mudd et al., 2013; USGS, 2016). In many regions of Chile, significant water scarcity or overexploitation of water resources is occurring (see BWS and WSI data in Table 3). As a result the Chilean copper industry has achieved considerable improvements in water efficiency. From 2009 to 2014, water use decreased from 0.67 to 0.53 m^3/t ore for copper concentrate producers and from 0.12 to 0.08 m^3/t ore for electrowon copper producers (COCHILCO, 2015a). Despite these efficiency improvements, growth in the Chilean industry has required significant investment in seawater desalination and pipeline capacity (e.g. the US$ 3.4 billion desalination plant to supply the Escondida mine site; BHPB, 2013) to meet water requirements. Desalinated water use by the Chilean copper industry is anticipated to grow by 14% annually between 2015 and 2026 (COCHILCO, 2015b). Raw seawater is also occasionally used in Chilean mineral processing circuits, such as at the Las Luces copper-molybdenum concentrator (Moreno et al., 2011). Further use of seawater in mineral processing circuits is possible, but key issues remain with ore mineralogy, processing constraints, and the limited understanding of flotation chemistry and performance in saline waters (Castro, 2012; Moreno et al., 2011; Wang and Peng, 2014).

The water balance of a mine-site may shift between being water positive and water negative in response to changing seasons, weather events and site water management decisions. This variability can pose operational challenges for mining operations, particularly when regulatory or practical constraints limit the ability to withdraw or discharge water. A variety of strategies are available to mitigate these risks, such as the development of new infrastructure or changes to operating procedures (Kunz and Moran, 2016).

Drought conditions may exacerbate existing water availability and allocation issues for mining operations. The development of additional water storage infrastructure may provide some buffer against these issues. However, generally these conditions will require mines to source additional water from surrounding surface or ground water systems to offset increases in net evaporative water losses (i.e. evaporation minus rainfall). Increases in net evaporative losses from ponds and dams may effectively limit the maximum achievable recycling rate of water at mineral processing operations. Firstly, the physical decrease in water volumes requires external surface or groundwater to be sourced to offset these losses. Additionally, the increased evaporation rates may lead to accelerated accumulation of salts in water storage facilities, requiring additional water treatment or dilution with freshwater to achieve the water quality requirements of ore processing facilities. Drought can also impact mining operations post-closure, through limiting the success of site revegetation and rehabilitation efforts (Halwatura et al., 2014).

Perhaps more concerning for many mining operations are the risks associated with having too much water. Heavy rainfall (or snowfall) can result in sections of mines becoming inaccessible or unsafe to operate, which can lead to supply disruptions. Alternatively periods of excess water may result in unplanned or uncontrolled water discharges to surrounding environments, as happened at the Lady Annie copper mine in Australia in 2009 (Taylor and Little, 2013). The financial implications of flood events in mining regions can be significant. During the 2010/2011 Queensland floods (Australia), 20% of the coal mines in the Bowen Basin became inoperable and a further 60% faced operating restrictions. The associated financial costs were AU$ 5–9 billion (Sharma and Franks, 2013; QRC, 2011) and emergency water discharges were approved to prevent further adverse consequences and infrastructure risks (QFCI, 2012).

Köppen-Geiger climate classifications only provide insight into average seasonal hydrologic variability. However assessing the exposure of the mining industry to drought and flood risks requires more acute measures of hydrologic variability. This could be assessed using data for rainfall frequency and intensity, or the recurrence intervals of drought or flood events. Previously these risks to the mining industry have typically been assessed at the scale of individual mine sites. However some studies are now considering these risks at the scale of major multi-national mining companies (Bonnafous et al., 2016).

4.2. Climate change and infrastructure risks

Mine site infrastructure is typically designed and built based upon historic weather and climate patterns (Pearce et al., 2011). For instance a tailings storage facility may be built to withstand a 1-in–500 year rainfall event. However, as weather patterns and climate change into the future, the assumptions used to develop infrastructure at current and historic mining operations may no longer be appropriate. Thus, risks associated with existing infrastructure in the mining industry may increase or decrease through time depending upon local climate changes. When these climate related risks are relatively small for individual mining operations, a company or institutional investor that has a stake in multiple mining operations may still be exposed to significant technical or financial risks when considered at a portfolio level (see Bonnafous et al., 2016).

The stability of slopes in open pit mines and tailings storage facilities are a potential hazard throughout the world and there are many contributing risk factors, especially in the context of a changing climate. Among the various factors affecting slope stability, water is known to be a major trigger for failure (Azam and Li, 2010). At mine sites, the stress relieving moments are quite commonly observed, and these moments can range anywhere from a few millimetres to a couple of metres, depending on where a particular mine is situated. These are largely influenced by the in-situ stress conditions and the deformation characteristics of the material in which the excavations are made. These movements will increase in-ground strain on the slopes, which may in turn result in localised cracking of the slopes. The rain fed surface water entering these cracks may then result in block instabilities, placing the safety of equipment and personnel at risk.

In many natural mining environments, rocks (ore bearing and overlain) are subjected to cyclic drying-wetting conditions because of repeated water absorption and evaporation. This phenomenon is commonly referred to as 'creep' and prolonged creep leads to 'fatigue' (Özbek, 2014) which in turn causes rocks to undergo weathering, which can lead to the deterioration of its mechanical properties resulting in slope instability (Vergara and Trianafyllidis, 2015). Studies have established that rocks exposed to repeat drying-wetting cycles deteriorate more rapidly when compared to rocks in saturated conditions (Hua et al., 2015). Moreover, expansive and reactive soils (clay) that undergo periodic swelling and shrinkage during the alternate wet and dry environments, can result in severe damage to the slope stability (Erguler and Ulusay, 2009). Therefore alterations to local climates may alter the risk profile of slopes.

Climate change may lead to a thawing of permafrost over time in some regions, which is particularly concerning as thawing of permafrost can affect the stability of existing slopes and dams that were designed under the assumption of being continually frozen (Pearce et al., 2011). Melting permafrost also has implications for other water risks, such as contributing to flood waters or allowing unwanted migration of ground water. Some examples being the Diavik diamond mine that has resorted to actively freezing permafrost to prevent inundation of the site by surrounding lake waters (Haley et al., 2011), and the Red Dog zinc-lead mine in Alaska that has had higher than expected seepage due to heat generation from waste rock oxidation (Clark et al., 2011; Haley et al., 2011).

The potential impacts of climate change on transportation infrastructure, both on-site for mine operations and off-site for product export, have been highlighted as a particular concern for the Canadian mining industry (Ford et al., 2011). Hydrologic variability and climate change may impact transport infrastructure in a variety of ways. Mine wall collapses caused by high rainfall may damage or block access roads. Drought conditions can lower river levels and prevent barges accessing remote mine sites, as occurred at the Ok Tedi copper and gold mine in Papua New Guinea. Ice roads providing access to mines may experience decreases in their operating season, as has been observed at some Canadian mines (Haley et al., 2011). Breaking up of ice sheets may create hazards for shipping of mineral concentrates (Haley et al., 2011). Alternatively there may be some benefits to the industry's transport system from climate changes, such as the opening and development of the north-west passage that may shorten export shipping routes.

The recession of glaciers may make new mineral deposits accessible or easier to develop. Examples of this exist in Canada such as the Red Mountain gold deposit, the Brucejack gold-silver deposit and the Mitchell gold-copper deposit that forms part of the KSM project (the region containing this deposit may transition from being classified as Snow (Dfc) to being Warm Temperate (Cfb) by 2100 under the A1FI and A2 climate scenarios). On balance though the recession of glaciers has potentially significant impacts for mine-sites, due to the significant changes in runoff volumes and frequency that may occur through the life of the mine.

Examples also exist of mining directly reducing the extent of glaciers, such as the Kumtor gold mine in Kyrgyzstan where 39 million m^3 of ice was removed by 2011 (this equates to approximately 5 m^3 per ounce of gold produced; Kronenberg, 2013). In addition, although the Pascua-Lama mine on the border of Chile and Argentina initially intended to remove 0.8 million m^3 of ice during development, the removal was prevented after widespread outcry through the development of government policies to protect endangered glaciers. This led to the sterilisation of some 7% of the deposits contained gold (1.3 million ounces) (Kronenberg, 2013).

Extreme climate events such as cyclones or typhoons carry significant risks for mining operations. The impacts of typhoons and heavy rainfall in the Philippines have led to mine worker deaths, flooding of mines, production halts, transport and processing infrastructure damage, landslides and tailings spillages (Holden, 2015). Overall, 44% of mining projects in the Philippines are located in regions with at least a medium typhoon risk. This is particularly concerning as the expected impacts of climate change − such as sea level rise and increasing sea-surface temperatures − may increase the severity of storm surges, rainfall and winds associated with typhoons (Holden, 2015).

4.3. Water quality risks

The generation of water quality risks from mining operations is heavily dependent upon the local hydrology and climate, the geochemistry of the deposit being mined, techniques used for water and mine waste management, and the chemical and ecological nature of surrounding water bodies. A major source of water quality issues associated with the mining industry is acid mine drainage (AMD). AMD is generated when sulfide minerals − such as pyrite (FeS_2), pyrrhotite (FeS) or chalcopyrite ($CuFeS_2$) – undergo surface weathering and chemical or bacterial oxidation processes (Dold, 2014; Lottermoser, 2010; Nordstrom et al., 2015; Amos et al., 2015). Generation of AMD increases sulfate concentrations, lowers pH and can solubilise salts and metals from surrounding rock, thereby causing adverse consequences when these solutions migrate into surrounding environments. Today, the majority of the world's copper, lead, zinc and a large proportion of nickel are extracted from sulfide ores. Consequently the problem of AMD is relatively widespread throughout the mining industry (Akcil and Koldas, 2006; Benner et al., 2002; Da Rosa et al., 1997). A range of active and passive techniques are available for the prevention, treatment or remediation of AMD (Johnson and Hallberg, 2005; Lottermoser, 2010; Ziemkiewicz et al., 2003). The success or suitability of these techniques depends upon a variety of factors, including the local climate.

Climate change and variability can influence AMD risks through a variety of processes (Anawar, 2013; Lin, 2012; MEND, 2011; Nordstrom, 2009; Phillips, 2016), including:

1. Higher temperatures affecting rates of biologically- or chemically-driven sulfide oxidation, with a probable tendency to increase overall reaction rates,
2. Changes to precipitation altering pollutant migration pathways and rates,
3. Changes to pit lake levels arising from changes to evaporation and rainfall patterns (i.e. changed water balance) altering the direction of groundwater flow towards or away from open pits,
4. Increases in evaporation reducing or eliminating water bodies or moisture-retaining covers on mine waste facilities,
5. Capping layers on waste dumps cracking or degrading (see Section 4.2),
6. Changes to flow rates in receiving bodies altering rates of contaminant dilution (for better or worse),
7. Changes to the length of dry spells altering the magnitude of first flush effects on water quality (e.g. potentially increased solute concentrations due to extended evaporation periods).

Phillips (2016) identified that further work is required to assess the potential impacts of climate change on mining and surrounding regions– and also posed the question of whether there exists a climate threshold that could lead to large-scale potential for AMD generation. As part of our study we have provided some datasets that may contribute to understanding climate change-related AMD risks. The exposure to changes in Köppen-Geiger climate classifications provides some basic understanding of potential changes to temperature and precipitation that will influence a range of risk factors at mining operations. When combined with knowledge of mineral deposit types, the common mineralogy of these deposit types could be used to further understand mine-site water quality risks. As an example, Rayne et al. (2009) provided a conceptual framework to consider potential changes in run-off water quality by considering the rate of geochemical weathering of mine wastes and changes in climate and hydrology. Some other water quality issues such as erosion and transport of sediment can also be significant from surface mining, with potential to impact downstream ecosystems and to exacerbate changes to fluvial sedimentation processes caused by climate change (Phillips, 2016).

At present, it is very difficult to quantify the effects of changes in climate regimes on these aspects of mining and water resource risks. Changes in the climate of regions containing base metal resources (Figs. 5 and 6) can reasonably be assumed to alter water quality risks at mining operations through the processes described in the preceding paragraphs. However it is not possible to generalise whether climate change will lead to an overall increase or reduction in water quality risks across the industry due to the differing geochemistry, waste containment and management practices of individual mine sites. This means that further assessment of these risks at individual mine-site or regional scales is required before any industry-wide conclusions could be reached.

4.4. Climate change adaptation in the mining and minerals industry

The historical climate change focus of the mining industry has been on greenhouse gas emission mitigation efforts and reducing the potential exposure to carbon pricing schemes (ICMM, 2013; Ford et al., 2010, 2011). Comparatively, there has been relatively little attention given to assessing the potential climate change adaptation requirements of the mining industry. The International Council on Mining and Metals provided some guidance to the industry on how to assess the potential impacts, risks and adaptation requirements posed by climate change (ICMM, 2013). Mason et al. (2013) also developed a guide that showed through a range of case studies how the industry can adapt to climate risks and extreme weather events. Despite this the perception of climate change as a risk varies between industry stakeholders and across regions (Loechel et al., 2013; Ford et al., 2010, 2011).

A range of studies by the CSIRO have reported climate change adaptation in the Australian mining industry. This included activities such as assessing the potential climate changes in 11 different mineral producing regions of Australia (Hodgkinson et al., 2010), consultations with industry experts through workshop discussions (Hodgkinson et al., 2010), and surveying the perception of climate risks within the mining industry and surrounding communities (Loechel et al., 2013). Overall, adaptation efforts in the Australian mining industry have generally been reactive rather than anticipatory (Hodgkinson et al., 2014; Sharma and Franks, 2013), which is likely due in part to the need for improved information and understanding of the potential impacts to the industry from climate change (Loechel et al., 2013). Similar findings exist for the Canadian mining industry (Pearce et al., 2011; Ford et al., 2010, 2011).

Any climate change adaptation requirements in the mining industry are likely to be highly site specific. As an example, one adaptation measure may be the post-closure installation of a geo-synthetic cover to mitigate increased acid drainage risks associated with a changed local climate. This hypothetical scenario could result in a decrease of 13.9-14.9% of a copper project's net-present value (MEND, 2011). Further research and evaluation is required to understand the potential financial implications of climate change adaptation measures throughout the industry.

4.5. Implications for water footprint and life cycle assessment studies

Water footprint and life cycle assessment (LCA) methods attempt to quantify the relative impacts associated with production processes. Water footprint and LCA studies of the mining industry are increasingly utilising spatial water indices to assess the relative impacts associated with water use that occurs in different regions (Northey et al., 2016). The AWaRe, WSI, WDI, BIER, BIER-h and BWS indices have all been proposed for or used as part of these impact assessment procedures (Berger et al., 2014; Boulay et al., 2016a, 2016b; Hoekstra et al., 2012; Pfister et al., 2009; WULCA, 2016). There has been some debate over the best index to use for these purposes (Hoekstra, 2016; Pfister et al., 2017). As each index is measuring a different aspect of water use or hydrology, considering multiple indices simultaneously may lead to a greater understanding of water use impacts and trade-offs than when considering a single index in isolation.

Ideally, studies assessing water use impacts should use the highest level of spatial differentiation/resolution possible. However, indices developed based upon global hydrological modelling may not always be accurate when considering an individual region of concern for a particular study. For instance, modelling of the WSI for the Mississippi Basin at a sub-basin scale revealed significant differences when compared to the WSI estimate based upon a global hydrological model (Scherer et al., 2015). Therefore, studies considering water use in specific regions may benefit from recalculating indices based upon hydrologic models tailored specifically to the region under consideration. Boulay et al. (2015) identified regions where results determined using water indices are more sensitive to the spatial resolution, temporal resolution, water source differentiation (e.g. surface or ground water), water quality differentiation, data sources and the conceptualisation of the index.

Berger et al. (2014) proposed that the BIER and BIER-h be used to modify evaporation data in life cycle inventories to more accurately account for the contribution to regional water consumption. The BIER only accounts for evaporation that re-precipitates in the same drainage basin. However, evaporation may re-precipitate and replenish water stores in other water basins that also contain base metal resources. The global average continental evaporation recycling ratio has been shown to be approximately 57% (van der Ent et al., 2010), considerably higher than the 1% global average BIER (Berger et al., 2014). Due to this the perceived impacts associated with evaporation caused by base metal resource development may be very different depending upon if the geographic boundaries of assessment are drawn at local, regional, national, or continental scales.

Northey et al. (2014a) showed how the exposure of regions to water risk (measured using WSI) varies throughout metal supply chains, as different stages of the production process (e.g. mining and mineral processing, smelting, refining, etc.) may be located in different regions. Although based upon national production and WSI data (so not considering where mines are located in those countries), it was shown that on average copper production is occurring in countries with a higher WSI than lead, zinc or nickel production – which is broadly consistent with the results for the location of deposits presented by this study. Understanding how mineral supply chains are spatially distributed will enable more comprehensive assessment of water use impacts as part of LCA.

As mineral exploration continues and individual deposits are mined and depleted, the spatial distribution of identified and remaining mineral resources will change through time. Concurrently, studies have assessed potential changes in the future distribution of water use, stress and scarcity as a result of climate change and socio-economic developments (e.g. Alcamo et al., 2007; Ercin and Hoekstra, 2014; Hejazi et al., 2014; Kiguchi et al., 2015). This means that any assessments of future or long-term resource extraction should ideally account for these changes during inventory development and impact characterisation.

4.6. Implications for long-term resource development

The copper resource dataset has previously been used to assess scenarios for long-term copper supply and demand (Northey et al., 2014b). Incorporating regional data for contextual water risks into this style of assessment and combining this with estimates of water use requirements would provide valuable insights into the potential water constraints and impacts associated with developing mineral resources.

A number of other metals commonly occur within copper, lead-zinc and nickel resources and these are often produced as by-products. Thus to some extent, the inferences we have drawn for these four base metals extend also to their by-products. With approximately one quarter of the periodic table supplied as by-products or co-products of the mining of copper, lead-zinc, and nickel (Nassar et al., 2015), this suggests that the outcomes of our study are potentially quite far reaching. However, a limiting factor is the extent to which each by-product is reliant on each base metal for supply, and the differential spatial distribution of the by-products to the base metals. A full assessment of water-related risks for the by-product metals would therefore require a more detailed understanding of the quantities of by-product metals contained within the deposits identified in this study. This can be problematic to determine as they are often not reported within the mining industry (Mudd et al., 2016). However, steps are being made towards quantifying these metals (Werner et al., 2017), meaning that it may be possible to infer water related risks in detail and the implications for resource development for many more metals in the future.

5. Conclusions

The societal costs and benefits associated with developing mineral resources are not solely related to the nature of individual deposits and prevailing market conditions, but rather they are also a function of the local contexts of the location of these deposits. This study has provided a quantitative assessment of the water and climate contexts associated with global copper, lead-zinc and nickel resources. These resources are located across a diverse array of hydrological and climate contexts. The various indices show that copper resources are, on average, located in regions with more water stress, scarcity and risk than regions containing lead-zinc or nickel resources. In addition, regions containing copper and lead-zinc resources are potentially more exposed to changes in climate classification over the coming century than those containing nickel resources.

The impacts of climate change may be adverse for mines in some regions (e.g. increased evaporation and external water requirements), whereas in other regions these changes could be beneficial for managing various water risks at mine sites (e.g. potential decreases in AMD risks). Further work is required to understand the full extent of these risks and the likely impacts for mines in individual regions. However, we emphasise that reactive approaches to mine site water management, such as assuming a continuation of historic climate conditions and responding to weather conditions as they occur, should be avoided. Rather, mining operations should further embrace anticipatory risk management strategies and plan to be resilient to − or potentially even benefit from – the expected changes in regional climates. These plans should be developed well before mining commences.

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Appendix A. Supplementary data

Supplementary data associated with this article can be found, in the online version, at http://dx.doi.org/10.1016/j.gloenvcha.2017.04.004.

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resources

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Metal Criticality Determination for Australia, the US, and the Planet—Comparing 2008 and 2012 Results

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Abstract: Episodic supply shortages of metals and unsettling predictions of potential supply constraints in the future have led to a series of recent criticality evaluations. This study applies a consistent criticality methodology to the United States, Australia, and to the global level for both 2008 and 2012. It is the first time that criticality assessments are presented for Australia, a country that contrasts with the United States in terms of its mineral deposits and metal use characteristics. We use the Yale criticality methodology, which measures Supply Risk (SR), Environmental Implications (EI), and Vulnerability to Supply Restriction (VSR) to derive criticality assessments for five major metals (Al, Fe, Ni, Cu, Zn) and for indium (In). We find only modest changes in SR between 2008 and 2012 at both country and global levels; these changes are due to revisions in resource estimates. At the country level, Australia's VSR for Ni, Cu, and Zn is 23%–33% lower than that for the United States, largely because of Australia's abundant domestic resources. At the global level, SR is much higher for In, Ni, Cu, and Zn than for Al and Fe as a consequence of SR's longer time horizon and anticipated supply/demand constraints. The results emphasize the dynamic nature of criticality and its variance between countries and among metals.

Keywords: aluminum; iron; nickel; copper; zinc; indium; metal demand; supply risk

1. Introduction

There have been many efforts in recent years to evaluate the criticality of elements and materials in light of material scarcity and predictions of potential supply constraints. These evaluations are attempts to identify materials that have a comparatively high economic importance combined with a comparatively high risk of supply disruptions [1]. Among the governmental criticality assessments most widely recognized are those of the U.S. National Research Council [2], the U.S. Department of Energy [3], the European Commission [4,5], and the British Geological Survey [6,7]. Some of these studies were used by the Australian Government as a foundation to examine critical resources for Australia, given this country's relevance in the global supply of mineral commodities [8]. Complementary academic studies are rapidly evolving and various versions of criticality methodology have been presented [9–17].

Most criticality research has been restricted to lists of metals, with a few exceptions that examined several non-metal materials (e.g., coking coal, natural rubber, pulpwood) [4,5]. Although a definitive list of "critical metals", stable over time, would be very helpful to industries and governments, such a list is unrealistic. New ore deposits are discovered, revisions occur in geopolitics and legislation, technological evolution induces changes in demand, and so forth. Therefore, because criticality is a dynamic state, it is desirable that the results from criticality assessments be periodically updated [18]. The European Commission has attempted to respond to this situation by performing an initial evaluation in 2010 and an update in 2014 (the results showed little change) [4,5]. The British Geological Survey [6,7] also performed evaluations four years apart, but for metal supply only. There was some shuffling of rankings between the two assessments, but a change in methodology makes comparison of results from the two years problematic.

An additional attribute of criticality is that the degree of concern for a given metal depends on who is making the inquiry, as results can legitimately differ at corporate, national, regional, and global levels [19].

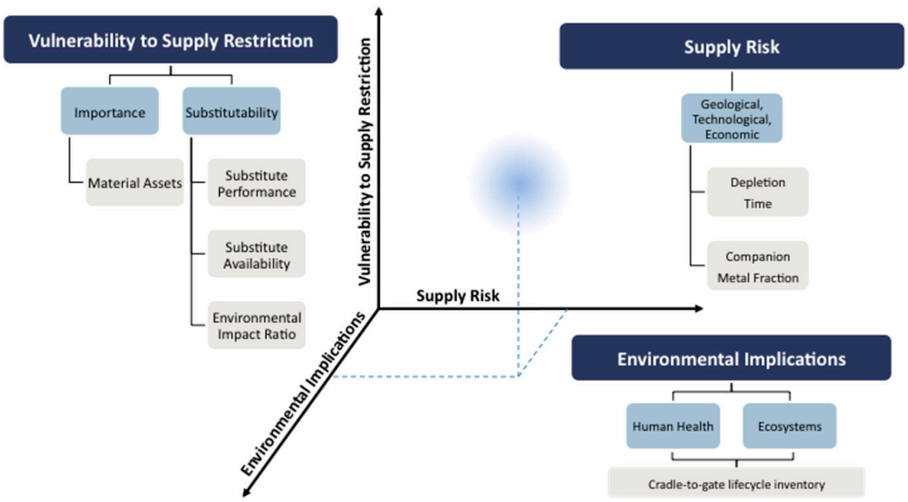
In the present work, we respond to some of these issues in the following ways:

- We update our previous 2008 results [19–22] to 2012 for the United States and for the planet;
- We generate criticality assessments for 2008 and 2012 for Australia;
- We compare the results over time at the global level, at the country level, and metal to metal;
- The assessments are performed for six metals: aluminum, iron, nickel, copper, and zinc, used in large quantities throughout the world in major industrial sectors such as construction, infrastructure, electrification, and transportation, plus indium as an example of a metal with higher scarcity concerns and employment in very specialized applications.

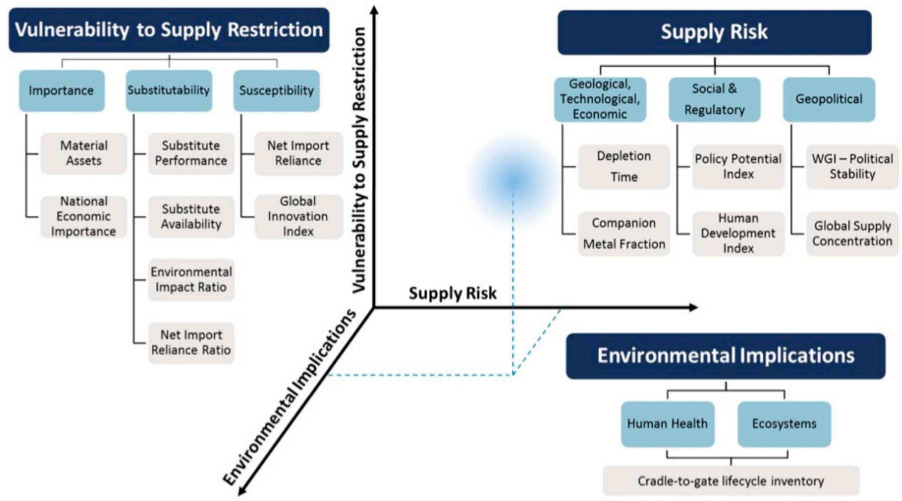
2. Methodology

The Yale methodology, which we employ here for metal criticality assessment, is shown in Figure 1a (global level) and Figure 1b (country level). The methodology addresses three criticality dimensions: Supply Risk (SR), Environmental Implications (EI), and Vulnerability to Supply Restriction (VSR). Each of these is the result of the aggregation of several indicators, and those in turn are aggregations of several metrics. A detailed description of the methodological details is given in Graedel et al. [11] with updates given in Harper et al. [21].

A feature of the criticality methodology worth noting is that the evaluations of the SR and EI axes at the global level depend entirely on the geological and environmental properties of the metal. The EI axis evaluation at the national level is identical to that at the global level because the lack of mine-specific environmental impacts limits the EI assessment to inherent toxicological and energy use impacts. The national SR axis takes into account not only geology of ore deposits but also various properties of the countries from which the metal is extracted and processed (Figure 1b). These values are weight-averaged by each producing country's share of worldwide production. As a result, national SR ratings for all countries are identical. Using country variability is addressed by the VSR axis in such metrics as Economic Importance and Import Dependence, as shown in Figure 1b.



(a)



(**b**)

Figure 1. The Yale criticality methodology: (**a**) global level (reprinted by permission from Graedel et al. [20]; (**b**) country level (courtesy of Nedal T. Nassar, and as reported in Harper et al. [21].

A review of a number of methodological approaches to the assessment of criticality [18] generated a list of ten desirable attributes such as transparency, evaluation of uncertainty, and breadth of scope. The Yale methodology as applied in previous work for different metal groups [19–22] satisfied nine of the ten. The tenth desirable attribute, periodic updating, is served at least in part by the present work. An alternative discussion of methodological challenges in measuring criticality is given by Helbig et al. [17].

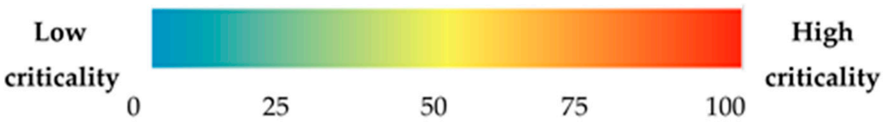
3. Results

For the present study, we have evaluated criticality for 2012 for Australia, the United States, and the world for aluminum, iron, nickel, copper, zinc, and indium. The numerical values for the three criticality dimensions (on a scale of 0–100, with higher values indicating more concern) are given in Table 1, which also reproduces previous 2008 results for the United States and the world [19–22], and adds those for Australia generated in this work. To our knowledge, criticality results for Australia (for both 2008 and 2012) are the first determinations for the six metals for this country. In fact, the Australian Government's assessment relied on previous criticality results of overseas studies [8], while Australia-specific data and metrics are used as inputs in this analysis.

We compare the results for different years, different countries, and different metals in the sections below. The numeric values for the indicators and metrics that underlie the Table 1 aggregated values are given and discussed in the Supplementary Materials.

Table 1. Values of the criticality dimensions (0–100 scale, with higher values indicating higher concern) for six metals: United States (US 2008, 2012), Australia (AU 2008, 2012), World (2008, 2012). SR: Supply Risk; EI: Environmental Implications; VSR: Vulnerability to Supply Restriction.

	Criticality Dimensions				Criticality Dimensions		
	SR	**EI**	**VSR**		**SR**	**EI**	**VSR**
Aluminum				**Copper**			
US (2008)	43	3	58	US (2008)	54	17	60
US (2012)	44		47	US (2012)	55		57
AU (2008)	43		47	AU (2008)	54		53
AU (2012)	44		43	AU (2012)	55		49
World (2008)	0		36	World (2008)	24		48
World (2012)	0		37	World (2012)	29		48
Iron				**Zinc**			
US (2008)	46	1	52	US (2008)	59	3	53
US (2012)	56		55	US (2012)	61		52
AU (2008)	46		53	AU (2008)	59		43
AU (2012)	56		54	AU (2012)	61		41
World (2008)	0		57	World (2008)	46		51
World (2012)	0		55	World (2012)	47		52
Nickel				**Indium**			
US (2008)	50	10	47	US (2008)	77	22	54
US (2012)	55		52	US (2012)	78		56
AU (2008)	50		41	AU (2008)	77		54
AU (2012)	55		40	AU (2012)	78		55
World (2008)	1		42	World (2008)	98		55
World (2012)	1		43	World (2012)	98		56



3.1. Comparing Criticality over Time

We utilize the information derived in the present study to compare criticality values at two points in time. At the global and United States levels, we have a 2008 determination available from previous work [19–22] and the 2012 determination accomplished herein. In the case of Australia, we have generated 2008 and 2012 determinations for consistency purposes. We note that the EI evaluation has not changed between 2008 and 2012 for any of the metals because the underlying databases have not been revised during that period, and because the processes that are employed during metal extraction and processing have long lifetimes. Therefore, we can ignore the EI axis in displaying and discussing the results of this study.

In Figure 2 (and Table 1) we give the SR-VSR results for the six metals as evaluated for the world, the United States, and Australia for the two different years. Most of the differences are certainly not dramatic. Overall, there is a modest increase in SR for a couple of the metals, and VSR becomes more uniform at levels between 50 and 60.

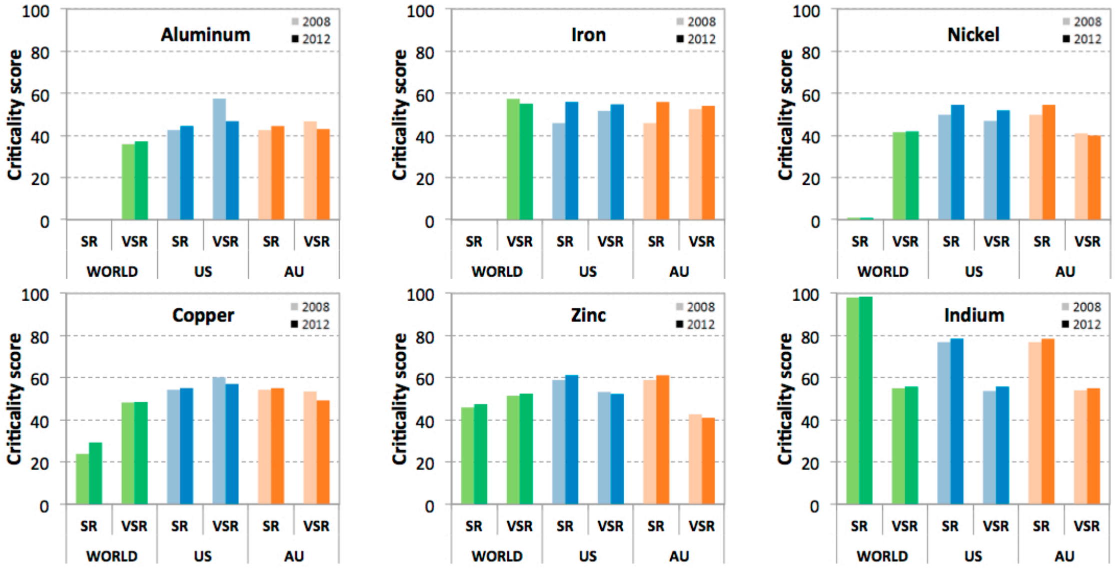


Figure 2. 2008 and 2012 results for Supply Risk (SR) and Vulnerability to Supply Restriction (VSR) at the global, the United States (US), and Australian (AU) levels for the six metals. For each pair of bars, the 2008 result is on the left and that for 2012 on the right.

The global level results (green bars shown in Figure 2) are dramatically different from those of the United States and Australia. In general, SR is less of a concern at the global level, with the exception of indium and the numeric values of the metals are considerably more diverse. There is little concern of SR for iron, aluminum, and nickel. SR for copper is markedly higher (though lower than is the case for the United States) and SR of zinc is higher still, near that of the United States. SR for indium remains of very high concern at the global level, but substantially lower scores are computed for the United States and Australia. There is little variation in either SR or VSR between 2008 and 2012 at the global level.

3.2. Comparing Country Level Criticality

The results displayed in Figures 2 and 3 enable a comparison of the criticality assessments between the United States and Australia in 2012 as a snapshot of the "current" criticality determinations. As mentioned above, SR and EI values for all countries are identical (SR is weight-averaged by using global-average metal supply rather than using an individual country's import source mix because potentially a using country's import sources are not constant). For VSR, there are small differences (both in Australia's favor) for aluminum, iron and indium. For copper, zinc, and nickel, however, VSR differences are larger, with Australia's VSR for these metals 23%–33% lower than those for the United States. This is due largely to Australia's large domestic ore stocks of these metals [23]. The United States also has geological stocks of copper and zinc (but not of nickel); however, the domestic metal production is not sufficient to avoid imports of all three metals.

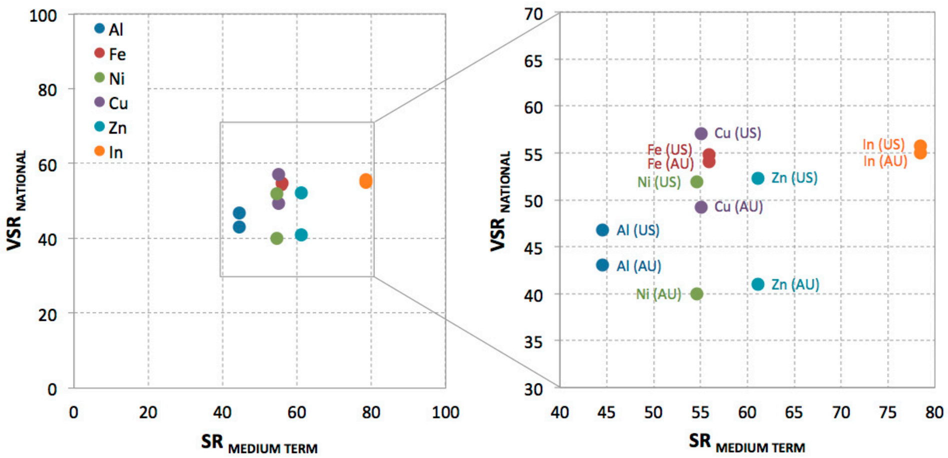


Figure 3. Criticality results for the six metals for Australia (AU) and the United States (US) for 2012, with full axes view on the left and expanded view on the right.

3.3. Comparing Criticality Metal to Metal

The criticality values for the United States and Australia illustrate various features of more general results for the metals evaluated in our research (Figure 3). The five major metals in our study are extensively used in global buildings and infrastructure: because these sectors have long lifetimes of products in use and a well-established demand-supply pattern worldwide it would be anticipated that vulnerability to supply restriction would be moderate or lower at both spatial levels and for both countries, and that is what is observed in the present study. In the United States, for the six metals and the two years of assessment, all VSR values are between 47 (aluminum) and 60 (copper). Australia's VSR values are lower for both years because this country dominantly relies on domestic metal sources, falling between 40 (nickel) and 55 (indium).

SR results for the medium-term perspective, used in conjunction with the country level assessment [11], are influenced by several geological, technological, economic, social, regulatory, and geopolitical metrics (see Supplementary Materials). In particular, the relative availability of a metal is determined by the amount of time that it would take to deplete the geological reserves (according to the USGS definition [23]) at the current level of demand and secondary metal supply (i.e., recycling). For the five major metals of our study, SR values are in the 40–60 range. SR for indium is much higher (78), as (i) its reserves are lower than those of the other metals; (ii) it is mainly extracted as a by-product of zinc, tin, and copper; and (iii) because indium recycling at end-of-life is very limited.

Similar variations exist among the metals at the global level (Figure 4). Supply risk for aluminum, iron, and nickel is negligible as a consequence of their large global abundance (in the global level assessment, the reserve base is used [11]), small for copper, and moderate for zinc. Supply risk for indium is much higher as its reserves are lower than those of the other metals, confirming the challenge of securing access to this (by-product) metal worldwide.

VSR levels for four of the metals (iron, copper, zinc, and nickel) are squarely in the midrange values of 37–56. Iron has the highest VSR as a result of its very wide use in buildings and infrastructure. VSR is lowest (though quite moderate) for aluminum and nickel, which is somewhat less widely used than the other metals. Indium has a VSR score comparable to that of other metals investigated. One could expect a higher result following the outcomes for SR, but indium is much less prevalent in modern society compared to the major metals, which reduces concerns due to global vulnerability to potential supply shortages.

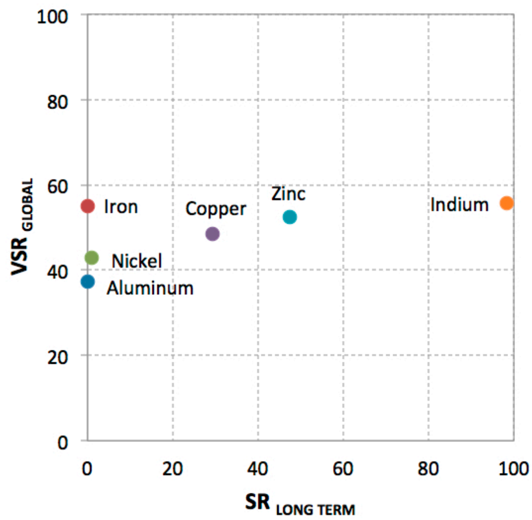


Figure 4. Criticality results for the planet for 2012.

4. Discussion

The modest criticality changes seen over time in this study of five major metals and the high criticality evaluation for indium are consistent with European Union studies, in both cases using a consistent methodology over a four-year interval [4,5]. Results from the British Geological Survey are also similar over a four-year period [6,7], although changes in methodology make that conclusion less robust. In any case, these results from three different research groups provide strong support for assuming that changes in criticality over a several year time period (and perhaps longer?) are likely to be relatively small.

When comparing criticality across countries at a similar time period, significant differences are seen in the present work for major metals in Australia and the United States, and in Nassar et al. for rare earths in China and the United States [24]. The country level differences are largely related to the presence or absence of domestic ore deposits sufficient to meet domestic demand and to the presence or absence of a well-established production chain for the most common forms of metals employed in the creation of finished products. Thus, Australia possesses low criticality for major metals, for which it holds large deposits and it is a major global exporter, and China for the rare earths, with which it is well endowed. However, notwithstanding that Australia has the world's largest reserves of zinc and bauxite from which indium and gallium (which is potentially a good substitute of indium in solder and alloy applications, electrical components and semiconductor manufacturing [20]) are mainly extracted, the current Australian indium and gallium refinery capacity is very limited and constrains the exploitation of this potential. As a result, indium criticality is greater than what would likely be determined in the event that extensive indium recovery took place to match the possible magnitude of indium resources in this country [8]. This is, in turn, influenced by several technological, economic, and policy developments that will likely change over time and could influence the country's reliance on domestic sources to provide goods and services (and hence Australian vulnerability to potential indium supply restriction). On a wider level, because Australia is a major contributor to global base metal production, such a change would also have effects in the global supply of critical commodities.

Comparing criticality evaluations for different metals is somewhat more nuanced than between years or countries. In particular, metals have very different properties with regard to principal uses, recycling potential, substitution potential, sourcing countries, by-product production, and so forth. For abundant metals, especially those with deposits widely distributed around the planet, a high degree of criticality is unlikely, and indeed neither this study nor those of the European Commission [4,5] or the British Geological Survey [6,7] find aluminum, iron, nickel, copper, or zinc to be of substantial concern. The situation changes, however, when considering by-product metals (e.g., indium or rhenium), or metals with no workable substitutes for their major uses (e.g., rhodium or selenium). Given these additional complexities, methodological differences assert themselves to a greater degree, and the relative level of concern for elements such as vanadium and molybdenum varies significantly among different assessments. However, even though some variation would be expected, it is clear that methodological variation is a legitimate concern in assessing the criticality of the scarcer elements.

5. Conclusions

This study has demonstrated that criticality results for the six metals assessed do not appear to change markedly over a four-year interval at the global level. The results do, however, differ significantly at the country level. Analysis of a more diverse set of countries using a consistent methodology is necessary to determine the possible extent of differences between countries. Various studies of the criticality of different metals demonstrate differences among geographical entities (countries, regions); some of this is understandable as inherent country/region differences, but variations may be attributable as well to different analytical methodologies, as noted in the Introduction.

Finally, we note that even though the changes in the criticality parameters for Australia, the US, and the planet are not great over this four-year time period, some were significant enough to be noticeable in the results obtained. This fact emphasizes that criticality is not a fixed set of parameters, although criticality may be relatively stable year to year and perhaps even longer time periods. Periodic updates, perhaps at five years intervals, would seem a judicious practice.

Supplementary Materials: The following are available online at www.mdpi.com/link.

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Author Contributions: T.E.G. and B.K.R. conceived and designed the experiments; L.C., P.N. and T.T.W. performed the experiments; L.C. analyzed the data; T.E.G., B.K.R., L.C. and T.T.W. wrote the paper.

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The world's by-product and critical metal resources part I: Uncertainties, current reporting practices, implications and grounds for optimism

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ABSTRACT

Many of the metals extracted as by-products and rated as 'critical' are vital to important modern technologies but are seldom reported as extractable commodities by the mining industry. This creates numerous uncertainties and challenges in estimating the global resources of these metals; here we outline the current approaches (or lack thereof) used in resource reporting and identify and discuss the uncertainties surrounding critical co- and by-product reporting in detail. We present a review of ore reserve and mineral resource reporting in the mining industry, including the identification of various methods by which resource accounts have been constructed, a discussion of examples and differences between countries, states and organisations that conduct these assessments, and a discussion of the various ways in which these resource data have been interpreted. This is followed by a series of case studies that document and discuss the reporting of selected critical metals that suggest that the lack of reporting is dominantly as a result of the perceived or actual low monetary value of by-products at the mine site rather than as a result of any lack of these metals, indicating that different types of uncertainties can arise in estimating the global resources of these critical commodities.

This paper also addresses (and continues in parts II and III) the numerous issues and uncertainties identified in this study by outlining approaches and alternative data sources that can be used to develop more comprehensive assessments of critical metal resources from deposit to global levels. The hybrid methodologies proposed in this paper provide reasons for some optimism in that by-product and critical metal resources can still be identified to a reasonable degree of accuracy, and these resources are likely to be sufficient to meet demand for some decades. However, there is still a strong case for improved reporting of by-products and critical metals in mineral deposits to assist in these efforts and to clarify the true global position in terms of the future security of supply of co- and by-product metals, invariably including the critical or e-tech metals.

1. Introduction

There are a wide range of factors that a potential mining company needs to consider and carefully assess during the determination of whether (or not) a particular mineral or metal is economically worth the effort of extracting it from the earth in some manner (e.g. Edwards, 2001). At a basic level, this includes finding a body of material enriched in one or more elements or minerals that are mineable (by open cut, underground or in situ extraction methods), processable into a saleable product (e.g. concentrate or metal), profitable (i.e. markets must demand the product at a price greater than the costs of mining; unless mining for other strategic reasons such as national defence or socio-political goals), and which meet the demands of a range of other potential technical, environmental, financial, governance and social factors. Over time, particularly in the last 50 years, the mining industry has developed considerable technical capacity and experience in such assessments, and the requirements for these are now typically enshrined in codes, instruments or guidelines by law, regulation or good industry practice.

In general terms, mining uses the concepts of ore reserves and mineral resources to assess the potential for exploiting a mineral deposit – and for the major metals and minerals, this is now routine and straightforward thanks to the adoption of formal codes and guidelines. There are, however, issues when it comes to metals which are not normally the target of mining projects. For example, indium (In[1]) is often associated with zinc (Zn) deposits and typically deports to Zn concentrates, with the indium potentially recoverable at a Zn refinery (if they have an indium recovery circuit; e.g. Werner et al., 2015) – yet to a mine, the value of indium is typically negligible to very small and is (in the main) not recognised in prices paid for Zn concentrates. In this case, indium is often termed a 'companion metal', as it occurs with Zn but is not

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[1] Although the chemical symbol for the majority of elements are used throughout this paper, we have elected to spell out indium to avoid confusion with the word "in" or the abbreviation of the natural logarithm, "ln".

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the primary metal of interest to a mining company (e.g. Nassar et al., 2015). Many of the metals increasingly used in modern technology are companion metals, including indium, gallium (Ga), tellurium (Te), germanium (Ge), selenium (Se), cadmium (Cd), cobalt (Co), hafnium (Hf), rhenium (Re), and the rare earth elements (REEs, more commonly referred to as rare earth oxides or REOs), amongst others. Furthermore, current supply or resources of many of the companion metals are concentrated in just a few countries (e.g. indium, REOs and China; Re and Chile), leading to concerns over security of supply to continue to support modern civilian and military technologies – with such metals commonly referred to as 'critical metals'. In the mining industry, however, there is very little data reported on such companion or critical metals in ore reserves, mineral resources or mine production statistics – leading to false concerns or confusion between current supply and future supply potential.

Here, we provide a background to ore reserve and mineral resource reporting in the mining industry, review different sources of such estimates and methods of compiling national or global assessments, and present resource-reserve data for various deposits that do report critical metal data before finally discussing the implications overall for more robust global estimates of the resources-reserves of critical metals. Rather than adopt a pessimistic outlook, we highlight the many technical grounds for optimism for adequate supply of critical minerals and metals for some decades. The following sections form the theoretical background to a multi-part series of studies examining the global resources of by-product and critical metals. In part II (Werner et al., under review a), we develop a methodology to estimate by-product/critical metal resources which explicitly overcomes the uncertainties identified in this study that is then demonstrated in greater detail for the case of indium in part III (Werner et al., under review b).

2. Background and rationale

In general, the primary concepts to assess the potential exploitation of a mineral deposit are ore reserves and mineral resources, as outlined in codes or instruments such as Australia's Joint Ore Reserves Committee (JORC) Code (AusIMM et al., 2012), South Africa's Mineral Resource Code (SAMREC; SAMRCWG, 2009), Canada's National Instrument 43-101 (aka NI43-101; OSC, 2011), and the associated CIM[2] Code (CIM, 2014), or the Pan-European Reserves and Resources Reporting Committee (PERC, 2013), with similar codes in other countries (e.g. USA, Chile, and Russia). The 1994 establishment of the Committee for Mineral Reserves International Reporting Standards (aka "CRIRSCO") through the Council of Mining and Metallurgical Institutes has enabled international cooperation on codes for the reporting of mineral reserves and resources, with the specific codes listed above being CRIRSCO-approved. Ore reserves are well studied, have a definite economic basis and typically form the basis of a pre-feasibility study, feasibility study or detailed mine plans, whereas mineral resources are less certain in some respects (e.g. economically) but are well understood in others (e.g. geologically, metallurgy) and have reasonable prospects for eventual economic extraction. For a particular deposit, this generally leads to a tonnage being reported at a given grade for a contained amount of a metal(s); e.g. mineral resources of 100 million tonnes (Mt) at 1% copper (Cu) for 1 Mt contained Cu with ore reserves of 25 Mt at 1.2% Cu for 0.3 Mt contained Cu.

Wider assessments of national or global endowments are often undertaken by government agencies (such as geological surveys or mining/resources departments) or industry associations; these organisations often compile data sets for their given country or less frequently for selected regions or the entire globe, yielding estimates of total endowment at that time for a particular metal or mineral (e.g. the United States Geological Survey (USGS), Geoscience Australia (GA), Natural Resources Canada (NRC), or the Indian Bureau of Mines (IBM)). For some commodities, especially energy, there are also international agencies and industry associations that conduct similar assessments, such as the International Energy Agency, International Atomic Energy Agency, Chamber of Mines of South Africa (CMSA) or the World Coal Association. Although some corporate groups also conduct resource assessments by country or globally (e.g. for financial, exploration, policy or other purposes), these are rarely made public, as these data are seen as valuable and are often sold to businesses for profit. These factors mean that national or global level assessments almost always only report contained metals or minerals rather than detailed breakdown by mineral deposit types or even by individual deposit.

The inherent problem with this approach (i.e. only reporting contained tonnes of commodities) is that without expert knowledge of resource-reserve reporting and the detailed data and assumptions behind government or similar assessments, it is easy to misunderstand (or perhaps deliberately misinterpret to make a point) resource endowments and their ability to supply likely future demands. For example, Cohen (2007) in New Scientist presented an infographic suggesting that the world had 'X to Y' years of commodity 'Z' remaining, with examples including 4 to 13 years left for indium, 38 to 61 years left for Cu or 42 to 360 years left for platinum (Pt), while for some metals such rhodium (Rh), Ge, Ga and Hf there was no data available to estimate 'how many years left' (despite Rh being part of the platinum group elements (PGEs) and well quantified for such a crude estimate), with a similar story published by Moyer (2010) in Scientific American. It is now some 9 years since Cohen (2007) and indium has not been 'depleted' (or arguably even close to it, especially as global indium production continues to grow). More recently, Ragnarsdóttir and Sverdrup (2015) somewhat controversially[3] suggested that the known resources of strategically important metals, materials and fossil fuels, all of which are crucial to modern life, are already exhibiting "peak behaviour", where production of a given resource reaches a maximum before decreasing into the future due to resource depletion (see also Bardi, 2014, Giurco et al., 2012, Mohr et al., 2015, Northey et al., 2014). This is somewhat echoed by an earlier white paper by Nickless et al. (2014), who suggest that peaks in the production of a number of the most commonly produced commodities could occur within the next three decades, although a more balanced report by Meinert et al. (2016) outlines some of the issues with this hypothesis and the findings of Ragnarsdóttir and Sverdrup (2015) and other more sceptical publications on this topic. The more sceptical of these reports are based on a number of admittedly crude assumptions (e.g. current reported reserves or similar with no future increase, constant demand, ignore additional mineral resources) and also generally (especially Cohen, 2007, and Moyer, 2010) fail to articulate any understanding of mineral deposit resource-reserve reporting, the long-term trends in such national and global estimates, ongoing discovery success, new technology, the role of economics, or geological considerations (such as lower grades generally being associated with vastly bigger deposits) – much of which is well understood by companies and the mining industry as a whole (e.g. as discussed by Meinert et al., 2016). The overall economics of mineral exploration is also generally poorly understood by the broader scientific community and the general public, as discussed by Nickless et al. (2014) and Meinert et al. (2016); for example, scaremongering relating to a perceived exhaustion of a given commodity is frequently based on the fact that mineral exploration and mining companies only delineate enough of a deposit in resources or reserves that might be extracted in the short to medium term rather than spending exorbitant amounts to drill-out and define an entire deposit prior to commencing mining, an approach that would provide more accurate estimates of life-of-mine and contained metal (and hence more accurate estimates of "how much 'Z' is remaining") but would never be economic!

[2] CIM is the Canadian Institute of Mining, Metallurgy & Petroleum.

[3] As evidenced by the letters in response to this article published in Geoscientist; see www.geolsoc.org.uk/letters.

All of this means that there are very few published studies which compile detailed global data sets on reported ore reserves and/or mineral resources on a deposit by deposit basis. The closest example is the CMSA, which used to publish every quarter year the proven gold (Au) ore reserves by company and deposit/project but for its corporate members and their South African operations only (e.g. CMSA, var.) – thereby excluding all other Au sources such as probable ore reserves, additional mineral resources at other deposits as well as the Au contained in the extensive Bushveld PGE deposits (see Mudd, 2012). Recent studies which do compile such detailed resources data sets include: Cu (Mudd et al., 2013a), Co (Mudd et al., 2013b), uranium (U; Mudd, 2014), nickel (Ni; Mudd and Jowitt, 2014), REOs (Weng et al., 2013, 2015), lead (Pb)-Zn (Mudd et al., 2016), indium (Werner et al., under review b) and PGEs (Mudd, 2012) – with these studies typically showing greater reported resources than almost every other source. For example, CMSA states South Africa's Ni reserves as 3.50 Mt Ni (CMSA, 2013), yet data from Mudd (2012) and Mudd and Jowitt (2014) shows that the Bushveld has reported ore reserves containing 4.95 Mt Ni with an additional 31.27 Mt Ni in mineral resources (all as a co/by-product to PGEs). Similarly, using data for Ni-containing PGE deposits from Mudd and Jowitt (2014), 2013 South African ore reserves of PGEs are estimated at 9884 t PGEs with an additional 62,455 t PGEs in mineral resources (i.e. total ~ 72,339 t PGEs), yet both CMSA and the USGS report South African reserves as 63,000 t PGEs (CMSA, 2013, USGS, 2014).

Furthermore, mining companies typically focus on the dominant economically valuable metals or minerals at a given project, such as Au or Cu, and frequently may ignore commodities which have a relatively small market (e.g. Se, Co), low value (e.g. silver [Ag]) or are potentially difficult to extract economically (such as indium, Ga, Ge, Cd, Hf, etc.). One extreme example is provided by the value of REOs contained in the deposit at Olympic Dam in South Australia; depending on price assumptions, the value of these elements within Olympic Dam is of a similar magnitude (or greater) than Cu–U–Au–Ag production and resources-reserves (see Weng et al., 2015). BHP Billiton, however, has chosen not to pursue the extraction of REOs, primarily as "the technology available to recover these is not economically viable at this point in time" (page 81, BHPB, 2011) – although no technical details or economic justification of this claim has been provided to date. A similar case is presented by Werner et al. (under review b), who demonstrate that reported indium mineral resources significantly underestimate the true picture, primarily as the vast majority of companies do not report indium grades, even from mines that produce indium as a valuable by-product. In addition, this situation is exemplified by the fact that many Ni laterite mines focus on Ni and ignore Co (often related to their ore processing configuration; see Mudd, 2010, Mudd et al., 2013b).

It is also common for many by-product metals to exhibit significantly more price volatility than their primary metal hosts (e.g. Co; Mudd et al., 2013b), meaning mining companies are often wary of entering such volatile commodity markets. In essence, this means that global by-product metal production rates are intrinsically tied to their primary metal host in physical terms but the economic drivers behind increasing production of a by-product metal are complex and not always closely correlated. For example, if there is a reduction in demand for a primary metal at the same time as the by-product demand increases, this will limit the supply of a by-product metal (unless other sources or stockpiles can be sourced).

As society's demands for a wide range of minerals and metals continues to grow to meet ever-increasing technological complexity (e.g. renewable energy, pollution control, safety, strength, etc.), this has led to certain metals being labelled as 'critical' due to their crucial role in meeting modern needs, although the precise definition of criticality differs from organisation to organisation and from country to country (e.g. Jowitt, 2015; Sykes et al., in press). For example, Pt is used as a catalyst for pollution control in vehicles and to improve production efficiency in oil refineries but is mostly mined by South Africa (Mudd, 2012); REOs are crucial in consumer electronics, specialty alloys and certain military technologies but China maintains a near global monopoly on supply (Weng et al., 2015); global indium supply is also dominated by China (from Zn mining and refineries; Werner et al., 2015); while Re is dominated in supply by Chile from their extensive by-product molybdenum (Mo) extracted from Cu mines (Polyak, 2016). Thus concerns have been raised over the ability to reliably supply the various metals needed for growing technological needs – leading to a variety of studies on which metals are indeed 'critical', and to whom.

In 2008, the US National Academy of Science (NAS) released its study, "Minerals, Critical Minerals, and the U.S. Economy" (USNRC, 2008), using two aspects to examine whether a mineral or metal was at critical risk of affecting the US economy, namely the importance of use and supply (i.e. availability). Based on this approach, a mineral would be classified as critical if supply was at high risk or the use was fundamental in specific technologies (e.g. renewable energy, military, etc.). The preliminary work suggested that Rh, Pt, manganese (Mn), niobium (Nb), indium and REOs were critical. Recent work by Graedel et al. (2012) has further developed and enhanced the NAS methodology for determining the 'criticality' of a particular mineral or metal, expanding the aspects to supply risk, environmental implications, and vulnerability to supply restriction. Many other countries or regions have also examined the issues surrounding critical minerals and metals, often from their own perspective as a material consumer or manufacturing centre rather than as a large mining supplier. However, what elements are considered critical varies from country to region, as is clearly demonstrated by the recent spate of reports on critical metals (e.g. BGS, 2012, EC, 2014, Skirrow et al., 2013, USDoD, 2014, USDoE, 2011), although in general the REOs, Ga, indium, tungsten (W), PGEs, Co, Nb, magnesium (Mg), Mo, antimony (Sb), lithium (Li), vanadium (V), Ni, tantalum (Ta), Te, chromium (Cr), and Mn are all considered "critical" to our way of life in some manner. This inherent uncertainty in what truly is critical is further compounded by two common themes that are present in many of these reports and similar studies, namely (i) a lack of focus on detailed resource-reserve data, and (ii) confusion on the difference between existing mine supply versus reported mineral resources. As noted above, a mining company typically ignores minor metals such as indium or Cd due to their low (or even negative if processed at a smelter where penalties may apply) value compared to major metals like Cu, Au or Zn, meaning that the ability to compile detailed resource-reserve and associated mining production data can be very limited – and this is often confused with limited known ore reserves, mineral resources or supply; i.e. confusion may exist between scarce data and a scarce commodity. Thus, although there is a limited ability to understand and quantify the global resources-reserves and potential supply of many of the so-called critical metals, this does not mean they are 'scarce' or limited over the coming several decades – despite the infographics by Cohen (2007) or Moyer (2010).

The extent of critical metals is represented on a periodic table of the elements in Fig. 1, using the most recent BGS Risk List (BGS, 2015) to provide shading to the various elements. As can be seen, numerous elements are at significant supply risk, often due to the dominance of a particular country (e.g. China), by-product status (REEs, Ge, Ga), or concerns over socio-political status (e.g. Co from the Democratic Republic of the Congo). All of this indicates a fundamental need to understand the data sources and methods behind ore reserve and mineral resource reporting in the mining industry and how this intersects with the complex issues surrounding critical metals. This paper provides such an assessment and links closely with new methodologies for compiling global assessments of critical metals (Werner et al., under review a) as well as a case study of global indium resources (Werner et al., under review b).

3. Mineral resource accounting at the national, state/provincial or corporate level

3.1. Different sources, different methods

The uncertainties inherent in the assessment of national mineral resources and subsequent government policy on exporting and/or

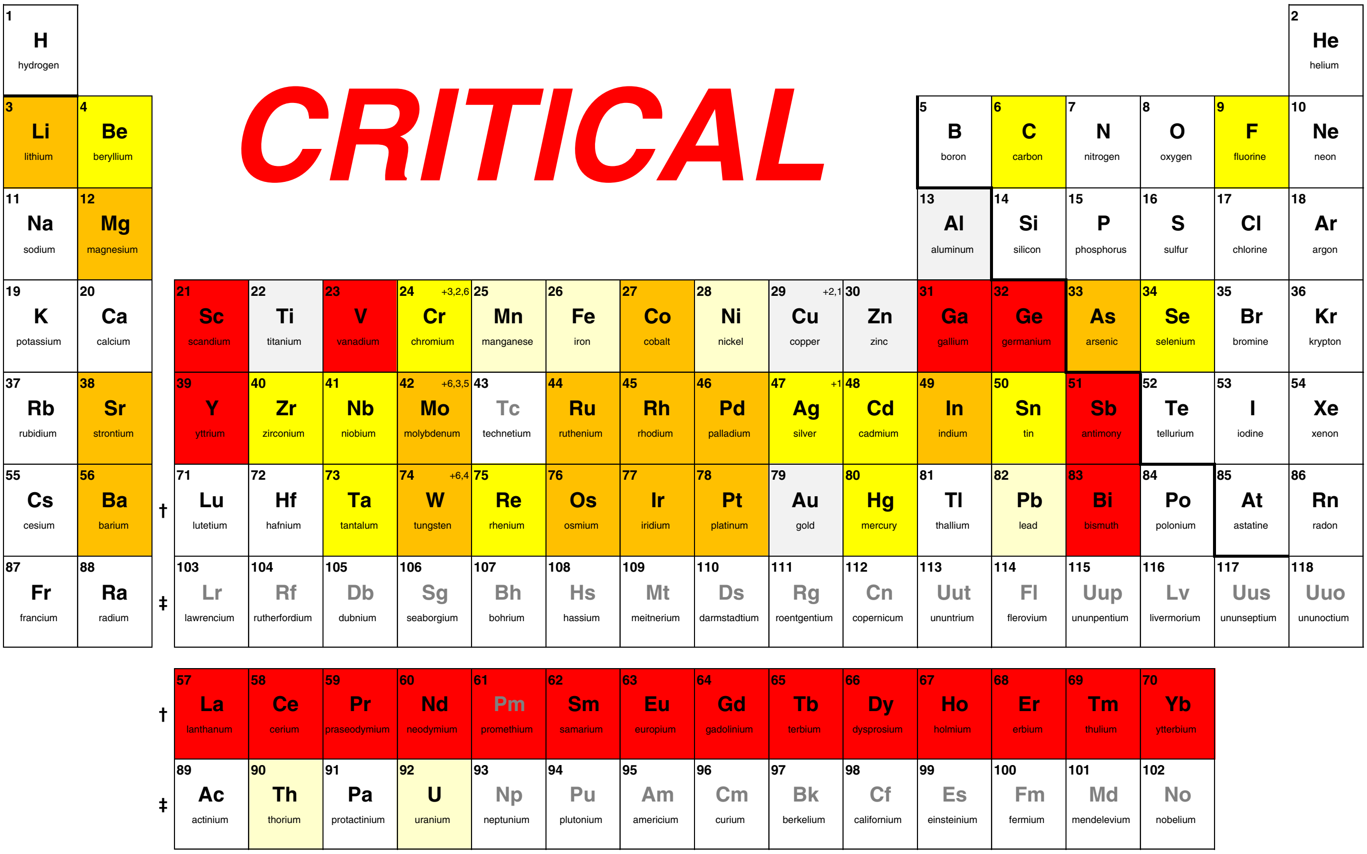

Fig. 1. The extent of critical elements represented by the periodic table of elements, using the most recent BGS Risk List (BGS, 2015) (note: colours adapted from the BGS relative supply risk index, red = 8.5–10, orange = 7.5–8.5, yellow = 6–7.5, light yellow 5–6). (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

stockpiling of minerals are exemplified by the Australian Government's policy on iron (Fe) ore between 1938 and 1960. The Australian government in 1938 decided to embargo the export of Fe ore for two reasons; firstly, as a strategic decision to prevent the development of known Australian Fe ore deposits by the Japanese for export to Japan in the tense period leading up to World War II; and secondly, on the perceived grounds of resource scarcity, with accessible Fe ore reserves estimated at some 350 Mt (Lee, 2013). This embargo, and an associated embargo on the pegging of Fe ore claims in Western Australia, continued (with the support of BHP, who had a monopoly on the Australian Fe ore and steel industry) until 1960, when regulations began to be relaxed, mainly due to emerging discoveries and more positive technological developments regarding Fe ore processing (Blockley et al., 1990, Mudd, 2009). Although in 1960 the Bureau of Mineral Resources' estimate of Australian Fe ore reserves at 368 Mt of Fe ore was only slightly more than the 1938 estimate, the main shift was in focus, from conservation of resources to exploration and exploitation, even though considered wisdom would have suggested that Australia was poorly endowed with Fe ore (Blockley et al., 1990). The removal of the export and claim pegging embargos in stages between 1960 and 1966 eventually led to the development of the current major Australian Fe ore industry, with 2013 exports of ~579 Mt of Fe ore (OCE, 2014; significantly higher than the estimate of the entirety of the Fe ore reserves within Australia from 1938 to 1960!) and the delineation of some 52.58 Gt of economically demonstrated Fe ore resources in Australia as of December 2013, with a further 80.69 Gt of sub-economic and inferred resources of Fe ore (GA, 2014). Although this is obviously an extreme example, it also clearly highlights the uncertainties in national resource reporting and the influence that misunderstood or mistaken resource accounts can have on national policy over many years.

There are a wide variety of approaches in compiling national or global estimates for mineral and energy reserves and/or resources, and here we provide different examples to highlight the critical importance of understanding the different sources of data and methods used to prepare such estimates.

- National Reserves versus Resources: some groups strictly use ore reserves only while others compile measured and indicated mineral resources as economic (or 'demonstrated') resources but keep inferred mineral resources separate. Although the recoverable mineral or metal is critical to understand (i.e. not just contained minerals or metals in the deposit), such factors are rarely included or clearly justified. Some examples include (with Cu resources-reserves by selected countries shown as examples in Fig. 2):
  - *Canada* (*NRC, var.*): NRC only reports contained metals in ore reserves at operating mines and those projects under development, excluding ore reserves and/or mineral resources reported for other undeveloped projects as well as additional mineral resources at operating and developing projects. For 2010, NRC's reported Cu and Ni reserves were 10.7 Mt Cu and 3.07 Mt Ni that compare to total mineral resources of 54.12 Mt Cu (Mudd et al., 2013a) and 21.92 Mt Ni (Mudd and Jowitt, 2014). In 2010, a detailed deposit-by-deposit breakdown of projects by proved and probable reserves was published for the first time, and included data on additional metals in measured and indicated resources at operating mines and those projects under development, showing an additional 2.38 Mt Cu and 0.385 Mt Ni (Drake, 2012). Although Canada's Cu

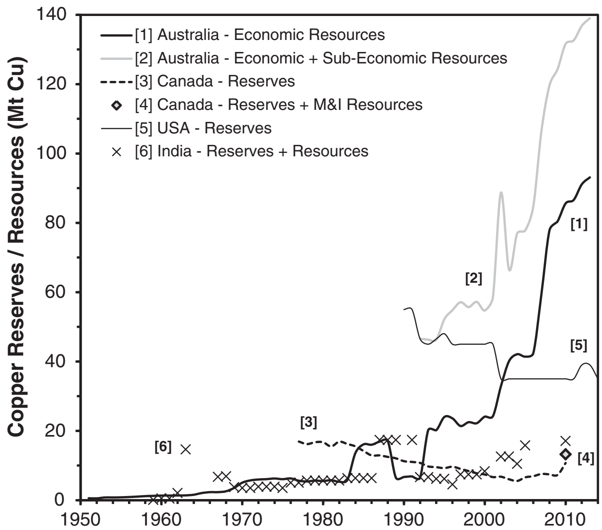

Fig. 2. Long-term trends in national estimates of copper reserves and/or resources (data combined from Drake, 2012, GA, var., IBM, var., Mudd et al., 2013a, NRC, var., USGS, var.-a,b).

and Ni (and possibly other) sectors over time could be perceived to be in decline based on ore reserves alone, when considering the potential future conversion of known mineral resources to ore reserves, it is clear that Canada has considerably more room for optimism.

- *USA*: The USGS only reports estimates of minerals or metals from ore reserves, mainly through the annual Minerals Yearbook (USGS, var.-b; including its earlier version published by the former US Bureau of Mines; USBoM, var.) and Mineral Commodity Summaries (USGS, var.-a), although as shown earlier for PGEs, the basis and data are not always clear or appear to be consistent with code-based compilations. The USGS used to report a 'Reserve Base' category (see USGS, var.-a) that was on measured and indicated resources and is inclusive of reserves, but this was stopped from 2010 due to the issues and extent of work required to develop such estimates annually. An important ramification of this change in practice and reporting was that it created the clearly wrong perception, mainly by people unfamiliar with resource-reserve reporting, that either the previously reported Reserve Base estimates were too optimistic or that the mineralisation was not actually there or realistically mineable. For example, USGS (var.-a) cites South African Au reserves of 6000 t Au in the 2009 edition (a value which has been reported from 2004 to 2014, with cumulative production during this time of ~2400 t Au) with an additional 25,000 t Au in the now unreported reserve base category, but with 2010 to 2014 values only indicating reserves of 6000 t Au. Recent work has shown that reported Au ore reserves for South Africa are 5518 t Au with additional mineral resources of some 21,517 t Au – and still the largest reported Au mineral resource endowment in the world (data updated from Jowitt and Mudd, 2014).
- *Australia* (GA, var.): GA reports data for 'economically demonstrated resources' (EDR), akin to measured and indicated resources, as well as sub-economic, paramarginal and inferred resources, collectively similar to inferred resources (see Lambert et al., 2009). Although GA has published an annual assessment of Australia's mineral resource endowment since 1992 (which includes annual data from 1975), they only report contained minerals or metals. Detailed resource-reserve data by individual project is made available through their online database and mapping system, the Australian Mines Atlas (GA et al., 2015), arguably one the best such systems globally – although it lacks data on most critical metals. Curiously, as shown in Fig. 2, the sudden decline in Cu resources in 1989 (from 17.0 to 6.5 Mt Cu) is purely a function of the introduction of the 1989 edition of the JORC Code and stricter requirements for developing mineral resource estimates – this meant extra drilling and other work had to be completed at large projects such as Olympic Dam and Northparkes before this mineralisation could be reclassified as mineral resources, with Cu resources again rising to 20.2 Mt Cu in 1993.
- *India* (IBM, var.): The Indian Bureau of Mines (IBM) reports ore and contained metals by reserves and resources both nationally and by state, although these data are only updated every few years. For 2010, reserves were 4.79 Mt Cu with additional resources of 12.28 Mt Cu (total 17.07 Mt Cu). The reporting of ore and contained metal allows an estimate of ore grades over time, a unique feature in global resource-reserve reporting.
- *China*: Chinese government agencies regularly report reserve estimates for various commodities, including many critical metals, but the data that these estimates are based on remain generally unclear; even though individual Chinese companies have started reporting resources-reserves using CRIRSCO-approved codes (e.g. the Inner Mongolia Sanguikou Pb—Zn deposit and others owned by Zijin Mining Group, or the Liziping, Menghu and Shizishan Pb—Zn projects owned by China Polymetallic Mining), these remain the exception rather than the rule. For example, in 2008 the Chinese Geological Survey reported estimated Au resources of between 15,000 and 20,000 t of contained Au (Klapwijk, 2014) and several publications have reported contained tonnages of Au for individual mines or districts (e.g. Jiaodong with an endowment of >3000 t Au). However, it is unclear whether these values are for remaining or total endowments (e.g. resources + reserves + past production), a situation that is confounded by the long history of mining in some of these areas (e.g. Jiaodong has a mining history of >1400 years; Goldfarb and Santosh, 2014) and the fact that the vast majority of Au production across China is from small mines. This is again exemplified by the Jiaodong district, where some 159 mines produce a minimum total of 30–50 t Au/year (Goldfarb and Santosh, 2014, Guo et al., 2013), yielding an average production per mine of 0.18–0.32 t Au/year (~5.8–10.3 koz Au/year), although even this doesn't reflect the true situation as Shandong Gold Corporation produces ~28 t Au per year from three mines in this region (Goldfarb and Santosh, 2014, Jowitt and Mudd, 2014). The fact the same uncertainties exist within a wide range of commodities in China (e.g. Mudd and Jowitt, 2014, Mudd et al., 2013b; Weng et al., 2015) means that these uncertainties are in turn propagated into any prediction of future supply involving China, be it from an individual mine or from the country as a whole.

- State/Provincial Agencies: Some major state/provincial government agencies or departments maintain detailed databases of resources and reserves, while others publish brochures summarising various projects with the associated data (e.g. Yukon; Anonymous, 2008) or detailed annual review reports (e.g. Alaska; Athey et al., 2013).
  - *MINEDEX (WA) and MINDEP (SA)*: Western Australia's (WA) Department of Mines and Petroleum and South Australia's (SA) Department of State Development actively maintain online searchable databases, called MINEDEX and MINDEP, respectively, of ore reserves and/or mineral resources for all deposits throughout the state and across all commodities. For MINEDEX, one of the summary report options allows a compilation of all projects by tonnage and ore grade, including reserves and resources. MINEDEX includes all minerals and metals as reported by the mining company, including many metals listed as 'critical' (e.g. Ge, Ga, Nb, Pt, etc).
- Corporate Compilations: Given the rise of the internet and digital publishing, it is now considerably easier for any individual or group to undertake global assessments of resources-reserves for numerous minerals and metals, although the methodology is rarely explained and justified (eg. NRH's annual Au resource report; NRH, 2014). A variety of other corporate groups maintain extensive data sets on resources, reserves and mine production, typically based on public

reporting but often including confidential data provided by a company or industry association. Examples of such groups include MinEx Consulting and AME Mineral Economics (Australia), SNL (USA), Mineral Economics Group (MEG, Canada), Raw Materials Group (RMG; Sweden), Intierra (Canada), Roskill (UK) (amongst others; note: MEG, RMG and Intierra are now part of SNL). The reliability of this approach is most probably high, but invariably these compilations and the databases behind them are commercial-in-confidence and not published (although many offer paid subscriptions to their data and information services). The work of Richard Schodde, through MinEx Consultants, is an admirable exception, as his work is summarised and presented at various conferences and industry seminars (e.g. Schodde, 2010, 2011, 2012, 2013).

3.2. Need to reconcile resources-to-reserves-to-production

Although the controls on discovery success once ground has been acquired are generally geoscientific, converting mineralisation to mineral resources to ore reserves to mine production is reliant on a number of different factors (akin to the modifying factors used in the McKelvey system and associated resource reporting codes such as JORC, NI43-101, SAMREC and others). Some of these factors are scientific, environmental and engineering-related, while others are political and social in nature, the latter being more difficult to quantify than the former. A brief list includes:

- deposit size, grade and tonnage and complexities involved in resource estimations;
- mineral deposit type;
- location (i.e., political security and logistical issues);
- geometallurgy and mineral processing, and advances in associated technology during the life of a mine;
- mining methods (open cut vs underground mining methods such as narrow vein or block caving)
- geotechnical conditions (especially for block caving mines);
- impurities and the presence of penalty elements (e.g. As, Cd, Hg, U);
- production of individual metals as primary and secondary/by-products and the relative (and changing) importance of by- and co-products on the economics of a given mine;
- global and local economic factors;
- environmental factors, including environmental and energy costs of production and the cost of clean up after mine closure (although the latter may not be considered as comprehensively as it should be);
- time taken from discovery to production and effects on internal rate of return (IRR; i.e. profitability) etc.;
- overall supply chain from mine through mill to smelter and refinery, especially whether a smelter/refinery has the capacity to extract particular by-products (e.g. indium at a Zn refinery, Ga at an alumina refinery).

Other factors such as resource nationalism and the cut-offs in terms of amounts of contained metal within a given deposit type or project that are applied by both major and middle tier companies to projects to see if they are economic from their viewpoint (i.e. can generate enough revenue to be meaningful to a given company) may also need to be considered in some areas and for some projects. Obviously the economics of all possible developments are considered by all companies. Consideration of the latter by mining companies is undoubtedly already based on an assessment of the mining industry and the state of the economy, but given the proprietary nature of these data it is unclear what factors these companies consider important and which are seemingly less crucial, and (more importantly) whether the considerations taken by a given company actually reflect reality.

3.3. Lack of resource-reserve data for critical metals

Although the above knowledge and approaches are well developed for more common metals (e.g. Cu, Au, Pb—Zn, Ni, PGEs, Fe), grade data for most critical metals are rarely reported in mineral resources, often because these critical metals are not considered a primary or by-product metal by mining companies as they receive minimal to no economic incentive for them in outputs such as concentrates. This means that building an extensive compilation of projects reported to contain the numerous critical metals to estimate their global reserves and resources is challenging. Hence, other methods are often used, and these are reviewed briefly below along with some selected case studies, with newer methodologies also being developed in this area; this is exemplified by the new methods used to estimate potential resources of critical metals outlined by Werner et al. (under review a), and the subsequent application of these methods to global indium resource estimates (Werner et al., under review b).

3.4. Lack of clear code guidance on co/by-products or payable metals

At present, there is no explicit guidance or definitions of a primary product, co-product or by-product in the various mineral resource reporting codes. There is, of course, an expectation that metallurgical recovery factors should be incorporated into resource-reserve modelling and estimates (even including allowance for deleterious elements which may affect mineral processing or the quality and saleability of products such as concentrates). Resource reporting codes also allow for the use of metal equivalents in estimating total value, with the most valuable metal generally used as the basis for results. As an example, the 2012 JORC Code expects that a company should justify "… that all the elements included in the metal equivalents calculation have a reasonable potential to be recovered and sold" (page 24, AusIMM et al., 2012). The problem for many critical metals is that they are often perceived not to contribute 'materially' to a mine's recoverable (or payable) metals and therefore revenue (since they are often not paid for in concentrates), and are only extracted at a smelter or refinery (if that facility has the appropriate processing capacity for a given critical metal, which is frequently not the case). In reality, the markets for many critical metals are modest in scale compared to their primary metal hosts, albeit with strong growth prospects given the uses of critical metals in sustainable energy technologies, consumer electronics, specialty alloys, etc. – thus it remains difficult to assess and predict the economics of additional metals in resource-reserve and production reporting. In addition, the relative prices of various metals can fluctuate significantly over time, leading to changing reporting of a resource. For example, reporting by Overland Resources of mineral resources for the Andrew Zn deposit in the Yukon province of Canada included Ag—Ge grades in their 2007 and 2008 annual reports but have not done so since (OR, var.), although they did report concentrate Ag—Ge grades in a 2011 corporate presentation (see OR, 2011). This highlights the critical need for resource-reserve codes to explicitly address co/by-products, an approach that would facilitate much more extensive reporting of many critical metals that at present are very minor potential by-products but could represent more value if they were better recognised and estimated in resources and reserves.

Another perhaps minor but related issue is the effects of cut-off grades on estimates of the ore grades of critical metals. Although deposits are modelled on various cut-off grades to inform economic assessments and decisions, there is considerably less reported data for many critical metals, primarily as these metals are perceived as (or may actually be) a low margin by-product. Rare examples of reporting by varying cut-off grades include the Anna Lake U-Mo-Re deposit in northern Labrador, Canada (Fraser and Giroux, 2009), the high grade Merlin-Little Wizard Mo—Re deposit in western Queensland, Australia (Warren and Bucci, 2010), as well as for the Baal Gammon Cu–Sn–Ag-indium project in northern Queensland (see corporate reporting by Monto Minerals Ltd. and Kagara Ltd.), although these are certainly the exception rather than the rule.

4. Case studies: demonstrating different types of uncertainties

This section focuses on a series of case studies that outline the differing types of uncertainties in resource reporting, building on the information presented above.

4.1. Ore reserves vs mineral resources

As above, it is worth pointing out here that some global mining companies, due to their need to conform to US Securities & Exchange Commission (SEC) requirements, only report ore reserves but choose to exclude reporting of additional mineral resources as 'mineralised material'. For example, after Vale successfully took control of Canada's Inco in 2006, the size of the Sudbury, Thompson, Voisey's Bay and Goro operations looked decidedly smaller (see Mudd and Jowitt, 2014) – yet this only related to the fact that Vale reported ore reserves and excluded any additional mineralised material previously reported by Inco as mineral resources, an approach that has also been used by Hecla Mining and Peabody Energy. However, numerous mining companies report both ore reserves and additional mineralized material but are consistent with SEC requirements (e.g. Cliffs Natural Resources, Newmont, Grupo Mexico, Freeport–McMoRan Copper & Gold, Coeur d'Alene Mines), suggesting that an approach like that undertaken by these companies (rather than by Vale, Hecla Mining or Peabody Energy) would enable the determining of a clearer picture of global metal resources.

4.2. Reported mineral resources of critical metals

A brief global survey of reported resources by project for some critical metals is compiled in Table 1; this is developed from larger resource data sets on Cu, Co, U, Ni, REOs, Pb—Zn, Au and indium (see earlier), as well as general research for such data. Additional data sets for Hf and scandium (Sc) are given in Tables 2 and 3, respectively, with both focused on Australian projects.

- Indium: Indium is used in the manufacture of liquid crystal displays (LCD) for mobile phones, tablets and laptops and also has applications in solar photovoltaic (PV) panels, yet recent research into indium has shown that only 12 known deposits globally currently include indium in their resource-reserve reporting (see Werner et al., under review b). Although some former mines have reported indium grades in resources or mine production, current prices and typical grades applied to deposits not reporting indium grades indicate that indium most likely represents ~8% of the combined value for even the highly most indium-enriched Pb–Zn–Ag deposits (data from Werner et al., under review b) – making it hard for mining companies to currently justify the effort involved in including it in resource-reserve estimates and mine production statistics for most indium-containing deposits. At present, some 95% of indium supply is derived from Zn refineries, although there is potential to extract more indium from Zn concentrates (since some indium-enriched Zn concentrates do not go to an indium-enabled Zn refinery) as well as minor amounts of indium from Pb, Sn and Cu smelters and refineries (see Werner et al., 2015). The deposits in Table 1 represent contained metal of 9449 t indium – and although the USGS no longer report estimates of global indium reserves, their last estimate from 2007 was reserves of 11,000 t indium with an additional 5000 t indium in reserve base (USGS, 2008). Given the extent of indium being extracted from existing Zn concentrates, this suggests that if just 12 deposits represent resources of 9449 t indium alone, there would be considerable potential for indium to be classified as a valuable product from numerous Pb—Zn projects as well as other Sn and Cu projects around the world, as well as the numerous projects that are described as containing indium but give no indium grade data in resource estimates (e.g. East Kemptville in Canada).
- Tellurium: despite being a crucial metal for electronics, especially some types of solar PV panels, Te is rarely ever mentioned in mineral deposit resource reporting, despite the fact that the majority of this metal is produced from anode slimes as a by-product of electrolytic Cu production. One rare exception is the Boliden-Kankberg deposit in Sweden, owned by Boliden, which reports Te with Au—Ag grades (4.246 Mt at 178 g/t Te to contain 756 t Te; Boliden, 2015). Another notable example is two mines in China that are primary Te producers (Anderson, 2015), although the uncertainties surrounding resource reporting in China (as discussed above) means that constraining production or resources from these mines is problematic. In a current study of global Au reserves and resources (Jowitt and Mudd, 2014), Boliden–Kankberg was the only deposit reporting Te grades – despite for example the significant numbers of intrusion-related and orogenic Au deposits hosting tellurides (and therefore potentially significant amounts of Te, e.g. Kalgoorlie, Australia; Kumtor, Kyrgyzstan; Fort Knox, Alaska, USA; Muruntau, Uzbekistan). In particular, although Au resources-reserves for the giant Muruntau Au mine (owned by the Government of Uzbekistan) are considered a state secret and not publicly reported, the ore is known to be Te (and Se, W) rich, averaging ~20 g/t Te (Bierlein and Wilde, 2010, Graupner et al., 2001). Assuming an average grade of ~2.5 g/t Au and total Au endowment (cumulative production plus resources-reserves[4]) of ~5287 t Au (~170 Moz) (Graupner et al., 2001, Wilde et al., 2001), this gives an estimate of ~2115 Mt ore – meaning an approximate Te endowment of ~42,300 t Te alone, despite the fact that it is believed that Te has never been extracted at Muruntau (although clearly the lack of public reporting limits such insights). In comparison, the 2014 USGS estimate of global Te reserves was 24,000 t Te (USGS, 2015), showing the significant uncertainty in quantifying resources or reserves for these critical metals.
- Rhenium: Re is fundamental to high performance specialty alloys, especially those used in jet turbines as well as in catalysts and a range of other uses, and is almost exclusively extracted as a by-product from the processing of Mo concentrates derived from porphyry Cu deposits (effectively making the majority of Re production a by-product of a by-product). Typically, Mo might represent some 5–10% or more of a deposit's value, and is extracted as a by-product to Cu ($\pm$Au), but the value of Re within a deposit can vary from <1 to 15% (see Table 1). Two different regressions are given in Fig. 3, with our compiled reported resources suggesting a relationship of ~17.6 g/t Re/%Mo (R^2 value of ~0.93), with the reported Re—Mo assay data for the Merlin deposit (Australia; Valk, 2014) also shown and suggesting a relationship of ~16 g/t Re/%Mo for the range 0.1–50% Mo (no correlation coefficient provided), suggesting our reported data is reasonable given the paucity of formal mineral resource data for Re. Although the 2014 USGS estimate of global Re reserves was 2500 t Re (USGS, 2015), our compiled data suggest that some 1625 t Re is contained in just 9 deposits (Table 1), and the fact that the vast extent of Mo is produced as a by-product from porphyry Cu systems (Mudd et al., 2013a) that also contain Re suggests that global Re resources are considerably larger than currently understood.
- Rare earth elements: the REEs are increasingly critical to a range of electronics, chemicals and alloys and are a family of elements that include the lanthanides as well as yttrium and Sc; although the latter two are not strictly counted as REEs according to IUPAC designations are often grouped and considered with the lanthanide elements (Jowitt et al., 2013b). Although some REEs are similar in abundance to some major metals such as Cu or Pb or precious metals such as Au (see Rudnick and Gao, 2014), the REEs have to date been a small and often obscure sector of the global mining industry. A detailed study of global REEs by Weng et al. (2015) showed that there are extensive known REE

[4] The various published estimates of total metal endowment for Muruntau do not distinguish between cumulative production and remaining resources-reserves.

Table 1
Mineral resources with reported data for some critical metals.

Project, Country	Total mineral resources										Proportional value (%)[a]			
	Mt ore	%Pb	%Zn	g/t Ag	%Cu	g/t Au	g/t In	%Mo	g/t Re	Other	Pb–Zn–Ag–Cu–Au–Sn	Mo	Indium/Re	Other
Pingüino, Argentina	10.58	0.62	1.37	62.9		0.38	11.5				92.2	0	7.8 In	0
Baal Gammon, Australia	2.800			40	1.0		38			0.2%Sn	83.6	0	16.4 In	0
Conrad-King Conrad, Australia	3.13	1.26	0.56	95.4	0.18		5.7			0.21% Sn	97.4	0	2.6 In	0
Kalman, Australia	75.1			1.1	0.4	0.2		0.05	1.24		68.1	25.0	6.9 Re	0
Merlin-Little Wizard, Australia	6.715	0.02	0.14	8.3	0.34	0.08		1.34	23.2	0.0081%Co	7.6	77.0	14.9 Re	0.5 Co
Mt Dore, Australia	144.3	0.05	0.30	5.9	0.52	0.10		0.007	0.03	0.0080%Co	91.0	3.2	0.2 Re	5.6 Co
Zeehan Group Slag, Australia	0.42	1.5	13.6	55			48				91.4	0	8.6 In	0
Malku Khota, Bolivia	485.01	0.08	0.04	23.8	0.02		5.0			4.3 g/t Ga	78.0	0	15.4 In	6.5 Ga
Andrew (Lad), Canada	7.762	1.4	5.8	9.49						14.86 g/t Ge	84.4	0	0	15.6 Ge
Anna Lake, Canada	7.64							0.013	0.174	0.036% U_3O_8	0	11.6	1.7 Re	86.7 U_3O_8
Keg Main Zone, Canada	39.76	0.28	0.77	30.25	0.15		5.77			0.027% Sn	93.3	0	6.7 In	0
Mt Pleasant (North Zone), Canada	18.485		0.80		0.10		67.4	0.036		0.34%Sn, 0.91%As, 0.09%WO_3, 0.07%Bi	43.2	4.2	20.6 In	32.0 As-WO_3-Bi
Snowfield, Canada	2203.3			1.79	0.08	0.49		0.0079	0.48		88.3	7.0	4.7 Re	0
Berlin, Columbia	8.7		2.8	3.4				0.062	6.8	U–P–V–Y–Ni–Nd–CaF_2[c]	14.4	3.8	4.7 Re	77.1 rest[c]
Geyer Southwest, Germany	12.6		0.58				35			0.46%Sn, 28 g/t Ga	76.4	0	16.7 In	6.9 Ga
Tellerhäuser, Germany	32.2		0.74				71			0.35%Sn	64.8	0	35.2 In	0
Namib, Namibia	0.917	2.4	5.7	44.8			29[d]				90.8	0	9.2 In	0
Ayawilca, Peru	13.3	0.2	5.9	14			68				74.7	0	25.3 In	0
Yandera, PNG	1293.5			1.35	0.26	0.08		0.0091	0.07		89.3	9.8	0.8 Re	0
Ak-Sug, Russia	722				0.47	0.11		0.012	0.20		90.8	7.7	1.5 Re	0
Boliden-Kankberg, Sweden	4.246			14.3		4.31				178 g/t Te	89.9	0	0	10.1 Te
Muratdere, Turkey	51.6			2.4	0.36	0.12		0.0125	0.34		87.7	9.4	2.9 Re	0
West Desert (Crypto), USA	71.113		1.88		0.21		23			42.3%Fe[b]	48.5	0	14.1 In	37.4 Fe
Total contained metal tonnage		824 kt	4.28 Mt	21,019 t	10.24 Mt	1326 t	9449 t	533 kt	1625 t					

Notes: All mineral resources data (year 2014 or similar) derived from technical and/or annual reports.
[a] Based on 2014 commodity prices from OCE (2014), USGS (2015).
[b] Fe in magnetite.
[c] The Berlin deposit also contains 9.3% P_2O_5, 0.5% V_2O_5, 497 g/t Y_2O_5, 0.2% Ni, 0.11% U_3O_8, 101 g/t Nd_2O_5, 37.3% CaF_2, with most of the value associated with CaF_2 (31.9%), U_3O_8 (18.2%) and V (14.3%).
[d] Indium grade not reported in 2013 resource, indium adjusted from 33 g/t indium from the maiden resource in 2012 after allowing for the reduction in %Zn in the slightly larger 2013 resource (i.e. 2012 was 6.6% Zn while 2013 was 5.7% Zn).

Table 2
Some mineral resources with reported data for Hafnium.

Project, Country	Total mineral resources										Proportional value (%)[a]					
	Mt ore	%Zr	%Nb	g/t Hf	g/t Ta	g/t Ga	g/t Dy	%Y	%TREO	Other	Zr	Nb	Hf	Ta	Ga	Rest
Hastings-Brockman, Australia	36.2	0.660	0.248	260	149	116	149	0.088	0.21		65.0	10.6	15.0	3.0	4.3	~2.1 TREO[b]
Dubbo-Toongi, Australia	73.2	1.45	0.32	327	246				0.75		76.0	7.3	10.0	2.6	–	4.0 TREO[b]
Narraburra, Australia	73.2	0.093	0.009	38		40		0.011	0.033	54 g/t Th, 55 g/t Li	65.4	2.7	15.8	–	10.6	2.4 TREO[b]
Strange Lake, Canada	492.5	1.35	0.11	390					0.89		78.6	2.8	13.3	–	–	5.4 TREO[b]
Norra Kärr, Sweden	36.82	1.01		243					0.592		83.2	–	11.8	–	–	5.0 TREO[b]

Notes: All mineral resources data (year 2014 or similar) derived from technical and/or annual reports.
[a] Based on 2014 commodity prices from OCE (2014), USGS (2015).
[b] Based on approximate average TREO prices with no allowance for variation in LREE vs HREE or specific REE prices (i.e. proportional values are approximate).

mineral resources – some 619.5 Mt REO compared to 2014 production of ~0.11 Mt REO. However, these resources are dominated by the light REE (LREE), whereas the majority of demand for the REE is for the rarer (both in terms of typical abundance and in terms of known resources) heavy REE (HREE; e.g. Jowitt et al., 2013b). This mismatch of supply and demand has a number of consequences; for example, the similar behaviour of the REE means that any deposit that contains significant quantities of the rarer and more sought-after HREE will also undoubtedly contain significant amounts of the LREE, meaning that any increase in HREE production will almost certainly cause a coincident increase in LREE production. This balance problem may lead to oversupply, a decrease in the price of the LREE, and a significant change to the economics of REE producing mines. In addition, the reason why REEs are considered as 'critical metals' is the near monopoly China has on global supply, although this is often confused with lack of resources elsewhere around the world – which is clearly not the case (e.g. Weng et al., 2015).

- Hafnium: Hf is extracted as a by-product from zirconia (ZrO_2) production, although detailed resource and production data is rarely reported. Table 2 shows the few known projects which report Hf grades, with a Hf—Zr regression of grades (Fig. 4) showing a coefficient of correlation of 0.83 (although caution is required given the availability of only 5 data points).
- Scandium: since Sc has a very small market at present (widely believed to be ~10 t Sc/year; see Duyvesteyn and Putnam, 2014), it is generally perceived to be a very scarce metal. Although Sc can often be found in REE deposits, it is not widely reported in or extracted from existing REE deposits (despite Sc formally being a member of the REE family; Weng et al., 2015). Sc is also present within some Ni laterite deposits in Australia (Table 3) as well as being present in a range of other deposit types (e.g. U, apatite, fluorite, Sn, W, Ta, etc). For some REE deposits, Sc grades include 12, 28 and 5.1 g/t Sc in the Buckton-Buckton South, Niobec and Eco Ridge projects, respectively (all Canada) as well as 194 g/t Sc in the U tailings at Port Pirie in Australia (see Weng et al., 2015). In addition, the Kovdor baddeleyite–apatite–magnetite deposit in Russia contains some 420 t contained Sc at a grade of 780 g/t Sc (Kalashnikov et al., 2016).[5] At prices ranging from US$43 million/t Sc to US$263 million/t Sc for samples of 2 or 5 g of acetate, chloride, fluoride, iodide, oxide or purified metal (data from USGS, 2015), this makes Sc typically more expensive than Au (at US$43 million/t Au, or US$1270/oz Au) – and thus dominates the potential economic value of Sc-bearing Ni laterite projects. Given potential Sc uses in Al—Sc light-weight alloys, solid oxide fuel cells and specialty chemicals, it remains a potentially important element if a large, reliable supply can be developed. Of course, with such a small market, if a new mine was able to produce Sc reliably on a large scale, it may affect prices and thereby economics – but it is abundantly clear that where Sc occurs, it presents substantial potential value.

[5] This was a geological estimate and not a formal code-based estimate.

5. Critical metals: assessing global resources-reserves

As highlighted already, one of the prime reasons why many critical metals have poor estimates of global reserves and resources is that they are typically not reported in resources-reserves and mine production by mining companies – with many such metals simply extracted at a smelter or refinery with even less publicly reported data (e.g. indium, Se, Cd, Te, Re, etc). This in turn means that it is unclear whether the concerns over the security of supply of these metals (e.g. Naden, 2012) is founded in evidence, is worse than we perceive it to be, or is based on uncertainty and is to some extent unfounded. Clearly, this has implications in terms of assessing future supply, the security of this future supply, and what areas should be prioritised for exploration and research, especially as these metals are often outside of the core business of most major mining companies and are therefore considered lower priorities than other base or precious metals or bulk commodities. Equally, there are also any more issues which need to be considered in developing global resources-reserves estimates of the various critical metals than typically need to be considered when undertaking the same exercise for more common base metals, bulk commodities, or even for precious metals. All of this means that there are a number of factors to consider when compiling and/or generating estimates of global critical metal resources and reserves.

5.1. Recovery rates and deportment studies during mining, milling and smelting/refining

A key factor in assessing and modelling a mineral resource is understanding the ore processing and recovery factors for all relevant metals within a given resource. For example, a deposit containing Pb–Zn–Ag–Cu–Au ore will generally produce concentrates (Cu, Pb, Zn) potentially along with Au doré (or perhaps gravity Au in rare circumstances) and tailings (the gangue remaining after ore processing). Given the mineralogy of particular critical metals, especially substitution behaviour in primary metal sulphides (e.g. indium in ZnS, Re in MoS_2), the majority of these metals may report to a specific concentrate, with minor amounts to other concentrates and the remaining to tailings – it is critical to

Table 3
Australian mineral resources with reported data for Scandium.

Project, Country	Total mineral resources				Proportional value (%)[a]		
	Mt ore	%Ni	%Co	g/t Sc	Ni	Co	Sc
Sconi-Kokomo	29.5	0.49	0.08	55	1.1	0.3	98.6
Sconi-Greenvale	27.4	0.6	0.04	40	1.8	0.2	97.9
Sconi-Lucknow	13.8	0.31	0.07	116	0.3	0.1	99.5
Owendale-Cincinnati	10.1			338	0	0	100
Gilgai-Nyngan	12.012			261	0	0	100
Hurll's Hill	19.2	0.63	0.1	40	1.9	0.6	97.5
Syerston	25.4			414	0	0	100

Notes: All mineral resources data (year 2014 or similar) derived from technical and/or annual reports.
[a] Based on 2014 commodity prices from OCE (2014), USGS (2015).

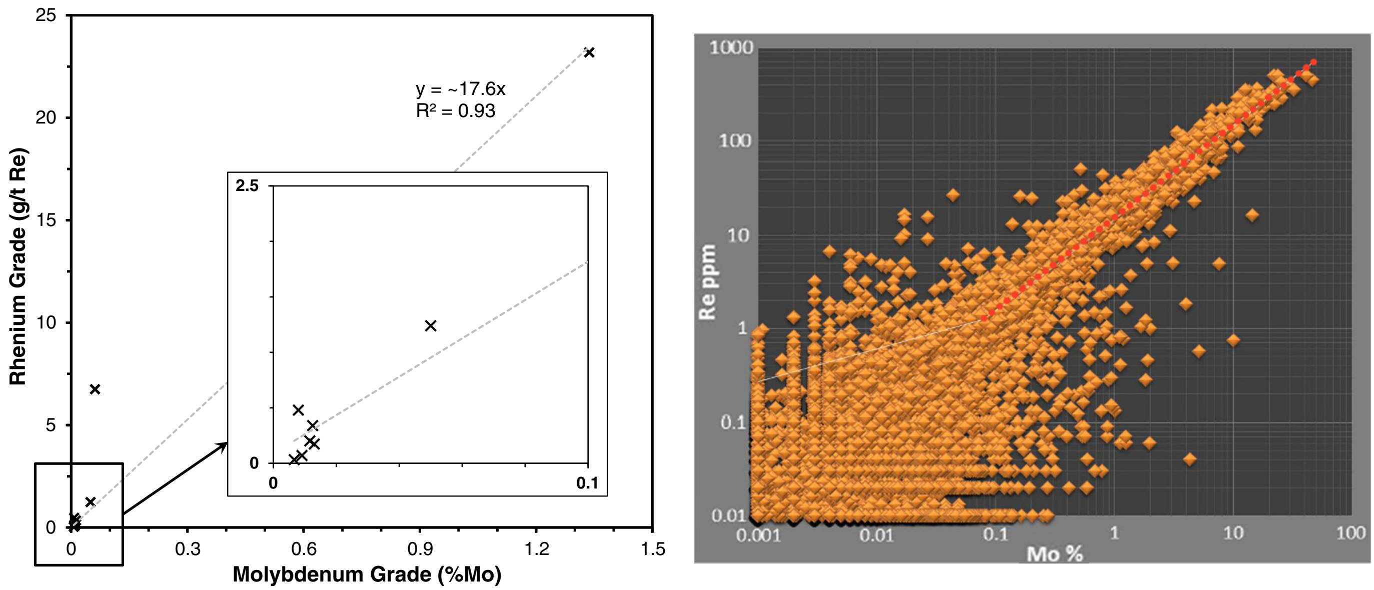

Fig. 3. Rhenium versus molybdenum grade for (i) reported mineral resources at selected projects (left, data from Table 1), and (ii) Re—Mo assay data for the Merlin deposit, Queensland Australia (right, Valk, 2014) (note: although the original is hard to discern, an approximate red-dashed line is included in this figure for the higher grade data >0.1% Mo, with a ballpark regression of Re (g/t) = ~16 × %Mo; see page 8, Valk, 2014). (For interpretation of the references to colour in this figure legend, the reader is referred to the web version of this article.)

understand the flow or deportment during ore processing (milling) to understand the material flows and economics of overall recovery during mining and milling. Furthermore, as with the primary metals of interest, it is possible to operate a mill to favour some metals over others (e.g. Au over Cu, Ni over Co) – but it is generally very difficult (or practically impossible) to optimise a mill to recover all metals of interest at high efficiency. There are very few papers and studies which document deportment and recovery factors for many (if not most) of the critical metals, with some examples discussed below.

In the case of indium, deportment studies are indeed rare, with the two best examples having been carried out over 30 years ago by Chen and Petruk (1980) and Petruk and Schnarr (1981). These studies indicate that 28–35% of the indium present in the milling feed from the Heath Steele and Brunswick 6–12 mines in Canada reported directly to tailings, with some 50–60% deporting to Zn concentrates. As these concentrates were not processed in indium-capable facilities, these quantities now likely reside in slag heaps (see Chen and Petruk, 1980, Petruk and Schnarr, 1981), meaning that these mining wastes most likely represent indium (plus potentially other metal) resources, as is also the case elsewhere (e.g. Zeehan in Tasmania, Australia; Table 1). The understanding of deportment provided by these studies has enabled a more detailed analyses of materials flows and waste accumulation, as a recent study showed many thousands of tonnes of indium have accumulated in tailings and slags associated with these two sites alone, and remain untouched (see Werner et al., under review b). Comparing this to global primary production of indium, which sits at ~820 t indium/year (USGS, 2015), these cases reveal that mine waste may be an important future source of indium, and that estimates of global indium in mine wastes may be orders of magnitude higher than previously thought. If this is the case for indium, it is likely that the same results may be replicated for other critical metals, meaning there are likely to be significant volumes of tailings and mine waste (never mind electronic or e-waste and landfill waste) that represent potential sources of critical and other metals.

Additional information about the deportment of indium can be found in some NI 43-101 technical reports, as some include sections outlining the results of locked cycle flotation tests, whereby mine samples are used to determine elemental deportment at the laboratory scale. Notably, the reports of the Keg (Giroux and Melis, 2013) and Mount Pleasant (McCutcheon et al., 2013) deposits in Canada, and the Pingüino deposit, Argentina (Ristorcelli et al., 2014) include indium grades, tonnages and concentrations. However, the majority of sites that are known to contain (and may even produce) indium do not report indium grades or tonnages by any sort of code-based reporting or explicit deportment studies. In these cases, deportment may alternatively be indirectly calculated using studies reporting the grade of critical elements in concentrates as long as production quantities are also known. Studies by Ishihara and Endo (2007), Ishihara et al. (2011) and Rodier (1990) focus on indium, and also reveal more about the economic value of critical metals in the concentrates examined during this research. A final class of studies, such as Bloodhurst et al. (2007), developed more empirical approaches to determining the deportment of by-product metals. However, although these studies are often specific to certain deposit types and/or processing configurations, or otherwise

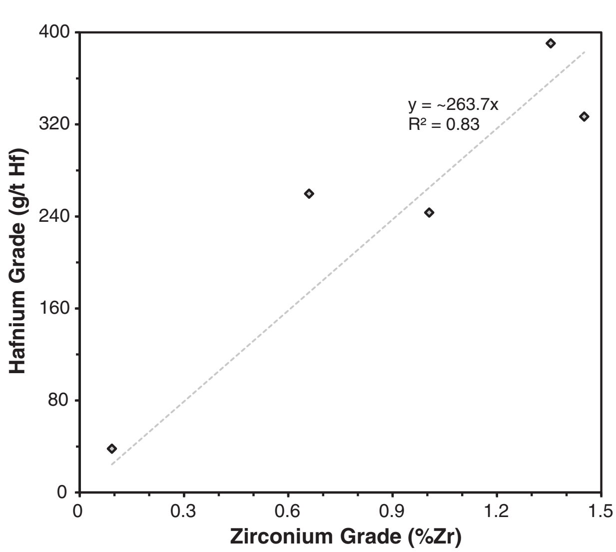

Fig. 4. Hafnium versus zirconium grade for reported mineral resources at selected projects.

entail large ranges of uncertainty, they may assist in understanding the absolute ranges for elemental deportment of critical metals.

The Canadian Minerals Yearbook (NRC, var.) used to provide excellent reporting on a range of critical metals, such as indium, Cd, Se, etc., although in recent years this publication and the associated detailed statistics relating to the Yearbook have not been continued. The 1973 edition noted that Canadian Zn ores contained up to ~0.07% Cd, with Zn concentrates containing up to ~0.7% Cd compared to global Zn concentrates containing ~0.1–0.3% Cd (page 65, NRC, var.). In the 1975 edition (pages 79–82), detailed Zn concentrate grades are provided, including tonnage, %Cd, %Zn, %Pb, %Cu and g/t Ag by individual project, an extremely rare example of such detailed reporting. Although a thorough analysis of historical NRC (and earlier) data for various companion or critical metals is beyond the scope of this paper, along with similar USBoM or USGS and other reporting, these data highlights the strong value in detailed statistics and reporting, even for apparently minor or low value metals, as they remain important in modern technology and society.

Some additional examples of current studies which have investigated or show the potential for companion or critical metal recovery include:

- Rhenium:
  - Yandera Cu–Au–Ag–Mo–Re porphyry deposit, Papua New Guinea, owned by Marengo Mining, with Cu and Mo concentrates from laboratory testing reporting <50 and 230 g/t Re, respectively (Hyland et al., 2012);
  - CuMo Mo–Cu–Au–Re porphyry deposit, USA, owned by American CuMo Mining Corporation (formerly Mosquito Consolidated Gold Mines), with overall Re recovery rates of between 20 and 52% and potential for additional value also noted from W and Ga (Jones et al., 2012);
  - Kalman Cu–Mo–Au–Re iron oxide Cu—Au (IOCG) deposit, Australia, currently owned by Hammer Metals, with overall expected Re recovery rates of 77% (HM, 2014);
  - Merlin Mo-Re-Cu IOCG project, Australia, currently owned by Chinova Resources (formerly a project of Ivanhoe Mines Australia), estimates a Re recovery rate of 75.7% (Horton, 2010).
- Hafnium:
  - Dubbo–Toongi Zr–REO–Nb–Hf–Ta alkaline complex project, Australia, under development by Alkane Resources, contains numerous critical metals, with Hf recovery during ore processing assumed to be 50% (Alkane, 2015);
- Tellurium:
  - Boliden–Kankberg Pb–Zn–Ag–Cu–Au–Te mine, Sweden, owned by Boliden, reports data for all 6 metals from 2012 to 2014, showing Te recoveries improving from 40.8 to 49.%, respectively (data from Boliden, 2015).
- Germanium:
  - Andrew Pb-Zn–Ag–Ge mesothermal vein deposit, Canada, owned by Overland Resources, with Pb and Zn concentrates from laboratory testing reporting 8 and 131.6 g/t Ge, respectively (OR, 2011).

At present, there remains a dearth of detailed deportment studies during the resources-reserves, mining, milling and smelting-refining stages for the full range of companion and critical metals – increasing the uncertainty and limiting our ability to understand and assess metal flows more precisely and accurately during primary extraction processes.

5.2. Hybrid methods for quantifying resources-reserves of critical metals

In the absence of detailed project by project compilations of resources-reserves, other methods could be used to provide approximate estimates of global resources-reserves of various critical metals. These are discussed briefly below, including examples where possible.

- Production Ratios: where it is known that a metal is derived almost exclusively as a by-product from a primary metal, it is possible to use the ratio of such production combined with global estimates of resources-reserves for the primary metal to approximate a critical metals' resources-reserves. Another important consideration is the critical metal capacity of current refineries and possible future expansions in capacity, as this provides a potential upper bound on annual production. For example (using statistics and information from Kelly et al., 2016, USGS, 2015):
  - *Indium*: Indium is mostly obtained from Zn refineries (~95%), with the ratio of global refined indium to Zn production shown in Fig. 5. Annual growth for refined indium and Zn production from 2000 to 2013 averaged ~11.2% and ~3.9%, respectively. The indium–Zn ratio has increased from a low of 3.7 g indium/t Zn in 1983 up to 62.0 g indium/t Zn in 2013, reflecting the considerable growth in demand for indium in recent years. At present, it is estimated that only 30–35% of indium is actually recovered from Zn concentrates, meaning that there remains strong growth opportunities for increasing indium recovery at Zn and other base metal refineries. Assuming a value of say 60 g indium/t Zn from current refineries, allowing for this to be one-third of possible indium present in Zn concentrates (i.e. 180 g indium/t Zn) and a recovery of 80% of the contained indium yields a value of 144 g indium/t Zn. Using USGS 2014 global Zn reserves of 230 Mt Zn, this gives an approximate global indium reserve of ~33,100 t indium – or using the global Zn mineral resource estimate of Mudd et al. (2016) of 534.1 Mt Zn, this gives ~76,900 t indium: both values are considerably higher than older USGS estimates as discussed earlier. Indium is analysed in greater detail in part II and is the primary focus of part III of this series of papers (Werner et al., under review a,b).
  - *Tellurium*: Te is mostly obtained from Cu refineries, with the ratio of global refined Te to Cu production included in Fig. 5. The annual growth of refined Te and Cu production averaged ~4.7% and ~3.1% per year from 2000 to 2013, respectively. The Te—Cu ratio has increased from a low of ~1 g Te/t Cu in the early 1930s to a peak of 44.7 g Te/t Cu in 1960, but gradually declined from this period onwards to a current range of ~7–15 g Te/t Cu. It is not clear why this ratio has declined (in contrast to the growth observed for indium),

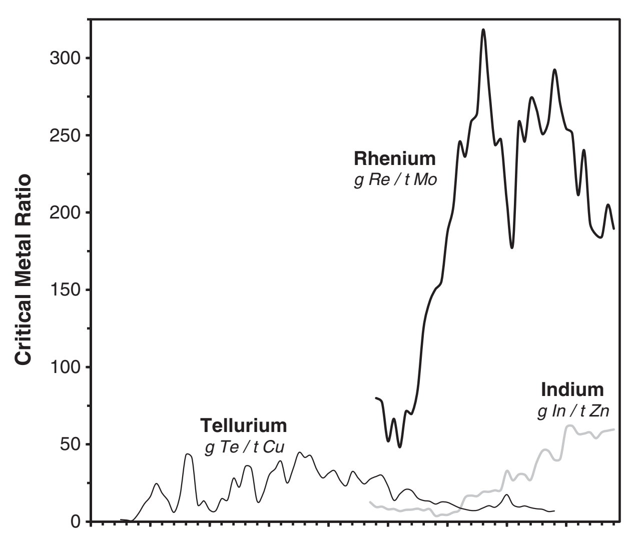

Fig. 5. Ratios of selected refined critical metal production to their dominant primary metal source (data from Kelly et al., 2016).

be it related to economics, recovery rates or declining concentrations in Cu concentrates and thereby primary deposits or the increasing use of solvent extraction-electrowinning (or SX-EW) technology (especially at refineries and heap leach mines). Assuming a value of say 10 g Te/t Cu from current refineries and using USGS 2014 global Cu reserves of 700 Mt Cu (USGS, 2015), yields an approximate global Te reserve of ~7000 t Te – or using the global Cu mineral resource estimate of Mudd et al. (2013a) of 1861.3 Mt Cu, this gives ~18,600 t Te: both values are somewhat lower than the 2014 USGS estimate of 24,000 t Te (USGS, 2015).

- ○ *Rhenium*: Re is almost exclusively obtained from Mo refineries, with the ratio of global refined Re to Mo production included in Fig. 5. Between 2000 and 2013, annual growth for refined Re and Mo production averaged ~2.9% and ~5.4%, respectively. The Re—Mo ratio has increased from a low of 48.2 g Re/t Mo in 1977 to a peak of 318.3 g Re/t Mo in 1991, but showing a gradual decline since this time and now averaging ~190 g Re/t Mo. As with Te, it is not clear as to why the ratio has declined (in contrast to the growth observed for indium), although this could be due to the stronger growth in Mo production in more recent years. Assuming a value of say 200 g Re/t Mo from current refineries and using USGS 2014 global Mo reserves of 11 Mt Mo, this gives an approximate global Re reserve of ~2200 t Re, similar to the 2014 USGS estimate of 2500 t Re (USGS, 2015).

- Deposit Types: a variety of mineral deposit types are likely to contain useful quantities of many critical metals. At present, almost all estimates of global resources-reserves simply report contained metals and do not breakdown by deposit types (e.g. volcanogenic massive sulphide, porphyry, sediment-hosted, secondary, magmatic sulphides, etc.). A more complete picture of deposit types for the major metals, and a more detailed consideration of deposits that have a single resource but contain multiple deposit types, each with different grades, tonnages, and contained metals (e.g. Jowitt et al., 2013a) would allow more accurate relationships between critical metals present in various deposit types and global estimates of the resources-reserves of critical metals.
- Cost Curves: most of the critical metals are produced as by-products, or recovered opportunistically at smelters and refineries. Although not widely used, an economic approach of estimating the available long-term resources by quantifying the costs of production can be performed, with one example being Li (Yaksic and Tilton, 2009). This approach has some appeal, but it requires extensive data on costs versus different sources, deposit types, ore (or solution) processing configurations and recovery rates – data which is rarely published or released by the companies involved.

6. Discussion and implications

It is clear that a lack of reporting of the numerous critical metals has led to a paucity of data relating to the global resources of these metals and beyond. This is not a reflection of exploration success (or lack thereof), but rather the historical and present-day economics of critical metal extraction and processing. The fact that these metals are often not the core business of mining companies means that these companies are not incentivised to report or even examine these commodities which may be marginal or sub-economic, or in the case of some elements may attract penalties during smelting (Cd) and may be seen (whether true or not) to adversely affect the share price of a company. This has resulted in very few code-based reports focusing on by- or co-products (inevitably including the critical metals), meaning in turn that high profile publications almost certainly have underestimated the total availability of these elements. One example where this is not the case is the recent study by Weng et al. (2015), who demonstrate the significant amount of (often mining code) quantified REE resources that are currently known, indicating that there are significant amounts of known and quantified resources of these critical metals. However, this is the exception, rather than the rule; as such, we would strongly encourage the mining community to at least discuss the possibility of including more by- and co-product information in resource (and ideally reserve, although given the economic certainty attributed to reserves rather than resources this may not be possible) reporting, even if this is just to outline future rather than current potential for a given resource to produce various co- or by-products. Recent developments have recognised the trends and uncertainties of mineral resource accounts in the literature (e.g. Crowson, 2012, Sykes et al., in press), with the effect of reducing some (but not all) of the questioning of 'how much is left'. More recent works on critical metals appear to place greater emphasis on anthropogenic traits such as supply risks, companionality and recycling rates (e.g. Nassar et al., 2015). One could, however, still argue that the reporting of critical metal resources has influenced these factors, as resource accounts can provide an indication of the adaptability of critical metal supply chains in the face of geopolitical supply constraints. This therefore highlights that the data provided by code based reports can refine our views of the location and processability of the deposits that may meet critical metal demand in the future, and hence the inherent value in reporting of critical metal resources should not restricted to resource accounting alone.

Given that many critical metals are likely to remain under-reported using current mineral resource reporting codes (although we hope future developments may address this), it is certainly worth reiterating the uncertainties associated with other types of resource accounts as outlined above, notably those by national/provincial agencies and corporate groups. As exemplified by the case of Fe ore in Australia, a lack of data combined with political and strategic objectives can influence governmental resource estimates (and associated restrictions) by multiple orders of magnitude. Yet these governmental estimates are more heavily relied upon for global resource estimates than accumulated code-based reports, with both Cohen (2007) and Moyer (2010) using USGS data (albeit potentially incorrectly) in their assessments of critical metal depletion. In the absence of code-based reports or detailed datasets compiled through hybrid methods as outlined previously, governmental reports are still likely to remain a key source of information for those without the time or resources to conduct new empirical studies. An additional discussion of the uncertainties of USGS data and sources is presented by Crowson (2011), who highlights that the USGS themselves have contended with limited data and organisational resources, have presented differing levels of analysis over time and between metals, as well as the change in mineral resource assessments from the former USBoM to the USGS (and the challenges implied in such major organisational change). Despite this, the USGS remains the most commonly used (and often the best or only) source of information like this, and one of the reasons that they ceased of reporting of reserve base was the extent of work required to develop these estimates (i.e. possibly under-resourcing the USGS mineral resources program as well as their international equivalents). This leaves policy makers in a quandary, especially as although corporate reports can provide more reliable estimates, the provenance of the data within these reports and information on the methodology used to compile these data are lacking, primarily as they often rely on sensitive proprietary information.

Moving forward, it appears that there are changes necessary both within the public domain and the research community to address the uncertainties of resource-reserve accounting for companion or critical metals. In the public domain, it must be clearly understood that it is only in a mining company's interest to prove what amount of ore (and grades etc.) justifies their investments into extraction and the development of processing capabilities, not to conduct an altruistic study that contributes to scholarly efforts in global resource estimation or to spend exorbitant amounts drilling out the entirety of a deposit before commencing mining, an understanding that needs to be incorporated into research into the security of supply of all commodities, not just the critical metals. Some have likened resource-reserve reporting to a baker reporting what bread exists on shelves for that day. This is a relatively simple message to communicate, and could be more explicitly noted, in some way, in future resource accounts. Guidelines for increased reporting of strategic minerals would be of considerable value within the research community, could inform related studies (e.g. of criticality) and would provide more nuanced views of critical mineral resources. Similar clarity on the classification of primary, co-product and by-product commodity status would also provide great utility within the research community. It is therefore a major recommendation of this study that existing mineral resource reporting codes be reviewed for their potential to more formally define the economic status of reported commodities, and to incorporate the identification of critical metals to a greater extent. In the meantime, and in the event that such changes are not implemented we would argue that the research community should embrace the possible hybrid methodologies outlined above (and discussed in more detail in Werner et al., under review a) that enable the use of existing information to better infer critical metal resources in the absence of quality reporting, and which permit data quality to be evaluated.

Mineral deposits are of course not the only potential source of metals. Once extracted and fabricated, critical metals are often integrated into specialised technological applications, which themselves experience very low recycling rates and hence accumulate as anthropogenic stocks in society or as e-waste in landfills (see UNEP, 2013). The development of infrastructure for end of life recycling has been of particular interest in recent years, with a number of e-waste reprocessing facilities established in the EU (see Rombach and Friedrich, 2014) as well as ongoing research into the mining of landfills (e.g. ELMC, 2012, Richards, 2014) and other alternative sites to primary mineral deposits. There are both challenges and opportunities for secondary resource recovery. Firstly, the technical challenges of extracting critical metals from difficult to process e-waste components creates significant economic challenges (Khaliq et al., 2014). Additionally, the technology and legislative background for recovery of critical metals from these sources is perhaps underdeveloped, especially when compared to the developments in base metals (Hagelüken, 2014). The potential volumes available for recycling are also of concern, as previous research has indicated that extracting all of the indium from Australia's unused smartphones in 2014 would only produce <0.5 t indium (Werner et al., 2015), clearly not comparable to even a single deposit as listed in Table 1 or 2014 world production of ~820 t indium (USGS, 2015). However, we note that there is still much work to be done to better quantify secondary resource potential for many metals globally (see Chen and Graedel, 2012). The assessment and reporting of end of life resources is further complicated in that e-waste is managed by a number of different stakeholders and organisations, who may play various but complementary (and occasionally conflicting) roles in collection and processing (Lane, 2014). There are therefore questions over who would take responsibility for resource accounting, an issue not faced for accounting of virgin resources. There may be social benefits for the future exploitation and recycling of e-waste, such as the reduction of demand for conflict metals (which are often critical, such as Ta; e.g. Bloodworth, 2015), however it is also possible that instead of the current trade in conflict minerals we develop a new trade in conflict e-waste, although e-waste flows (and associated pollution and other issues) are typically moving from developed to developing countries, rather than the other way (Robinson, 2009, Widmer et al., 2005). These consequences certainly require further study.

Despite these challenges, the longer term environmental outcomes of extraction from secondary resources are driving their continued consideration. Some evidence of the comparative environmental benefits is provided by Yamasue et al. (2009), who compared the extraction of metals from so-called "urban mines" with that of conventional mines by examining the total amount of materials required to produce a pure metallic product. They showed that the total material requirement (TMR; i.e. the direct and indirect material inputs required to extract a given mass of metal) is generally lower for metals sourced from urban mines, with indium being a notable exception. While this does not directly suggest that production from secondary sources is more economically attractive, it does highlight the material, and hence environmental, benefits of secondary extraction, which may influence the relative economic viability of recycling in future. Recognition of the environmental benefits of recycling today through improved reporting of secondary resource potential could be strategically advantageous, particularly for critical metals, although given the significant quantities estimated for some critical metals in mineral resources (e.g. Werner et al. under review b), it is unlikely they will be significantly exploited in the short to medium term.

In summary, although a number of recent studies have highlighted possibly peak behaviour in by-product and critical metals (and a variety of other commodities), we here pose the question of whether this is based on reality or rather a paucity of information. As such, we call on the global mining and resources community to come together to a) compile the information needed to clearly define known critical metal resources in a robust and open manner, b) discuss a new framework for the assessment of all aspects of the supply and security of supply of these resources, including both secondary and primary sources, and c) use this information to outline priorities for future research. In many ways, the global mining and resources communities have started answering questions about the security of supply of critical metals without fully understanding the questions (such as the recent European Commission conference on by-product metals in Brussels, Belgium, in November 2015). This is as important (if not more important) as the current 'UNCOVER' initiative to examine exploration under cover in Australia (AAS, 2012), primarily as if we cannot understand the resources we currently have, then how can we understand the resources we discover in the future? This means that unless research is undertaken to address this now, we may be forever be examining the criticality (or lack thereof) of these resources through a veil, and what is more, governments, mining companies, and other organisations will be basing policy, exploration, environmental and social decisions on this potentially occluded data.

7. Conclusions

This paper documents and discusses numerous factors which commonly lead to by-products and critical metals not being reported as a potential commodity in the mining industry, and the implications of this lack of reporting. In particular, the lesser economic value of these metals reduces their chances of being extracted, despite their geological presence and potential strategic or technological value. Under the mineral resource reporting codes commonly employed in the mining industry, there are no strict guidelines for reporting such metals. For those commodities which have been assessed via the typical institutions and methods identified, there are still many issues, e.g. relating to politics, economics or corporate confidentiality, which hinder data availability and traceability. These issues have created ambiguity around the scarcity of many by-products and critical metals, however we note that these factors are not related to discovery success. There have been recent efforts to explore the secondary resource potential of many critical metals, and these have not been explored in detail in this study, although the limited available data on mineral resources suggests that secondary production is not essential to ensure future critical metal demand is met.

This study has identified multiple ways in which by-product and critical metal resources can be estimated, including leveraging from studies on deportment, using production ratios and through the development of global deposit databases. Although these approaches are yet to be extensively applied, they present an important opportunity to overcome the lack of resource data for many technologically and strategically important metals, and permit a shift from the use of incomplete data to inform policymaking. It is hoped that this study may be used to stimulate further discussion amongst the academic, mining and resource communities that addresses the uncertainties identified here.

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Product flow analysis using trade statistics and consumer survey data: a case study of mobile phones in Australia

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ABSTRACT

This study describes an integrative approach to product flow analysis of (waste) electrical and electronic equipment using trade statistics and consumer survey data. We demonstrate this approach with a case study of mobile phones. Using statistical and empirical data for Australia over 1997–2014, we have shown how different sources of information can be collated and cross-checked to estimate the product in-use stocks and flows, product lifespan and lifespan structure, as well as to detail the product age structure in stock and at the end of life.

From our results, the total number of mobile phones in in-use stocks in Australia has been estimated at 46 million at the end of 2014, or about 2 phones per capita. The proportion of phones kept in storage (not being in use) has been constantly rising, reaching 50% in 2012–2014. The average expected lifespan for a mobile phone sold in Australia decreased from about six years in the late 1990s to about five years in the early 2000s, and then stabilised at around four years (±0.5 years). The average time of active use for mobile phones was estimated in the range of 2.0–2.6 years (which includes first use and reuse). The estimated lifespan profile for mobile phones in Australia has been confirmed to be relatively similar to that reported in Japan.

While this methodology presented here provided meaningful results, the accuracy and relevance would be improved by better quality of original data. Therefore, in conclusion, we also highlight potential improvements in consumer surveys that would help to enhance the analysis.

1. Introduction

Electronic products such as mobile phones, laptops, TVs and tablets utilise the physical properties of highly specialised and geochemically scarce metals to function. These metals (e.g. Ag, Au, In and many others) must be mined and refined, sometimes at significant environmental and social cost, to be integrated into these products. Yet many electronic products are wasted at the end of their useful lives, appearing in landfill or in some cases illegally exported to developing nations, fostering further economic, environmental and social problems for these countries (Balde et al., 2015). The recovery of valuable components in electronic products has attracted significant interest in recent years as a means to reducing these risks (e.g. Li et al., 2015; Pickren, 2014), particularly amidst growing environmental impacts and regulations facing the mining industry (e.g. Mudd, 2010). However, the economic recovery of metals from e-waste requires some understanding of the location, composition and volume of products available for future extraction, so that investments into recovery operations can be properly informed. Approximations and modelling are necessary to obtain such information in the absence of direct measurement and reporting.

The materials in electronic products, and indeed all metals in society, whether active in use or dormant and not yet disposed of, are known as 'in-use stocks'. In-use stocks have been indirectly or directly estimated through various approaches and methodologies, each with different emphases, for example in input–output accounts, national capital accounts, life-cycle assessments (LCA) and material flow analyses (MFA) (Pauliuk et al., 2015). Within the MFA studies, there are two primary approaches by which in-use stocks

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of products or their contained materials have historically been estimated: top-down and bottom-up. The top-down approach essentially entails the collection and analysis of data on material inputs and outputs for a specified system. The difference between inflows and outflows (e.g. imports plus domestic production minus exports) over a specified time period can indicate in-use stocks via mass balance. The bottom-up approach entails the collection of data on the number of products/commodities within a given area and summing these to estimate the total in-use stocks.

Both the top-down and bottom-up approaches have advantages and disadvantages. For example, while bottom-up studies permit the spatial distribution of in-use stocks to be estimated, they are often temporally restricted to one year. Here, top-down studies can shed more light as they permit multi-year analyses and hence trends in stock accumulation, although they often rely on highly aggregated data that do not relate to specific products, and are further problematic to disaggregate spatially. These and other methodological uncertainties are described in numerous previous studies, e.g. (Chen and Graedel, 2015a; Gerst and Graedel, 2008; UNEP, 2010).

The multi-year analyses under the top-down approach are referred to as Dynamic MFA (DMFA) and have been applied to describe historical material flows and stocks of various metal resources. Several DMFA studies have projected possible future developments and related resource flows at both national and global levels based on scenarios. Muller et al. (2006) and Wang et al. (2007) focused on the anthropogenic iron and steel cycle, Daigo et al. (2007) estimated both the in-use and the total steel stock, which includes hibernating stock in Japan, and Reck et al. (2008) analysed the nickel stock and flows at the national and global scale. While the above studies have focussed on specific metals, other MFA studies have emerged which focus on the flows of specific products. Oguchi et al. (2008) analysed the circulation of major consumer durables in Japan, Harper (2008) analysed global flows of tungsten-containing products, and Chen and Graedel (2015b) estimated in-use stocks of 91 products in the United States.

Studies in MFA have additionally used monetary Input Output (IO) tables even for relatively small flows; an example is found in the work of Nakamura et al. (2007). IO analysis is one of the most widely used tools for describing economy-wide activities and their environmental implications (Suh, 2009). IO based MFA models, for example Waste IO-MFA, analyse the compositions of the materials or substances in products and scrap. Nakamura and colleagues provided several studies on IO based MFA (Nakamura et al., 2008, 2009; Ohno et al., 2014). Nakamura et al. (2014) also provided a worthy methodological framework, MaTrace model to enable visual tracking of the fate of materials whether accumulated in in-use stocks or dissipated in waste streams. In addition, Wang et al. (2013) provided a detailed overview of different IO models used for product flow analyses and e-waste estimation.

If elements of multiple MFA methodologies are applied to the same commodity under the same system boundaries, more can be revealed about the nature of that commodity. For example, both the spatial distribution of the commodity and potential trends over time in stock accumulation could be determined, and further the uncertainty associated with each method could be compared and interpreted. Very few studies have conducted multiple assessments, e.g. both top-down and bottom-up assessments of the same commodity or product, with Hirato et al. (2009) being a notable example. This is likely due to the time taken to conduct MFA studies, and given that regardless of the specific MFA method employed, almost all MFA studies must contend with a lack of up to date and spatially relevant data. Indeed, it is relatively accepted that the contribution offered by an MFA study is that it synthesises available data to characterise the flows of a new commodity, and/or to represent previously un-studied spatial/temporal aspects, but not necessarily that it employs raw data collection.

The limited data sources which are available for MFA studies can often be re-used through multiple generations of studies, which ultimately become less spatially and temporally relevant to the source data. For several electronic products including mobile phones, we have seen increases in value and utility, and considerable hoarding behaviour developed (ACMA, 2015; Read, 2015), which affects the in-use stocks and average lifespans of the products. There are therefore considerable uncertainties for future projections of e-waste volumes associated with using fixed product lifespan and distribution parameters (e.g. Weibull function) based on previous investigations. Furthermore, the limited number of studies currently used to inform in-use stock behaviour may provide source data that is spatially explicit (i.e. reflecting usage behaviour in a certain country), making them problematic to infer for other locations.

Empirically collected data, which reflects the system boundaries of the MFA study itself, can assist in reducing these uncertainties, and hence this study focuses on how such data can be integrated into multiple methods of product in-use stocks and flows estimation. In the following sections we describe this approach in detail and demonstrate its application with the case study of mobile phones in Australia.

2. Methodology description

Estimating (waste) electrical and electronic equipment ((W)EEE) circulation is a difficult task due to often low quality and incomplete data, meaning that multiple assumptions are required for input–output modelling (Wang et al., 2013). The annual sales of EEE (in monetary value and units) are usually well recorded through national and international systems and institutions (e.g. UN Comtrade database), while the information on in-use stocks and end-of-life (EoL) products is not directly documented. To uncover the latter, detailed consumer (and institutional) surveys, information from professional associations and authorities (e.g. telecommunication regulatory bodies), recycling and waste management companies data, as well as special investigations are needed (Fig. 1).

The top-down approach in this study uses aggregated information at the country level. The modelling of in-use stocks and flows in this approach can be solely based on trade statistics. Using estimations of products average lifespans from previous studies allows an approximation of the overall circulation of (W)EEE in the economy (Balde et al., 2015). However, for many EEE categories the lifespans significantly differ over time and/or between countries. The up-to-date (dynamic) country/region based information derived from bottom-up approaches can significantly improve and/or help validate the modelling.

The overall approach to estimating the circulation of mobile phones in this study is presented in Fig. 2. Most individual parts of this approach are generic and can be applied to any EEE, however the integration of top-down and bottom-up components is permitted by the available data, which is an uncommon feature in MFA. The system boundaries for this study are limited by EoL products generation, although the (historic) consumer surveys also indicate the likely pathways for mobile phones at the end of life.

Our methodology first requires the compilation of two major datasets: sales and in-use stocks, based on trade statistics and consumer surveys accordingly. The information on mobile services subscriptions can be used for comparative purposes to support the mobile phones in active use estimation. The number of EoL phones (outputs) can be estimated via mass balance between inputs (phone sales) and in-use stocks for every respective year.

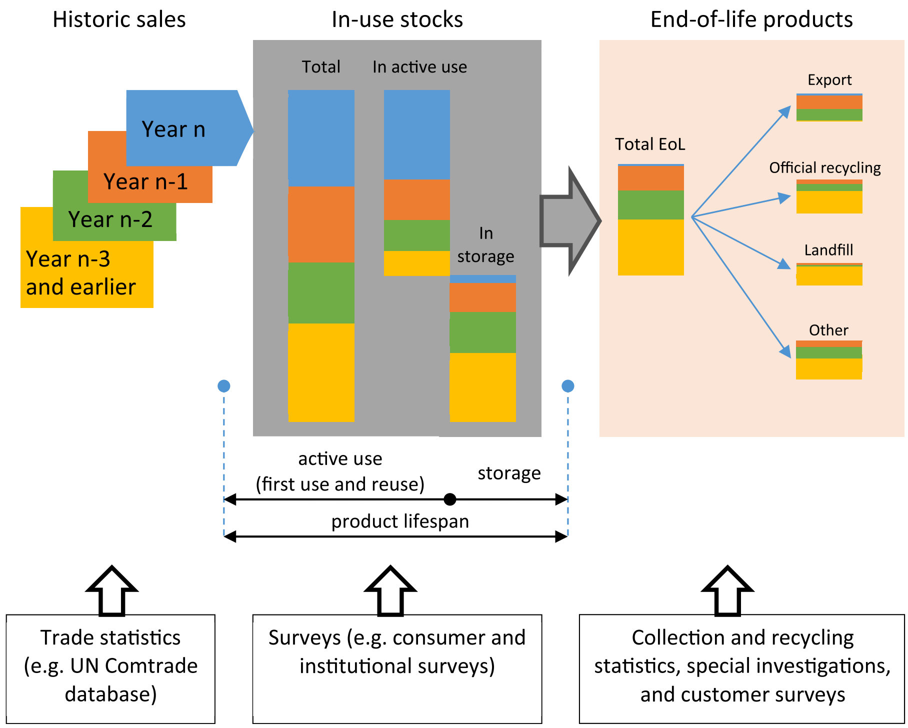

Fig. 1. Illustration of in-use stocks and flows estimation across different product age groups in product flow analysis for (W)EEE. Note: the size of individual bars is for illustrative purposes and may not match the balance in total.

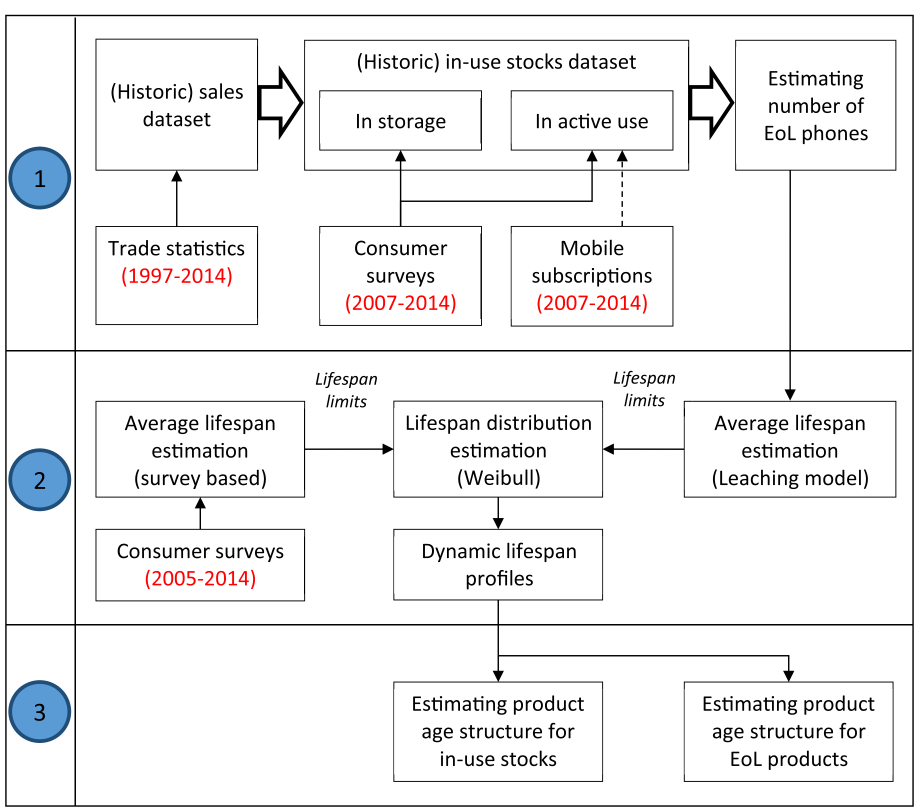

Fig. 2. Using empirical data to model in-use stocks and flows of mobile phones in this study.

Second, the product lifespan is estimated by different methods and cross-checked. Different scopes of a product or commodity circulation within the economic system can be used to define the lifespan. Murakami et al. (2010) and Oguchi et al. (2010) provided a detailed overview of lifespan scopes and classified different methodologies for estimating the lifespan distribution. The product lifespan which we use in this study can be referred to total lifespan for consumer durables in the classification by Murakami et al. (2010), and is measured by the following techniques:

- **Average lifespan estimation based on the Leaching model**. The input–output data from the previous step allows the use of the Leaching model (assuming that the product has reached the market saturation level). The average lifespan can be estimated as the total stock divided by the EoL products generation (Wang et al., 2013).
- **Average lifespan estimation based on consumer surveys**. The consumer survey data, namely information regarding expected time of use for a new mobile phone (and/or time of use for the previous phone) and expected destiny for this phone after use (and/or destiny for the previous phone), are used to reconstruct the average lifespan of a phone.
- **Lifespan distribution estimation based on the use of the Weibull function**. The models above provide estimation of average lifespan only, while statistical functions such as the Weibull function also determine the lifespan distribution. The use of lifespan range (limits) allows for optimizing the search for suitable distribution function parameters.

Finally, the Weibull distribution parameters can be used to reconstruct the product age structure for the in-use stocks and EoL flows.

The use of a Weibull function provides better results for modelling stocks and flows of EEE products than other statistical functions (Wang et al., 2013). In general, the estimation of distribution parameters requires detailed information, not only on the number of devices coming into and being in stock, but also on the age structure of products (in stock and/or at the end of life). Without the latter, there is still a possibility to find suitable parameters, however there may be multiple solutions satisfying the requirement of matching the inflows and stocks (and/or inflows and outflows). As shown in Fig. 2, we suggest a possible way to resolve this issue by limiting the lifespan range, based on findings from other methods for product lifespan estimation, while searching for the best-fit Weibull function parameters. This drastically decreases not only the number of required iterations but also the number of satisfactory solutions. The non-linear regression analysis along with the solver function in MS Excel can help to define the best-fit parameters for Weibull distribution (Wang et al., 2013). However, there is still a possibility that the solution does not exist or does not provide an adequate result; in this case the original data needs to be checked for possible errors and/or alternative statistical distributions considered.

3. Case study – mobile phones in Australia

In this section, we demonstrate the application of the developed methodology to mobile phones in Australia. First, the existing trade statistics and consumer survey data are compiled to estimate the number of mobile phones in stock and at the end of life. Second, the average lifespan for a mobile phone is estimated by different methods, including the Leaching model, survey based approach, and Weibull distribution fitting. Third, the Weibull distribution parameters are used to reconstruct the in-use stocks and EoL product flows age structure. Finally, the adequacy of the data and the reliability of results are discussed, including suggestions for improvement. Most data in this investigation are represented on a one-year basis, approximated to the end of the year where applicable (also see Supplementary document for details).

3.1. Estimating in-use stocks and end-of-life products

The export–import statistics for every country can be obtained from the UN Comtrade database. The mobile phones are represented by the code 851712, Harmonised System, “Telephones for cellular networks/for other wireless networks, other than line telephone sets with cordless handsets”. There is no known production or assembly of mobile phones in Australia, and the domestic EEE sector is relatively small overall (IBISWorld, 2015). Therefore, in this study we assume imports being equal to sales of mobile phones. There is an uncertainty on how to interpret the export data, which forms up to 10% of imports of mobile phones (by the number of units). By comparing average prices between imported and exported devices, it can be concluded that a significant part of mobile phone exports may be represented by old devices for resale and reuse overseas (Fig. 3); this issue was also highlighted by Wang et al. (2012) for EEE in general.

In Australia, the consumer surveys on mobile phone possession and use have been regularly performed by Mobile Muster, the Australian mobile phone industry's official product stewardship program (Read, 2015). The Mobile Muster's reports also provide data to similarly estimate the number of phones in active use (see Supplementary materials to this article for details), which can be further compared with mobile phone subscriptions statistics from the Australian Communication and Media Authority (ACMA). Since 2009, there has been a growing disparity between the estimated, based on surveys, number of phones in active use versus the number of mobile subscriptions, which has reached about 8% (or two million units) in the last three years (Fig. 4). This could be explained by the presence of mobile phones supporting two SIM cards at the same time. However, there is also a growing number of children owning mobile phones, which is not covered by current consumer surveys focussing primarily on adults (at least 15+ years old). For example, the recent investigation initiated by Telstra in Australia highlighted that the average age to receive the first mobile phone was 12 in 2014 (SMH, 2015).

The numbers of phones in active use and in storage equate to the total number of phones in in-use stocks; comparing the latter with annual sales allows us to estimate the number of EoL phones (Fig. 5). The mobile phone annual sales in Australia were in the range from 10 to 13 million units in 2008–2014, peaking at 13.1 million in 2010 (Fig. 5). The total number of mobile phones in in-use stocks had been increasing until 2012, and stabilised at around 46 million units or about 2 phones per capita (or 5.1 per household). The mobile phones not in use (kept in storage) form a significant part of the total in-use stocks, in the last three years they accounted for about 50% of all phones hold by Australian households (Fig. 4). The generation of EoL phones has reached parity with new phone sales, and estimated at 12 million units in 2014 (Fig. 5).

3.2. Estimating product lifespan

3.2.1. Leaching model

The Leaching model provides an adequate estimate of lifespan if the product has reached the market saturation level (Wang et al., 2013). The mobile phones have been widely introduced to the market in the second half of 1990s. By the mid-2000s the product penetration level, based on mobile subscriptions, has reached 100% in Australia, on average covering every person aged 15+ (ACMA, 2006).

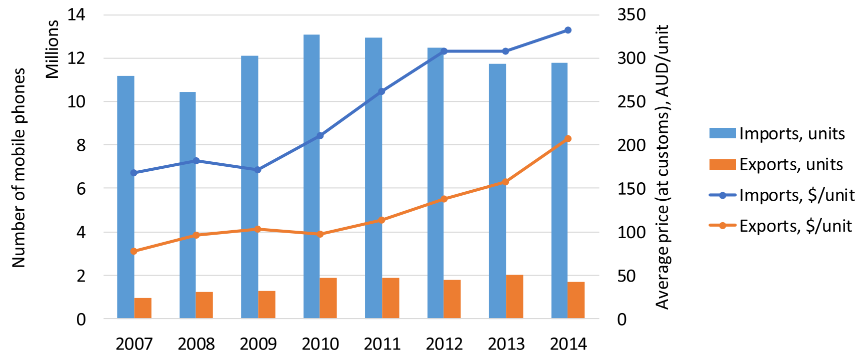

Fig. 3. Import and export statistics for mobile phones in Australia. Data source: (UN Comtrade, 2015).

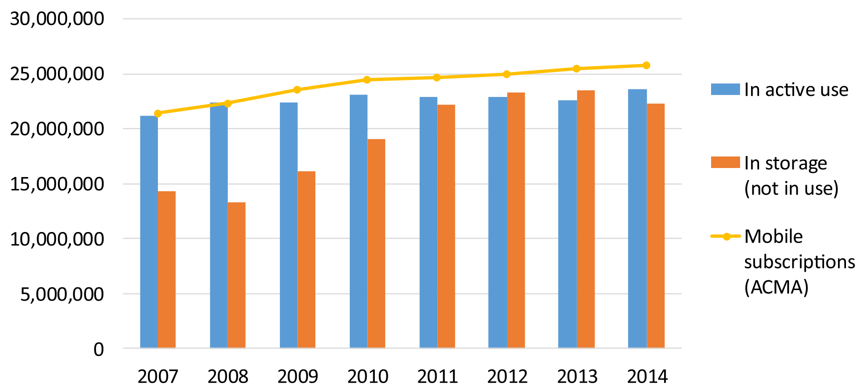

Fig. 4. Number of mobile phones in Australia. Data sources: (ACMA, 2015; Read, 2015).

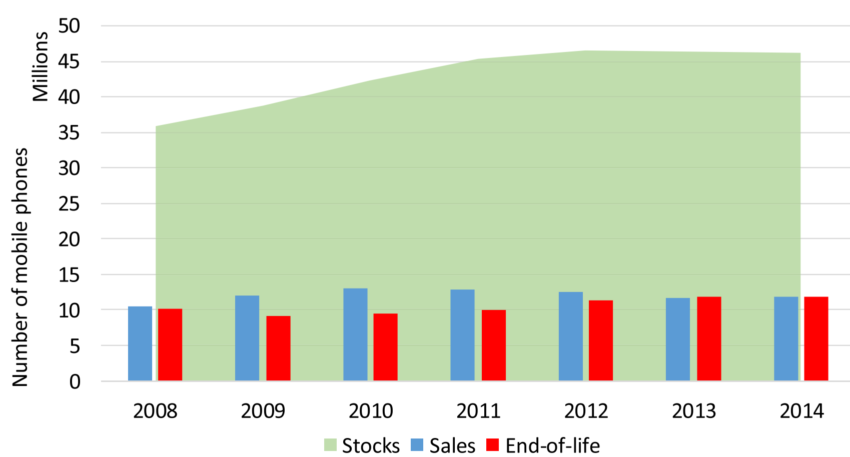

Fig. 5. Stocks and flows of mobile phones in Australia.

Using the number of mobile phones in stock and at the end-of-life from the previous section, the average lifespan has been estimated and presented in Fig. 6. In 2008–2014, it was fluctuating around 4 years with plus or minus of a half-year difference.

3.2.2. Consumer surveys based approach

The consumer surveys can provide invaluable insights into estimating the life of electronic devices in the society. These include patterns of ownership and use, hoarding behaviours, awareness and attitudes of recycling, and ways of disposing. The data from such surveys can be used to estimate product in-use stocks and lifespan. When performed regularly, the survey data also allow the temporal analysis of patterns. In this study, Mobile Muster's consumer survey data from 2006 to 2015 (Read, 2015) were used to estimate mobile phone lifespan and its changes over time.

Mobile Muster has conducted consumer surveys on mobile phone use and recycling every year since 2006. The survey respondents were selected randomly from an online panel, who were 15 years old or older and owned a mobile phone (Read, 2015). The sample size for each survey ranged from 600 to 1100 people. The survey questions include those about the current phone ownership and use patterns, the length of the actual and expected time period consumers use their phones, when and why they get a new phone, what they are doing with old phones, and how they recycle their

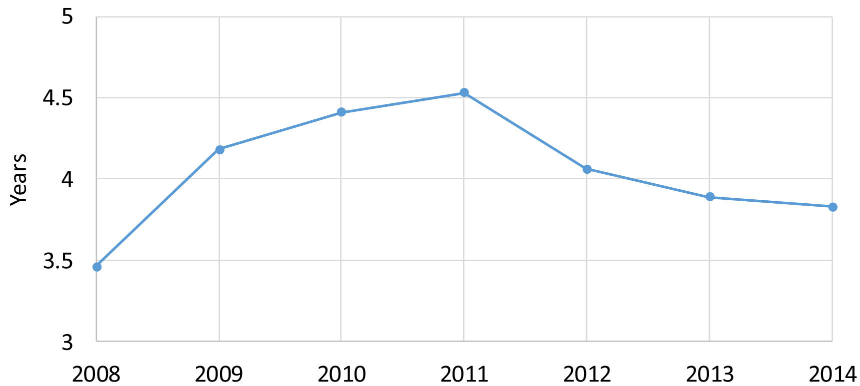

Fig. 6. Average lifespan of a mobile phone in Australia (Leaching model).

phones. Read (2015) provided a detailed description of the survey methods. The data on the destiny of consumers' previous mobile phones, and their expected period of use for new phones from the surveys were used to reconstruct the structure of mobile phone lifespan.

The expected lifespan of mobile phones (L) can be calculated as the sum of three components: time of the first use (T_F), time of reuse (T_R), and storage time (T_S).

$$L = T_F + T_R + T_S$$

Mobile Muster's surveys divided the length of the use of new mobile phones into 6 groups (or time intervals): <6 months, 6–11 months, 12–18 months, 19–24 months, 2+ years and “don't know”. The time of the first use and time of storage after the first use can be estimated by multiplying percentages of respondents and average values of time intervals for the first use/storage, based on the survey data. The average value of a time interval is defined as its middle point. For example, if a time interval is from 12 to 18 months, the average value of the time interval would be equal to 15 months (or 1.25 years). For the time interval of 2+ years, the middle point is set as 2.5 years. The percentage of people who answered “don't know” is added to the answers for the “2+ years” group.

There is no available information for estimating a phone's reuse time and storage time after reuse. Therefore, we applied the estimates of first use (storage) with additional 0.8 ratio, assuming that second hand phones are of less value for consumers, and thus the average time of reuse and following storage is likely to be shorter compared to a new phone.

The estimated results are presented in Fig. 7. It is interesting to note that the estimated average expected lifespan for a mobile phone did not fluctuate much in the period 2005–2014, namely from 3.6 to 3.8 years. This could be partly explained by mobile phones reaching the saturation level and maturity in the market, but may also be due to some inaccurate or unreliable answers from the surveys. In this investigation, while assuming that available data are reliable, we also cross-checked the results using different approaches to the lifespan estimation. The results from the previous section (Fig. 6), while being slightly higher, generally align closely with the estimation presented in Fig. 7.

As it can be seen in Fig. 7, the period of active use of mobile phones (first use plus reuse) typically comprises about two thirds of the life of a mobile phone. This should result in a similar ratio between mobile phones in active use and in storage (i.e. two to one). However, the consumer surveys have showed that this proportion is about one to one (Fig. 4). A possible explanation to this could be that people tend to underestimate the expected time of mobile phone storage after use (and the number of phones stored after use), and/or overestimate the expected time of new phone use (i.e. replacing phones more often than originally expected).

According to the consumer survey results, every second phone in Australia is expected to be stored after the first use, every fourth gets a second life with relatives/friends or resold for reuse, every 10th goes to recycling, and one out of 20 is lost or disposed (Read, 2015). Some important questions arise from a more detailed analysis of the survey data. These include the lack of information regarding the destiny of older phones in storage, e.g. while most respondents admitted that they would prefer to store their last phone, the same may or may not be valid for older phones in storage, which in turn affects the average storage time and overall product lifespan. An additional question regarding how long the consumers were actually using their previous phone (apart from

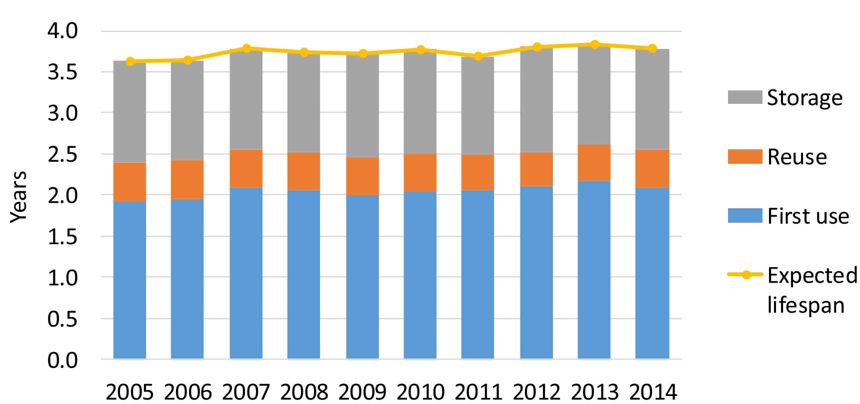

Fig. 7. Expected lifespan for a mobile phone in Australia (based on consumer surveys).

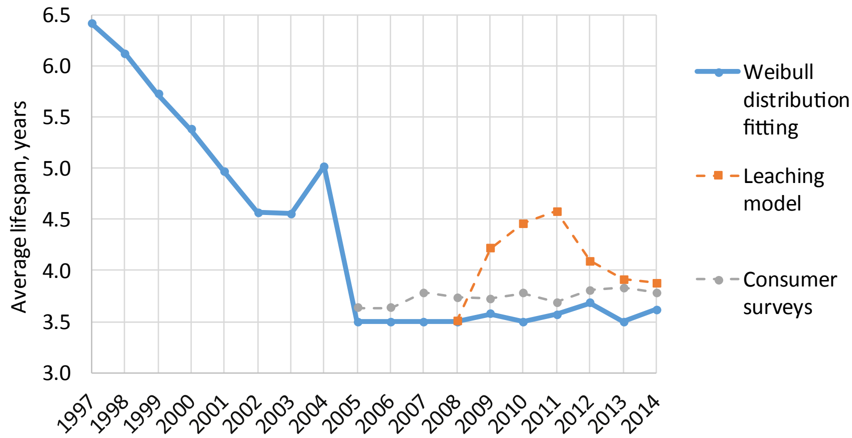

Fig. 8. Comparison of estimated average lifespans for mobile phone by different methods.

what happened to it) would also be helpful to verify the lifespan of a mobile phone. There is also a lack of details about the behaviour of consumers who prefer replacing their mobile phones more often (e.g. every year). A critical question is: does this show a different pattern, e.g. would a mobile phone be more likely to be reused/resold rather than stored if replaced more often?

3.2.3. *The use of Weibull function distribution*

The Weibull distribution has been demonstrated as the most suitable function to describe the obsolescence of consumer durables, including mobile phones. The generation of EoL products can be estimated with the use of a cumulative function, while product's average lifespan is represented by the mean value of Weibull distribution (Oguchi et al., 2008; Polák and Drápalová, 2012).

The lifespan estimates from previous sections can be used as limits in non-linear regression analysis to define the best-fit parameters for Weibull distribution, with the use of solver function in MS Excel (Wang et al., 2013). The following limits have been applied to average lifespan (mean value): 3.5–4.5 year range over 2005–2014, and extended to 3.5–7 years over 1997–2004 (no survey data are available for this period). Additionally, parameter α (shape) of a Weibull function has been limited to values from 0.7 to 3.1 based on the range of values for this parameter defined in previous studies for different EEE (Oguchi et al., 2008; Wang et al., 2013); we believe that this can representatively cover different potential rates of ageing for mobile phones.

For the computation of stocks and flows with the use of a Weibull function, we applied the midpoint values for every year (e.g. 0.5 for year #1, 1.5 for year #2 etc.), similar to the approach used by Wang et al. (2013). We also believe that this is more relevant for products with shorter lifespan, i.e. a certain part of product sales is accounted for the EoL flows starting with the first year. The results from estimating the best-fit parameters for Weibull distribution over 1997–2014 are presented in Table 1, and also compared with estimations by other methods in Fig. 8.

The estimated parameters for the Weibull distribution (Table 1) show a constant decrease in parameter β until stabilising at around 4.0 since 2005, while there is a significant fluctuation in parameter α – from 1.03 to 3.10 (within the applied limits of 0.7–3.1). This may be caused by potential inconsistencies in the original data. It can also be explained by the fitting process itself, i.e. there may be several solutions (combinations of Weibull parameters values) that meet the requirements within the applied limits. For comparability with other studies, we suggest to average the parameters in Table 1 over the 5-year period. The lifespan distribution curves for mobile phones in Australia, averaged for 2001–05 and 2010–14, are compared with the results from studies in other countries in Fig. 9 (a) and (b).

The proportion of mobile phones reaching the EoL status earlier has increased in Australia over time: in 2001–05 about 37% of mobile phones were expected to reach the end of life within 3.5-year time versus about 50% in 2010–14 (Fig. 9b). Based on the shape of curves, the lifespan distribution in Australia is relatively close to that from the Japanese study for 2003, while the results for the Netherlands in 2005 and Czech Republic in 1996–2008 are standing apart (Fig. 9).

The Czech study was based on the analysis of the EoL mobile phones' age structure in 2008 derived from the official collection systems and special campaigns (Polák and Drápalová, 2012). This could result in a bias due to the fact that a significant number of EoL phones can be exported for reuse, end up in landfills, and/or go through unofficial collection and recycling systems (see also Fig. 1 for details on possible destinations for the EoL products). The use of only one source of information (i.e. official collection systems) and only one selected year in reconstructing the age structure of EoL products would not provide an adequate result in the lifespan modelling for mobile phones, thus has to be avoided.

Table 1
The best-fit parameter values of Weibull distribution for mobile phones in Australia.

Year	Weibull parameters		Average lifespan (mean value), years
	α (shape)	β (scale)	
1997	3.10	7.17	6.41
1998	2.99	6.85	6.12
1999	2.97	6.41	5.72
2000	3.05	6.02	5.38
2001	3.06	5.55	4.96
2002	3.10	5.10	4.56
2003	2.03	5.14	4.56
2004	0.97	4.94	5.01
2005	2.38	3.95	3.50
2006	1.57	3.90	3.50
2007	1.28	3.78	3.50
2008	2.87	3.93	3.50
2009	1.65	4.00	3.58
2010	3.10	3.91	3.50
2011	1.82	4.01	3.57
2012	3.10	4.11	3.68
2013	2.66	3.94	3.50
2014	3.05	4.05	3.62

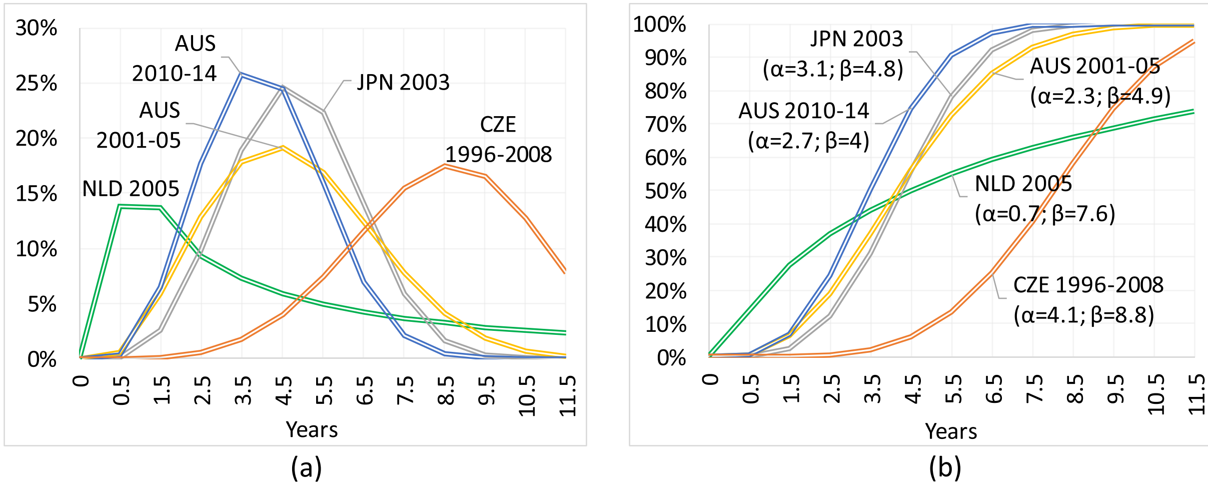

Fig. 9. Comparison of Weibull lifespan distribution curves for mobile phones from this study (average for Australia in 2001–05 and 2010–14) with previous estimations in Japan, Netherlands, and Czech Republic: a) annual EoL product rate; b) accumulated EoL product rate.

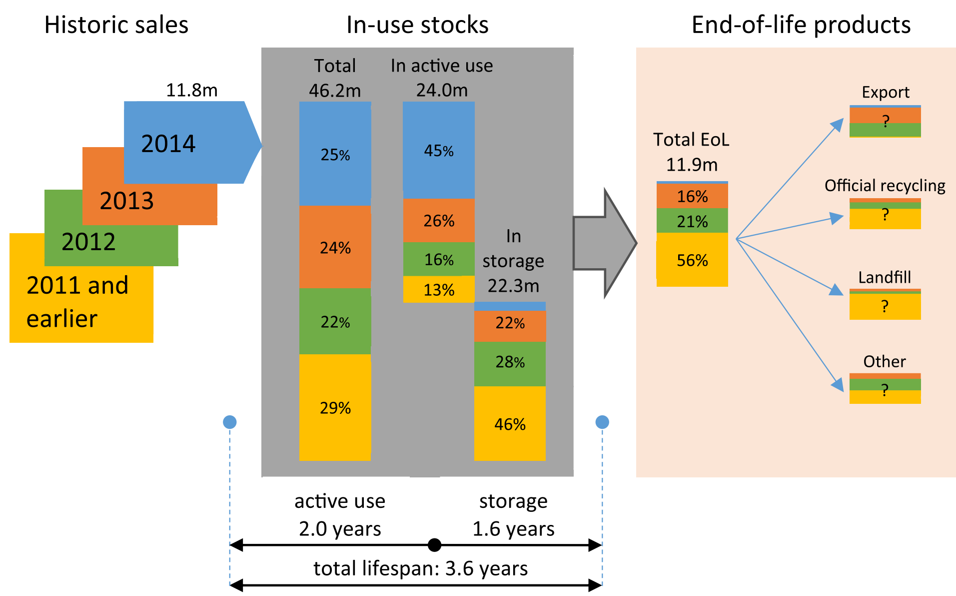

Fig. 10. Estimated in-use stocks and flows across different product age groups for mobile phones in Australia in 2014.

The lifespan estimate in the Dutch study (Wang et al., 2013) shows a relatively high EoL product rate in the first two years followed by a drastic decrease resulting in about 30% of mobile phones being in stock even 10 years later after purchase (Fig. 9b). One of possible explanations for this rather unusual result could be that it is based on the consumer surveys. These surveys often indicate the expected time of use for new bought phones, but usually detail only the first 2–3 years. Another important point for analysing the consumer survey results is that the first use of mobile phone is not equal to its total lifespan, which also includes the reuse and storage components.

3.3. Estimating product age structure for in-use stocks and end-of-life products

The lifespan distribution parameters allow the modelling of the product age structure for mobile phones in stock and at the end of life. Similar to Section 3.2.3, the Weibull distribution fitting can be applied to define the age structure of mobile phones in active use (see Supplementary document for details). Subtracting the latter from the total stocks would indicate the age structure of mobile phones kept in storage. The results are summarised and presented in Fig. 10.

About one quarter of all mobile phones in stock were brand new (less than a year old) at the end of 2014, but these phones form close to a half (45%) of mobile phones in active use (Fig. 10). Similarly, the mobile phones less than 2 years old cover about 50% of total in-use stocks, and more than 70% of phones in active use. At the same time, relatively old phones (dated 2011 and earlier), while representing 29% in stock, account for only 13% of mobile phones in active use, but 46% of phones in storage. About 56% of EoL mobile phones in 2014 were also represented by old phones.

The visual representation of product stocks and flows (Fig. 10) can help to better understand the circulation of different EEE in the economy, informing the development of appropriate policy measures to increase the collection rates and improving the accuracy for estimating the value associated with recycling of EoL products. If statistical and empirical data allow, the analysis can be extended further for specific brands and/or models of electronic devices.

4. Conclusion

The use of empirical data in modelling the product in-use stocks and flows can help overcome inconsistency and reduce uncertainties attributed to a lack of official information sources and statistics. The key novelty of the developed methodology in this article is the combination of top-down and bottom-up approaches, based on trade statistics and consumer survey data respectively, to assess the product lifespan, in-use stocks and flows, including reconstructing the product age structure if this information is not available from primary sources. It has been successfully demonstrated with the case study of mobile phones in Australia over 1997–2014.

The total number of mobile phones in in-use stocks in Australia has been estimated at 46 million, or about 2 phones per capita, being relatively stable since 2012. The proportion of phones not being in use (kept in storage) has been constantly rising, accounting for about 50% of all phones in Australian households in 2012–2014. The generation of EoL phones has reached 12 million units, being equal to new phone sales and indicating the saturation level in the market.

The average lifespan of a mobile phone in Australia in 2005–2014, estimated by different methods, is in the range between 3.5 and 4.5 years, with a consensus estimate of 3.8 years for the last five-year period from 2010 to 2014. The estimated Weibull distribution shows that the average lifespan has been shortening over time – from about six years in late 1990s to five years in early 2000s, and then stabilised at around 3.6 years which further supports the fact that mobile phones have reached the market saturation level. The comparison with previous studies in other countries revealed that the lifespan distribution profile for mobile phones in Australia is relatively close to results reported in Japan, while the available European studies are inconsistent with each other and standing apart from our results.

Based on existing customer surveys for 2005–2014, a mobile phone's average expected lifespan includes 2.1 years (55%) of first use, 0.5 years (12%) of reuse, and 1.2 years (35%) of being in storage. The use of the Weibull function for modelling the lifespan distribution indicated a slightly different result. The average time of active use for mobile phones was estimated at about two years (which includes first use and reuse) (56%), while the storage time was about 1.6 years (44%). The estimation of stocks and flows across different product age groups showed that the mobile phones less than 2 years old cover about 70% of all phones in active use, and 50% of total in-use stocks. At the same time, relatively old phones (4+ years old) account for only 13% in active use, but 46% of kept in storage and about 56% of EoL mobile phones in 2014.

The analysis of existing consumer survey data revealed a lack of details regarding potential differences in consumers' decisions towards previous mobile phones depending on how often phone is replaced. This would help to verify the Weibull function distribution, namely higher/lower EoL products generation in the first (and second) year due to higher/lower rates of phones resale, reuse, or temporal storage. An additional question regarding how long the consumers were actually using their previous phone (apart from what happened to it) would also be helpful to verify the lifespan of a mobile phone.

Mobile phones contain a significant recovery value compared to other electronic devices, thus the consumers' hoarding behaviour in Australia means an accumulation of significant potential resources for future (metal) recovery. A better collection and recycling system would help capture this value. On the other hand, the facilitation and wider enabling of mobile phone reuse can help mitigate the shortening of the lifespan of these devices, while minimising the overall environmental impacts over the product life cycle.

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Appendix A. Supplementary data

Supplementary data related to this article can be found at http://dx.doi.org/10.1016/j.jclepro.2016.05.117.

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